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MINERAL RESOURCES
14 FEB 2001
TASMANIA

VOL 1 of 10

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

SCOPING STUDY

SUMMARY REPORT

MICROFILMED
FICHE No. 015502-13

28 January 2001

MINERAL RESOURCES
FILE REF EL28/88
14 FEB 2001
DOC. REF
OFFICER | FOR ACTION | FOR INFO
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01_4526

Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL*; Newnham Exploration and Mini EL28/1988
Newnham, L.A.

VOL 1 of 10

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PREFACE

In order to better understand the commercial potential and financial demands of the Avebury nickel sulfide deposit discovery, Allegiance Mining NL decided to complete a scoping study on the project.

An appropriate team of professionals was assembled to undertake this task.

It is important to recognise the limitations of a scoping study, which is adequately defined in *"Cost Estimation Handbook for the Australian Mining Industry"* (Pub AIMM) as follows:

> "After initial identification of a potential resource, scoping or conceptual studies are undertaken to identify:
>
> - general features and order of magnitude parameters of the project
> - technical issues requiring further investigation or testwork
> - the costs and time required to undertake further pre-development
>
> Scoping or conceptual studies are not sufficiently accurate to carry out a meaningful assessment of the economic viability of any project, only whether, and how much, further pre-development efforts are warranted". (p 252)

This Scoping Study does not seek to quantify the commercial viability of the conceptualised operation at Avebury. It is not intended for use in a prospectus and should not be published in full or in part without the written permission of the authors.

1. SUMMARY

Drilling programs by Allegiance Mining NL have discovered a nickel sulfide resource at Avebury near Zeehan in western Tasmania.

The identified resource, which occurs in two deposits, is estimated as:

Deposit	Resource Category	Tonnes	% Ni
North Avebury	Indicated	990,000	1.7
South Avebury	Inferred	2,110,000	1.7
TOTAL		**3,100,000**	**1.7**

Technical studies completed at a scoping study level suggest this resource is capable of supporting a medium sized operation with the following statistics:

Annual throughput	:	300,000 tpa
In-situ resource grade	:	1.7% Ni
Mill head grade	:	1.6% Ni
Mill recovery	:	88.9%
Nickel recovered	:	4,270 tpa
Concentrate grade	:	22% Ni
Concentrate production	:	19,400 dry tpa
Shipped concentrate	:	21,000 wet tpa
Life-of-mine	:	10 years

Mining would be an underground combination of bench and open stoping accessed by a single decline.

Nickel would be recovered in a conventional sulfide flotation mill.

Concentrates would be road and rail freighted to Burnie for shipment to a smelter.

Infrastructure would be developed on a flat, relatively open area adjacent to the deposit and the workforce based in Zeehan.

The operation would employ 80-90 people during both construction and operation.

This scoping study estimates total operating costs as $60/tonne ore = $18M pa, averaged over the life-of-mine.

Capital costs are estimated as $32.2M prior to production, and a further $3.7M during operations.

The high nickel grade of the concentrates makes them a premium product, and a competitive sales agreement could be anticipated.

Sales revenue is strongly affected by metal price, currency exchange rates, smelting costs, royalties and other ex-mine deductibles.

At a nickel price of US$7,000/tonne and an exchange rate of A$ = US$0.55, net sales revenue is estimated as A$35.8M pa.

At a nickel price of US$ 6,750/tonne and an exchange rate of A$ = US$0.60, net sales revenue is estimated as A$31.12M pa.

From this net sales estimate, mine operating, capital and taxation and depreciation costs must be deducted.

Additional technical and financial studies are required to complete a full/final feasibility study on which a development decision could be based.

Such a feasibility study is estimated to take one year to complete, at a cost of A$3.2M.

Upon completion of a favourable feasibility study, a permitting, construction and mine development period of two years is required.

Substantial potential exists in the Avebury area to identify additional nickel sulfide resources at a number of prospects. Any such discoveries would benefit development of Avebury in terms of both reducing costs by increasing annual throughput and extending mine life.

A two-year exploration program costing A$0.9M pa is required to evaluate these prospects in a timely fashion with the Avebury evaluation program.

2. CONTRIBUTING REPORTS

This Scoping Study consists of a series of technical reports prepared by appropriate and well-qualified people. The contributing reports are listed below and the authors' bibliographies appear as Appendix A.

Identified Mineral Resource Estimate:

"Allegiance Mining NL. Avebury Nickel Project Mineral Resource Report, November 2000" prepared by Michael V McKeown of McKeown Mining Pty Limited

Mineral Resource Potential Assessment:

"Allegiance Mining NL. Avebury Nickel Project Scoping Study. Mineral Resource Potential Report" by Lindsay A Newnham, Newnham Exploration & Mining Services, January 2001

Mining:

"Avebury Prospect Mine Scoping Study, December 2000" prepared by AJ Bruce Consulting Pty Limited

Milling:

(a) *"Avebury Nickel Project Metallurgical Scoping Study for Allegiance Mining NL"* by GP Sheldon of Metcon Laboratories, December 2000

(b) *"Avebury Nickel Project Metallurgical Scoping Study for Allegiance Mining NL"* by John E Butler, December 2000

Infrastructure:

"Allegiance Mining NL. Avebury Nickel Project Scoping Study Infrastructure Development and Support Services" by Lindsay Newnham, Newnham Exploration & Mining Services, January 2001

Environment:

"Allegiance Mining NL. Avebury Nickel Project Scoping Study - Environment" by NSR Environmental Consultants Pty Limited, December 2000

Marketing:

(a) *"Avebury Nickel Project - Marketing Scoping Study. Report for Allegiance Mining NL, November 2000"* by John O'Shea and Associates

(b) *"Avebury Nickel Project Marketing Scoping Study for Allegiance Mining NL"* by John E Butler, December 2000

3. INTRODUCTION

Allegiance Mining NL has discovered a nickel sulfide deposit, called Avebury, near Zeehan in western Tasmania.

Tenements covering the deposit and adjacent prospective ground are beneficially held 100% by Allegiance, with Rio Tinto Exploration Pty Limited holding a 2% net smelter royalty over future production.

To date, $3.5M has been spent on evaluation of Avebury.

To better define both the commercial potential of this discovery and the requirements necessary to further evaluate the project, Allegiance decided to complete a scoping study which had four principal objectives:

(i) conceptualise a style of operation which might be developed on the currently identified mineral resources

(ii) estimate capital and operating costs attributable to such an operation

(iii) identify requirements necessary to advance the project to completion of a full feasibility study

(iv) estimate the cost of completing a full feasibility study

An appropriate team of professionals was assembled to develop the study.

This summary report reviews and draws together the contributing reports of this team as the basis for addressing the study objectives.

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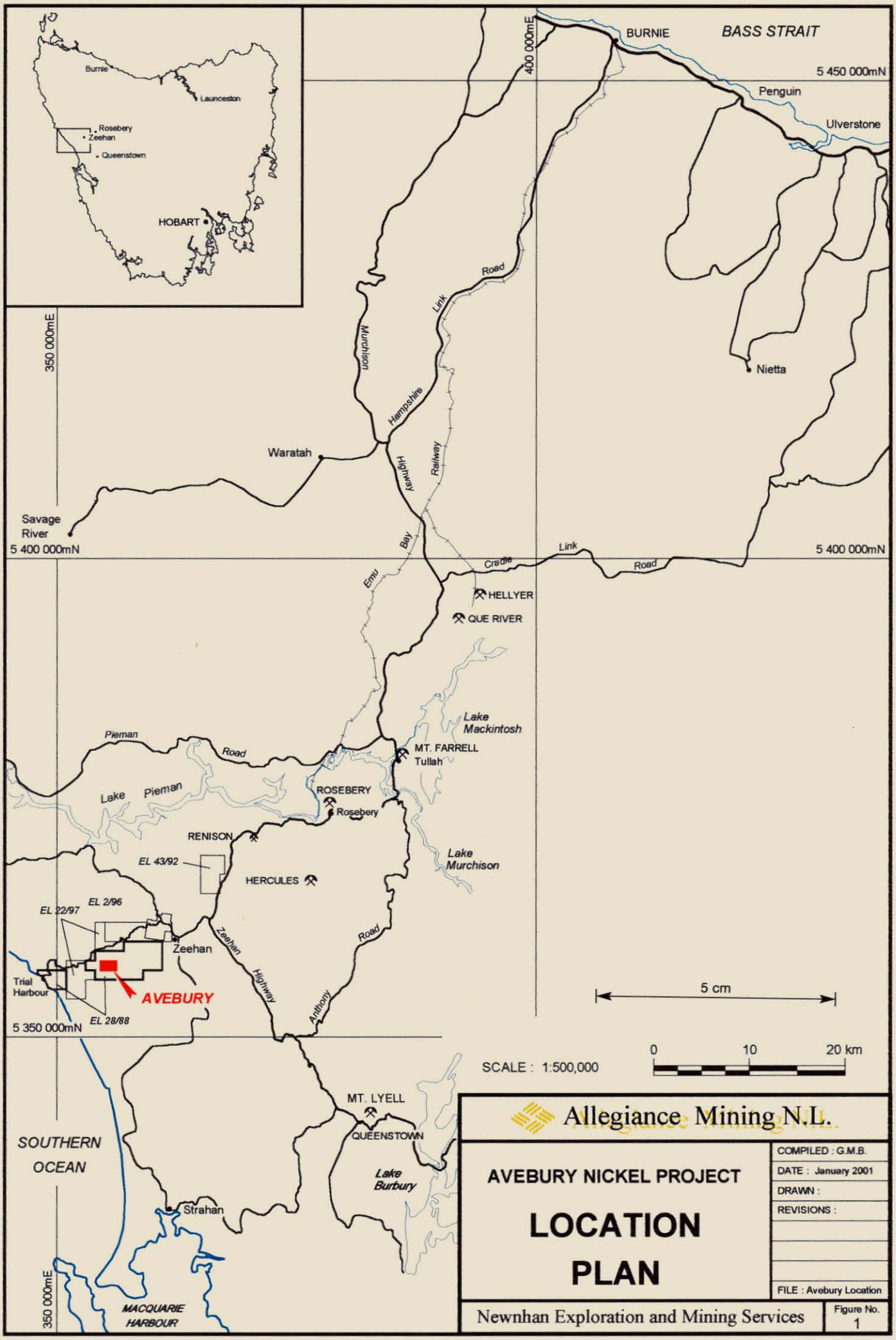

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

LOCATION PLAN

COMPILED : G.M.B.	
DATE : January 2001	
DRAWN :	
REVISIONS :	
FILE : Avebury Location	

Newnhan Exploration and Mining Services

Figure No. 1

4. MINERAL RESOURCES

Mineral resources for the Avebury project are reviewed in two groups:

- **identified resources** as defined by the JORC code
- **potential resources**

The technical components of the scoping study are based solely on the identified resources.

The potential resources are reviewed purely as an "up-side factor" which could have a positive impact on any operation established on the identified resources.

4.1 Identified Resources:

The Avebury identified resources were estimated by Michael V McKeown in accordance with the *"Australasian Code for Reporting of Mineral Resources and Ore Reserves (The JORC Code)"*.

Details of this estimation are presented in the attached report:

> *"Allegiance Mining NL. Avebury Nickel Project Mineral Resource Report, November 2000"* by Michael V McKeown of McKeown Mining Pty Ltd

This identified resource at a 0.75% Ni cut off grade is estimated as:

Deposit	Resource Category	Tonnes	% Ni
North Avebury	Indicated	990,000	1.74
South Avebury	Inferred	2,110,000	1.73
Total		3,100,000	1.73

The estimate is based on the results of 26 cored drill holes.

The resource lies in two adjacent steeply dipping deposits, **North Avebury and South Avebury,** developed along the margins of a serpentinised mafic-ultramafic body within a sequence of Cambrian sediments, themselves intensely altered by the intrusion of a Devonian granite.

North Avebury strikes east-west and dips 70-80° to the north. It is currently defined over a strike length of 300 m, a down-dip length of 300 m, and varies in width up to 30 m. Top of the main mineralised zone is 70 m below surface. It is terminated to the west by a fault, to the east by a grade boundary and is open at depth.

South Avebury strikes east-west and dips north at 50-60°. It is currently defined over a strike length of 150 m, and a down-dip length of 140 m. Drilling completed since the resource report was written suggests the mineralisation top is approximately 75 m below surface, and it may have a steep westerly plunge with a sharp grade boundary to the east, open to the west and down plunge. Mineralisation in South Avebury is currently interpreted as occurring in two lenses - one on the northern margin of the serpentinite host rock and one on the southern margin.

Mineralisation in both lenses is dominated by a coarse pentlandite-magnetite association with only minor pyrrhotite, pyrite and nickel arsenides.

The resource estimate was based on a cut-off grade of 0.75% Ni which was selected on the basis of anecdotal mining costs in the region, anticipated operating costs and market factors.

McKeown points out in his report that the contacts between sediments and mineralised ultramafics (hangingwall contacts) are generally sharp and represent the start of high grade mineralisation. The footwall, or internal contacts of the mineralisation are less well-defined and, in many instances, there are very thick intersections of lower grade mineralisation which have not been included in the resource estimate above.

McKeown further highlights the necessity for substantial further drilling programs to elevate the identified resource to higher categories of resource and reserve status, necessary for a full feasibility study. Details of this requirement are presented in Section 11 of this report.

4.2 Potential Resources:

This scoping study deals with a conceptualised operation based on the identified resources above.

However, exploration programs completed on other Allegiance tenements have highlighted the potential to identify further resources.

Confirmation by way of further exploration of any such additional resources could have a fundamental impact on the development of Avebury, in terms of life-of-mine, annual production, and mine development scheduling.

This additional resource potential is detailed in the attached report:

> *"Allegiance Mining NL, Avebury Nickel Project Scoping Study, Mineral Resource Potential Report"* by Lindsay A Newnham, Newnham Exploration & Mining Services, January 2001

A number of potential resource opportunities have been identified and each assigned a prospectivity rating:

Prospect	Prospectivity Rating
South Avebury Resource Extension	Very high
Avebury district:	
- East Avebury	Very high
- Bismark	Very high
- Bison	High
- Pontiac	Moderate
- West Avebury	Moderate
Trial Harbour	Very high
Melba Flats	Very high

The locations of the prospect areas relative to Avebury are shown on Fig 5. Work programs, budgets and schedules required to further evaluate these areas are detailed in the above report, and summarised in Section 11 of this report.

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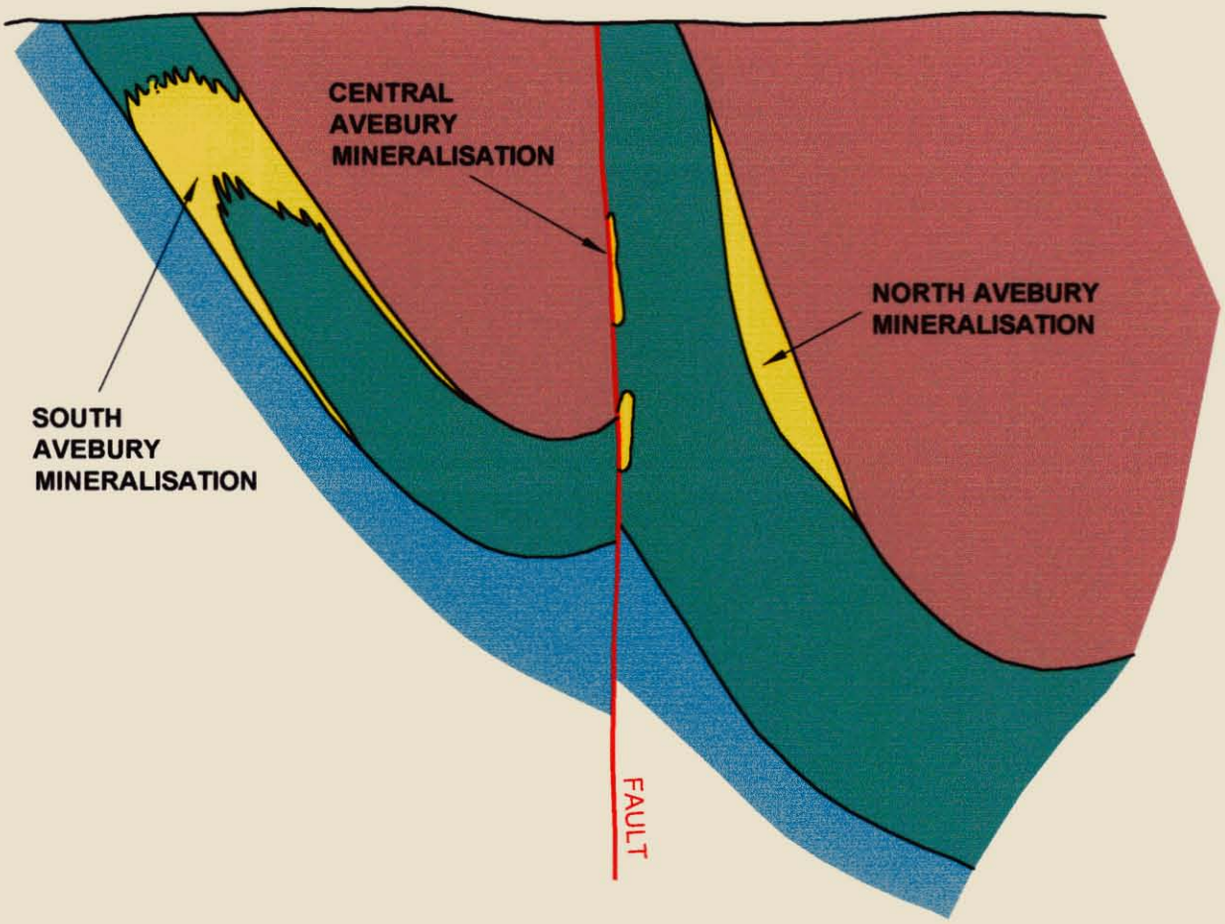


LEGEND

- Nickel mineralisation
- Serpentinite
- Hornfels and chert
- Chert, conglomerate, very altered rocks
- Fault

5 cm

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

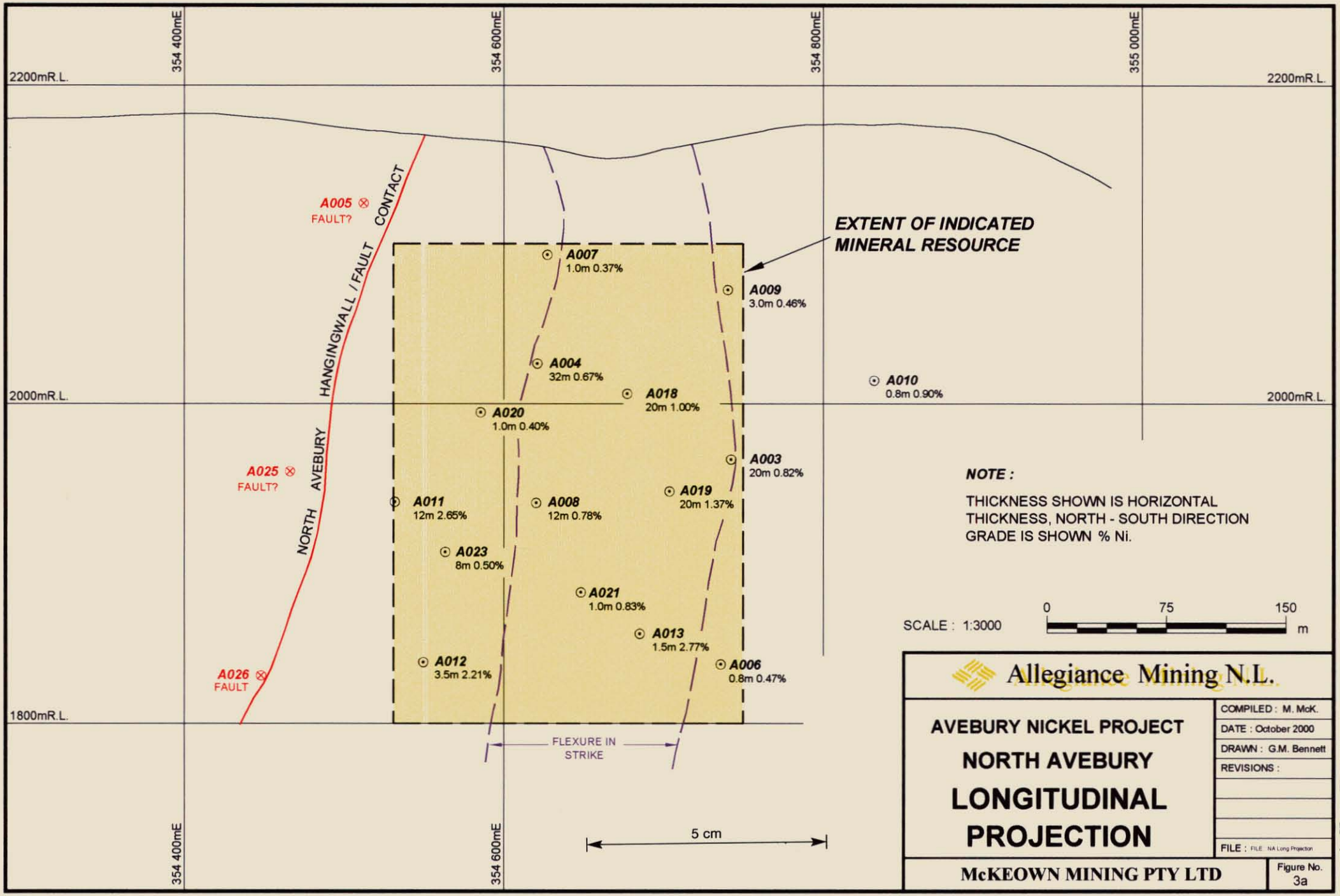
GEOLOGICAL MODEL

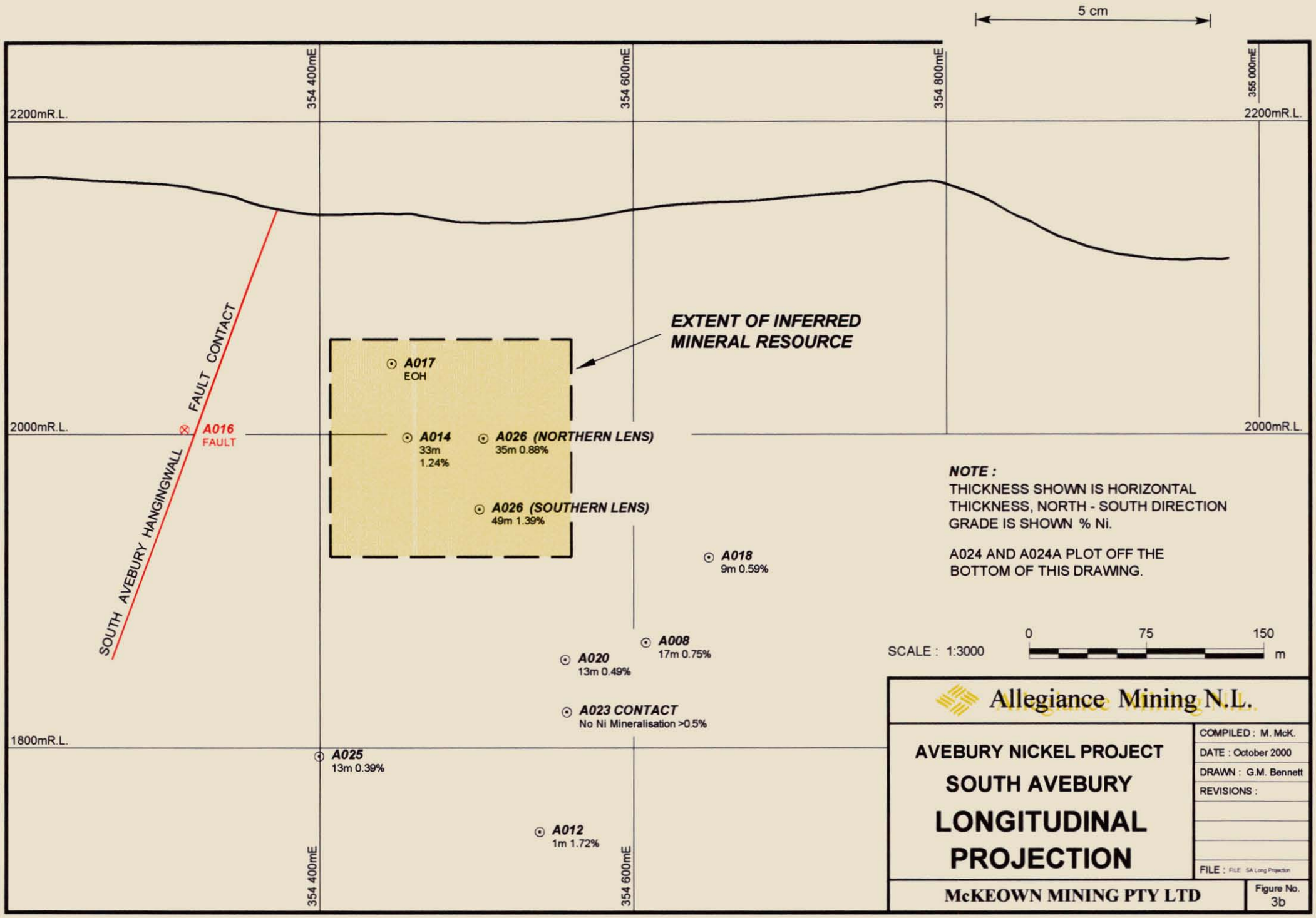
SKETCH CROSS SECTION

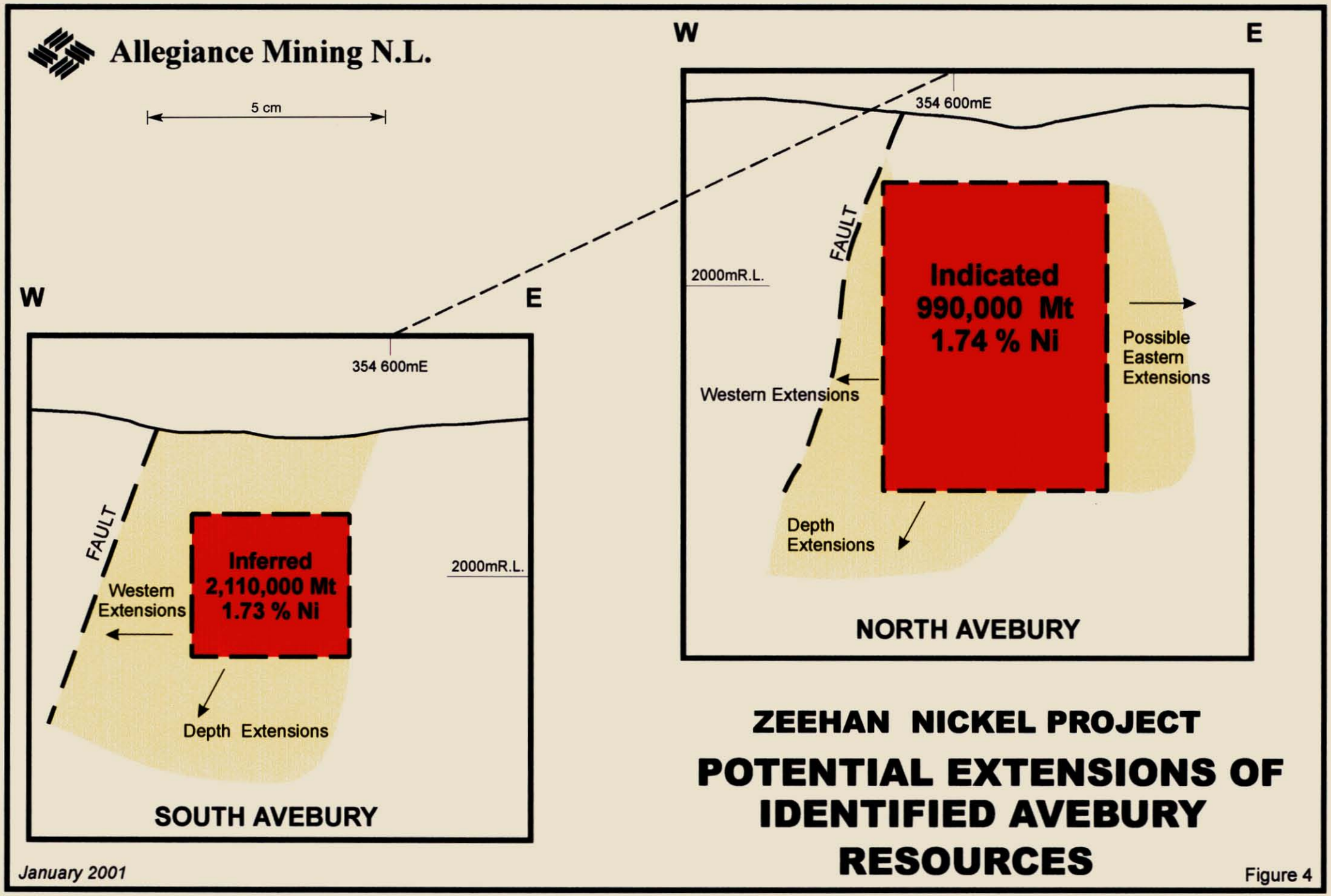
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : Geo Model SXS

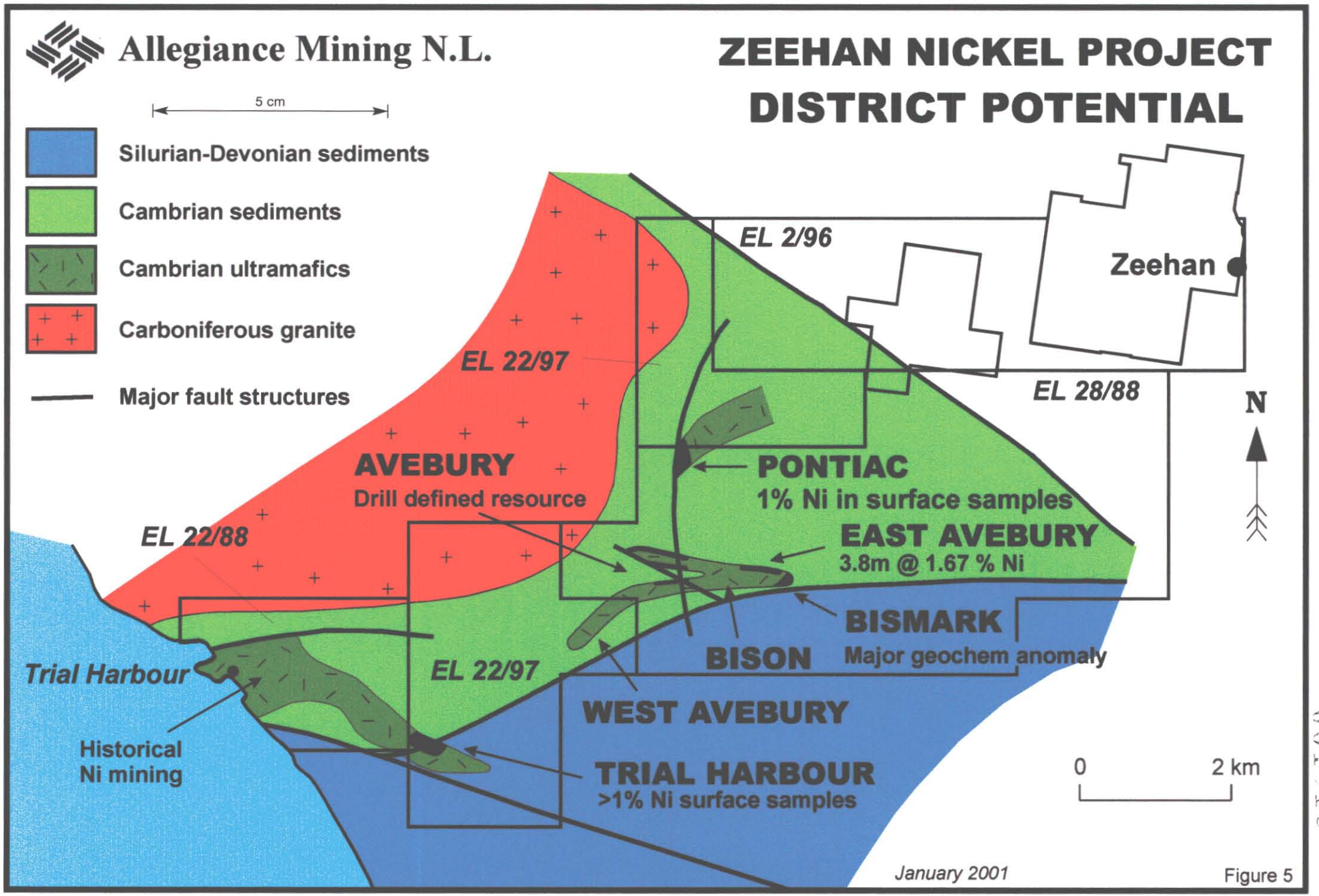
McKEOWN MINING PTY LTD

Figure No. 2


354 400mE
354 600mE
354 800mE
355 000mE
2200mR.L.
2000mR.L.
1800mR.L.
A005
FAULT?
CONTACT
HANGINGWALL / FAULT
NORTH AVEBURY
A025
FAULT?
A026
FAULT
EXTENT OF INDICATED MINERAL RESOURCE
A007
1.0m 0.37%
A009
3.0m 0.46%
A004
32m 0.67%
A018
20m 1.00%
A010
0.8m 0.90%
A020
1.0m 0.40%
A003
20m 0.82%
A011
12m 2.65%
A008
12m 0.78%
A019
20m 1.37%
A023
8m 0.50%
A021
1.0m 0.83%
A013
1.5m 2.77%
A012
3.5m 2.21%
A006
0.8m 0.47%
FLEXURE IN STRIKE
5 cm
NOTE :
THICKNESS SHOWN IS HORIZONTAL THICKNESS, NORTH - SOUTH DIRECTION
GRADE IS SHOWN % Ni.
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
NORTH AVEBURY
LONGITUDINAL PROJECTION
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : FILE : NA Long Projection
McKEOWN MINING PTY LTD
Figure No. 3a


5 cm
354 400mE
354 600mE
354 800mE
355 000mE
2200mR.L.
2000mR.L.
1800mR.L.
EXTENT OF INFERRED MINERAL RESOURCE
A017
EOH
A014
33m
1.24%
A026 (NORTHERN LENS)
35m 0.88%
A026 (SOUTHERN LENS)
49m 1.39%
A016
FAULT
FAULT CONTACT
SOUTH AVEBURY HANGINGWALL
A018
9m 0.59%
A008
17m 0.75%
A020
13m 0.49%
A023 CONTACT
No Ni Mineralisation >0.5%
A025
13m 0.39%
A012
1m 1.72%
NOTE :
THICKNESS SHOWN IS HORIZONTAL THICKNESS, NORTH - SOUTH DIRECTION
GRADE IS SHOWN % Ni.
A024 AND A024A PLOT OFF THE BOTTOM OF THIS DRAWING.
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
SOUTH AVEBURY
LONGITUDINAL PROJECTION
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : FILE SA Long Projection
McKEOWN MINING PTY LTD
Figure No. 3b


Allegiance Mining N.L.
5 cm
W
E
354 600mE
FAULT
2000mR.L.
Indicated
990,000 Mt
1.74 % Ni
Possible
Eastern
Extensions
Western Extensions
Depth
Extensions
NORTH AVEBURY
W
E
354 600mE
FAULT
Inferred
2,110,000 Mt
1.73 % Ni
2000mR.L.
Western
Extensions
Depth Extensions
SOUTH AVEBURY
ZEEHAN NICKEL PROJECT
POTENTIAL EXTENSIONS OF IDENTIFIED AVEBURY RESOURCES
January 2001
Figure 4


Allegiance Mining N.L.
ZEEHAN NICKEL PROJECT
DISTRICT POTENTIAL
5 cm
Silurian-Devonian sediments
Cambrian sediments
Cambrian ultramafics
Carboniferous granite
Major fault structures
EL 2/96
Zeehan
EL 28/88
N
EL 22/97
AVEBURY
Drill defined resource
PONTIAC
1% Ni in surface samples
EAST AVEBURY
3.8m @ 1.67 % Ni
EL 22/88
BISMARK
Major geochem anomaly
BISON
EL 22/97
Trial Harbour
WEST AVEBURY
Historical
Ni mining
TRIAL HARBOUR
>1% Ni surface samples
0
2 km
January 2001
Figure 5

5. CONCEPTUALISED OPERATION

Based on the identified mineral resource of 3.1 Mt 1.7% Ni, a conceptualised mining plan and milling operation were developed along with an infrastructure scheme to support the project.

This section reviews this development concept.

5.1 Mine Plan:

Details of the conceptualised mine plan are contained in the accompanying report:

> *"Avebury Prospect Mine Scoping Study December 2000"* by AJ Bruce Consulting Pty Ltd

This report addresses three objectives:

(a) review various underground and open-cut mining options

(b) establish indicative capital and operating costs

(c) recommend and cost further work necessary for a full feasibility study

Main conclusions of the report are:

(i) The identified resource is capable of supporting a 300,000 tpa operation over a 10-year mine life.

(ii) Underground mining of both the North and South Avebury deposits is recommended. Open-cut mining of South Avebury is possible but rejected for high stripping ratio and environmental reasons.

(iii) The preferred mining method for North Avebury is benching, and for South Avebury a modified open stoping method with cemented fill.

(iv) The two resource blocks would be accessed by a common decline (Fig 6).

(v) Because the resource blocks are relatively deep, first production would not be achieved until Year 2 of the development schedule, and full production would be achieved in Year 3; ie, there is approximately 18 months

of development prior to ore production.

(vi) The mine could be run on an owner-operator basis, or a contract basis with owner supervision and management. The cost differential is not great.

(vii) $11.6M capital is required prior to full production for mine development and associated facilities. A further $1.7M capital is required during the life-of-mine for further capital development and equipment.

(viii) Operating costs will vary during the life-of-mine and will reflect the balance between open stoping and bench stoping, haulage depth, etc. Overall, the operating cost will be approximately $31-32/t ore.

(ix) A significant factor which will affect the operating cost is ground support requirements. Initial work on drill cores suggests highly variable ground conditions, with the mineralised zones being quite weak.

AJ Bruce Consulting identified three principal requirements to advance the project to a full feasibility study mine design stage:

- acquire high quality geotechnical and hydrological data
- detailed evaluation and integration of this data into the final mine design
- more resource definition data

These requirements would be addressed in future drilling programs.

The cost of acquiring the recommended geotechnical and hydrological data, and completing a mine feasibility study is estimated by AJ Bruce at $166,000. This does not include the cost of specialised drilling, which is allowed for in the resource definition budget.

With additional site costs and contingencies, a budget to complete the feasibility study of $0.2M is recommended.

5.2 Metallurgical Testing and Mill Design:

Metallurgical test work was completed on three intersections of whole drill core - two from North Avebury and one from South Avebury. Data from this work was used to design a flow sheet for a conceptualised milling operation.

The metallurgical test work was undertaken by Metcon Laboratories in Sydney, and their results are presented in the accompanying report:

> *"Avebury Nickel Project Metallurgical Scoping Study for Allegiance Mining"* Report 00900 by GP Sheldon, December 2000

The work was supervised by consulting metallurgist John E Butler who then developed the initial flow sheet and costed the conceptualised mill. His work is presented in the accompanying report:

> *"Avebury Nickel Project Metallurgical Scoping Study for Allegiance Mining NL"* by John E Butler, December 2000

The Metcon testwork indicated the mineralisation was amenable to concentration by conventional sulfide flotation methods.

High nickel recoveries into high grade concentrates were achieved.

In developing a simplified flow sheet (Fig 7), Butler assumed a throughput of 300,000 tpa 1.6% Ni, with recoveries of 88.9% into 22% nickel concentrates for an annual output of 4,270 t Ni in 19,400 dry tonnes of concentrate.

Butler estimates a mill developed in accordance with this flow sheet would cost approximately $15.25M to construct on a prepared site. Operating costs would be $19/t.

Tailings would be alkaline because of the low sulfide content and would be stored in an adjacent tailings dam. Butler's report provides $3M for this dam. Staged construction should be possible with an estimated pre-production Stage 1 expenditure of $1M and later stages costing $2M during the life of operations.

The Metcon testwork indicates the only possible deleterious element in the concentrates is arsenic. However, Butler believes that the levels are controllable and may be reduced with process refinements developed from further testwork.

Two items in the process flow sheet have queries attached; viz, ball mill and tailings thickener. They have been included at this preliminary stage but further testwork may demonstrate cost savings in these areas.

Additional test work will be required to complete a detailed flow sheet for a feasibility study (section 11.3 below), and $0.5M is recommended for this work.

5.3 Infrastructure Development:

The proposed Avebury development lies only seven kilometres west of Zeehan, which is an established mining centre with under-utilised services and infrastructure.

Infrastructure issues and costs are addressed in the accompanying report:

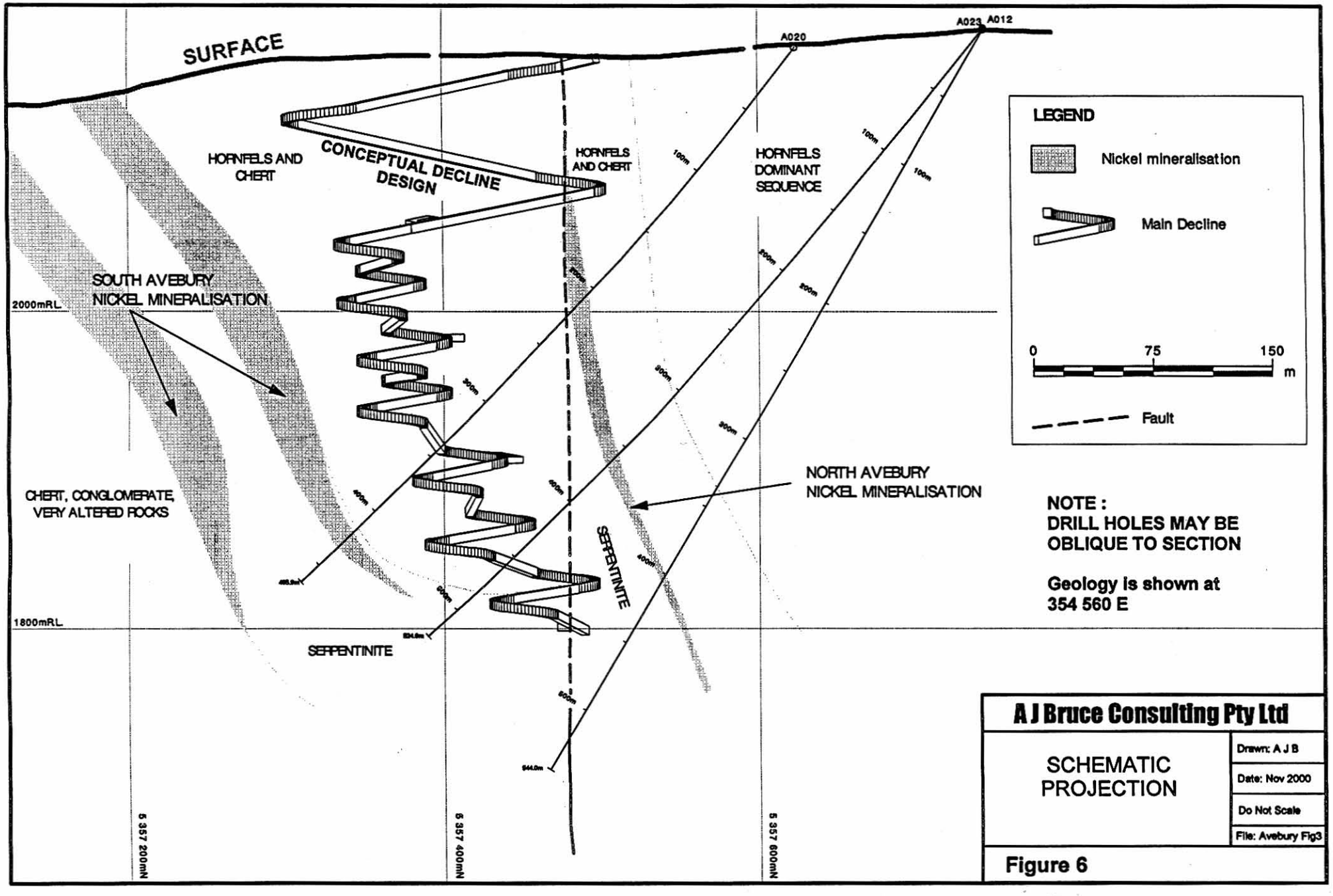
> *"Allegiance Mining NL. Avebury Nickel Project Scoping Study Infrastructure Development and Support Services, January 2001"* by Lindsay A Newnham, Newnham Exploration & Mining Services

It is envisaged that the workforce would be based in Zeehan and access the development site via an upgraded section of the Trial Harbour road.

All site facilities, mill and tailings dam would be developed in a flat, relatively open area south of the Trial Harbour road.

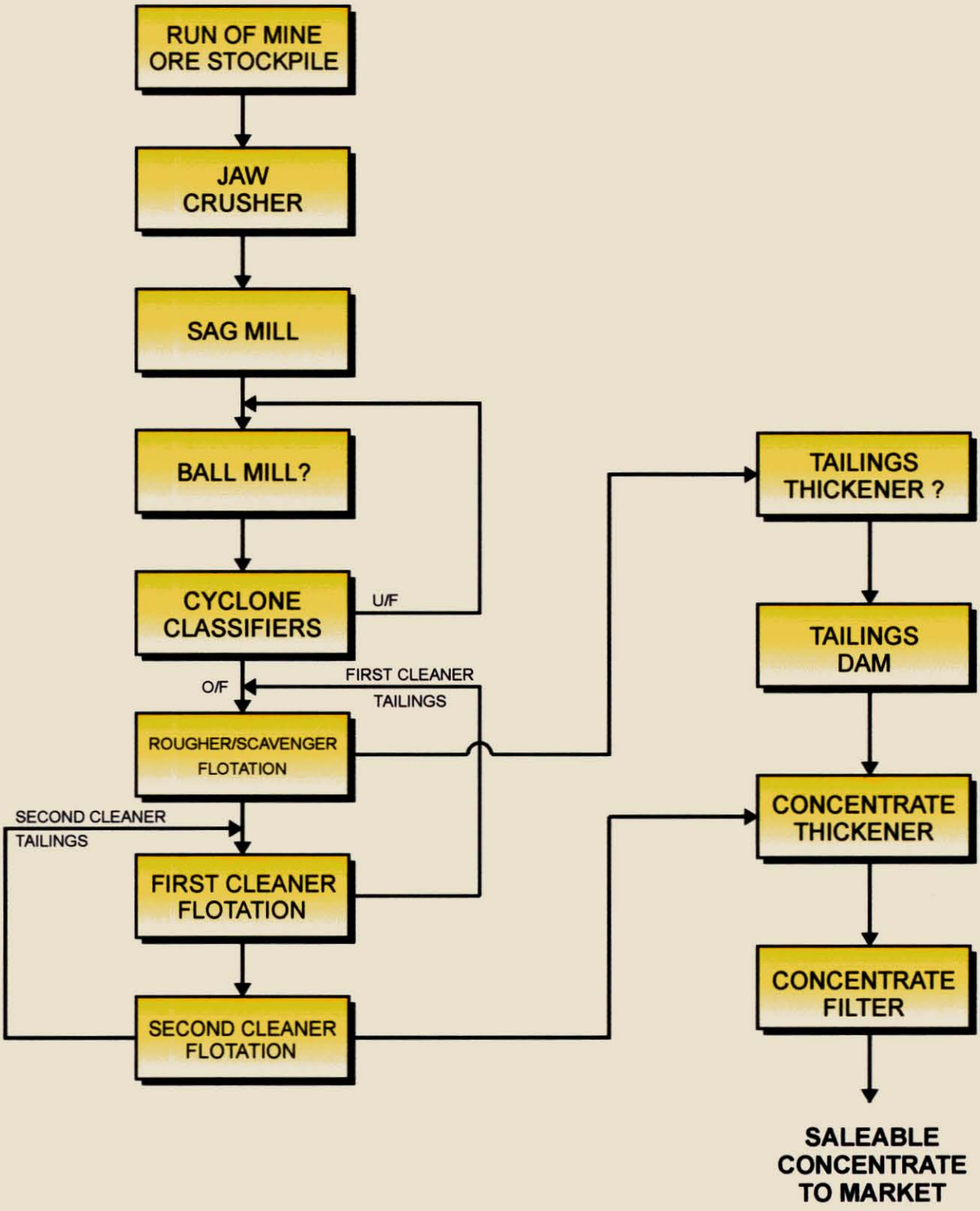
Power would be provided from the State grid and require construction of a new line from Zeehan to an Avebury sub-station.

No negative infrastructure issues were identified. Development costs were estimated at $4.3M.


SURFACE
A023
A012
A020
HORNFELS AND CHERT
CONCEPTUAL DECLINE DESIGN
HORNFELS AND CHERT
HORNFELS DOMINANT SEQUENCE
SOUTH AVEBURY NICKEL MINERALISATION
2000mRL
1800mRL
CHERT, CONGLOMERATE, VERY ALTERED ROCKS
SERPENTINITE
SERPENTINITE
NORTH AVEBURY NICKEL MINERALISATION
LEGEND
Nickel mineralisation
Main Decline
0
75
150
m
Fault
NOTE :
DRILL HOLES MAY BE OBLIQUE TO SECTION
Geology is shown at 354 560 E
5 357 200mN
5 357 400mN
5 357 600mN
A J Bruce Consulting Pty Ltd
SCHEMATIC PROJECTION
Drawn: A J B
Date: Nov 2000
Do Not Scale
File: Avebury Fig3
Figure 6
5 cm

66523

661024



KEY

O/F Overflow

U/F Underflow

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

SIMPLIFIED PROCESS FLOWSHEET

Compiled :
Date : January 2000
Drawn : G.M.Bennett
Revisions :
File :

John E. Butler - Consultant Metallurgist

Figure No. 7

661025

Water Supply

Trial Harbour Road

Kynance

Contiguous Creek

Coleman Creek

Tailings Dam

Access Road

Creek

Comstock Creek

Settling Dam

Hemp Creek

Water Tanks

Sub Station

Mill

Offices

Stores

ROM

Wash

Comp

Fuel

Magazines

East Avebury Track

Portal

Pad

NORTH AVEBURY

West Avebury Track

SOUTH AVEBURY

5 cm

SCALE : 1:10,000

0 250 500 m

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

SCOPING STUDY

SITE LAYOUT

Compiled : L.A.N.

Date : December 2000

Drawn : G.M.Bennett

Revisions :

File : Infrastructure Base

Newnhan Exploration and Mining Services

Figure No. 8

6. ENVIRONMENTAL MANAGEMENT

The Avebury deposit occurs in a moderately rugged, forested area that previously has been variably logged and burnt by bushfires. However, it has no previous mining history and, hence, no substantive surface disturbance.

The opportunity therefore exists for the deposit to be developed with high environmental standards, largely unencumbered by previous activities.

NSR Environmental Consultants Pty Limited was engaged for the purposes of this scoping study to advise on a range of environmental management matters associated with Avebury development:

- quantify the environmental setting of the deposit
- identify environmental issues associated with the conceptualised development
- advise on the approval and permitting process attached to the development
- recommend and cost future environmental work programs necessary to advance the project to a full feasibility study stage and into an operational phase

The accompanying NSR report on these matters is titled:

> *"Allegiance Mining NL, Avebury Nickel Project, Scoping Study - Environment"* by NSR Environmental Consultants Pty Limited, December 2000

This report identifies the key environmental issues as:

- land alienation
- water quality
- impact of adjacent mine operations
- public opinion
- waste management
- socio-economic effects

- floral impacts
- rehabilitation
- noise and air impacts
- fibrous rock factors
- archaeological/cultural factors

If appropriately managed, none of these issues were identified as substantive impediments to the project. Given the nature of the proposed operation, the environmental matters requiring greatest attention will probably be water and waste management.

The development and permitting process identified by NSR is transparent and standard, and no difficulties are foreseen, provided the prescribed sequencing of the process is closely followed.

NSR recommended a range of necessary short and long lead time environmental programs which should be implemented. Most of these are scheduled to commence following a decision to advance the project to full feasibility study stage. However, an almost immediate start on a water quality monitoring program was recommended.

NSR estimated the cost of preparing and supporting a DPEMP as $0.4M. This is a capital cost necessary for completion of a full feasibility study.

They estimated an environmental operating budget of $0.60/t ore ($180,000) for employment of a permanent environmental and OH & S officer, support services and consultative inputs.

7. CONCENTRATES TRANSPORT

The conceptualised operation will produce 19,000 dry tonnes of concentrate, averaging 22% Ni and containing 4,270 t Ni pa for a 10-year mine life.

On the basis of an 8% moisture content, this represents approximately 21,000 wet tonnes concentrate per annum.

The study assumes the following:

(a) concentrate will be sold to a buyer either overseas or on mainland Australia

(b) concentrate will be shipped from Burnie on a FOB Burnie basis

(c) the concentrate buyer will organise and pay all freight and insurance costs ex-Burnie

Options for transporting concentrate from Avebury and placing in a ship at Burnie were detailed in the accompanying report:

> *"Allegiance Mining NL. Avebury Nickel Project Scoping Study. Infrastructure Development and Support Services"* by Lindsay Newnham, Newnham Exploration & Mining Services, January 2001

The preferred option is to contract the total operation to Tasrail who would take the concentrate from a loading facility at Avebury and place in a bulk cargo ship at Burnie for an indicative price of $25/wet tonne.

The annualised transport cost is $525,000, plus insurance of $25,000 = $550,000.

This is treated as a lump sum ex-mine cost because it is a deductible for royalty calculation purposes.

The risk in this proposal is that the rail operator owns the bulk concentrate sheds at Melba Flats and Burnie and the ship loader at Burnie.

To retain a cost competitive and efficient service, a full feasibility study should identify an cost alternative transport systems.

This study assumes bulk shipment of concentrates. It is the most cost-effective method but has two potential disadvantages:

- contamination of concentrates
- infrequent shipments, possibly quarterly

The contamination factor can be addressed through appropriate quality control systems.

The infrequent shipment factor can produce cash-flow problems and may be addressed through negotiation with the buyer in terms of partial advance payments based on seller assays.

These factors would be addressed in detail in a full feasibility study.

8. PERSONNEL

The Avebury project lies within the major western Tasmanian mining district. As such, there is a wide range of appropriate skills available within the local population to construct and operate the conceptualised Avebury project.

Some specialised personnel, particularly at the senior management level, may have to be recruited to the area.

The operation is viewed as a mixed "owner-contractor" workforce.

The owner/operator would provide approximately 51 key personnel as listed below:

Mine	18
Mill	21
Geology	4
Environment	2
Administration	6
Total:	51

Contractors would provide personnel for:

- mining operations
- concentrate transport
- miscellaneous services

The number of full time contract personnel would be approximately 30-40.

The total workforce would therefore be approximately 80-90.

Scope exists in both the mine and mill to reduce the number of owner-operator employees to 43 by contracting duties such as surveying, tradesmen.

Further reductions could be achieved if decisions were made to largely run the mine and mill with a contract workforce. Workforce operating costs (salaries, wages, benefits, etc) are provided for in the sectional operating budgets.

This study assumes that the workforce would be accommodated in Zeehan where there are adequate existing facilities. No additional housing would be required and, thus, there is no capital provision for personnel accommodation.

Housing rental assistance for key personnel and workforce transport to the mine site are provided for in the administration operating cost.

9. COSTS

Cost estimates for the conceptualised operation described above are divided into **capital costs** required to develop the operation and **operating costs** required to run the operation.

At a scoping study level, these estimates are broadly ± 30% and are based on industry standards and general experience, rather than definitive specifications and quotations.

Potential exists to reduce both the capital and operating costs estimates. The latter would be most affected by any substantial upsizing of the operation. Capital costs would be most affected by the "style of operation".

9.1 Capital Costs:

Capital cost can be subdivided into:

- pre start-up capital - expenditure necessary to commence operations
- post start-up capital - capital expenditure required during the life-of-mine

Item	Pre start-up ($'M)	Post start-up ($'M)
Infrastructure	4.3	-
Mine	11.6	1.7
Mill	16.3	2.0
Totals	**32.2**	**3.7**

Avebury Nickel Project - Capital Cost Estimates

Infrastructure capital provides for adequate road, power, serviced site and water management systems.

There is only limited potential to reduce this estimate. Estimate over-runs would occur if work was not scheduled for summer periods.

The estimate includes provision for an environmental bond and construction management.

Mine capital is estimated on a contract mining basis with owner supervision/management. It assumes purchase of new mobile and fixed plan equipment. The pre start-up estimate represents all mine costs required to get to full production; viz, Years 0, 1 and part of 2 in the mine schedule.

Post start-up capital is required in Year 3 for further equipment and mine development.

Potential savings could be effected both by the purchase of some second-hand equipment and a lean approach to mine infrastructure development.

Mill pre start-up capital includes the full cost of constructing the mill and the first stage of the tailings dam.

Post start-up capital is required for later stages of tailings dam development.

The estimate includes provision for engineering, procurement and construction management.

Capital savings could be effected principally in two ways. Firstly, if appropriate secondhand, rather than new, equipment was used. Secondly, if further testwork indicated the ball mill and/or the tailings thickener were not required.

9.2 Operating Costs:

The following operating costs are based on information derived from the attached reports:

Item	$/tonne ore
Mine	31.40
Mill	19.00
Geology	3.70
Environment	0.60
Infrastructure	0.80
Administration	4.50
Total	**60.00**

Avebury Nickel Project - Operating Cost Summary

This estimate is for **site operating costs** only, and does not include ex-mine costs such as concentrate transport, smelter and royalty costs. Ex-mine costs are accounted for as revenue deductibles and are quantified in the following section ("Revenue").

The **mine** operating cost assumes contract mining and varies during the life-of-mine. The average figure (above) is derived by dividing the total estimated life-of-mine operating expenditure by the total tonnage mined.

The estimate will vary depending on prevailing labour conditions, stoping methods selected, ground conditions, etc. Average mine operating cost is most affected by mine throughput.

The **mill** operating cost is most affected by the nature of the ore and the annual throughput.

The **geology** estimate provides for grade control management and ore definition services.

Environmental cost provides for one full time environmental officer who could also address OH & S management.

Infrastructure provision is for upgrade and maintenance of roads and the site.

Administration includes provision for an Operations Manager, site administrative staff, public relations functions, administrative vehicles, etc. The cost estimate is determined as 8% of total non-administrative operating costs which is an industry standard for a lean, medium sized operation.

The factor which can most affect the operating cost estimate is throughput.

9.3 Cost Summary:

Pre start-up capital	$32.2M
Post start-up capital	3.7M
Site operating costs	$60/t ore = $18M pa

10. MARKETING and REVENUES

The conceptualised Avebury operation indicates the following production:

Throughput	300,000 tpa
Mill head grade	1.6% Ni
Mill recovery	88.9%
Nickel recovered	4,270 tpa
Concentrate grade	22% Ni
Concentrate production	19,400 dry tonnes pa
Concentrate moisture	8%
Shipped concentrate	21,000 wet tonnes pa
Life-of-mine	10 years

This study assumes the concentrate would be transported to Burnie and shipped by a buyer either internationally or to mainland Australia.

The value of the metal in the concentrates would be reduced by a number of ex-mine deductable costs.

10.1 Marketing and Sales Options:

A brief review of the nickel and nickel sulfide concentrates market are presented in the accompanying report:

> *"Avebury Nickel Project Marketing Scoping Study: Report for Allegiance Mining NL, November 2000"* by John O'Shea & Associates

Concentrate sales options and their implications for Avebury were further addressed in the accompanying report:

> *"Avebury Nickel Project Marketing Scoping Study for Allegiance Mining NL"* by John E Butler, December 2000

Because of the predicted relatively high nickel grade of the Avebury concentrates, it appears there may be several organisations (both smelters and metal traders) who would be prepared to purchase the concentrates on favourable and competitive terms.

O'Shea identified and ranked the most likely buyers. Concentrates will be priced in US dollars, and normally based on the LME nickel price.

Payment options are negotiable but, typically, most agreements provide for a percentage payment shortly after ship loading in Burnie, and final payment after arrival at the smelter.

10.2 Gross Revenue:

Sales of nickel sulfide concentrates are based on ***payable metal***, which refers to an agreed percentage of the assayed nickel content of the concentrate.

O'Shea recommended payable metal be calculated as 97% of nickel in concentrates, but Butler believes 97.5% should be attainable in a competitive selling environment, provided the concentrate grade and quality are high.

For this study, 97.5% payable metal factor is used.

Payable metal : 4,270 x 97.5 = 4,163 t

10.3 Ex-mine Deductibles:

Gross revenue is reduced by ex-mine deductibles to provide a net sales revenue or net smelter return.

The two principal ex-mine deductibles are:

- freight to Burnie
- treatment costs

10.3.1 Freight to Burnie:

This study assumes Avebury concentrates will be purchased FOB Burnie.

The accompanying infrastructure and services report estimates transport costs from the mine to Burnie ship at A$550,000.

This is not accounted for as a mine operating cost because it is a

deductible for the purposes of calculating net smelter/net sales royalties.

10.3.2 Treatment Costs:

There are normally two basic options available to the seller:

Option A:

A flat rate expressed as a percentage of the payable nickel. O'Shea recommends 35% of payable nickel be used for this study. The rate is negotiable.

Option B:

A flat charge per dry tonne of concentrate plus a refining charge per pound of payable nickel. These rates are negotiable and can vary on concentrate grade and metal price.

Butler recommends this study use a flat charge of US$200 per dry tonne of concentrate and a refining charge of US$0.50 per pound of payable nickel in concentrate, with concentrate grade of better than 20% nickel.

The two sales options need to be carefully analysed in a full feasibility study, because there are many factors which determine their relative benefits.

An initial analysis as detailed in the following table suggests Option B is preferred:

Nickel price US$/tonne	Gross Revenue US$'M	Treatment Options			
		Option A (US$'M)		Option B (US$'M)	
		Charges	Net	Charges	Net
6,500	27.06	9.47	17.59	8.47	18.59
6,750	28.10	9.84	18.26	8.47	**19.63**
7,000	29.14	10.20	18.94	8.47	**20.67**
7,250	30.18	10.56	19.62	8.47	**21.71**
7,500	31.22	10.93	20.29	8.47	22.75

Table 1: Effect of price and treatment charges on net revenue

10.4 Net Sales:

At a metal price of US$7,000/t, net smelter return (before domestic freight charges) is US$20.67M.

The currency exchange rate effect on this net smelter return is:

A$	Net Smelter Return (A$'M)
0.500	41.34
0.525	39.37
0.550	37.58
0.575	35.95
0.600	34.45

Applying an exchange rate of A$1 = US$0.55, and deducting A$0.55 for ex-mine freight charges:

Net smelter return/net sales = A$37.03M pa

10.5 Royalties:

There are royalties payable on net smelter returns to:

- Rio Tinto
- Tasmanian Government

10.5.1 Rio Tinto:

By agreement with Allegiance, Rio Tinto is entitled to a 2% net smelter royalty on Avebury production.

= 2% of A$37.03M pa

= A$0.74M pa

10.5.2 Tasmanian Government:

The Tasmanian Government royalty is a two-components royalty based on net sales and profitability. It is calculated on the following formula:

$$\text{Royalty} = \frac{1.2 \times \text{net sales}}{100} + (P \times C)$$

where P = profit from previous year

C = 0.3 x profit/net sales

Profit is determined in any manner the Government considers fair and reasonable.

Maximum royalty in any one year is the equivalent of 5% of net sales.

Government royalty, therefore, consists of two components:

- 1.2% of net sales
- profit-related royalty capped at the equivalent of 3.8% of net sales

The first component (1.2% net sales) can be quantified as an ex-mine deductible:

= 1.2% of A$37.03M pa

= **A$0.44M**

The profit-related component is essentially a State Government tax on profits and can range from A$0'M to **3.8% of $37.03M = $1.41M pa.**

10.5.3 Royalty Summary:

It is recommended that the State Government profit-related royalty component be accounted for as a taxation provision, and the non-profit-related component be regarded as an ex-mine cost.

Applying the metal price and exchange rates in the net smelter return illustration above, the ex-mine royalty costs are:

Rio Tinto:	A$0.74
Tasmanian Government:	A$0.44
Total:	**A$1.18**

Profit-related royalty range: A$0-A$1.41M pa.

10.6 Net Revenue:

Net revenue = gross revenue - ex-mine costs.

Using the base case data above, a nickel price of US$7,000/t and an exchange rate of A$ = US$0.55:

Net Revenue = A$35.85M pa

By way of illustration, in more adverse circumstances, an exchange rate of A$1=US$0.60 and a nickel metal price of $6,750 would give a net smelter return of A$31.12M pa.

10.7 Sensitivity Analysis:

This scoping study does not include any sensitivity analyses for major variables such as production factors (grade, recovery, tonnes, etc) or financial factors (metal price, exchange rates, etc).

Sensitivity analyses must be undertaken in a full feasibility study.

11. FULL FEASIBILITY STUDY REQUIREMENTS

Two of the principal objectives of this Scoping Study were:

(a) identify requirements necessary to advance the project to completion of a full feasibility study

(b) estimate the cost of completing a full feasibility study

Each of the accompanying reports addresses these objectives, and their recommendations are summarised below.

There are many different types and definitions of "feasibility studies". For the purposes of this report, a ***full feasibility study*** is defined as a study which would enable the project proponents to do two things:

- decide whether or not to develop the Avebury deposits
- facilitate capital raising if they decide to develop

As such, the study is broadly synonymous with terms like "bankable", "final" and "definitive" feasibility study.

A full feasibility study implies a level of accuracy and contingency of approximately 10%, as compared with 30% for this scoping study.

The *AIMM Cost Estimation Handbook for the Australian Mining Industry (Monograph 20)* lists the objectives of a feasibility study as:

- establishing the proved and probable reserves within the overall measured, indicated and inferred resources
- proving the technical viability of the mine and extraction methods
- defining the features and capacity of the facilities
- estimating the development, capital and operating costs of the mine over the economic life of the resource
- establishing the market for the commodity
- completing economic assessments of the selected project configuration
- assessing the economic sensitivity of the proposed development to various factors

- setting a framework for the implementation of the capital investment in the development

Preparation of a full feasibility study for a small-medium sized development such as Avebury would require the formation of a project team headed by a full-time Project Manager and supported by appropriate technical and financial professionals.

11.1 Resource Definition:

Fundamental to the full feasibility study is the necessity to convert identifed resources to ore reserves. This requires a drilling program and other attendant geological studies.

McKeown estimates approximately 10,000 m of additional core drilling is required. This core would also be utilised for geotechnical and metallurgical purposes.

Such a program would cost an estimated $1.25M and could be completed in 5-6 months if drilling was accelerated with two rigs on 2 shifts/day, 7 days/week.

A compacted program such as that would require two full-time geologists, supported by appropriate data handling facilities and assistants.

This budget estimate is based on the following assumptions:

Personnel (2 geologists, 2 technicians, accommodation, vehicles, etc for 6 months)	200,000
Drilling 10,000 m HQ-NQ core	840,000
Core management	50,000
Assaying, drafting, surveying services	50,000
Data handling facilities	75,000
Management	35,000
	$1,250,000

11.2 Mining:

A full feasibility mining study requires two key inputs:

- geotechnical study
- mine design study

The geotechnical study relies heavily on acquisition of geomechanical and hydrological data from the resource drilling program, and would be undertaken by specialists concurrent with the drilling.

Andrew Bruce estimates this study would cost approximately $84,000.

The full mine feasibility study is estimated by Bruce to cost a further $82,000.

It is recommended a total budget of $0.2M be provided for the mining component of the feasibility study.

11.3 Metallurgical-Milling:

The metallurgical and milling component of the full feasibility study will have three main components:

- fully characterise the nickel mineralisation in the resource
- conduct laboratory scale recovery test work
- develop a flow sheet
- design and cost a mill

John Butler estimates that at least 250 kg of core would be required for the mineralogical and laboratory test work. The core would be derived from the resource definition program.

Butler estimates the cost to complete this test work and develop the flow sheet at $0.2M.

Because this scoping study suggests the mill will be relatively simple and conventional, the anticipated budget to design, specify and cost the mill should be modest. Butler estimates $0.15M should be adequate (pers comm).

A tailings dam site will need to be selected, tested and the dam designed, specified and costed ($0.15M).

It is recommended a total budget of $0.5M be provided for the metallurgical-milling component of the feasibility study.

11.4 Infrastructure:

The feasibility study will require the design, specification and costing of all infrastructure development. This work is estimated to cost $0.25M (see accompanying Infrastructure Report).

11.5 Environment:

Development of Avebury will require a Government approved Development Plan and Environment Management Plan.

The DPEMP requires completion of a range of studies followed by a community consultation program. Details are specified in the attached NSR scoping study.

The cost estimate is $0.4M.

11.6 Marketing and Financial Studies:

The feasibility study will require:

- full nickel market analysis
- "pro-forma" concentrate sales agreement
- sensitivity and financial analyses of the project

A marketing/sales specialist and a financial analysis specialist will be required for this work, which could take 4-5 weeks to complete and involve some travel.

A budget provision of $0.1M is recommended.

11.7 Project Management:

The feasibility study will require a full-time project manager, with secretarial support. Substantial office and travel expenses could be anticipated.

The project is estimated to take a year to complete and this component to cost $0.2M.

11.8 Budget and Schedule:

The above estimates are summarised as:

Feasibility Study Component	$'M
Resource Definition	1.25
Mine	0.20
Mill	0.50
Infrastructure	0.25
Environment	0.40
Marketing/Financial	0.10
Management	0.20
Sub-Total	2.90
Contingency 10%	0.30
Total	**3.20**

It is estimated the study would take one year to complete and a schedule and budget follows.

The opportunity exists to curtail the project following the resource definition drilling program if results are below expectations; ie, after the expenditure of $2.4M.

The schedule provides for the actual feasibility study to be completed within 10 months, followed by a two-month period for concurrent Government permitting and project financing.

12. PROJECT RISK-BENEFIT ANALYSIS:

Ongoing evaluation, development and operation of the Avebury deposits is subject to a number of risks. Principal risks are tabulated and described below.

Detailed risk analysis, sensitivity analyses and risk management strategies are not appropriate in a scoping study, but must be addressed in a full feasibility study.

Project Stage	Risk/Benefit Type	Risk/Benefit Data
Feasibility Study Evaluation	Technical	- resources - metallurgy - mining
Construction-development	Technical	- mine development - mill construction - infrastructure - tailings dam
	Financial	- construction costs
Operation	Technical	- mining - mill performance - resources
	Financial	- operating costs - ex-mine costs - gross revenues

12.1 Feasibility Study Evaluation:

The estimated cost of preparing a full feasibility study is $3.2M. This investment would be subjected to risks and benefits associated with various technical aspects of the operation.

Resources: The recommended detailed drilling program will result in either a decrease or increase in resource tonnes and grade. The implications of this are clear.

Metallurgy: Further metallurgical test work on drill cores may indicate either lower or higher nickel recoveries and

concentrate grades. Detailed design of a mill may result in higher or lower capital costs.

Mining: Detailed geotechnical studies and detailed mine design programs may identify higher or lower mining costs.

12.2 Construction-Development:

This study estimates that if a decision is made, following completion of the full feasibility study, to develop Avebury, a pre-production capital investment of $32.2M might be required. This investment would be subject to various technical and financial risks and benefits, most of which could be adequately managed.

Technical Factors:

Mining: Mine development could encounter better or worse ground conditions than anticipated.

Mill Construction: Problems or areas of redesign could be encountered during construction of the mill.

Infrastructure: Climatic and ground conditions could have a negative or positive effect on development of roads and site works.

Tailings Dam: Ground conditions and weather could influence dam construction.

Financial Factors:

The technical factors above would translate into higher or lower construction-development costs. Negative risks can be reduced in all cases by good management strategies.

12.3 Operations:

This study estimates that, once the project is fully operational, it will incur operating costs of $60/t ore, a number of ex-mine costs, and generate net smelter returns of approximately $35 M pa. A number of technical and financial factors will affect these costs.

Operating Technical Factors:

Mine: Resource grades and tonnages may be higher or

lower than estimated. Ground conditions may be better or worse than predicted.

Mill: Nickel recoveries and concentrate grades may be higher or lower than predicted. There may be operating or circuit modifications which are detrimental or beneficial.

Operating Financial Factors:

The above mine and mill operating factors will result in either higher or lower operating costs. Risk management strategies will reduce the potential for higher costs.

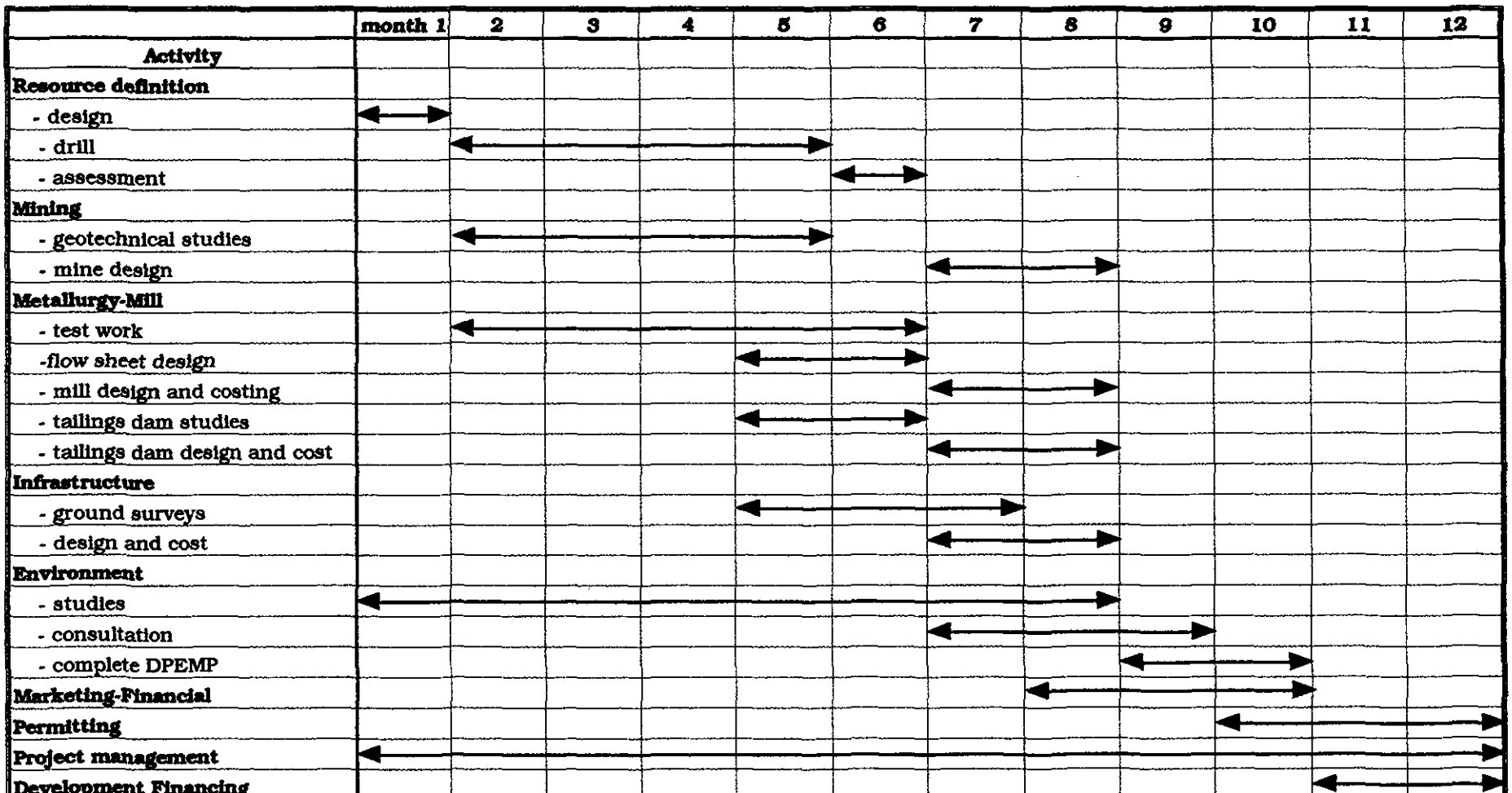
Ex-Mine Financial Factors:

Ex-mine costs are attributed to concentrate transport costs, treatment/smelter charges, RTE and Government royalties.

Risks are minimised in these areas by putting appropriate agreements in place.

Gross Revenue Risks:

The two principal factors influencing gross revenues are the nickel market price and the Australian dollar exchange rate. There are potential risks and benefits from both factors and these can be managed at a corporate level.



Avebury Nickel Project - Feasibility Study

Recommended Program Schedule

96 J 96

	Month 1	2	3	4	5	6	7	8	9	10	11	12	Totals
Activity													
Resource definition	100,000	250,000	300,000	250,000	250,000	100,000							1,250,000
Mining		30,000	30,000	20,000	20,000	20,000	30,000	50,000					200,000
Metallurgy-mill		50,000	50,000	60,000	60,000	80,000	100,000	50,000	50,000				500,000
Infrastructure					50,000	50,000	60,000	90,000					250,000
Environment	40,000	50,000	50,000	50,000	50,000	60,000	40,000	20,000	20,000	20,000			400,000
Marketing - financial								30,000	30,000	40,000			100,000
Project management	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	20,000	20,000	20,000	20,000	200,000
Contingency	16,000	40,000	45,000	40,000	45,000	33,000	26,000	27,000	13,000	9,000	3,000	3,000	300,000
Month Totals	171,000	435,000	490,000	435,000	490,000	358,000	271,000	282,000	133,000	89,000	23,000	23,000	3,200,000
Cumulative Totals	171,000	606,000	1,096,000	1,531,000	2,021,000	2,379,000	2,650,000	2,932,000	3,065,000	3,154,000	3,177,000	3,200,000	3,200,000

Avebury Nickel Project - Feasibility Study

Recommended Project Budget

0501799

661C51

APPENDIX 1

AUTHOR BIOGRAPHIES

661052

CURRICULUM VITAE

Lindsay A Newnham

August 2000

PERSONAL

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PHONE/FAX:	Ph: (03) 6394 3434; Mobile: 0418 386 229; Fax: (03) 6394 3435
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	e-mail: lnewnham@vision.net.au
AGE:	54 yrs
PLACE OF BIRTH:	Toowoomba, Qld
HEALTH:	Good
QUALIFICATIONS:	Bachelor of Science, University of Queensland, 1966
AFFILIATIONS:	Fellow, Australian Institute of Mining and Metallurgy; Member, Tasmanian Chamber of Mines (ex Councillor)

EMPLOYMENT HISTORY

1990 - Present

Consultant
Newnham Exploration and Mining Services

Current Projects:

- Management of all exploration activities for Allegiance Mining NL in Australia.

  This work is presently focused on the Zeehan Nickel project where the recent discovery of the Avebury nickel sulfide deposit has lead to a vigorous exploration program in the district. Currently Project Manager for the Pre-feasibility Study on these discoveries.

- Design and implementation of gold and base metal exploration programs in Eastern Australia for various public companies.

- Evaluation and assessment of a major magnesite project in Tasmania.

- Generation of new exploration project opportunities in Eastern Australia for various companies.

Other major activities to date include:

- Project Manager for development of the Beaconsfield Mine. This involved the management of a multidisciplinary team of contractors and consultants totalling 30 people and an annual budget of A$13 million. The work currently being undertaken has lead to the development of a 150,000 tpa, 20g/t narrow vein underground gold mine, within an environmentally sensitive urban area.

- Mineral resource potential assessments ranging from regional to mine scale for both public companies and Government. The promotion and reassessment of the Mt. Lyell resource has resulted in the continuation of this substantial mining operation by new owners.
- Mineral resource potential assessment of the Bolivian state-owned tin mines.
- Consultant to the Tasmanian Minerals Council on matters relating to exploration, land access, environment, industry promotion and industry-government relations.
- Minerals Industry project consulting work for the Tasmanian Government.
- Evaluation, implementation and management of exploration and mining projects in Tasmania, Victoria and Indonesia.
- Management of exploration projects in Chile for Allegiance Mining NL.

1988 - 1990

Manager - New Projects
Spectrum Resources Australia Pty Ltd, Tasmania

- Responsible for the identification and detailed assessment of new resource investment opportunities suitable for a medium sized public company, such as Spectrum.
- Management of the construction and operation of the 100,000 tpa underground Anchor Tin Mine in eastern Tasmania. This involved all aspects of developing and operating a new mine project, including construction of the mine, mill, infrastructure and tailings dams; day to day management of the operation, and marketing of concentrates.

1982 - 1988

Exploration Manager
RGC Exploration Pty Ltd

- Direction and management of exploration programs in Tasmania, New South Wales and New Zealand. This work was directed at a wide range of commodities, principally gold, base metals and tin. Major discoveries resulting from this work were the Lucky Draw gold deposit in eastern New South Wales (which developed into a significant open cut mine) and the Henty gold deposit in Tasmania (current underground production 90,000 oz pa.
- Mineral resource potential assessment of Koba tin deposit in Indonesia, prior to acquisition by RGC.
- Responsibility for overviewing mine geological services on the Mt. Lyell and Renison Mines. This work included a thorough reassessment of the reserves at Mt. Lyell in 1986 which resulted in the mine life being significantly extended beyond its planned 1989 closure date.
- Manager of the Henty Gold Project development, responsible for all project negotiations with statutory authorities, all infrastructure and site development and planning the first stage of underground development. This project occurs in a sensitive rainforest area and substantial factors of environmental impact had to be addressed.

1971 - 1982

Chief Geologist
Renison Ltd, Tasmania

- Management and direction of all geological functions on the rapidly expanding Renison Mine, including personnel recruitment and development, program design and evaluation, resource assessment, and functioning as an integral member of the mine management team. Major ore discovery and development during this period enabled Renison to expand into one of the largest tin mines in the world.

- Direction and management of programs designed to locate new tin resources in eastern Australia. This work was concentrated in Queensland, New South Wales and Tasmania and resulted in the discovery and delineation of several substantial deposits in Queensland (Gillian, Windermere), New South Wales (Doradilla) and Tasmania (Mt Lindsay, Anchor, Sweeneys and St Dizier).

1970 - 1971

Senior Geologist
Mt Lyell Mining & Railway Co Ltd, Tasmania

- Responsible for Mt Lyell's exploration activities throughout Tasmania, resulting in discoveries of gold-bismuth near Moina and base metal rich deposits at Howards and Henty (now the Henty Gold Prospect), north of Queenstown.

1969

Project Generation Geologist
Longreach Metals NL, Queensland

- Control of drill evaluation programs on base metal prospects in the Mt Isa region.

- Evaluation of tin-tungsten-moly and base metal prospects in southern Queensland and northern New South Wales.

1966 - 1969

Exploration and Mine Geologist
Mt Lyell Mining and Railway Co Ltd, Tasmania

- Responsible for exploration throughout Tasmania directed towards a wide range of commodities.

- Responsible for surface and underground mapping, sampling and drilling programs associated with Mt Lyell Mine.

TOURS & CONFERENCES

Over a 35-year period I have travelled extensively to mining districts in order to remain technically abreast of developments, with visits to most major fields in Australia and overseas in Malaysia, Indonesia, England, Czechoslovakia, Canada, USA, New Zealand, China, Bolivia and Chile.

I have also attended a wide range of technical and managerial conferences and courses throughout Australia and overseas in Malaysia, Indonesia and China.

Presentations on current projects have been made at various conferences and meetings over the past 12 months.

PUBLICATIONS

Geology of the Renison Bell Tinfield, Tasmania
LA Newnham; Australian IMM Conference, Western Australia 1973

Lower Cambrian Carbonate Sedimentation and Subsequent Mineralisation in the Bischoff-Renison Area, Western Tasmania
LA Newnham; GSA Symposium on Limestones, Hobart 1974

Exploration and Mineralogical Data on Several Australian Complex Tin Deposits
LA Newnham; Seatrad Technical Publication No 3, 1981

Helicopter Supported Drilling Programs Aid Environmental Protection in Tasmania
LA Newnham; Sixth Australian Drilling Symposium, Newcastle 1982

Development and Application of Energy Dispersive X-Ray Fluorescence Borehole Loggers and Drill Core Analysers at the Renison Tin Mine in Tasmania
LA Newnham and TW Tucker; Int'l J of App Radiat Isot, Vol 34, No 1, pp 345-352, 1983

The Western Tasmanian Tin Province with Special Reference to the Renison Mine
LA Newnham in "Geology of Tin Deposits in Asia and the Pacific"
Selected papers from Int'l Symposium on Geology of Tin Deposits, Nanning, China 1984
Ed. CS Hutchinson, Publisher Springer

Keeping Our Minds Open: Long Term Exploration Opportunities in Tasmania
LA Newnham, D Leaman, P Uttley; Tasmanian Chamber of Mines Publication, 1990

Models, Methods and Human Needs: The Case for Dynamic Assessment of Mineral Resource Potential
LA Newnham; Tasmanian Chamber of Mines Publication, 1990

Mineral Exploration - Minimum Impact
LA Newnham; Tasmanian Chamber of Mines Publication, 1992

CLIENT LIST

ACM Gold Ltd
Acacia Resources Ltd
Allegiance Mining NL
Allstate Explorations NL
Asarco
Beaconsfield Gold Mines Ltd
Beaconsfield Gold NL
Cominex Pty Ltd
CRA Exploration Pty Ltd
Dragon Mining NL
Goldstream Mining NL
Golden Triangle Resources NL
Golden Shamrock Mines Ltd
Mt Lyell Mining & Railway Co Ltd
Mineral Holdings Australia Pty Ltd
Newcrest Mining
Pacific-Nevada Mining Pty Limited
Poseidon Gold Ltd
Snowden Associates
Stephenson EMF Consultants
RGC Exploration Pty Ltd
Sinclair Exploration Pty Ltd
Tasmanian Chamber of Mines
Tasmanian Minerals Council
Tasmanian Department of Mines
Tasmanian Development Authority
Tasmanian Farmers & Graziers Association
Tasmanian Legislative Council
Titan Resources NL
University of Melbourne
West Tamar Council

JOHN EDWARD BUTLER

TECHNICAL BACKGROUND

Chemical and metallurgical professional affiliations:	Member Royal Society of Chemistry, formerly Royal Institute of Chemistry (MRSC)
	Chartered Chemist (UK) (C.Chem)
	Member Institute of Materials, formerly Institution of Metallurgists (MIM)
	Member Institution of Mining & Metallurgy (MIMM)
	Chartered Engineer (UK) (C.Eng)
	Member Institution of Engineers, Australia, including membership of associated Environmental Engineering Society (MIE Aust)
	Chartered Professional Engineer (CP Eng)
	Management Course 9 Manchester Business School, England
Additional qualifications	B.Sc (Metallurgy) and B.Sc (Chemistry) - Durham University
Date and Place of Birth	8th April 1929 Whitby, Yorkshire, England

EMPLOYMENT HISTORY

1951 - 1953	Metallurgical Chemist, Dorman Long & Co., Warrenby, Yorkshire, England. Iron and steel production control at a major blast furnace/ open hearth furnace/rolling mill complex.
1953 - 1956	Scientific Officer, United Kingdom Atomic Energy Authority, Windscale, Cumberland, England. Research and development associated with plutonium/fission products processing, tritium production, and power generation.
1956 - 1957	Technical Officer, British Titan Products, Billingham, Durham, England. Process development within a major titanium dioxide pigment manufacturing complex. Included basic research into the now standard ilmenite/ titanium tetrachloride/titanium dioxide technology, developed to overcome the environmental problems associated with the traditional sulphuric acid leaching system.
1957 - 1969	The following positions with the Rhodesian Selection Trust (RST) group, now part of ZCCM, in Zambia (formerly Northern Rhodesia). Research Manager, R & D Laboratories, Kalulushi. Responsible for metallurgical and chemical research associated with the mining, milling, smelting and refining of non-ferrous metals, primarily copper, cobalt, nickel, lead and zinc. Leach Plant Superintendent, Chambishi. Included copper metal/cobalt hydroxide flowsheet development, plant construction, commissioning and operational management. The flowsheet incorporated vat leaching of oxidised copper ores, fluidised bed roasting of copper sulphide concentrates, roaster calcine agitation leaching, sulphate solution purification and copper metal electrowinning. The cobalt hydroxide by-product technology was developed and adopted following copper metal plant commissioning, when the build up of soluble cobalt was identified.

1957 - 1969 (cont'd)	Concentrator Superintendent, Chibuluma. Responsible for crushing, grinding, froth flotation operations, etc. at an operation producing separate copper and cobalt concentrates. The implementation of flotation tailings mine backfill technology was undertaken during this period, utilising hydrocyclone classification.
	Concentrator Superintendent, Chambishi. Included plant design, construction, commissioning and operational management for production of high grade copper concentrate by means of froth flotation, from ores containing major chalcocite and bornite values.
	Special Projects Superintendent, Chibuluma copper/cobalt, Chambishi copper, and Kalengwa copper. Covered modifications to existing plant procedures at Chibuluma and Chambishi, and flow-sheet development for Kalengwa ore.
	Metallurgical Superintendent, Chibuluma, Chambishi, and Kalengwa. Operational control of the overall integration of the various metallurgical plants.
	Miscellaneous responsibilities included operating and management control at the Mufulira copper smelter and refinery, Roan Antelope copper smelter, Ndola copper refinery, and Ndola cobalt smelter.
1970 - 1973	Project Manager, New Enterprises, Shell International Petroleum Company, London, England, and The Hague, Netherlands. Evaluation of mineral development projects, and co-ordination of Shell's international metallurgical research activities.
1973 - 1974	Consulting Metallurgist, Foster Wheeler, London, England. Advisor on mineral project opportunities within the engineering contracting field.

1974 - Oct 1993 Group Metallurgical Consultant/Chief Metallurgist, Renison Goldfields Consolidated (RGC), formerly Consolidated Gold Fields Australia (CGFA), Sydney, Australia.

Responsible for group metallurgical operations and developments, primarily in the areas of copper, tin, gold, mineral sands and coal orebody processing.

Copper operations included Mount Lyell, Tasmania (copper concentrate and pyrite concentrate production) and Gunpowder, Queensland (copper concentrate). Following the cessation of copper concentrate production at Gunpowder an in-situ leaching pilot scale programme developed the parameters for a commercial cement copper operation, and a subsequent RGC designed in-situ leaching/ solvent extraction/copper electrowinning plant was commissioned by another mining company.

Tin activities embraced concentrate production at Renison, Tasmania and PT Koba, Indonesia, the former being particularly complex due to its lode deposit nature which necessitated carefully controlled comminution classification, heavy media and gravity separations, sulphide and oxide froth flotation, acid leaching of ferrous carbonate gangue, and concentrate pressure filtration. An associated duty was to evaluate the construction of smelters for the production of refined tin metal at both Renison and PT Koba, eventuating in the establishment of a smelter at the latter location.

Gold responsibilities included design, construction and operating duties at Pine Creek (Northern Territory), New Guinea Goldfields (PNG), Lucky Draw (NSW), and Nalesbitan (Philippines), process development and design for Henty (Tasmania), and flowsheet evolution for the Porgera Joint Venture (Placer/Mt Isa/RGC) in PNG. The latter is again a complex metallurgical operation, embracing comminution, gold bearing pyrite froth flotation, pyrite concentrate pressure oxidation, and gold bullion production.

1974 - 1993 (cont'd)

Mineral sands treatment at both NSW and Western Australia sites required regular monitoring for the recovery of separate ilmenite, rutile, synthetic rutile, monazite, and zircon concentrates. Various peripheral activities were undertaken, including the examination of synthetic rutile processing modifications, removal of traces of radio-active mineralisation from saleable titaniferous concentrates, and fine grinding of zircon concentrate for the production of opacifier/ ceramic grade feedstock.

Coal duties covered production operations at the Bellambi washery (NSW), coal preparation testwork and design for the Glendell deposit (NSW), and the production of thermal coal by the Narama Joint Venture (NSW).

Miscellaneous duties covered evaluation of a wide variety of technological developments, frequently requiring laboratory and pilot plant programmes with available RGC feedstock. An additional responsibility involved close liaison with RGC Exploration personnel with an annual budget of $15 - $20 million, their primary targets being copper, tin, gold, zinc and mineral sands deposits. The position demanded continuous updating of the complete spectrum of mineral processing and extractive metallurgy developments, this being achieved by means of literature searches, attendance at national and international conferences, plant visits, and contacts with relevant world mining industry personnel.

Miscellaneous executive duties included directorships on the boards of Renison Ltd (tin), Gunpowder Ltd (copper), and North West Acid Ltd (sulphuric acid).

Nov 1993 - Present

Metallurgical and chemical consultant.

Commissions have included projects with RGC, RGC's subsidiary Goldfields, Porgera Joint Venture, Adelaide Brighton Cement, Gold Mines of Australia, Copper Mines of Tasmania,

1993 - Present (cont'd)	Walhalla Mining, Allegiance Mining, Placer Dome, Mineral Resources Development (USA), Coeur d'Alene (USA), Lakefield Research (Canada), etc. These duties have embraced gold, silver, mineral sands, copper, tin, antimony, lead, zinc, nickel and fluorite processing technologies, and detergent synthesis evaluation, in Australia, PNG, USA, Canada, Mexico, Bolivia, Argentina, Chile, Peru, Indonesia, Borneo/Sarawak, Germany, England, and Siberia. Of special environmental value was the continuing role of metallurgical plant design, construction and commissioning consultant for the Henty gold project, an operation which subsequently received several state and national awards.
Miscellaneous industry duties	At various times engaged in the following capacities: - Founder Councillor of the Minerals Industry Research Organisation (MIRO), UK. - Alternate Councillor of the Australian Minerals Industry Research Association (AMIRA). - Member of the Advisory Panel of the Julius Kruttschnitt Mineral Research Centre, Queensland. - Member of the non-ferrous metals committee of the Australian Standards Association. - Member of the Editorial Advisory Board of "Minerals Engineering", UK, an international journal devoted to innovation and developments in mineral processing and extractive metallurgy.

July 1999

McKEOWN MINING PTY LTD
INCORPORATED IN TASMANIA

SERVICES TO THE MINING INDUSTRY
A.C.N. 009 586 814

MICK MCKEOWN

Mick McKeown is a geologist with over 30 years experience in mining and exploration geology.

After graduation, he worked in tin mines in Tasmania and New South Wales, including nine years for Renison Ltd. Mick has been consulting since 1988; during that time he has worked in Tasmania, Western Australia and China.

Mick's experience ranges from supervision of diamond drilling to mine planning. He has been supervising diamond drilling programs since 1970, including deep directional drilling at the Beaconsfield gold mine. He has lithologically and geotechnically logged tens of thousands of metres of diamond drill core from successful base metal, gold and iron projects in Tasmania.

Mick has a record of success in converting mineralisation to ore. He has long experience in the interpretation of structurally complex orebodies, including those at the Renison mine. His involvement in the estimation of ore reserves has ranged from the use of classical manual methods to geostatistical methods. Mick has been using DATAMINE for over ten years and is a licensed DATAMINE user. In the late 1980s, he undertook the first geostatistical ore reserve estimation at Mount Lyell which enabled selective, and profitable, mining of the Prince Lyell orebody. Over the years Mick has worked on ore reserve estimates for base metal, gold, iron and nickel deposits.

Mick has developed a comprehensive set of procedures for checking the validity of mineral resource and ore reserve estimates. He has undertaken ore reserve reconciliations in base metal, gold and iron mines.

Mick's geotechnical investigations have included nearly ten years' work at the Savage River iron mine dealing with pit wall stability in a high rainfall environment, and also the assessment of the caveability of the deposit during the feasibility study for an underground mine.

Mick has been involved in the Avebury nickel project since Allegiance Mining's first diamond drilling program in 1998.

Mick graduated with a Bachelor of Science degree in geology in 1969. Since then he has completed a Graduate Diploma in Mining and has been awarded a Master of Engineering Science for his thesis on selective mining at the Mount Lyell mine guided by geostatistical resource estimation. He is a Fellow of the Australasian Institute of Mining and Metallurgy.

MICHAEL V. MCKEOWN - GEOLOGICAL AND MINING CONSULTANT

1998 to present — **Allegiance Mining NL, Avebury project, Zeehan, Tasmania**

supervision of diamond drilling at the Avebury prospect and subsequent identification of high grade nickel sulphide mineralisation, mineral resource estimation, and recommendations for future work

2000 to present — **Tasmania Mines Ltd, Kara scheelite and magnetite mine, Tasmania**

estimation of ore reserves; preparation of Environmental Management Plan for the Kara operations

1999 to present — **King Island Council, King Island, Tasmania**

remediation of acid rock drainage associated with Council's quarries; exploration for suitable rock for road-making materials

1998 to present — **Murchison United Ltd, Renison Bell tin mine, Tasmania**

geological interpretation of structurally complex areas of the mine; ore reserve estimation using DATAMINE

1997 — **Allegiance Mining NL, Zeehan base metals project, Zeehan, Tasmania**

compilation of existing data on the Zeehan mineral field and recommendations for future work

1997 to 1999 — **Australian Bulk Minerals, Savage River iron mine, Savage River, Tasmania**

assistance with the re-development of the Savage River Mine.

1997 to 1998 — **RGC Ltd, Renison tin mine, Renison Bell, Tasmania**

development of geostatistically based resource and reserve system, including development of DATAMINE based macros and menus.

1997 to 1998	**Allegiance Mining NL, Andersons Creek nickel prospect, Beaconsfield, Tasmania** design and supervision of reverse circulation drilling programme to test lateritic nickel/cobalt prospect, and estimation of mineral resource.
1996	**Australian Bulk Minerals, Savage River iron mine, Tasmania** supervision of diamond drilling programme, and preparation of subsequent geological interpretation, ore reserve assessment, mine design and geotechnical evaluation; mining re-commenced at Savage River in 1997.
1996	**Tasmania Mines Limited, Kara scheelite and magnetite mine, Tasmania** preparation of Environmental Management Plan for the Kara operations.
1996	**Copper Mines of Tasmania, Prince Lyell copper mine, Queenstown, Tasmania** geotechnical assessment of the area proposed for a new northern ventilation shaft with recommendations for its location, shape and orientation.
1995	**Beaconsfield Mine Joint Venture, Beaconsfield gold mine, Tasmania** supervision of deep diamond drilling programme and preparation of mineral resource estimate; this mine is currently being developed for production in 1999 .
1995	**Asia Resource Capital Limited, Jiang Cha Ling gold mine, Sha'anxi, China** geological audit of the Jiang Cha Ling gold and nickel deposit, Sha'anxi Province, China; the gold deposit is now being mined with Australian management.
1995	**RGC Exploration Pty Ltd, Tasmania** valuation of RGC's Tasmanian Exploration Licences during restructuring of RGC/Goldfields companies.

1994 to 1995 **Savage Resources, Long Plains magnetite and ochre project, Tasmania**

geological interpretation of diamond drilling of magnetite deposits; opencut designs, ore reserve estimations, mine planning and scheduling for mining of ochre and magnetite deposits.

1994 to 1997 **Renison Limited, Renison tin mine, Renison Bell, Tasmania**

geological interpretation and ore reserve estimations of very structurally complex orebodies.

1993 **Telfer Gold Mines, Telfer mine, W.A.**

reconciliation of ore mined against ore reserves for the large, low grade gold deposit including the setting up of an ore reserve reconciliation system using DATAMINE, including macro and menu writing.

1993 to 1995 **Tasmania Mines Limited, Kara scheelite and magnetite mine, Tasmania**

responsible for all geological, mining engineering and environmental management functions, at Tasmines' operations at the Kara mine; recommendation for the production of fine magnetite for use as dense medium in coal washeries which is now in full production.

1993 **Spectrum Resources, Anchor tin mine, Lottah, Tasmania**

preparation of an ore reserve report which was used as the basis for the successful sale of the mine.

1993 to 1994 **Savage River Mines, Savage River mine, Tasmania**

evaluation of the caveability of North Pit and design of block cave mine with decline access, in association with Dr Kevin Rosengren and Dr Dennis Laubscher.

1988 to 1995 **Savage River Mines, Savage River iron mine, Tasmania**

responsible for all aspects of mine geology and geomechanics; evaluation of South Lens leading to mine life extension to 1994; evaluation of the causes and remediation of major toppling failure of west wall of Centre Pit, in association with Barrett Fuller and Partners; redesign of the North Pit leading to mine life extension to 1997.

1990 to 1992 — **The Mount Lyell Mining and Railway Company, copper mines, Queenstown, Tasmania**

calculation of the first useful geostatistical ore reserve for Prince Lyell; installation of DATAMINE software including macro and menu writing; setting up of longhole open stope ore reserve evaluation system using DATAMINE; preparation of comprehensive ore resource study used as a basis for the Tasmanian Government's search for a new operator; analysis of blast hole deviation and recommendations for its minimisation; analysis of the patterns of electricity consumption in the Prince Lyell Mine.

1989 to 1991 — **Spectrum Resources, Anchor tin mine, Lottah, Tasmania**

diamond drilling of the Anchor and Royal George tin deposits; interpretation of the geological structure of the Anchor mine and re-design of the mine workings; preparation of comprehensive ore reserve report and recommendations for further ore discovery.

1988 to 1990 — **Renison Limited, Renison tin mine, Renison Bell, Tasmania**

re-interpretation of the Envelope orebodies leading to change of mining method from long hole open-stoping to benching; interpretation of the Rendeep orebodies; testing of, and recommendation to purchase, DATAMINE software.

1982 to 1988 — Teacher of HSC level Geology and mathematics

1980 to 1982 — Technical Services Superintendent with Renison Limited at the Renison Bell tin mine.

1976 to 1980 — Senior Geologist with Renison Limited at the Renison Bell tin mine.

1973 to 1976 — Mine Geologist with Renison Limited at the Renison Bell tin mine.

1970 to 1973 — Mine Geologist with Aberfoyle Limited at the Storeys Creek tin mine.

Short Resume

PRINCIPAL for A J BRUCE CONSULTING PTY LTD

Mr. Andrew Bruce: Mining Engineer

Mr. Bruce is Principal of A J Bruce Consulting Pty Ltd.. Andrew, a graduate of the University of Melbourne, is a mining engineer with more than twenty years experience in underground operations all over Australia. He has particular experience in Tasmania, Western Australia and Queensland including Renison, Hamersley, Agnew and Mount Isa mines. Andrew has direct experience in a wide range of mining methods including; large-scale opencut, cut and fill, benching, open stoping, hand-held and sublevel cave. Most of the mining Mr. Bruce has been involved with is in complex geological settings, necessitating a close liaison between geology, mine planning and operations. Mr. Bruce is used to working closely with senior management and staff and employees and providing comprehensive advice and reports in the various areas of expertise of the Consultancy.

Career highlights for Mr. Bruce include:

Principal, A. J. Bruce Consulting Pty Ltd (2 years)
Successfully set up my own small business as a consultant mining engineer specialising in underground mine planning & ventilation. Completed Level IV certificate in Small Business Management. Ongoing provision of technical assistance to Kundana Gold, Beaconsfield Joint Venture, Renison Bell and Pasminco Rosebery mines. Recently completed a pre-feasibility study for Allegiance NL on the Avebury nickel prospect.

Chief Mining Engineer, Renison Bell Tin Mine (7 years)
Direction of mine technical services section including preparation of capital and operating budgets, life of mine production and development planning and scheduling. Mine planning and scheduling for mine expansion from 525,000 tonnes to 725,000 tonnes and a mine life extension from five to thirteen years. Conversion of the mine from jumbo cut & fill to longhole benching. Core member of the Renison shaft feasibility study. Contract management, mine design and specification of the $38 million Shaft Project. Personally planned all extraction aspects for the structurally complex Rendeep orebodies.

Senior Planning Engineer, Hilton Mine (5 years)
Design of the first Sublevel Open stopes. Mining design of the Hilton the crusher and conveyor system. Team leader for all major pumping and drainage projects. Acting Planning Manager with responsibility for planning, draughting and survey personnel. Development of a ventilation network program with 3D graphics-display. Design and project control of a major aquifer pump station. Preparation of a surface and underground water management strategy for Hilton Mine. Specification, selection and commissioning of a 1950 kW ventilation fan.

Senior Mining Engineer - Planning, Agnew Mining Company (1 year)
Introduced computer based modelling of ventilation using network analysis, set-up systems for grade control of sublevel cave mining and conducted model studies of sublevel cave ore flow.

Mining Engineer, Renison Limited (6 years)
Positions held included planning engineer, ventilation engineer, shift boss, miner. Gained a Certificate as Tasmanian Mine Manager.

Graduate Mining Engineer, Hamersley Iron Pty Ltd (3 years)
Completed the Hamersley Iron graduate training scheme and was promoted to pit planning engineer at Paraburdoo.

TERTIARY EDUCATION
Melbourne University
B.E. (Mining) with Honours (1976)

PROFESSIONAL MEMBERSHIP
Member Australian Institute of Mining and Metallurgy

EMPLOYMENT and EXPERIENCE SUMMARY
for
ANDREW JOHN BRUCE

January 1999 – Present
A J Bruce Consulting Pty Ltd
Position Held: Consultant Mining Engineer

Job Description:

A J Bruce Consulting Pty. Ltd. is based in Launceston but services all areas of the Australian mining industry. Areas of expertise of the Consultancy are mine ventilation, mine planning, feasibility studies and project management.

Personal Achievements:

- Achievement of the start-up business plan.
- A successful ongoing relationship with clients.
- Completion of Small Business Management certificate.

November 1991 – December 1998
Renison Bell Tin Mine
Most Senior Position Held: Chief Mining Engineer

Job Description:

Direction of the mine technical services section as Chief Mining Engineer. Supervision of mining engineers (planning, drill & blast, scheduling, ventilation, geotechnical) and surveyors. Preparation of capital and operating budgets, mine production and development plans and schedules to enable the mine department to meet production and budgetary targets. Mine projects were managed (including preparation of capital expenditure requests).

Personal Achievements:

- Implementation and documentation of systematic mine planning procedures.
- Development of models for the economic evaluation of all geological resource blocks.
- Core member of the Renison shaft feasibility study and subsequent project team.
- Completion of planning for the access and extraction of the structurally complex Rendeep orebodies.
- Contract management, mine design and specification of the $38 million Shaft Project which was completed three months ahead of schedule in June 1996. The project included high speed access to Rendeep orebodies, crushing, conveying and shaft hoisting.
- Mine planning and scheduling to facilitate mine expansion from 525,000 tonnes to 725,000 tonnes and a mine life extension from five to thirteen years.
- Conversion of the mine from jumbo cut & fill to longhole benching, thereby meeting the challenge of increased depth, mining remnants, increasing production and reducing costs.
- Planning and project management to successfully relocate a 500kW ventilation fan from surface to an underground chamber.

October 1986 - July 1991
Hilton Mine
Most Senior Position Held: *Senior Planning Engineer*

Job Description:

As one of a number of mine planning engineers reporting to the planning manager projects involved the design of the first Sublevel Open Stoping compartment including detailed assessment of the ground conditions, layout of orepasses, selection of sublevel intervals and design of ventilation circuits and a major project of coordinating the mining aspects associated with the design of the Hilton the crusher and conveyor system. This required close liaison with engineering design, construction and maintenance personnel.

Other more routine duties included: ventilation planning and the supervision of the Ventilation Officer, preparation of three month, two and five year production schedules, preparation of mining plans, including rail and trackless development design, team leader for all major pumping and drainage projects and project control of Hilton surface & underground raise boring program. The position of Acting Planning Manager with responsibility for planning, draughting and survey personnel during absences of the Hilton Planning Manager.

Personal Achievements:

- Development of a ventilation network program with 3D graphics-display for primary ventilation studies (using programs largely written by myself).
- A member of the computer-aided mine planning group that defined and recommended the technical computer based planning system for Mount Isa Mines (IMPS).
- Design and project control of a major aquifer pump station.
- Preparation of a surface and underground water management strategy for Hilton Mine.
- Specification, selection and commissioning of a 1950 kW ventilation fan.

October 1985 - August 1986
Agnew Mine
Most Senior Position Held: *Senior Mining Engineer - Planning*

Job Description:

Coordination of all mine planning and draughting activities. This involved the supervision of several mining engineers, the chief draughtsman and a tracer in preparation of mining layouts and designs, capital and operating budgets and technical reports.

Personal Achievements:

- Introduced computer based modelling of ventilation using network analysis.
- Developed spreadsheets for grade control of sublevel cave mining.
- Model studies of sublevel cave ore flow.

November 1979 - August 1985
Renison Bell Mine
Most Senior Position Held: Mining Planning Engineer

Job Description:

First employed as a mining engineer and for approximately eighteen months was spent working on small projects, the purchase and testing of mining consumables and explosives testing. The next eighteen months were spent as ventilation engineer and Registered Ventilation Officer responsible for statutory inspections and reporting.

Seven months as an underground miner, including nippering, Anfo mixing, rockbolting, off-siding on Jumbos, stope firing, truck driving and general underground labouring. This was followed by four months as a shift boss on a three shift roster supervising between eight and fifteen miners (production and development crews).

The remaining time with Renison was in the Mine Planning Section pre-dominantly involved in development of layouts for underground access and cut & fill stoping of the complex Renison ore bodies

Personal Achievements:

- Awarded a Tasmanian Mine Managers Certificate.
- Initiated drill consumable control and testing.

February 1977 - October 1979
Hamersley Iron Pty Ltd, Paraburdoo, WA
Most Senior Position Held: Mining Engineer

Job Description:

The following positions of Quarry labourer on continuous shift, assistant mining engineer in the Mining Engineering Department, Extraction & Blending Department and Production Departments while employed on the Hamersley graduate scheme. At the end of the graduate program a permanent position of mining engineer was accepted in the Mining Department doing short term planning, schedules, grade/tonnage reconciliations, blast and road work design.

Personal Achievements:

- Design of the 10 East final pit.

Uni Vacation Employment: *Mount Isa Mines Limited, Queensland 1975/76.*
Mt Lyell Mining & Railway Co., Tasmania 1974/75.

661074

PERSONAL DATA
for
ANDREW JOHN BRUCE

Age: 45

Tertiary Education:
Melbourne University
B.E. (Mining) with Honours (1976)
Final year thesis: Some Aspects of Kriging

Professional Membership: Member of Australian Institute of Mining and Metallurgy

Certification:
Tasmanian Mine Manager's Certificate of Competency
St John First Aid Certificate

Publications:
A History of Drill Consumable Control at Renison Mine,
Underground Operator's Conference, Kalgoorlie 1985.
Tin ore mining at Renison Limited,
Sir Maurice Mawby Memorial Volume, Monograph 19, 1992

Supplementary Courses:
- Accident Prevention
- KT Problem Solving
- Economic Evaluation
- Negotiation Skills Workshop
- Leadership and Motivation
- Ore Reserve Estimation
- Excavation Engineering for Underground Mines
- Datamine and Surpac
- Project Management
- Team Development
- Micro Planner X-Pert
- Beginners Japanese
- Financial Accounting for Non Accountants
- Middle Management Development
- Certificate IV in Small Business Management

661075

NSR General Information

NSR Environmental Consultants Pty Ltd was established in 1974, employs 24 people (seventeen professional staff) and continues to be run by its founding principals. The firm provides environmental services for the planning and permitting of new resource development projects, principally for the mining industry and for petroleum, forest products, urban development and industrial clients. Problem-solving and management and monitoring programs for operating projects are a substantial part of the firm's business. The firm also provides assessments and advice in relation to project financing, insurance and acquisitions.

More than 400 projects have been completed in 19 countries (Angola, Argentina, Australia, Canada, China, Cuba, Fiji, Ghana, Indonesia, Korea, Malaysia, Myanmar, New Caledonia, Norway, Papua New Guinea, Philippines, Solomon Islands, Tanzania, Thailand, Tuvalu, USA, Viet Nam).

Specialist expertise is offered in the following areas: the chemistry and biology of polluted waters; water quality sampling, analytical chemistry and quality control; aquatic biological surveys of fresh and marine waters; deep-water submarine tailing disposal; mining and industrial effluent characterisation, environmental simulation of effluents and ecotoxicology; meteorological, hydrological and oceanographic data collection in remote and inhospitable areas and computer data analysis and dispersion modelling. Over the past 24 years, the firm has established an extensive network of specialist subconsultants who have the expertise and experience required to cover fields not covered by in-house personnel.

Offices

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Indonesia (in association with)

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Facsimile : (62 21) 520-2359

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NSR Environmental Consultants Pty Ltd

Staff List 2000

Alastair Sharp-Paul	Director
Stuart Jones	Director
Michael Jones	Principal
David Gwyther	Specialist Consultant (Marine Biology)
Ian Hargreaves	Specialist Consultant (Oceanography, Hydrology and Sedimentology)
David Oram	Specialist Consultant (Data Analysis/Computer Systems)
Carmel Coyne	Senior Consultant
Michael Cramer	Senior Consultant
Luci David	Senior Consultant
Adrian Flynn	Senior Consultant
Kathryn Friday	Senior Consultant
Julianne McGeown	Senior Consultant
Clark Monahan	Senior Consultant
Michael Wright	Senior Consultant
Heath Doodie	Consultant
Tara Halliday	Consultant
Giulio Pinzone	Consultant
Gabrielle Pistone	Consultant
Liz Jacobsen	Technical Editor
Hanna Blaszkiewicz	Draftsperson
Margot Cleland	Office Manager
Jill O'Neil	Desktop Publishing Manager/Executive Assistant
Bronwyn Hughes	Secretary/Receptionist
Ros Dehnert	Project Administrator
Josie Bant	Accounts Clerk
Fay Reynolds	Administrative Assistant

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Lucinda David

Year of Birth:	1971
Nationality:	Australian
Qualifications:	B.Sc. (Hons in Geography), University of Melbourne
Present Position:	Environmental Consultant, NSR Environmental Consultants Pty Ltd
Speciality Areas:	Project management, data analysis and interpretation, GIS, report writing and environmental planning and management

Experience

1995–present

Joined NSR Environmental Consultants Pty Ltd in 1995.

Caloundra Downs II Informal Land Use Inquiry (1999) (Lensworth Group Limited)—Project Manager resident in Brisbane responsible for the administration and management of a $2.5-million, privately sponsored, public land use inquiry for a 3700-ha parcel of land on the Sunshine Coast.

Ramu Nickel Project, PNG (1998) (Highlands Pacific Limited)—GIS system development, compilation of technical information and management for preparation of the Environmental Plan.

Misima Mine, PNG (1998) (Placer Pacific Limited)—On-site assistance in development and preparation of Environmental Management System.

Savage River Iron Ore Mine, Tasmania (1996-97) (Australian Bulk Minerals)—Assisted in the preparation of the Development Proposal and Environmental Management Plan; on-site management of environmental investigations (including management of acid mine drainage, tailings and waste rock management) prior to re-opening of the mine.

Wemen Mineral Sands Project (1997-98) (RZM Pty Ltd)—Responsible for the initial research, consultation and preparation of the socio-economic assessment report; assisted in the preparation of the Environmental Effects Statement (EES) for the proposed mineral sands mine, Wemen, northwest Victoria.

Henty Gold Project (1996-97) (Goldfields Ltd)—Assisted in the preparation of the Environmental Management Plan and Cyanide Management Plan.

PNG Gas Project (1997-98) (Chevron South Pacific)—GIS system development, data input and management for preparation of the Environmental Impact Statement. Prepared review of climate data. Assisted in the preparation of the Environmental Plan.

Twelve Mile Project (1997) (RZM Pty Ltd)—Responsible for the research, consultation and preparation of the socio-economic study for the project.

Loei prospect (1995-96) (Niugini Mining Pty Limited, Thailand)—Responsible for data analysis of ongoing baseline stream monitoring.

Morning Star Mine Project (1996) (Placer Exploration Ltd)—Assisted with production of the Preliminary Environmental Assessment involving a site visit and consultation with relevant organisations.

Lihir Gold Project (1996-ongoing) (Kennecott/Niugini Mining, PNG)—Involved with the production of the Lihir Baseline Study.

Snowy Mountains Hydro Electric Authority (1996) (Water Studies Centre, Monash University)—Responsible for summary of water quality and hydrological data from SMHEA monitoring programs.

Lake Cowal Gold Project (1995) (North Ltd)—Involved with the report detailing Pit Decommissioning of the mine, analysing and interpreting density data.

1993–1995

Connell Wagner Pty Ltd, Melbourne

Environmental planner with responsibility for various projects:

Optus Communications - GSM (Mobile) Telecommunications Networks—Responsible for the preparation of environmental planning reports for base station sites with detailed investigations of constraints and opportunities regarding option sites and evaluation of significant environmental impact of each site.

Kinglake National Park Management Plan—Duties included assisting in planning and environmental considerations for the Kinglake management plan, liaising with the Department of Conservation and Natural Resources and local interest groups, and assisting in co-ordination and project management.

661079

31 May, 2000

Mr L Newnham
Project Manager
Allegiance Mining NL
PO Box 132
RIVERSIDE, Tasmania 7250

Dear Lindsay

ALLEGIANCE MINING NL - ZEEHAN NICKEL PROJECT

Thank you for the opportunity to submit a proposal and cost estimate concerning the preparation of a Pre-feasibility Environmental Study for the Zeehan Nickel Project. This letter confirms our willingness to participate in the study and provides additional relevant information as required in your letter of 17 May 2000.

Introduction to NSR Environmental Consultants Pty Ltd (NSR)

NSR is an Australian (Melbourne-based) environmental consulting firm that was established in 1974. The firm employs 24 people and continues to be run by its founding principals. NSR provides environmental services for the planning and permitting of new resource development projects, principally for the mining and petroleum industries. Due diligence investigations and auditing, especially for banks (including the World Bank/International Finance Corporation (IFC)) and sovereign risk insurers, are a growing part of the firm's business. General company information is provided in Attachment 1.

NSR plans, manages and conducts technical environmental investigations and impact analysis, substantiates and defends impact assessments in public forums, negotiates permit conditions, and services public information requirements for its clients. One of the firm's specialisations is in the environmental planning and assessment of mining projects, particularly in high-rainfall environments.

More than 300 projects have been completed in 18 countries (Australia, New Zealand, Papua New Guinea, the Solomon Islands, Fiji, New Caledonia, Indonesia, Malaysia, Thailand, Myanmar, the Philippines, Viet Nam, China, Korea, Ghana, Tanzania, Canada and Argentina). NSR's experience in the mining industry spans more than 90 mines and advanced mineral prospects, including over 50 in high-rainfall, tropical environments and over 40 in temperate and arid environments (Attachment 2).

NSR's permitting documentation for new mining projects is often reviewed by the World Bank/IFC and NSR is well known to both organisations.

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Victoria Australia 3123
Ph: 61 3 9882 3555 Fax: 61 3 9882 3533
e-mail: nsr@nsrenv.com.au
ACN 005 041 878

Experience in Tasmania

NSR has been actively involved in environmental assessments in Tasmania since 1980. In addition to providing in-house expertise in core areas such as project management, hydrology, water quality and aquatic biology, NSR makes extensive use of subconsultants who are chosen on the basis of relevant local knowledge.

In recent years, NSR has undertaken the following mining-related projects:

- Environmental Issues Report for the Main Creek Magnesite Project (1998-1999): This involved identifying the environmental issues for the project, discussion with DPIWE (Department of Primary Industry, Water and Environment) and developing an environmental works and permitting program.
- Permitting of the Savage River Iron Ore Mine (1996-1997): This involved preparation of the DPEMP (Attachment 3) and negotiation with DELM (Department of Environment and Land Management [now DPIWE]) concerning the Land Use Permit conditions imposed on the refurbished operation.
- Design and management of environmental investigations associated with the re-opening of the Savage River Iron Ore Mine (1997-98): One of the Land Use Permit conditions was that additional studies be undertaken concerning matters such as waste management, archaeology and air pollution. NSR designed and managed these investigations, with an NSR staff member operating out of Australian Bulk Mineral's (ABM's) Burnie office for six months in 1997. An assessment of the degree of existing soil, sediment and water degradation of both the Savage River mine area (including Main Creek and Savage River itself) and the Port Latta processing plant was also conducted as part of this program.
- Permitting of the Henty Gold Mine (1990-1997): This involved preparation of both the initial DPEMP and a reviewed and updated EMP (Attachment 4) after operations had commenced.
- Routine environmental reporting for the Henty Gold Mine (1993-1998): Annual EMRRs (Environmental Management Review Reports) have been prepared by NSR for submission to DELM (now DPIWE).
- EMP/EIP for the Renison Tin Mine (1994-95): NSR assisted in the preparation of both an Environmental Management Plan and Environmental Improvement Program.

Non-mining projects undertaken by NSR in Tasmania include planning, direction and management of an ecological risk assessment to assess whether the Fletcher Challenge pulp mill is causing 'environmental harm' to the Derwent River (1999–ongoing).

NSR is also currently undertaking a combined Federal/Victorian/Tasmanian impact assessment study for the Basslink Project, i.e. an HVDC interconnector between Victoria and Tasmania, for National Grid International Limited.

All of NSR's recent experience in Tasmania has been directed by one of the firm's principals, Michael Jones, whose CV is included in Attachment 5. We

661C81 01-4526A (Vol 2 of 10)

MINERAL RESOURCES
14 FEB 2001
TASMANIA

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

SCOPING STUDY

MINERAL RESOURCE POTENTIAL REPORT

MICROFILMED
FICHE No 015502-13

MINERAL RESOURCES
FILE REF: EL28\88P15
16 FEB 2001
DOC. REF:
OFFICER | FOR ACTION | FOR INFO
See folio 61
RESUBMIT TO | DATE

20 January 2001

01_4526 (A)
Vol 2 of 10
Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL*; Newnham Exploration and Mini
Newnham, L.A.
EL28/1988

Prepared by:
Lindsay Newnham BSc, FAIMM, CPGeo
Newnham Exploration and Mining Services
PO Box 183, Exeter, Tasmania 7275
Ph: (03) 6394 3434 Fax (03) 6394 3435

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1. SUMMARY

Allegiance Mining NL has identified a substantial nickel sulfide resource at Avebury - 3.1 Mt 1.7% Ni. This resource is currently the subject of a development Scoping Study.

Within the general Avebury area, Allegiance has also located eight (8) prospects with nickel sulfide potential. If substantial resources could be identified at one or more of these prospects, it would have a major beneficial impact on the development strategy at Avebury.

A prospectivity rating has been assigned to each of these areas.

Prospect	Prospectivity Rating
South Avebury Resource Extension	Very high
Avebury district:	
- East Avebury	Very high
- Bismark	Very high
- Bison	High
- Pontiac	Moderate
- West Avebury	Moderate
Trial Harbour	Very high
Melba Flats	Very high

A two year program of further exploration, costing just under $1M pa, is strongly recommended and is summarised below:

Prospect	2001 Budget ($)	2002 Budget ($)
South Avebury Extensions	300,000	
East Avebury	250,000	250,000
Bismark	130,000	120,000
Bison	30,000	60,000
Pontiac and West Avebury	40,000	120,000
Trial Harbour	140,000	200,000
Melba Flats	60,000	150,000
TOTALS:	**950,000**	**900,000**

Scheduling of this recommendation is driven by several factors:

- prospectivity rating
- likely Avebury development program schedule
- weather
- financial resources

Scope exists to accelerate or retard the recommended schedule.

The Year 2 schedule and budget would also be influenced by Year 1 results.

2. INTRODUCTION

Allegiance Mining NL has discovered a nickel sulfide deposit at Avebury near Zeehan, western Tasmania (Fig 1 (a)).

Avebury is currently the subject of a Scoping Study of the possible development of an underground operation mining 300,000 tonnes pa for the production of 4.270 t of nickel in sulfide concentrates over a 10-year mine life.

Any additional resources identified in the district would have a beneficial effect on the project in terms of either a higher annual throughput (reducing costs and increasing revenue), or an extended mine life.

Potential to define such additional resources exists within tenements held by Allegiance in the Zeehan area.

This brief report reviews that potential and comments on further evaluation requirements.

Greater detail on exploration data acquired to date on each area is contained within tenement Annual Reports and other technical reports.

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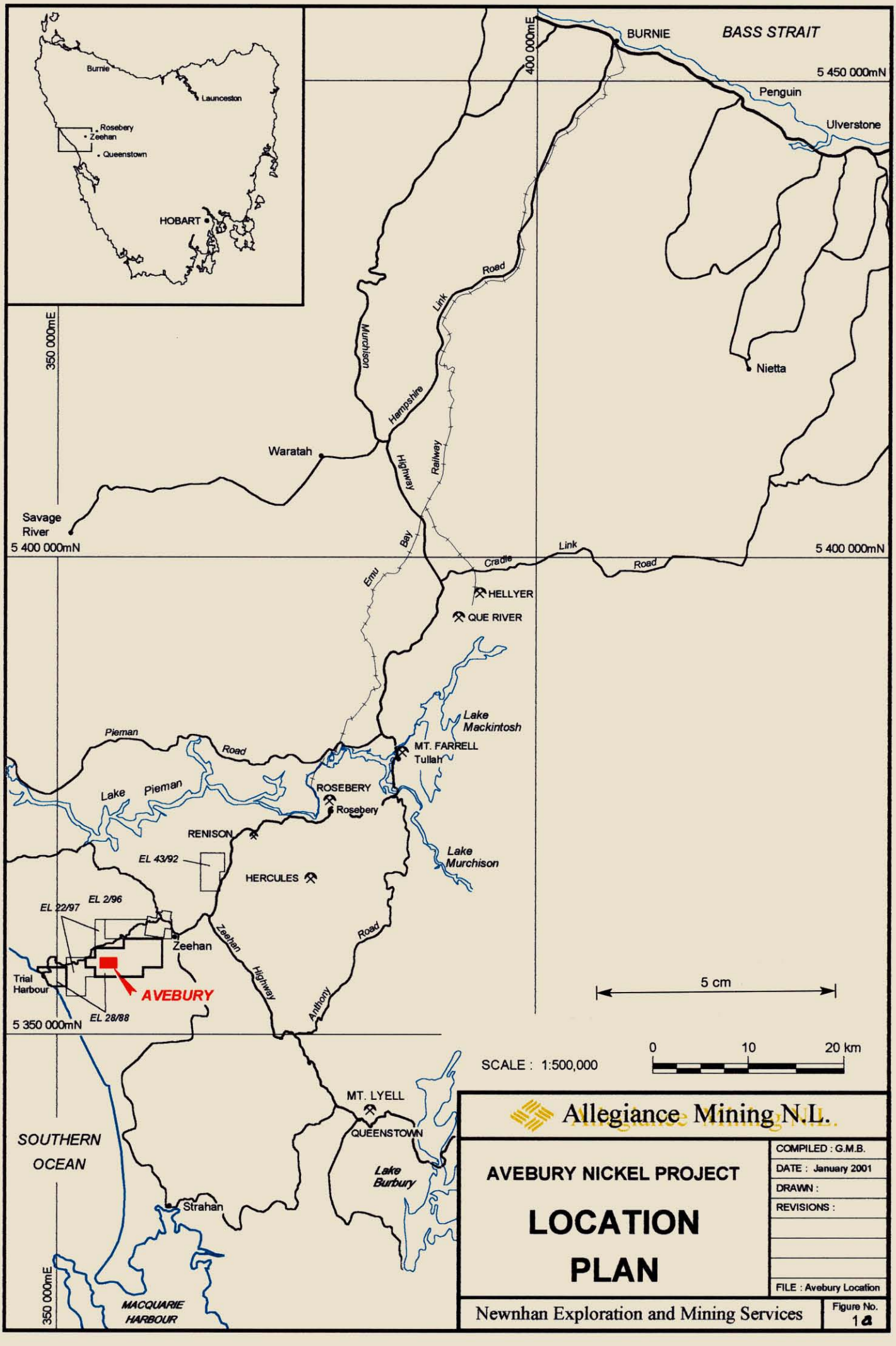

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

LOCATION PLAN

COMPILED : G.M.B.	
DATE : January 2001	
DRAWN :	
REVISIONS :	
FILE : Avebury Location	

Newnhan Exploration and Mining Services

Figure No. 1a

3. TENEMENTS

Allegiance Mining NL holds 100% of the following exploration licences in the Zeehan area, each regarded as having potential for discovery of additional nickel sulfide resources:

EL 28/1988	-	Avebury	29 sq km
EL 2/1996	-	Tenth Legion	8 sq km
EL 22/1997	-	Trial Harbour	14 sq km
EL 43/1992	-	Melba Flats	8 sq km

Rio Tinto Exploration Pty Limited holds a 2% net smelter royalty on production from these tenements.

Tenement locations are shown on Fig 1(b).

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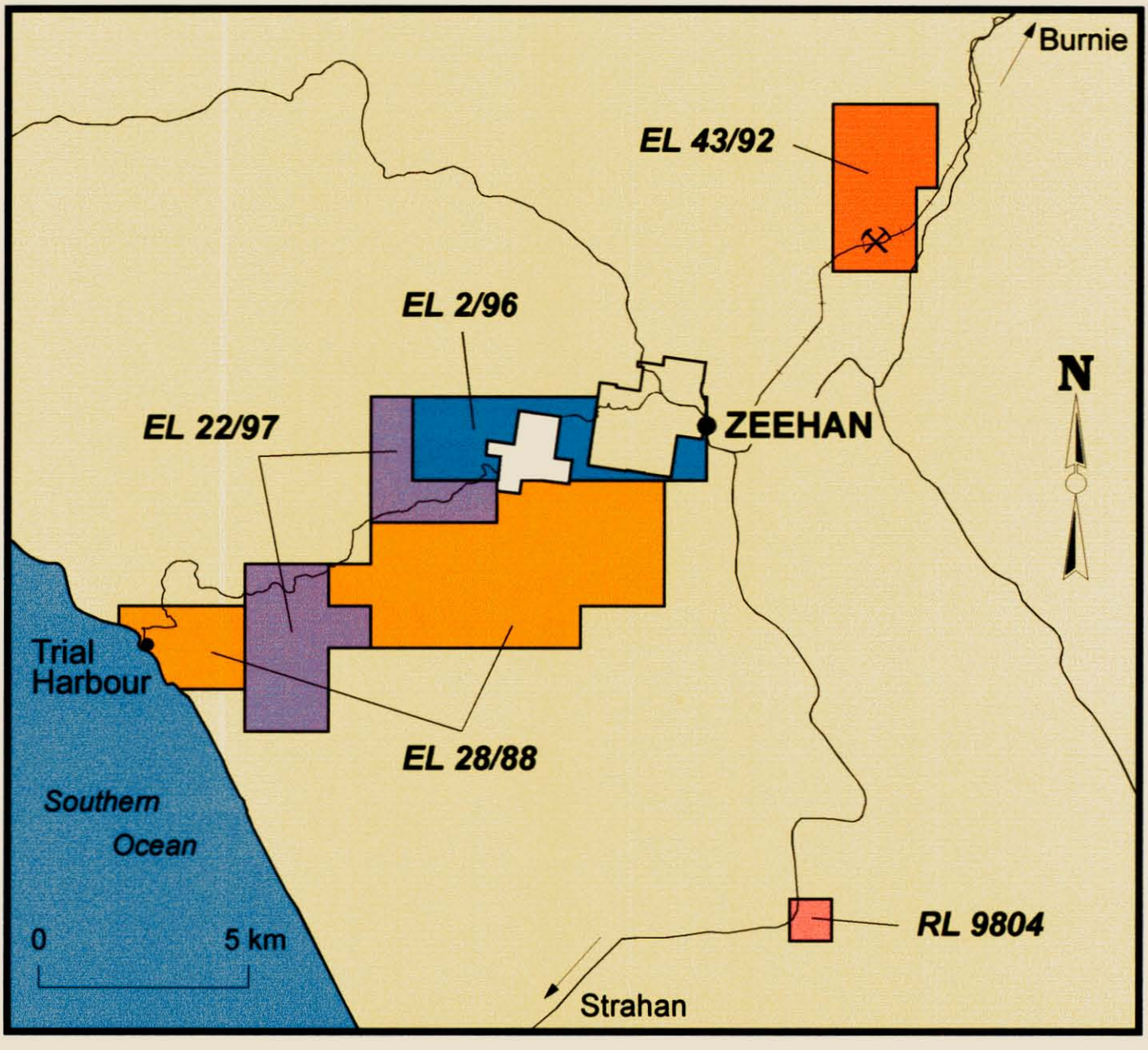
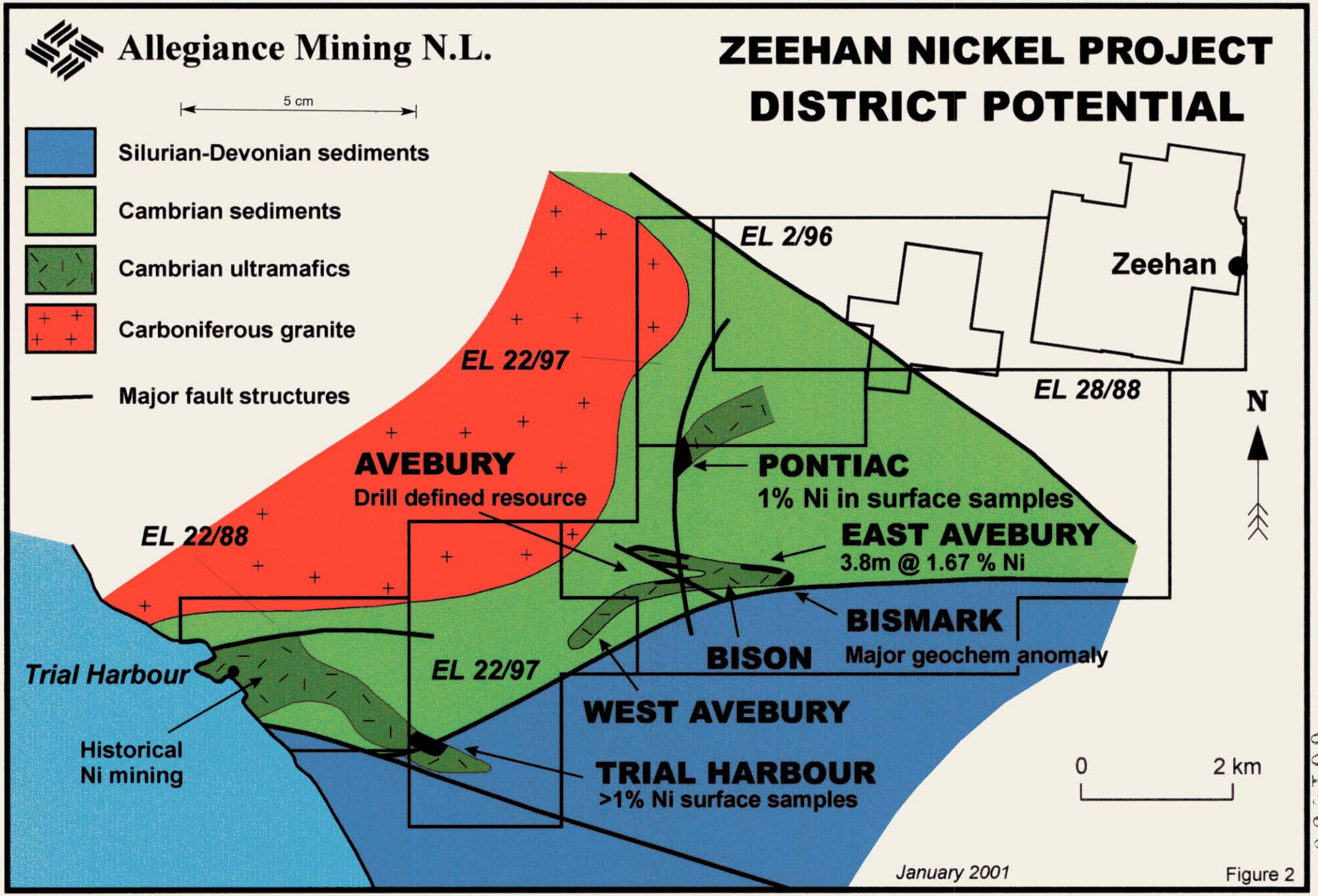


Allegiance Mining N.L.

Western Tasmania Projects - Tenement Locations

January 2001

Figure 1 b


Allegiance Mining N.L.
ZEEHAN NICKEL PROJECT
DISTRICT POTENTIAL
5 cm
Silurian-Devonian sediments
Cambrian sediments
Cambrian ultramafics
Carboniferous granite
Major fault structures
EL 2/96
Zeehan
EL 28/88
EL 22/97
AVEBURY
Drill defined resource
PONTIAC
1% Ni in surface samples
EAST AVEBURY
3.8m @ 1.67 % Ni
EL 22/88
BISMARK
Major geochem anomaly
BISON
EL 22/97
Trial Harbour
WEST AVEBURY
Historical
Ni mining
TRIAL HARBOUR
>1% Ni surface samples
N
0
2 km
January 2001
Figure 2

4. AVEBURY RESOURCE

At Avebury, Allegiance has drill-defined an identified mineral resource of 3.1 Mt 1.7% Ni (see Resource Report by MV McKeown). Potential exists for extensions of this resource.

4.1 Identified Resource:

A total of 10,000 m of core drilling in 26 drill holes at Avebury has identified two adjacent deposits of nickel sulfide.

McKeown has estimated an identified mineral resource as:

	Tonnes	% Ni
North Avebury (Indicated)	990,000	1.74
South Avebury (Inferred)	2,110,000	1.73
TOTAL	**3,100,000**	**1.73**

Details of this estimate are contained in the report:

> *"Allegiance Mining NL Avebury Nickel Project. Mineral Resource Report, November 2000"* by Michael V McKeown for Allegiance Mining NL

A plan location and longitudinal projection of the two resource areas is attached as Figs 3, 4.

The identified resource has been estimated in accordance with the *"Australasian Code for Reporting of Mineral Resources and Ore Reserves (the JORC Code)".*

Substantial further drilling as detailed in the McKeown report is required to elevate the status of this resource, and permit more detailed extraction studies.

4.2 Potential Resources:

In addition to this identified resource, potential exists to define additional resources in three locations:

(i) extensions of North Avebury

(ii) extensions of South Avebury

(iii) Central Avebury

Drilling will be required in each case to evaluate this potential.

4.2.1 North Avebury Extensions:

North Avebury is currently interpreted as terminating to the west against a NNW trending fault. Potential exists for modest resource extensions between the identified resource block and this western fault.

North Avebury is also currently interpreted as consisting of a number of high grade zones plunging to the north-west. Potential exists for extensions of this high grade zone at depth beneath the current resource block; ie, below A012 and A013.

Several drill holes east of the resource block suggest potential for modest extensions in that direction.

4.2.2 South Avebury Extensions:

The geological factors limiting the size of South Avebury are currently not well understood. Potential exists to locate extensions down-dip, east and west of the current resource block.

Surface geochemical and geophysical surveys (IP, EM) suggest modest up-dip extensions are also a possibility. Any such extensions would be advantageous to the project as early stage mill feed.

Drilll hole A016 pointed to a faulted western termination of South Avebury 75 m west of the resource block. This interpretation is supported by geophysical surveys, but surface mapping suggests the ultramafic host rocks continue further west, and modest resource extensions are possible in that direction. Extensions to the east of the resource block beneath A 027 are also possible.

South Avebury appears to be plunging north-west. Substantial potential for resource extensions exists down-plunge of drill holes A014 and A026.

4.2.3 Central Avebury:

Central Avebury is the name given to a zone of mineralisation developed between North and South Avebury.

It has been intersected in six (6) drill holes and identified over 300 m of strike length. It was not included in the resource estimate because some aspects of continuity and interpretation remain to be resolved.

A sectional view of Central Avebury is presented as Fig 5.

Main drill hole intersections to date are:

A001:	9.7 m	1.5% Ni
A018:	8.0 m	1.9% Ni
A020:	7.5 m	0.8% Ni
A023:	1.6 m	1.42% Ni

There is a high probability that, with further drilling, a modest additional resource can be defined in the Central Avebury zone.

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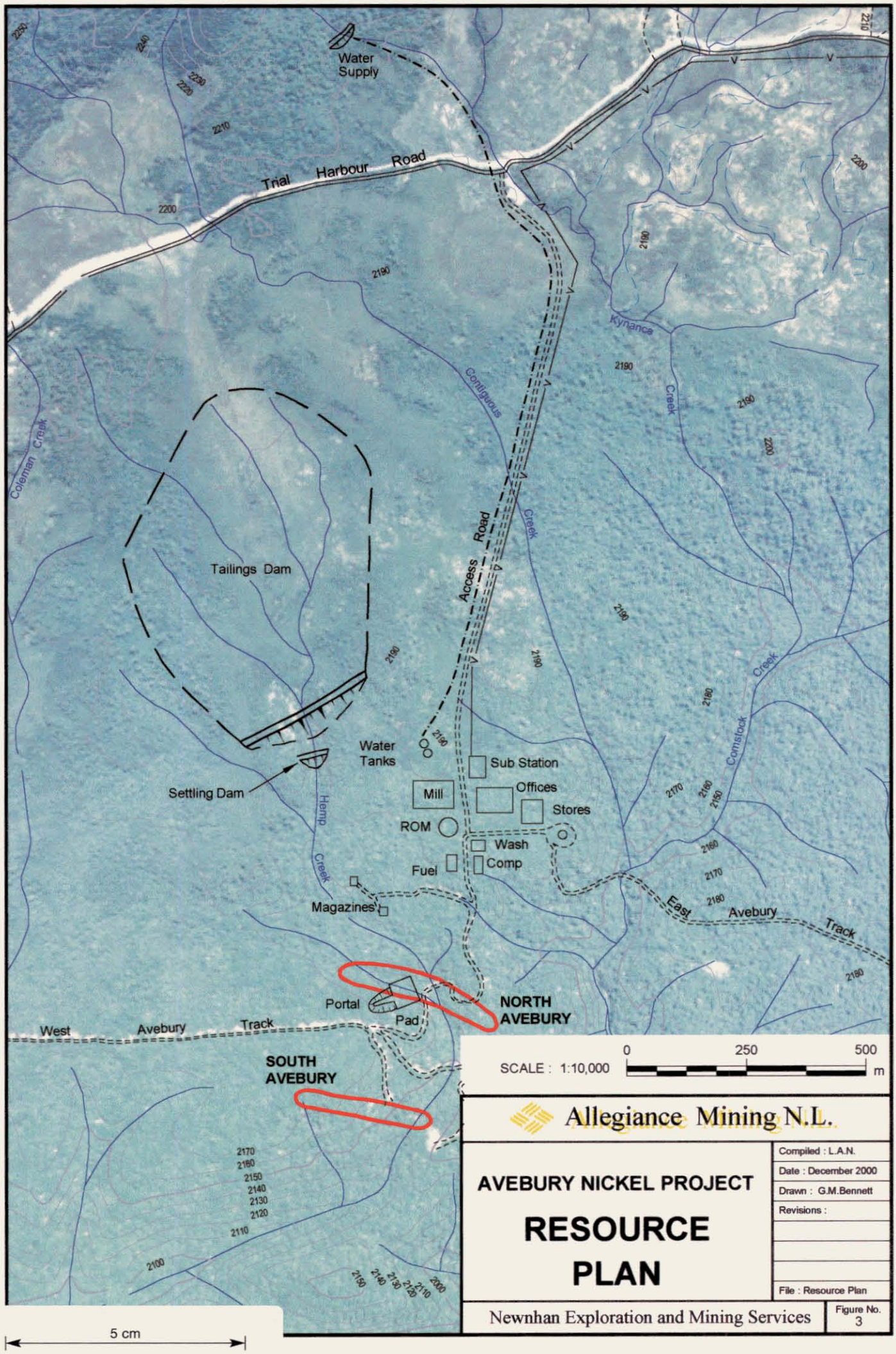
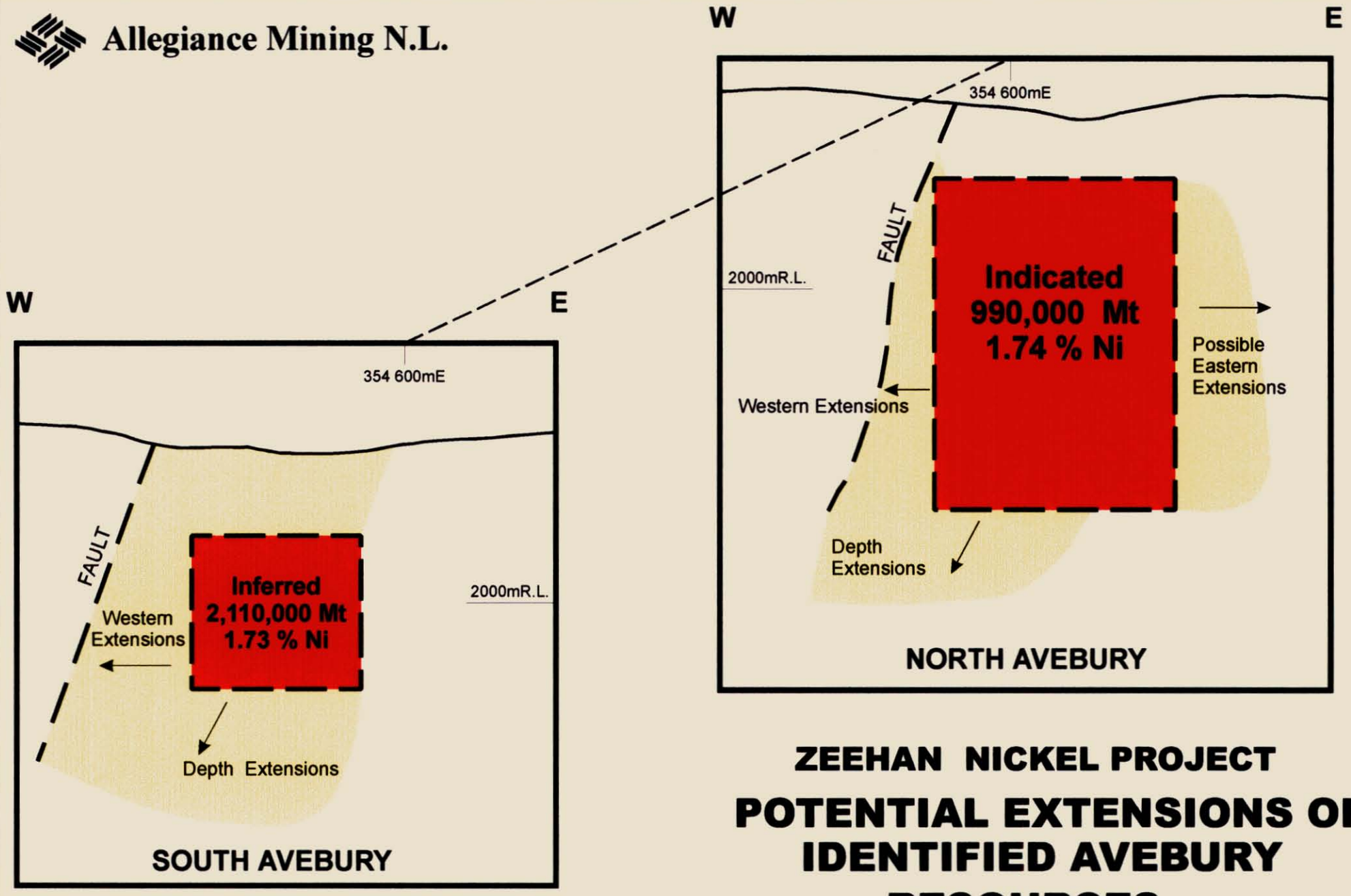

Water Supply
Trial Harbour Road
Kynance Creek
Contiguous Creek
Coleman Creek
Access Road
Tailings Dam
Settling Dam
Water Tanks
Sub Station
Mill
Offices
Stores
ROM
Wash
Comp
Fuel
Magazines
Hemp Creek
Comstock Creek
East Avebury Track
West Avebury Track
Portal
Pad
NORTH AVEBURY
SOUTH AVEBURY
SCALE : 1:10,000
0
250
500
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
RESOURCE PLAN
Compiled : L.A.N.
Date : December 2000
Drawn : G.M.Bennett
Revisions :
File : Resource Plan
Newnhan Exploration and Mining Services
Figure No. 3
5 cm

Allegiance Mining N.L.



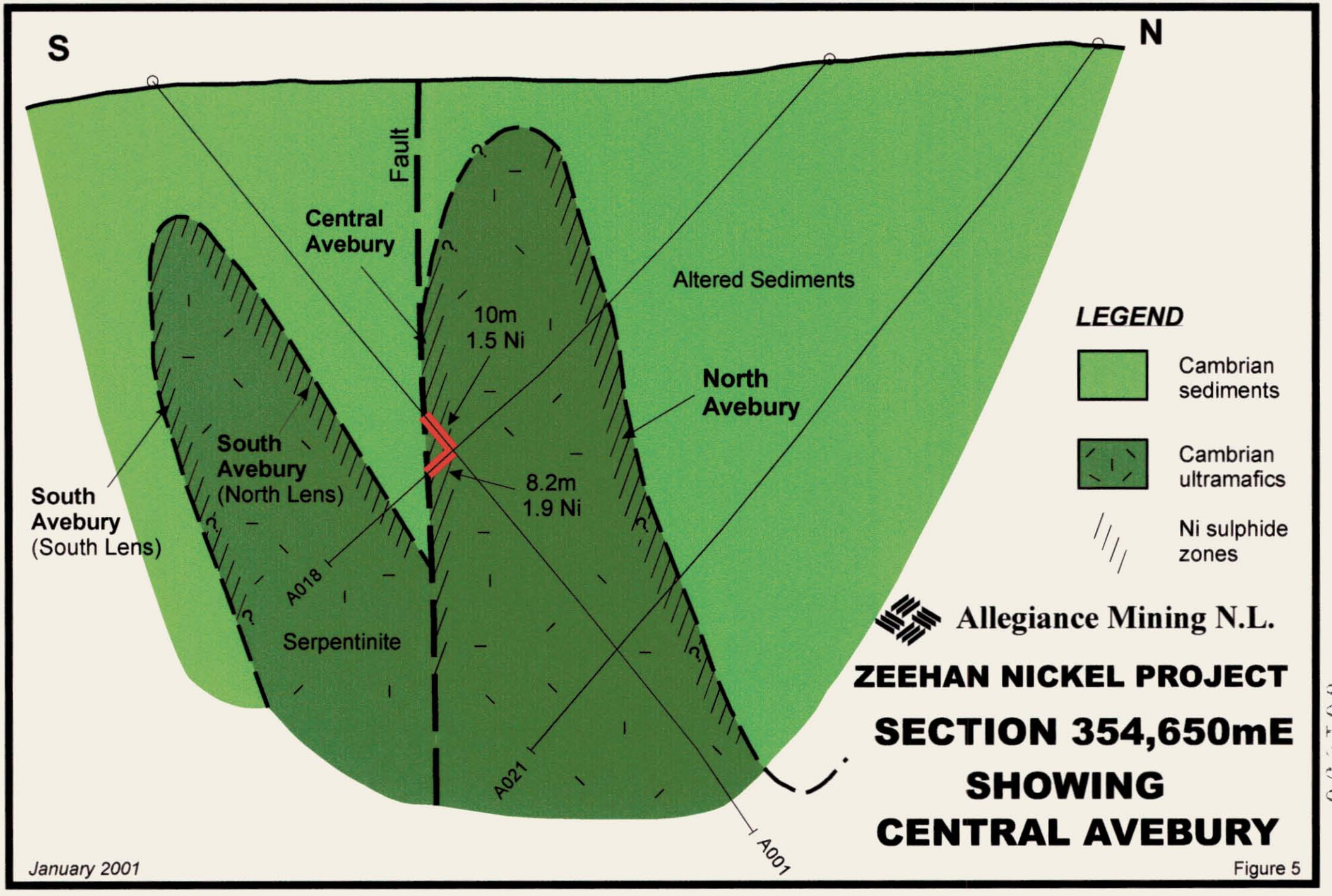
ZEEHAN NICKEL PROJECT

POTENTIAL EXTENSIONS OF IDENTIFIED AVEBURY RESOURCES

January 2001

Figure 4

661799


S
N
Fault
Central
Avebury
Altered Sediments
10m
1.5 Ni
North
Avebury
South
Avebury
(North Lens)
South
Avebury
(South Lens)
8.2m
1.9 Ni
A018
Serpentinite
A021
A001
LEGEND
Cambrian
sediments
Cambrian
ultramafics
Ni sulphide
zones
Allegiance Mining N.L.
ZEEHAN NICKEL PROJECT
SECTION 354,650mE
SHOWING
CENTRAL AVEBURY
January 2001
Figure 5

5. AVEBURY DISTRICT

In the general vicinity of the Avebury resource there are five (5) prospects considered to have potential for additional nickel resources, which could possibly supplement feed to any Avebury-based operation.

These areas are shown on Fig 2, and they are tabulated below, along with a prospectivity rating based on current exploration data.

Prospect	Prospectivity Rating
East Avebury	Very high
Bismark	Very high
Bison	High
West Avebury	Moderate
Pontiac	Moderate

Exploration knowledge of these areas is variable and, as additional data is acquired, their prospectivity ratings may vary.

The Avebury nickel deposits, as currently known, are associated with magnetite. Hence, magnetic surveys have been a useful, early stage exploration tool.

Results of a high resolution aeromagnetic survey flown by Allegiance in 1998 are presented as Fig 6. Avebury and the five prospects (above) are shown.

Detailed ground geological, geochemical and geophysical (IP) surveys have been completed, to compliment the aeromagnetic survey, over East Avebury, Bismark, West Avebury and Bison. Summarised results of these surveys are presented as Figs 7, 8, 9.

Mapping and rock geochemical programs are currently in progress over Pontiac and West Avebury.

5.1 East Avebury:

The East Avebury prospect lies 800 m due east of Avebury.

Its prospectivity is rated as very high and a three (3) hole initial drilling program is due to commence in the area in January 2001.

Work completed to date at East Avebury includes:

- Aeromagnetic survey which showed a strong response analagous to Avebury.
- Geochemical rock and soil sampling programs which defined nickel anomalies similar to Avebury.
- Geophysical IP surveys which defined a moderate anomaly similar to the shallow South Avebury deposit.
- Geological mapping which identified two ultramafic (host rock) units.
- Drilling one (1) cored helicopter supported hole (ZA2) which intersected two ultramafic bodies. The southern body was 30 m thick and contained drill intersections 50-60 m beneath surface of 3.85 m 1.67% Ni, 2.6 m 0.59% Ni, 2.3 m 0.89% Ni, and 1.4 m 0.56% Ni. The northern body was 20 m thick and contained drill intersections 100 m beneath surface of 2.2 m 0.57% Ni, 1.1 m 0.98% Ni.

The East Avebury prospect is very exciting. Any discovery here is likely to be substantial and could have a fundamental impact on development of Avebury.

5.2 Bismark:

The Bismark prospect lies 400 m south-east of East Avebury.

The prospectivity of the area is rated as high, for the following reasons:

- moderate to strong magnetic response
- very strong geochemical response in soils and rocks over a large area
- favourable host rock lithology
- one (1) drill hole (helicopter supported) which intersected a 30 m wide ultramafic unit including 0.4 m 1.09% Ni, and 2 m 0.55% Ni

The geochemical response over Bismark is the highest in the whole Avebury area. No further ground surveys are necessary ahead of further drill testing.

5.3 West Avebury:

Aeromagnetic surveys suggest the host rock lithologies which host Avebury and the structures which influence the mineralisation continue several kilometres to the west. The detailed Avebury grid ground surveys do not extend into this area.

Three (3) drill holes were completed 20 years ago, 1.5 km west of Avebury, but cores were not assayed for nickel.

Ground mapping and rock chip geochemical surveys are currently in progress.

A strong case exists to extend the Avebury grid into West Avebury and complete soil geochemical and electrical geophysical surveys prior to drill testing as warranted.

5.4 Bison:

The Bison prospect is defined by moderate airmag, rock and soil geochemical anomalies co-incident with an ultramafic unit.

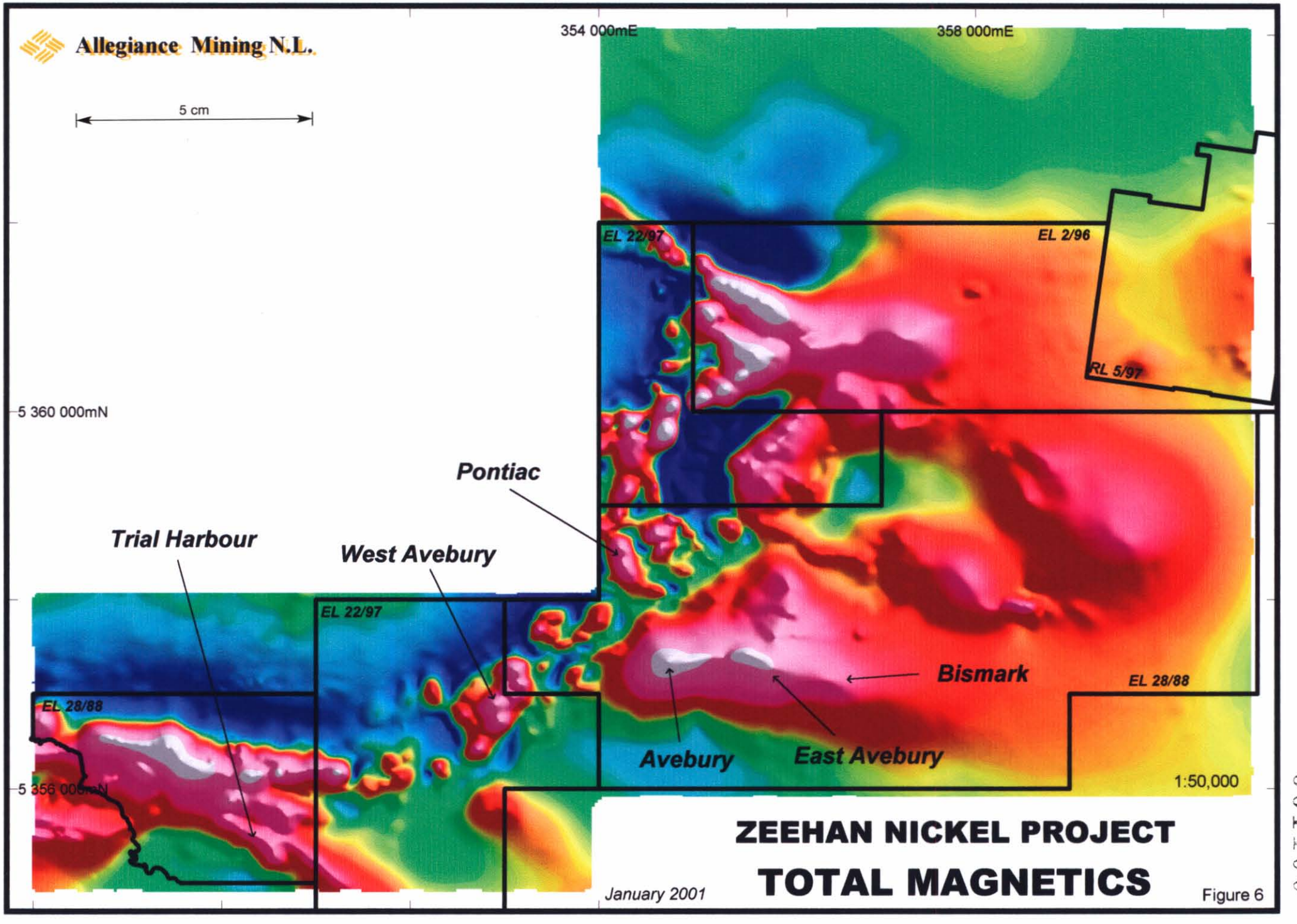
It lies in rugged country 500 m south-east of Avebury and may represent an easterly extension of South Avebury.

Electrical geophysical surveys did not adequately cover this area and it would be prudent to do so prior to drill testing.

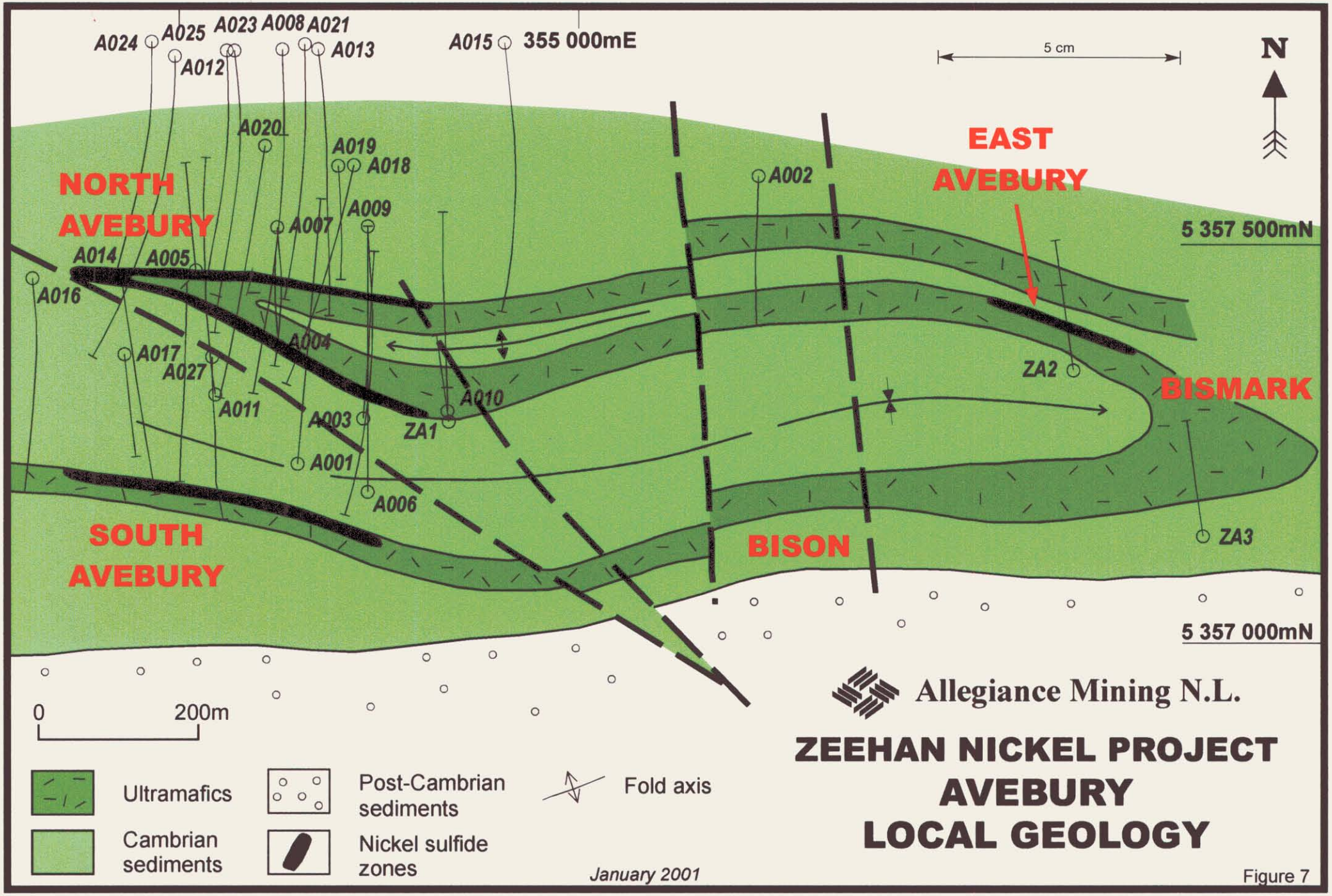
5.5 Pontiac:

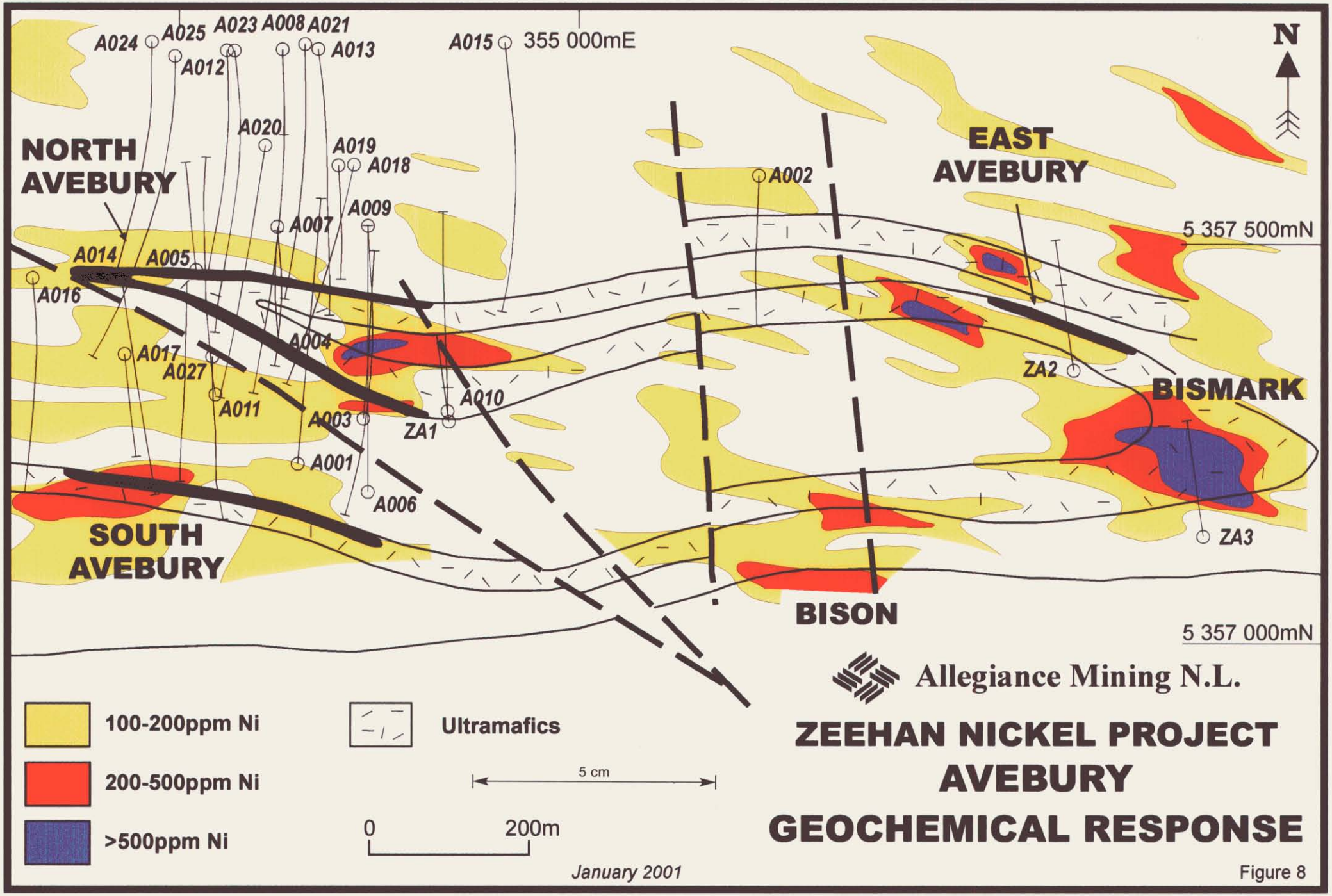
The Pontiac prospect lies north of Avebury and is defined by a series of modest airmag anomalies co-incident with ultramafic units affected by several major fault zones.

The area was recently mapped and geochemically sampled. Depending on the results of those surveys (results available early 2001), there may be justification for extending soil geochemical and ground electrical geophysical surveys into this area prior to possible drilling.

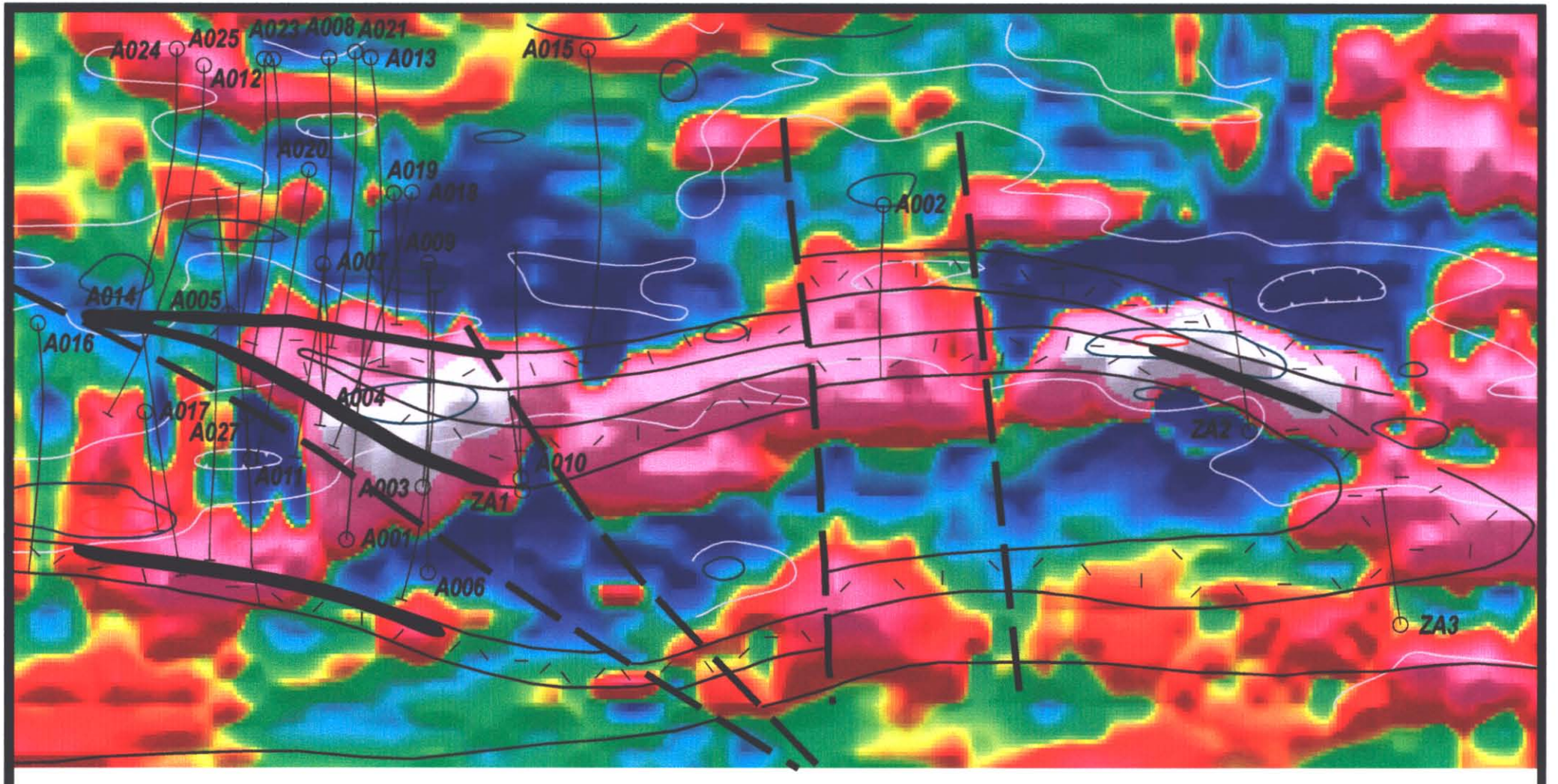

Allegiance Mining N.L.
5 cm
354 000mE
358 000mE
5 360 000mN
5 356 000mN
EL 22/97
EL 2/96
RL 5/97
EL 22/97
EL 28/88
EL 28/88
Pontiac
Trial Harbour
West Avebury
Bismark
Avebury
East Avebury
1:50,000
ZEEHAN NICKEL PROJECT
TOTAL MAGNETICS
January 2001
Figure 6

661100


A024
A025
A023
A008
A021
A012
A013
A015
355 000mE
5 cm
N
A020
A019
A018
NORTH AVEBURY
A009
A007
A014
A005
A016
A017
A027
A004
A011
A003
A001
A006
ZA1
A010
A002
EAST AVEBURY
5 357 500mN
ZA2
BISMARK
ZA3
SOUTH AVEBURY
BISON
5 357 000mN
0
200m
Ultramafics
Cambrian sediments
Post-Cambrian sediments
Nickel sulfide zones
Fold axis
January 2001
Allegiance Mining N.L.
ZEEHAN NICKEL PROJECT
AVEBURY
LOCAL GEOLOGY
Figure 7


A024
A025
A023
A008
A021
A013
A012
A015
355 000mE
N
NORTH AVEBURY
A020
A019
A018
A009
A007
A002
EAST AVEBURY
5 357 500mN
A014
A005
A016
A004
A017
A027
A011
A010
ZA2
BISMARK
A003
ZA1
A001
A006
SOUTH AVEBURY
ZA3
BISON
5 357 000mN
Allegiance Mining N.L.
100-200ppm Ni
200-500ppm Ni
>500ppm Ni
Ultramafics
5 cm
0
200m
ZEEHAN NICKEL PROJECT
AVEBURY
GEOCHEMICAL RESPONSE
January 2001
Figure 8

66-702



Allegiance Mining N.L.

ZEEHAN NICKEL PROJECT

AVEBURY AEROMAGNETIC SURVEY AND I.P. CONTOURS

January 2001

Figure 9

66L103

6. TRIAL HARBOUR

The Trial Harbour prospect lies approximately 4 km west of Avebury (Fig 2).

The geological setting is very similar to Avebury. A Cambrian ultramafic mass lies within the strongly faulted and altered contact aureole of the Heemskirk Granite. Nickel sulfides occur within the northern section of this mass and several small high grade deposits were mined early last century.

Aeromagnetics and geological mapping suggest the ultramafic mass trends south-east (discordantly ?) and the focus of exploration to date by Allegiance has not been the previously mined and explored northern section but, rather, the more remote previously unexplored area south of the Little Henty River.

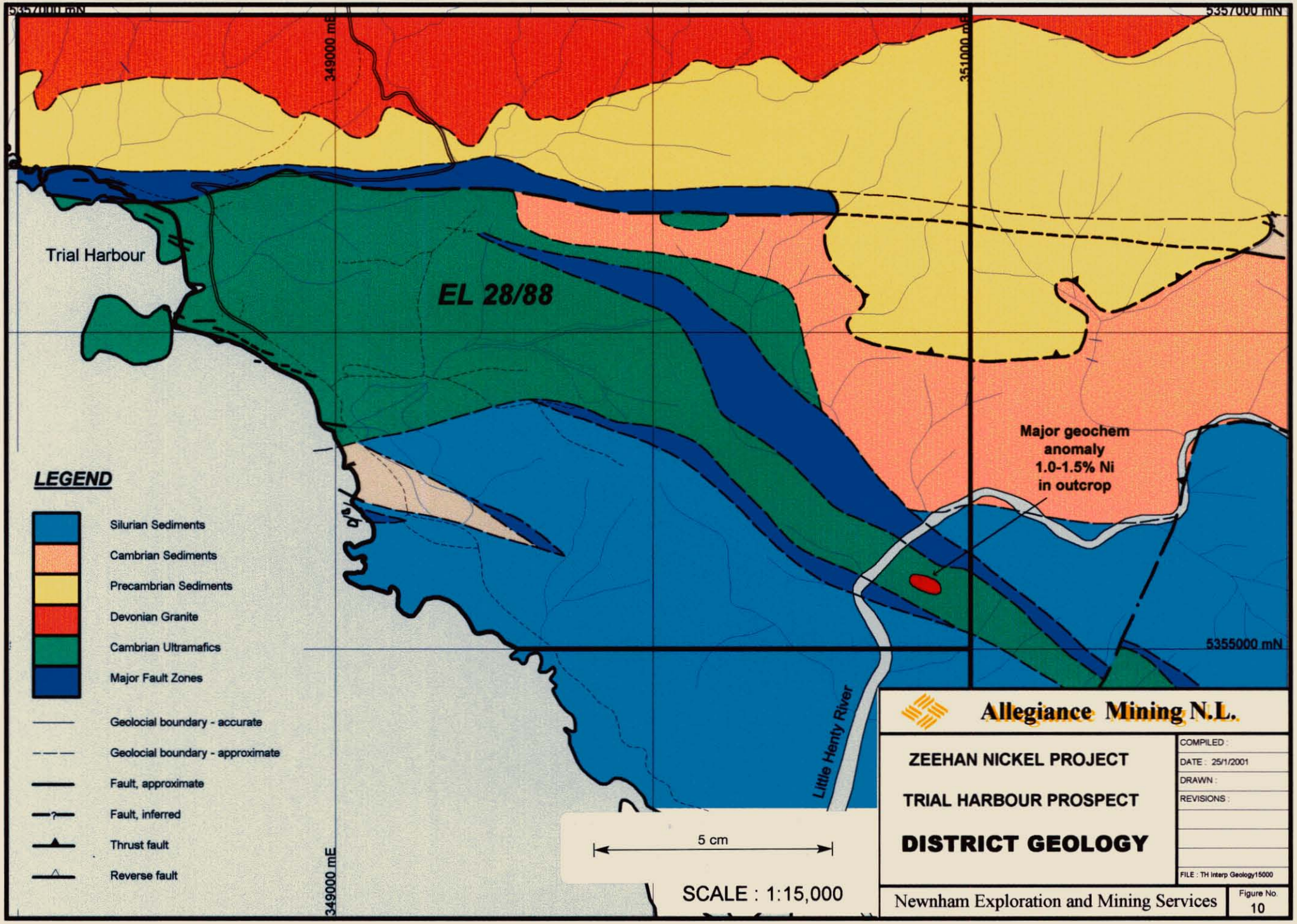
Reconnaissance mapping in this area early in 2000 located several large ironstone outcrops developed on the ultramafics, and initial rock sampling returned values in the 0.5-1.5% Ni range over a substantial area (Fig 10).

A more detailed mapping and sampling program was focused on this anomalous area in November 2000, and confirmed values of 1.0-1.5% Ni in ironstones developed on ultramafics over an area of 200 m x 100 m, near the intersection of two major fault zones.

This geochemical anomaly is larger than any previously acquired on the whole Zeehan Nickel Project area to date, and occurs in an area which has never been subjected to modern exploration evaluation.

The Trial Harbour prospect is assigned a very high prospectivity rating and justifies a significant future exploration effort.

The area is remote and can best be accessed by boat or helicopter. A helicopter supported drilling program is strongly recommended.


5357000 mN
349000 mE
351000 mE
Trial Harbour
EL 28/88
Major geochem anomaly 1.0-1.5% Ni in outcrop
5355000 mN
Little Henty River
LEGEND
Silurian Sediments
Cambrian Sediments
Precambrian Sediments
Devonian Granite
Cambrian Ultramafics
Major Fault Zones
Geolocial boundary - accurate
Geolocial boundary - approximate
Fault, approximate
Fault, inferred
Thrust fault
Reverse fault
5 cm
SCALE : 1:15,000
Allegiance Mining N.L.
ZEEHAN NICKEL PROJECT
TRIAL HARBOUR PROSPECT
DISTRICT GEOLOGY
COMPILED :
DATE : 25/1/2001
DRAWN :
REVISIONS :
FILE : TH interp Geology15000
Newnham Exploration and Mining Services
Figure No. 10

66-10

7. MELBA FLATS

7.1 Background:

The Melba Flats prospect lies seven kilometres north-east of Zeehan and was the site of Australia's first nickel mine early last century (Fig 1).

A three kilometre long swarm of narrow gabbroic dykes have "semi-discordantly" intruded a steeply dipping sequence of Cambrian sediments.

Small pipe-like shoots of high grade Cu-Ni sulfide mineralisation occur within these gabbros. Eight of these were mined to very shallow depths (<50 m) for total estimated production of 6,500 tonnes 9.7% Ni, 4.7% Cu direct shipping ore in the first half of the 20th Century (Figs 11, 12). This high grade mineralisation also carries significant gold, platinum, palladium and cobalt.

Exploration of the area since mining ceased has been limited to inappropriate geophysical programs and very shallow (<50 m) core drilling.

7.2 Exploration Targets:

Allegiance considers the Melba Flats area has potential for two principal deposit styles:

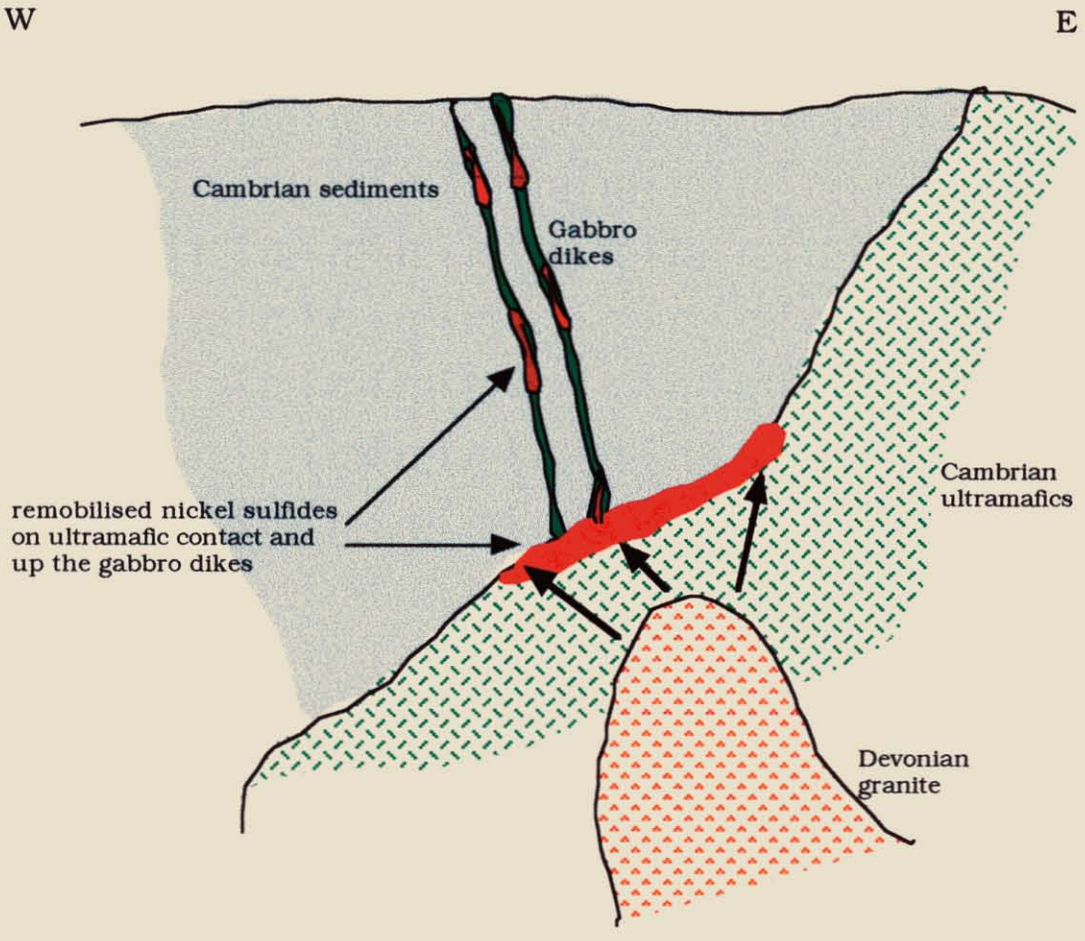
(a) High grade Ni-Cu-Au-Pt-Pd shoots or pipes:

These were the historically mined and explored for deposits of the area. The shoots are typically 0.5-5.0 m wide, 20-50 m strike length, and of generally unknown dip extent. Individually, they have the potential to contain up to 50,000 tonnes.

(b) Larger Ni-Cu deposits hosted by the underlying ultramafics:

The gabbros are viewed as derivatives of the ultramafic bodies which airmag surveys suggest may underlie this area. The ultramafics themselves have probably been intruded and affected by the underlying spine of Devonian granite suggested by gravity surveys as connecting the Heemskirk Granite outcrop to the south-west and Pine Hill outcrop to the north-east.

The potential remobilising effects of such a granitic intrusion on ultramafic hosted nickel sulfides is not unlike Avebury and may have been responsible for driving small amounts of nickel sulfides into the high grade shoots hosted by the gabbro derivatives.



Schematic Section of Melba Flats Genetic Model

Potential exists for larger Cu-Ni sulfide deposits to be developed in the ultramafics within the contact aureole of this granite intrusion.

7.3 Work Completed by Allegiance:

Allegiance has completed the following work:

- High resolution airmag survey. This survey was useful in defining the ultramafic outcrop/subcrop, as shown on Fig 11.

- Collation of previous (shallow) drilling data, with

particular emphasis on the Nickel Reward area where several earlier high grade but very shallow intersections were obtained.

- Two (2) cored drill holes (MF11, MF12) at Nickel Reward. Both holes intersected significant nickel sulfides (see Table following).

- Detailed orientation geophysical surveys over Nickel Reward, designed to quantify the more appropriate survey methods for these styles of deposits. Surveys included surface and down-hole EM and IP surveys.

- Extensions of the Nickel Reward IP survey to the north (Mosquito Grid). Results suggest further mineralised zones may exist in the Vaudeau Mine area.

- Completion of a third hole (MF13) at Nickel Reward. A narrow high grade intersection (see Table) suggests a steep south-east plunge on the Nickel Reward shoot to in excess of 100 m depth, as shown on the longitudinal projection at Fig 13.

7.4 Implications for Avebury Development:

If a decision was made to develop Avebury, the concurrent development of Melba Flats deposits could be important. On the basis of current knowledge, the following points are relevant:

- Melba Flats tonnages are relatively small, but high grade. Individual shoots may only contain 20,000-50,000 t and production rates would not be high.

- Melba Flats mineralisation is relatively shallow and production could be obtained very quickly, initially by small open-cuts.

- Avebury and Melba Flats are both sulfide deposits, but the mineralogy is very different. Avebury is essentially a pentlandite deposit with minor pyrrhotite, pyrite and nickel arsenides. Melba Flats is pentlandite-millerite-chalcopyrite-gold-cobalt-PGMs.

- Melba Flats resources may not stand construction of their own mill, but could be readily trucked to an Avebury mill for $6/tonne.

- Melba Flats ore could be treated:

  (a) concurrently with Avebury ore

(b) prior to Avebury ore whilst the Avebury mine was being developed

(c) in a batch mode in the Avebury mill.

Treatment concurrent with Avebury ore may result in the loss of some of the value of Au and PGMs.

7.5 Further Exploration:

Melba Flats is rated as having very high prospectivity for the discovery of both further near surface high grade shoots and deeper, larger tonnage deposits associated with the underlying ultramafics.

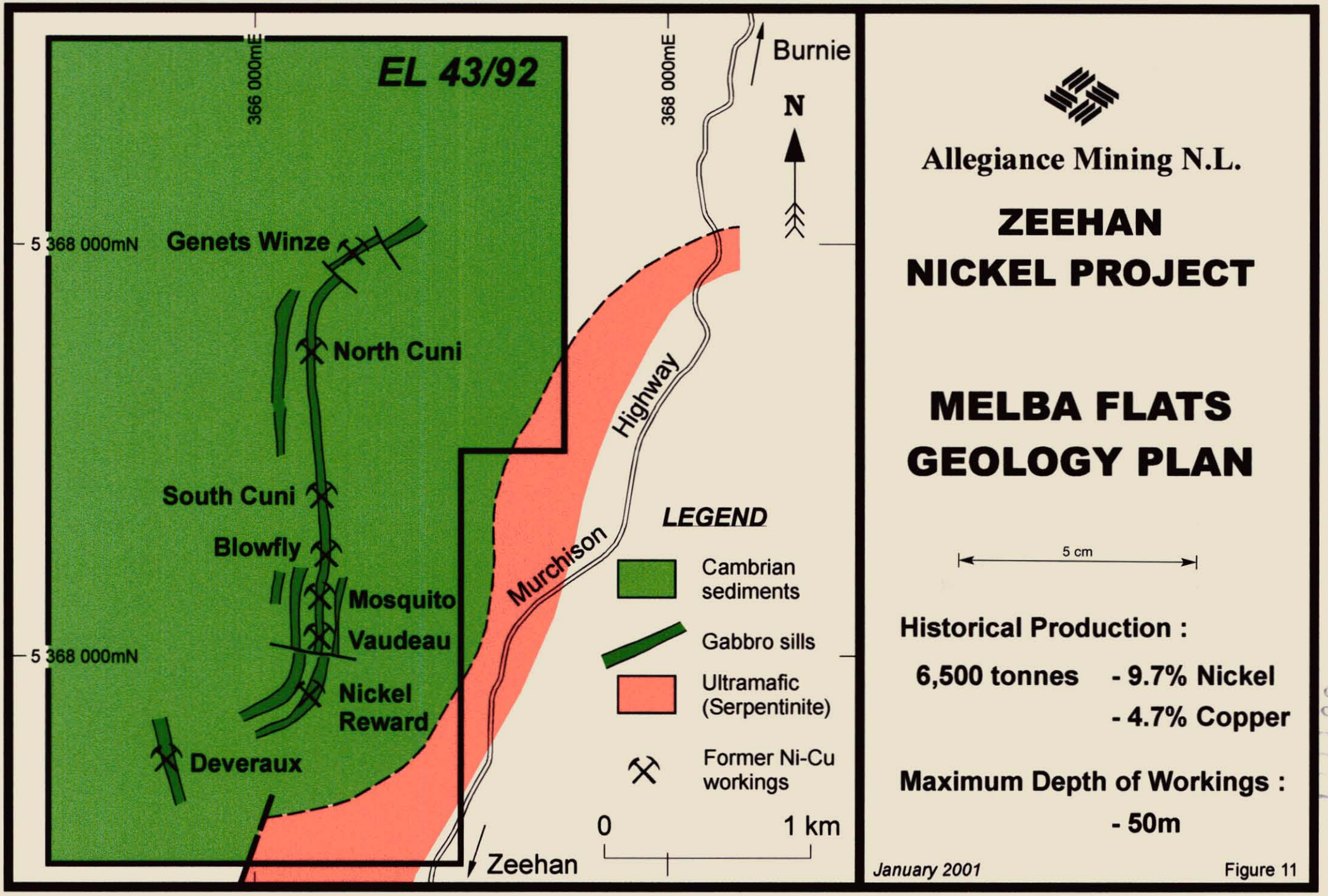
Systematic, sequential exploration should be continued in this area, and the following work is recommended:

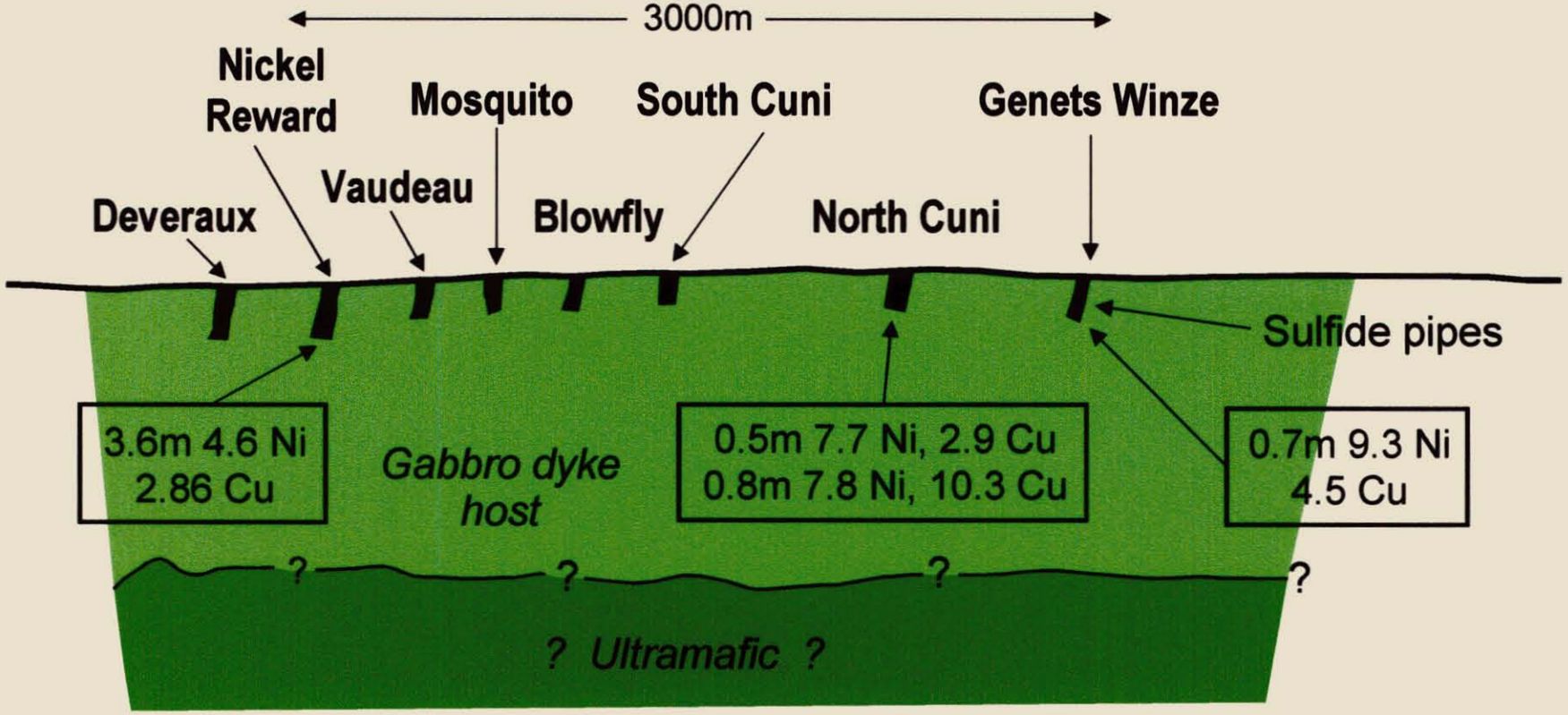
- detailed geological mapping of the licence area (no detailed maps exist)
- continue drilling at Nickel Reward
- extend geophysical surveys to the north
- drill geophysical anomalies to define more high grade shoots
- drill one or two deeper holes to establish the nature of the underlying ultramafic; the first hole should be drilled beneath Nickel Reward where airmag data suggests the ultramafics wrap around the gabbro dyke swarm

Location	Drill Hole	Drill Width (m)	% Ni	% Cu	g/t Au	g/t Pt	g/t Pd
North Cuni-Genets Winze	MF01	0.70	9.3	4.5	0.8	0.9	1.4
	MF03	0.50	7.7	2.9	1.1	1.1	1.1
	MF04	0.80	7.8	10.3	1.9	1.2	1.6
	MF05	1.95	0.4	0.6	0.2	0.2	0.3
Nickel Reward	MF11	3.65	4.3	2.8	0.3	0.43	0.52
	MF12	8.10	0.5	0.4	0.08	0.10	0.14
	MF13	1.40	7.26	2.10	0.11	0.23	0.48
	M18*	3.3	1.4	1.7			
	M19*	4.3	2.3	3.1			
	M13*	3.7	7.9	5.0			
	M15*	3.3	10.3	2.8			

* Very shallow 1950s holes drilled at Nickel Reward but exact positions not certain

Table 1: Melba Flats Inventory of High Grade Intersections


EL 43/92
366 000mE
368 000mE
Burnie
N
5 368 000mN
5 368 000mN
Genets Winze
North Cuni
South Cuni
Blowfly
Mosquito
Vaudeau
Nickel
Reward
Deveraux
Murchison
Highway
Zeehan
LEGEND
Cambrian sediments
Gabbro sills
Ultramafic (Serpentinite)
Former Ni-Cu workings
0
1 km
Allegiance Mining N.L.
ZEEHAN
NICKEL PROJECT
MELBA FLATS
GEOLOGY PLAN
5 cm
Historical Production :
6,500 tonnes - 9.7% Nickel
- 4.7% Copper
Maximum Depth of Workings :
- 50m
January 2001
Figure 11



Historical Production :

6,500t @ 9.7% Ni 4.7% Cu
Only minimal drill testing below 50m

5 cm

Allegiance Mining N.L.

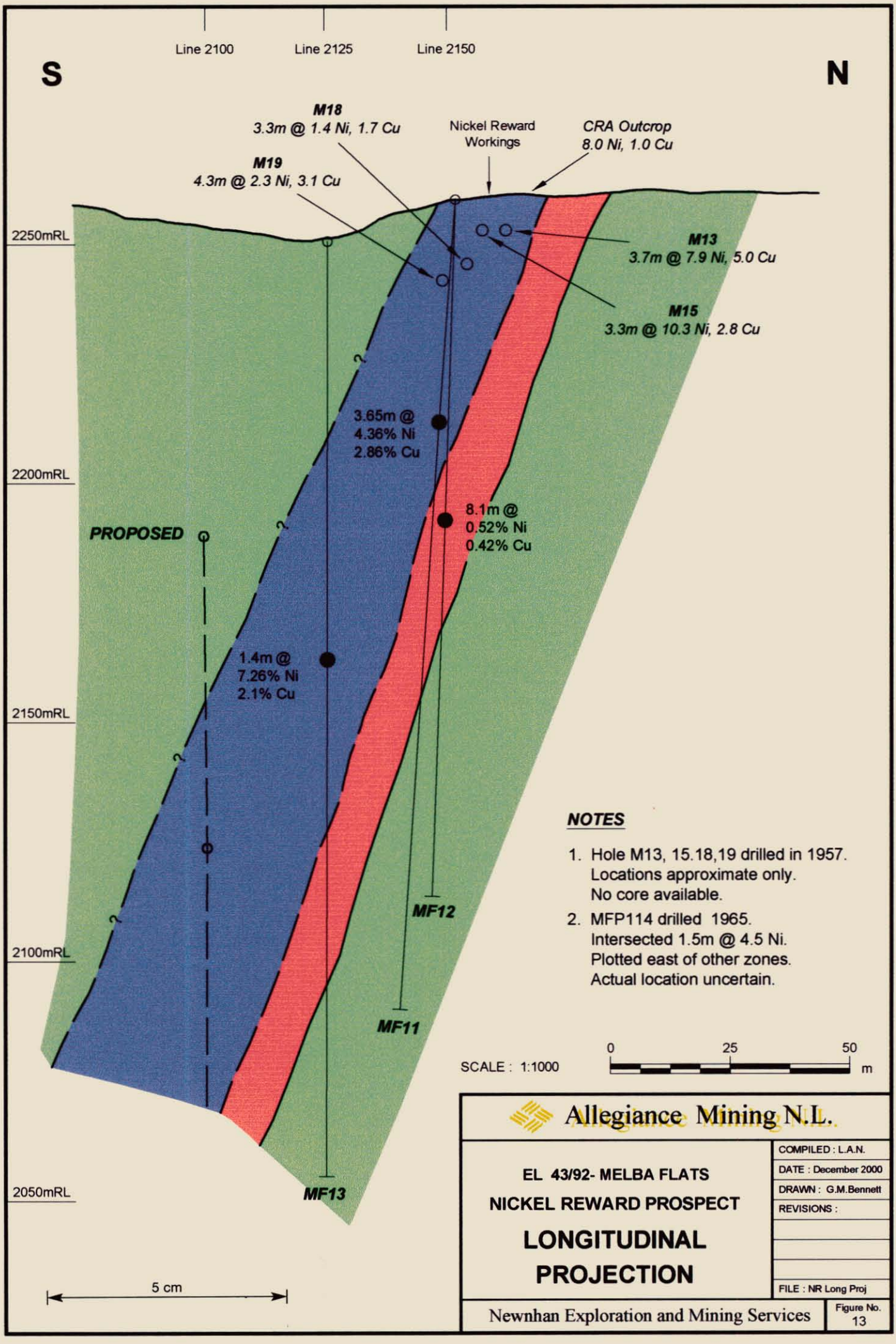
ZEEHAN NICKEL PROJECT
MELBA FLATS
LONGITUDINAL PROJECTION OF SULPHIDE PIPES

January 2001

Figure 12

6671-99

661113



Allegiance Mining N.L.

EL 43/92- MELBA FLATS

NICKEL REWARD PROSPECT

LONGITUDINAL PROJECTION

COMPILED : L.A.N.	
DATE : December 2000	
DRAWN : G.M.Bennett	
REVISIONS :	
FILE : NR Long Proj	

Newnhan Exploration and Mining Services

Figure No. 13

8. RECOMMENDATIONS

Allegiance has identified a significant nickel sulfide resource at Avebury which is the subject of ongoing assessment and commercial evaluation programs.

Within the general Avebury district, the company has also identified a number of prospects with nickel sulfide potential. If additional resources could be identified at one or more of these prospects, it could have a major beneficial impact on the development strategy for Avebury, either in terms of increased output or extended mine life.

It is, therefore, logical that exploration programs should be put in place to quantify this prospect potential. A two-year program is recommended, as this would key-in well with anticipated evaluation work at Avebury.

Each of the prospects is listed below with a recommended program of work.

Avebury Identified Resource Extensions:

The best opportunity for extending the Avebury identified resource in a manner which would substantially affect Avebury development decisions is as extensions of South Avebury to the west, east and down-dip.

A program of drilling should be completed in Year 1. Approximately eight (8) holes for 2,500 m is recommended.

Avebury District:

East Avebury represents an exciting opportunity to discover substantial resources which would affect Avebury development decisions. It is at a drill stage and two programs of drilling, each of 1,000 m, are recommended in Year 1, with possible additional drilling in Year 2 (say, a further 2,000 m).

Bismark should also be further drill tested towards the end of Year 1, with possible further drilling in Year 2.

Bison requires geophysical and geochemical surveys in Year 1, ahead of drilling in Year 2. Pontiac and West Avebury both require ground geophysical and geochemical surveys in Year 1, prior to possible drilling in Year 2.

Trial Harbour:

The nickel geochemical anomaly south of the Little Henty River is an exciting discovery and its prospectivity is rated very high.

It is available for drilling but, because initial drilling would be helicopter supported, this should only be done in summer, in Year 1. Further drilling could be envisaged in Year 2, given initial drilling encouragement A Year 1 program of three (3) holes, totalling 600 m, is recommended, with a Year 2 program of three (3) holes totalling 1,000 m.

Melba Flats:

Further discoveries of high grade shoots at Melba Flats could have a significant impact on Avebury development decisions, whilst discovery of deeper, larger resources could be important in their own right. A Year 1 program of drilling at Nickel Reward (two (2) holes, 400 m) and tenement mapping is recommended, followed by further drilling of high grade shoots and deeper targets in Year 2 (four (4) holes, 1,500 m).

PROSPECT	Prospectivity Rating	YEAR 1		YEAR 2	
		Program	Budget ($)	Program	Budget ($)
Avebury Extensions					
- South Avebury	very high	drill 2500 m	300,000		
Avebury district					
- East Avebury	very high	drill 2000 m	250,000	drill 2000 m	250,000
- Bismark	very high	drill 1000 m	130,000	drill 1000 m	120,000
- Bison	high	ground surveys	30,000	drill 500 m	60,000
- Pontiac, West Avebury	moderate	ground surveys	40,000	drill 1000 m	120,000
Trial Harbour	very high	drill 600 m	140,000	drill 1000 m	200,000
Melba Flats	very high	mapping drill 400 m	60,000	drill 1500	150,000
Budget Totals			950,000		900,000

Table 2: Prospect Evaluation Program

.....

PROSPECT	YEAR 1					YEAR 2				
	Mar Qtr	Jun Qtr	Sep Qtr	Dec Qtr	Totals	Mar Qtr	Jun Qtr	Sep Qtr	Dec Qtr	Totals
South Avebury										
- drilling	50,000	200,000	50,000		300,000					
East Avebury										
- drilling	120,000			130,000	250,000	130,000			120,000	250,000
Bismark										
- drilling				130,000	130,000	120,000				120,000
Bison										
- ground surveys				30,000	30,000					
- drilling									60,000	60,000
Pontiac / West Avebury										
- ground surveys				40,000	40,000					
- drilling							120,000			120,000
Trial Harbour										
- drilling				140,000	140,000				200,000	200,000
Melba Flats										
- mapping	20,000				20,000					
drilling			40,000		40,000			150,000		150,000
TOTALS	**190,000**	**200,000**	**90,000**	**470,000**	**950,000**	**250,000**	**120,000**	**150,000**	**380,000**	**900,000**

Table 3: Prospect Evaluation Program - Budget and Schedule

66144

661117 01-4526B (Vol 3 of 10)

Allegiance Mining N.L.

MINERAL RESOURCES
14 FEB 2001
TASMANIA

AVEBURY NICKEL PROJECT

SCOPING STUDY

MICROFILMED
FICHE No.015502-13

INFRASTRUCTURE DEVELOPMENT and SUPPORT SERVICES

01_4526 (B) Vol 3 of 10
Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL*; Newnham Exploration and Mining
Newnham, L.A.
EL28/1988

MINERAL RESOURCES
FILE REF: EL28/88 PT5
16 FEB 2001
DOC. REF:
OFFICER | FOR ACTION | FOR INFO
See folio 61H
RESUBMIT TO | DATE

20 January 2001

Prepared by:
Lindsay Newnham BSc, FAIMM, CPGeo
Newnham Exploration and Mining Services
PO Box 183, Exeter, Tasmania 7275
Ph: (03) 6394 3434 Fax (03) 6394 3435

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2. **INTRODUCTION**
3. **EXISTING INFRASTRUCTURE**
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   - 4.2 Environmental Factors
   - 4.3 Site Selection
5. **ROAD ACCESS**
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7. **Water and Waste Management**
8. **Concentrates Transport**

PREFACE

This report is a component of a comprehensive Scoping Study prepared for Allegiance Mining NL, which addresses the potential development of the Avebury nickel deposit in Tasmania.

The Scoping Study has four principal objectives:

- conceptualise a style of operation which might be developed on the deposit, based on current technical knowledge
- determine if there is commercial merit in advancing the project further towards a final Feasibility Study stage
- identify additional technical data requirements and other issues to be addressed in order to advance the project to completion of a final feasibility study
- estimate the cost of both acquiring the additional necessary information and completing a final feasibility study

Guided by these study objectives, this report deals with major infrastructure issues and cost estimates associated with the development of a conceptualised operation based on existing resource estimates and other technical information.

This infrastructure proposal should not be viewed as a final or preferred option. As additional future information is acquired on the project, the infrastructure components will change.

.....

1. SUMMARY

Allegiance Mining NL is completing a Scoping Study on the potential development of a nickel sulfide deposit near Zeehan in western Tasmania.

The conceptualised plan is to underground mine the resource at a rate of 300,000 tpa over a 10-year mine life, concentrating the nickel in a conventional nickel sulfide mill, and sell 21,000 wet tonnes concentrate per annum, containing 4,200 t of nickel.

This infrastructure report, completed at a Scoping Study level, has addressed the provision of access, power, a suitable site and various water and waste management issues associated with such an operation.

All infrastructure issues appear to be straightforward and no significant development impediments have been identified.

The capital estimate for infrastructure development is $4.3M. The operating cost estimate for maintenance and improvements of infrastructure is $0.80/t ore.

The capital works should be undertaken in two summer campaigns. The first would facilitate an early start to a 12-month period of underground mine development and the second would facilitate construction of the mill, tailings dam and site buildings.

The schedule assumes all permitting would be in place prior to the commencement of the first campaign.

As part of the completion of a full Feasibility Study, a team of appropriate engineers would have to be assembled for the purposes of designing, specifying and costing the infrastructure development. This study would take approximately five (5) weeks to complete at an estimated cost of $250,000.

Capital Estimate	$4.3 M
Operating Cost	$0.80/ t ore
Full Feasibility Study Cost	$0.25 M

Infrastructure Cost Estimates

2. INTRODUCTION

Allegiance Mining NL has discovered a nickel sulfide deposit eight kilometres west of Zeehan in western Tasmania (Fig 1).

Initial drilling has identified a probable resource of 3.1 Mt 1.73% Ni.

A conceptual mine plan provides for this resource to be extracted by an underground mining operation extracting 300,000 tpa over a 10-year mine life.

The ore would be treated in an adjacent processing plant, producing 21,000 wet tpa of sulfide concentrate averaging 22% Ni for an annual production of 4,200 t contained nickel.

This concentrate would be exported from Tasmania for further treatment.

Potential exists for Allegiance to define additional resources, which might form the basis of either a longer mine life or a higher annual throughput (or both).

A range of infrastructure developments are required to facilitate and support this envisaged operation.

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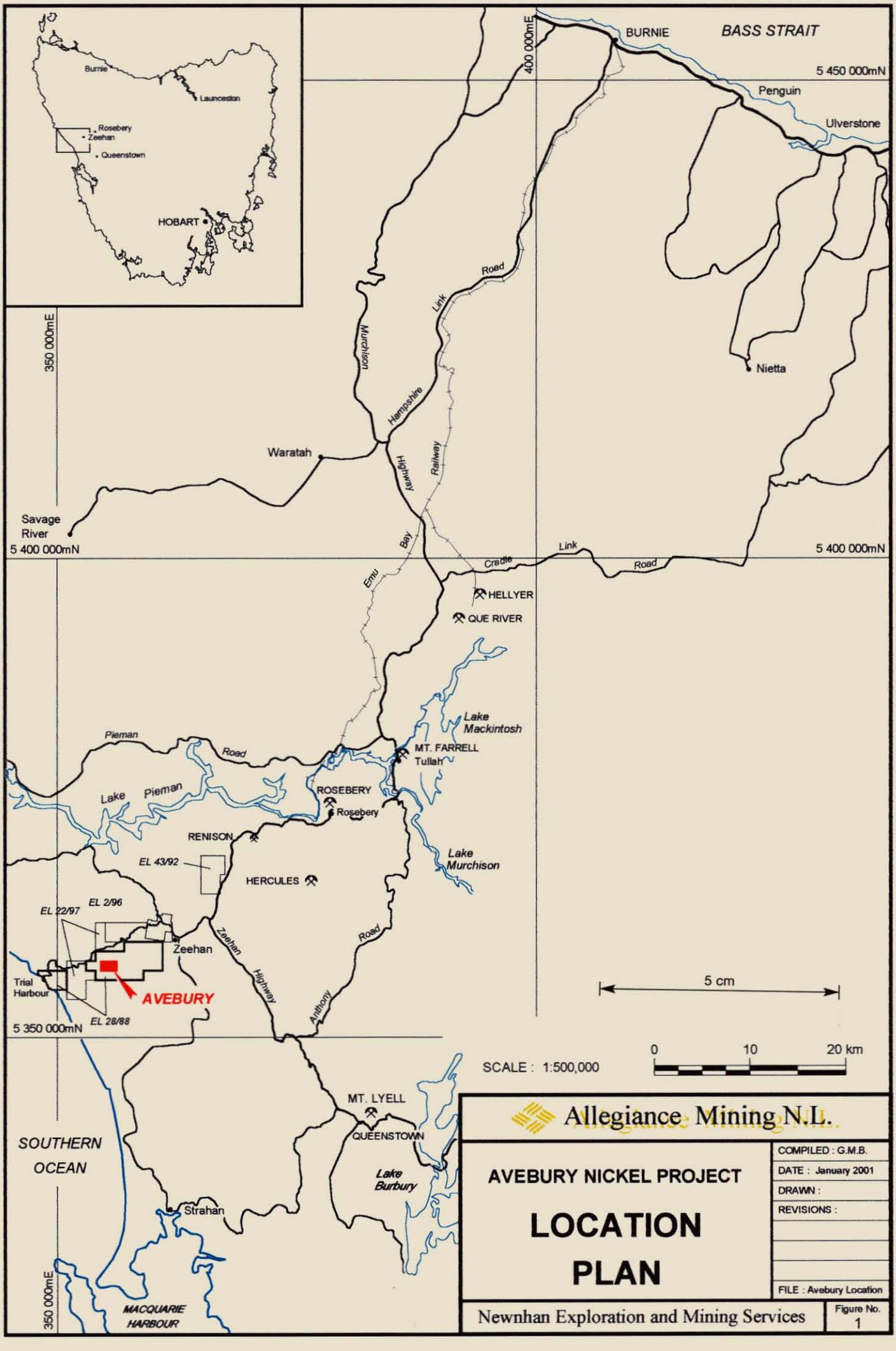


Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

LOCATION PLAN

COMPILED : G.M.B.	
DATE : January 2001	
DRAWN :	
REVISIONS :	
FILE : Avebury Location	

Newnhan Exploration and Mining Services

Figure No. 1

3. EXISTING INFRASTRUCTURE (Fig 2)

The Avebury deposit lies eight kilometres due west of Zeehan township in western Tasmania.

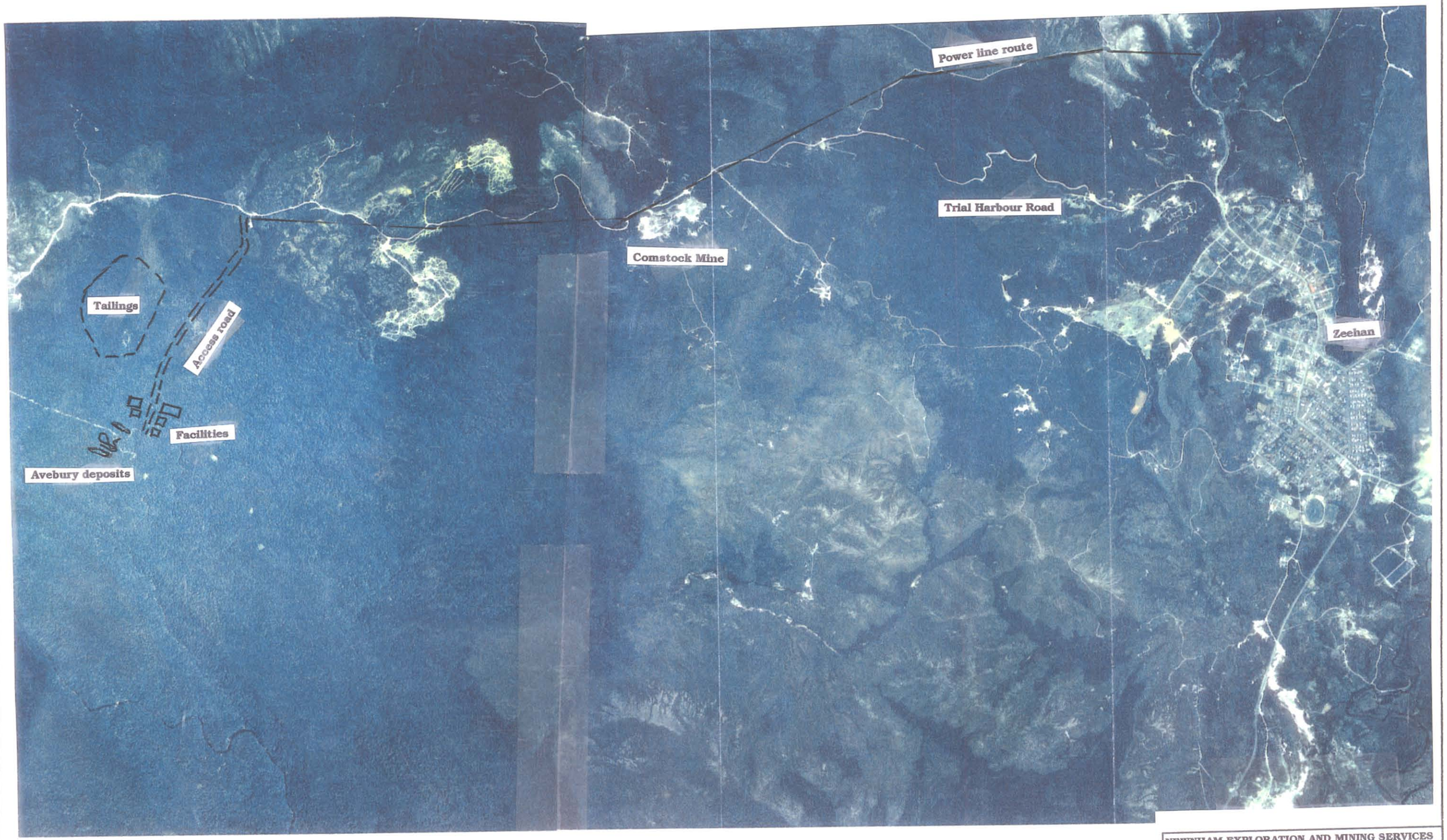
Zeehan is situated approximately 150 km south of the city of Burnie which supports a deep water domestic and international shipping port.

Zeehan is connected to Burnie by the sealed Murchison Highway, and a private railway from Burnie terminates at Melba Flats, 10 km north of Zeehan.

Zeehan is a well serviced township with a mining tradition. It is the principal support centre for the nearby Renison Mine. Current population is 1,100 but, until recently, the population was approximately 2,000. There is, therefore, an ability for the town to readily absorb an increased population from the possible development of Avebury.

Current access to Avebury is via 10 km of the road from Zeehan to Trial Harbour, then two kilometres of exploration access roading.

661125



5 cm

0 2 km.

N

NEWNHAM EXPLORATION AND MINING SERVICES

ALLEGIANCE MINING NL
AVEBURY NICKEL PROJECT
EXISTING and PROPOSED INFRASTRUCTURE

		Scale: 1 : 25,000
Drawn: LAN	Date: Jan 2001	Fig 2

4. SITE SELECTION

4.1 Geography:

The location of the Avebury resource is shown relative to Zeehan on Fig 2. The resource area is covered by a mixture of cool temperate rainforest, wet eucalypt forest and open "button grass" plains. Parts of the rainforest have previously been logged, and some areas have been affected by bushfires in the past 20-30 years. Rainfall is approximately 2,500 mm pa, falling mainly in May-October.

North of the deposits, the area is relatively flat and covered by open button-grass, juvenile eucalypts, and post-bushfire regeneration scrub.

South of the deposits, the area is more rugged and covered by rainforest.

The flatter northern area is ideal for site facilities and support infrastructure, whilst the more rugged southern area presents good opportunities for cost effective underground decline access development.

4.2 Environmental Factors:

A key factor affecting infrastructure site selection was the Company's stated objective of minimising environmental impacts, both short and long term.

Infrastructure development proposals were thus strongly influenced by the following considerations:

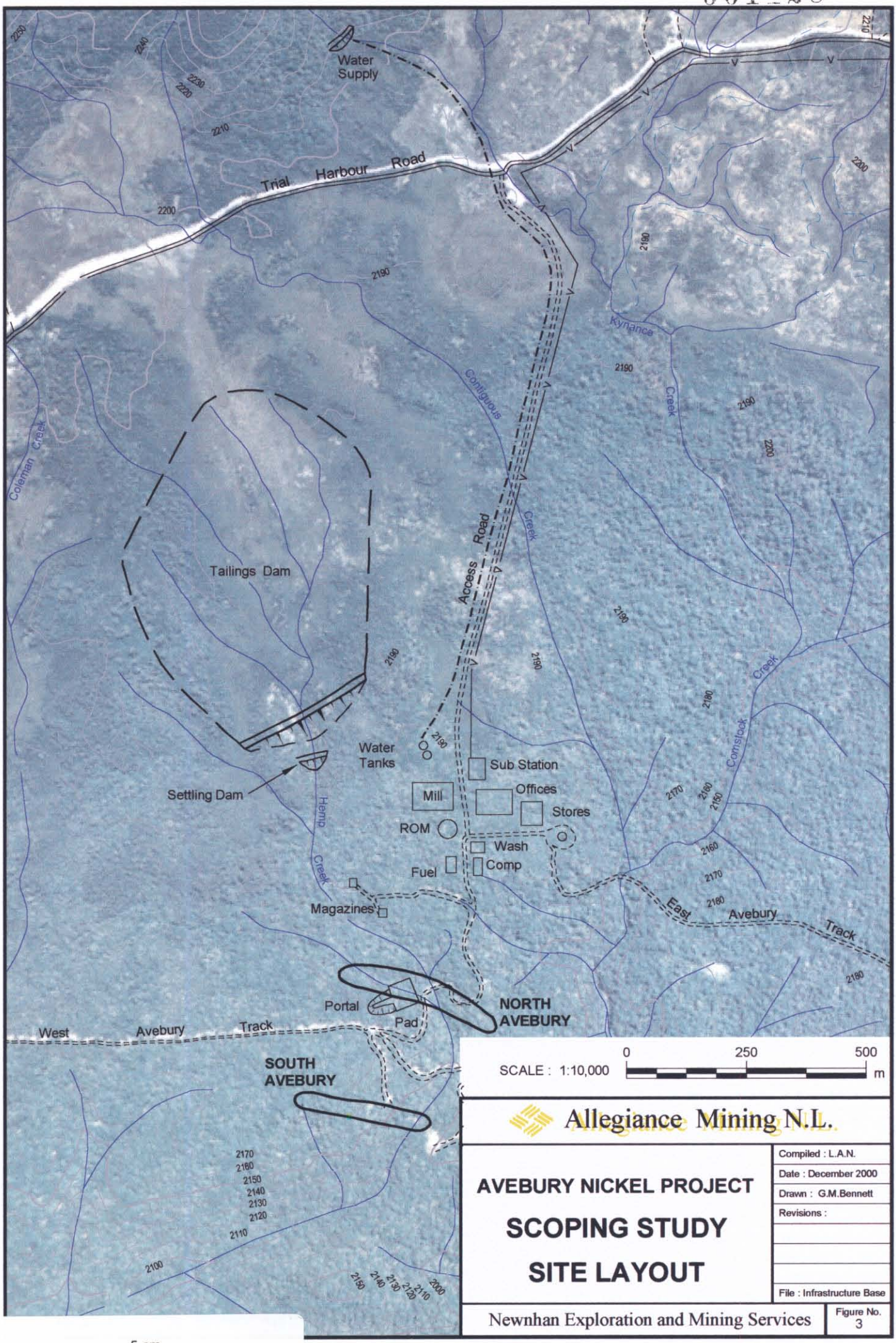
- minimise the footprint area
- minimise the amount of water leaving the site by appropriate water management schemes
- minimise visual impacts
- maximise the beneficial use of any waste products, such as development waste and tailings
- workforce would not live on site; ie, they would live in Zeehan or elsewhere

4.3 Site Selection:

The site recommended for infrastructure development is shown on Fig 2 and in more detail on Fig 3.

The total area affected is approximately 100 ha. This area lies between the Trial Harbour Road and the currently known resources. It is a relatively flat area largely non-visible from the Trial Harbour Road and covered by button-grass, scattered eucalypt and post-bushfire regrowth.

661128


Water Supply
Trial Harbour Road
Kynance Creek
Contiguous Creek
Coleman Creek
Tailings Dam
Access Road
Creek
Water Tanks
Sub Station
Mill
Offices
Stores
ROM
Wash
Comp
Fuel
Settling Dam
Hemp Creek
Magazines
Comstock Creek
East Avebury Track
Portal
Pad
NORTH AVEBURY
West Avebury Track
SOUTH AVEBURY
SCALE : 1:10,000
0
250
500
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
SCOPING STUDY
SITE LAYOUT
Compiled : L.A.N.
Date : December 2000
Drawn : G.M.Bennett
Revisions :
File : Infrastructure Base
Newnhan Exploration and Mining Services
Figure No. 3
5 cm

5. ROAD ACCESS (Figs 2, 3, 4)

Sealed State highway access exists to Zeehan from Burnie, Hobart and Strahan. No upgrading or modification of any of these roads is required.

The Avebury area lies eight kilometres due west of Zeehan. An examination of topographic maps suggests the best access route to the mine site from Zeehan is:

- eight kilometres along the existing Trial Harbour Road, then
- two kilometres of new road to the mine site

This road access would serve three functions:

(a) pre-production and construction access

(b) daily access during life-of-mine for personnel and goods

(c) concentrates transport

It would thus be required to handle frequent and heavy loads in a safe but efficient manner.

5.1 Trial Harbour Road:

The Trial Harbour Road component is best considered in two sections:

- first 4.5 km to the Comstock Mine
- second 3.5 km to Avebury Mine access road

The **first 4.5 km** to the Comstock Mine has recently been upgraded by widening, removing some corners, drainage and pavement improvement and regrading. The initial 1.5 km of this road is reasonably steep and contains several tight, high gradient curves.

However, it is considered adequate for initial mine development purposes, and no capital is allowed for upgrading.

An allowance should be made as a mine operating cost for annual upgrade and maintenance.

If this section of road is assessed as inadequate for regular heavy traffic, an alternative route may exist to the north along the route of the proposed power transmission route.

The **second 3.5 km** section from the Comstock Mine to the Avebury Mine access road junction is currently inadequate for frequent and heavy traffic. It requires upgrading, including:

- remove corners and reprofile in places
- widen pavement by up to two metres
- improve drainage
- basecourse development and regrading

Basecourse material would be sourced from re-alignment and widening works.

A capital estimate for this section is $255,000.

5.2 Mine Access Road:

Two kilometres of new road would be required from the Trial Harbour Road to the Mine Site, including some internal road development on the Mine Site.

The route selected would be relatively flat and sraight. Initially, it would be constructed to a minimum standard required to commence decline development. Decline development waste rock (non-sulfidic) would then be used (crushed and uncrushed) to upgrade the road to a high quality all-weather road. A small quarry may have to be established along the route for initial construction purposes.

The new road would have a 10 m wide formation, 3.5-4.5 m pavement and appropriate drainage.

A capital estimate for this section is $95,000.

5.4 Cost Estimates, Scheduling:

Trial Harbour Road upgrade:

- excavation & earthworks	100,000
- drainage	30,000

-	pavement	110,000
-	surveying	15,000
		$255,000

Mine Access Road:

-	excavation & earthworks	30,000
-	drainage	15,000
-	pavement (local crushed)	35,000
-	surveying	15,000
		$95,000

Total capital estimate for road access development is $350,000.

Provision should be made for annual maintenance and upgrade of $0.20/t ore = $60,000 pa.

Total operating cost: $0.20/tonne ore

No allowance is assumed for government assistance with road access development. Hopefully, some assistance would be forthcoming for the Trial Harbour Road upgrade.

The road development would take two months to complete and this should be undertaken in the summer months for cost and engineering reasons. Initial road development would be required prior to decline mine development. Final road development could be undertaken concurrently with site development works using decline development waste.

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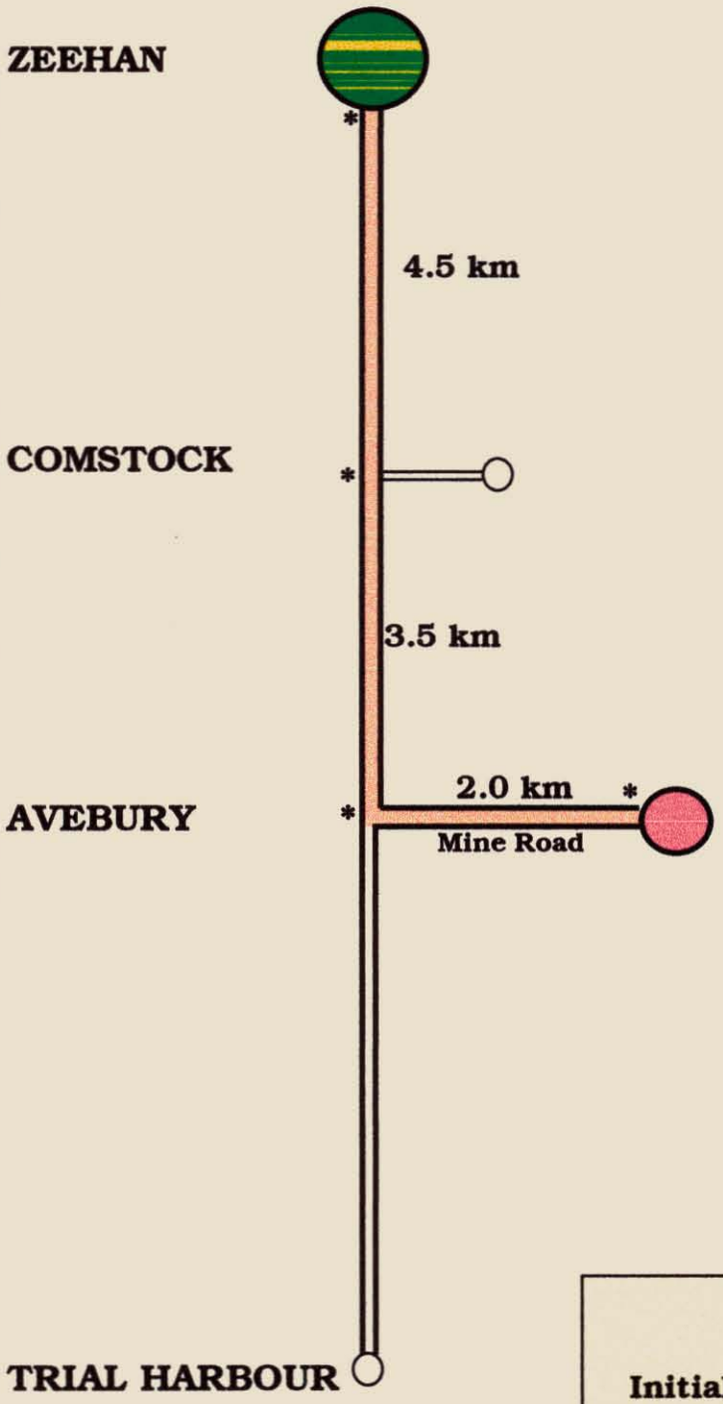

ZEEHAN - COMSTOCK
adequate for construction and early production; annual upgrade and maintenance

COMSTOCK - MINE ROAD
excavation
drainage
pavement $255,000

MINE ROAD
excavation
drainage
pavement $95,000

COST ESTIMATE:

Initial Capital:	$350,000
Annual Maintenance:	$60,000
	= $0.20/tonne ore

5 cm

ALLEGIANCE MINING NL
Avebury Nickel Project
Scoping Study

ROAD ACCESS DEVELOPMENT

Newnham Exploration and Mining Services

Fig. 4

6. ELECTRICITY SUPPLY (Fig 5)

The principal power source on the operation would be electric power provided by Aurora Energy.

Total site power consumption is estimated from other Scoping Study inputs as 4 MW.

6.1 Supply:

Power would be transmitted to Avebury from the Zeehan sub-station at 22 kV. The existing sub-station is adequate for the anticipated demand and Aurora advises that the power is available.

6.2 Line Development:

The transmission line from the State Grid to the Avebury Mine would be constructed by Transend. It would be a 22 kV branch line off the existing 22 kV line from Zeehan to Granville Harbour and Reece Dam. The route would follow an existing fire trail north of Oonah Hill to the Trial Harbour Road, thence along that road and the Avebury Mine road to the Avebury sub-station. Total distance is approximately nine kilometres.

Transend estimates construction costs of $50,000/km. If, at some future time, a branch is taken from this line to the Comstock Mine, or it is extended to service Trial Harbour, a substantial percentage of this capital cost would be refunded to Allegiance.

6.3 Sub-station Development:

An appropriate sub-station will be required on site.

The construction style and transformer equipment in this sub-station should reflect the nature and longevity of the operation. For example, there is little value in developing a 50-year life sub-station for a 10-year life operation.

Various transformers will be required to service the mine, mill and support infrastructure. Careful equipment selection can reduce transformer requirements but it could be anticipated that 11 kV, 3.3 kV, 440 V and 240 V would be required. For a 10-year operation, it should be possible to source good quality secondhand units.

6.4 Power Costs:

Capital:

-	line development to site 9 km @ $50,000/km	450,000
-	sub-station construction	50,000
-	purchase of transformers and site distribution	200,000
	Total Capital:	**$700,000**

Operating Costs:

Power would be supplied by Aurora on their high voltage demand tariff which consists of three charges:

-	fixed charges	100.978 c/day
-	peak demand charge	
	first 250 kW	$79.480/kW pa
	next 250 kW	$64.872/kW pa
	remaining kW	$60.423/kW pa
-	energy rate	
	all kWh	6.119 c/kWh

The power operating costs have been absorbed into other components of this Scoping Study. However, it is important to note the substantial cost benefits to an operation such as Avebury of high quality power management and auditing in order to minimise the peak demand factor.

An overall cost estimate for power on the basis of a 3 MW peak demand and a 2 MW average demand is (very approximately) $3.60/t ore.

6.5 Scheduling:

Power supply to the site is required at an early stage in project development. Whilst initial decline development could be supported with generators, it would be advantageous to have the power supply fully developed as soon as possible after the access road development was completed.

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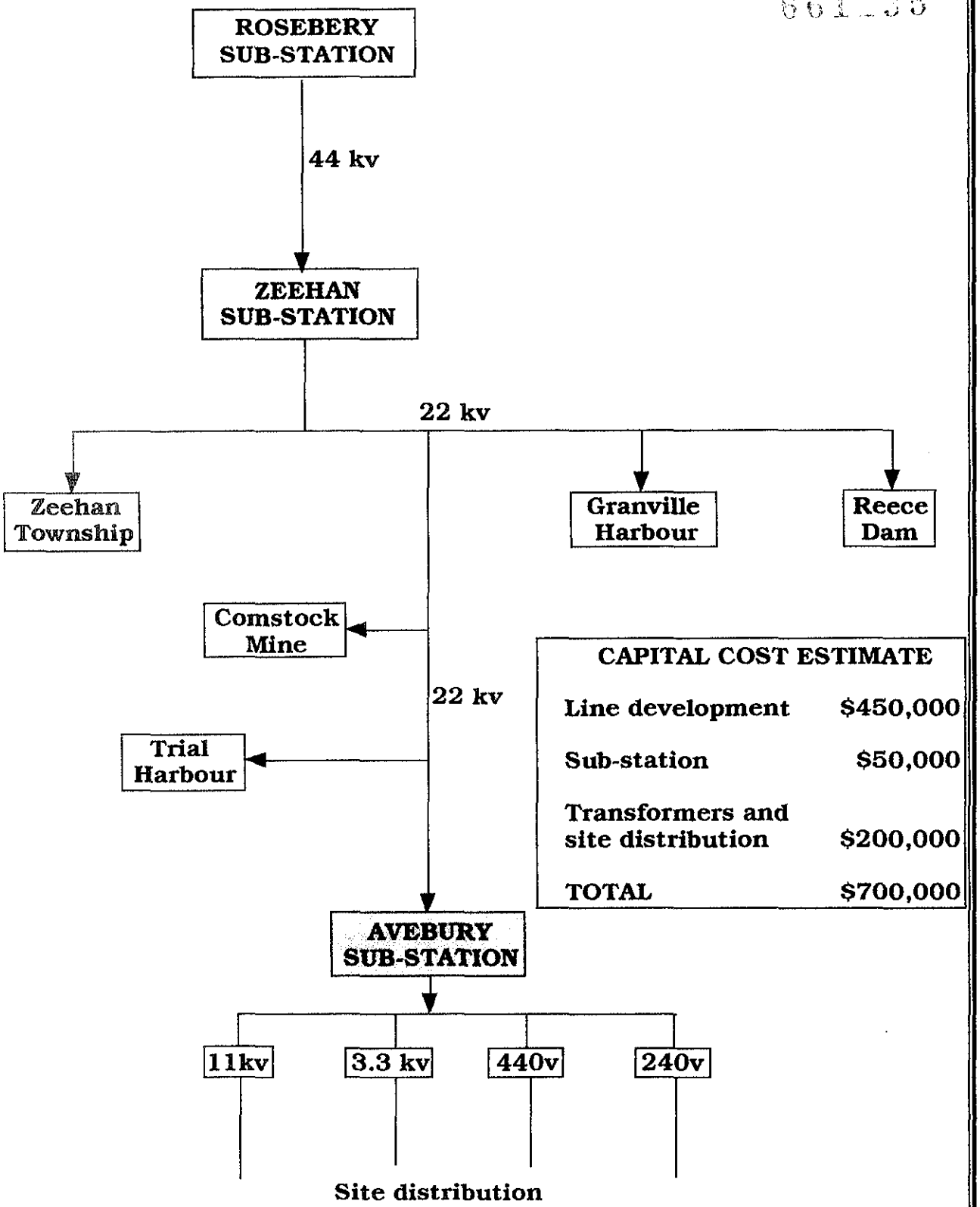


CAPITAL COST ESTIMATE	
Line development	$450,000
Sub-station	$50,000
Transformers and site distribution	$200,000
TOTAL	$700,000

POWER COST

Power supplied by Aurora at Standard High Voltage Tarrif: If peak demand is 3 MW, this equates, for the whole operation, to approx $3.60/t ore

ALLEGIANCE MINING NL
Avebury Nickel Project
Scoping Study

ELECTRICAL POWER SUPPLY

Newnham Exploration and Mining Services

Fig. 5.

7. SITE DEVELOPMENT (Figs 3, 6)

The project site development requires the preparation and servicing of an area sufficient for the construction of a 300,000 tpa mill, offices, changerooms, workshops and associated support facilities.

The site selected and shown in Fig 3 is relatively flat and covered by regrowth vegetation following logging and bushfires. It is underlain at shallow depth by generally hard, competent fresh rock.

Capital estimates to develop this site are divided into three groups:

- site preparation
- buildings
- services

7.1 Site Preparation:

An area of eight hectares would be cleared and prepared. This would involve:

- removing and stockpiling vegetation and soil for later rehabilitation purposes
- placing and rolling non-sulfidic mine development waste to create a solid and even base
- place and roll gravel
- prepare hardstand areas for major construction
- develop drainage

On the basis of $10/sq m this work would cost $800,000.

An amount should be provided for as an operating cost to progressively up-grade and maintain the site during the life-of-mine; eg, sealing. An allowance of $0.20/t ore would provide $60,000 pa for this purpose.

7.2 Buildings:

This item provides for construction of general offices, changerooms, stores, workshops and other minor facilities.

The cost of construction of the mill, specialised mine buildings, magazines are provided for elsewhere. The cost of a fuel installation will be borne by the fuel supplier.

The capital cost of buildings will depend greatly on the nature and style of the establishment. Factors such as new vs second hand, elaborate vs basic, will have a major influence on costs.

Overall floor space as itemised in Fig 6 is 2,500 sq m and, at a cost of $400/sq m, totals $1 M.

An operating cost provision of $0.20/t ore pa would provide $60,000 pa for maintenance and development.

7.3 Services:

Site reticulation of water and power is provided for elsewhere.

However, there are other services which must be provided for, including:

- communications (phone/data facility)
- security
- safety and training facilities
- laboratory for environmental and project control sampling

A cost estimate of these services is $200,000.

7.4 Cost Estimates and Scheduling:

Total site preparation capital cost = $2 M.

Site operating cost = $0.40/t ore pa.

The capital cost would be incurred early in the schedule to take advantage of available decline waste rock, and to be available in a timely fashion for construction of the mill and other infrastructure buildings.

The site operating cost can appropriately be absorbed into the overall administrative budget.

SITE PREPARATION:	8 ha = 80,000 sq.m	
CAPITAL	-clear site, stockpile vegetation, soil	
	- place and roll mine waste	
	- place and roll gravel	
	- prepare hardstand area	
	- drainage and services	
	INITIAL CAPITAL	$800,000
OPERATING	- maintenance - sealing	$0.20/t ore

BUILDINGS:			
CAPITAL	- office	(500 sq.m)	
	- change rooms	(500)	
	- stores	(500)	
	- workshops	(1000)	
	INITIAL CAPITAL		$1,000,000
OPERATING	- maintenance - development		$0.20/t ore

SERVICES:		
	- communications	
	- fuel installation (provided)	
	- security systems	
	CAPITAL	$200,000

TOTAL SITE WORKS CAPITAL	$2,000,000

ALLEGIANCE MINING NL
Avebury Nickel Project
Scoping Study
SITE WORKS
PREPARATION and INSTALLATION

Newnham Exploration and Mining Services

Fig 6

8. WATER and WASTE MANAGEMENT (Fig 7)

Because of Allegiance's requirement to minimise environmental impacts, the operation must give close and high regard to water and waste management.

Water is used in the:

- mine
- mill
- general infrastructure facilities

Water is generated by:

- direct rainfall
- streams in the project area
- ground water from the mine

Waste is generated by:

- mine rock
- mill tailings
- general infrastructure facilities (offices, changerooms)
- surface drainage (siltation)

Basically, waste should be minimised, used and managed in such a way that it has minimum impact on water leaving the site (see Scoping Study Environmental Report for more discussion).

8.1 Water Management:

It is recommended that fresh water be supplied to the operation from a supply dam, or intake pond, created upstream on Contiguous Creek or Comstock Creek. The water quality of the latter would have to be established to ensure it was of potable quality.

This water supply would be piped to header tanks with 200,000 l capacity for distribution to the mine, mill and other surface facilities.

Ground water and waste water from the mine would feed the mill, with any surplus pumped to the tailings dam.

Water from the tailings dam would pass through a settling pond, or wetlands, prior to returning to the mill for discharge.

Potable water used for ablutions, drinking, etc, would be supplied to site facilities as required and waste water from these facilities would be treated in a purpose-built facility prior to discharge.

The capital cost for water management is estimated as:

-	supply dam, pipes, header tanks	80,000
-	settling pond and pump	50,000
-	potable waste treatment	20,000
-	reticulation and siltation traps	50,000
	Total:	$200,000

An operating cost to maintain the above facilities of $0.20/t ore is estimated.

8.2 Mine Waste Rock Management:

With careful planning most mine waste rock will be non-sulfidic.

It can be used in various beneficial ways:

- for road construction and site works
- tailings dam construction
- mine back-fill

Waste developed in the early stages of the operation will probably be fully used during the construction stage.

Surplus waste can be stockpiled for either later mine fill or extensions to the tailings dam. Any sulfidic mine waste would be stored separately to be preferentially used for mine fill. Over the life-of-mine it is anticipated that there will be a deficit of rock waste, and a fill quarry may be required.

No capital or operating cost provision has, therefore, been made to deal with mine waste rock.

8.3 Tailings Management:

Tailings will be alkaline and will require storage in an appropriate dam. Some sand tailings will report back to the mine as back-fill.

The tailings dam location as shown on Fig 3 is of sufficient size to contain all tailings for a 10-year mine life. This location has several advantages:

- it occurs in an open, flat natural amphitheatre which has a very small catchment area
- it is close to the mill
- it is not visible from Trial Harbour Road

A potential disadvantage is that it is upstream of the mine portal. A solution to this is to move the mine portal into an adjacent valley so that there is zero risk to the mine.

It is envisaged that the tailings dam would be constructed in several stages during the life of the mine; ie, the capital cost would be spread over two or three incremental amounts.

The initial increment, which would be required prior to the start of mill production, is provided for in the Metallurgical Scoping Study report.

8.4 Cost Estimates and Schedules:

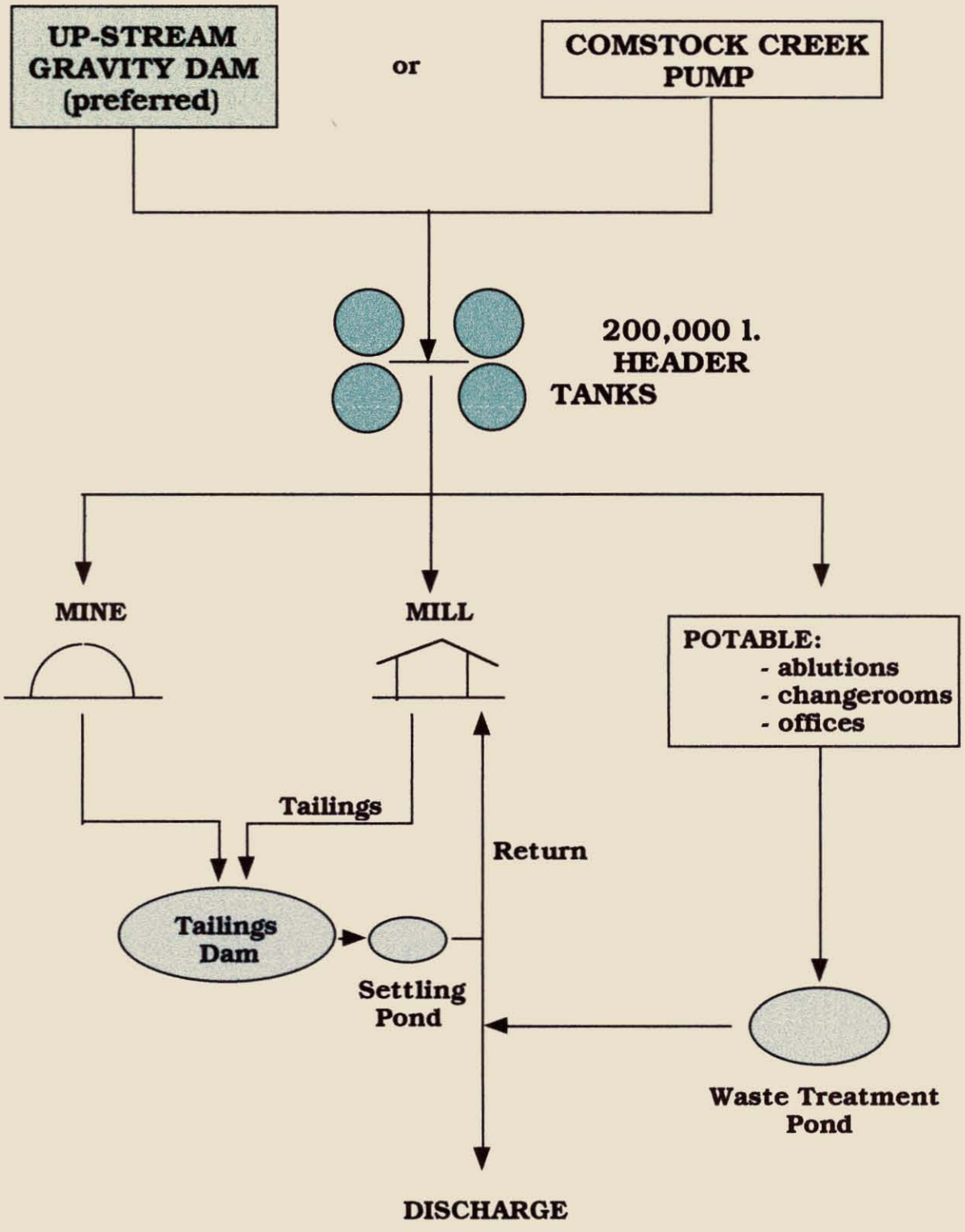
Capital estimates are:

Water management	$200,000
Mine waste management	Nil
Tailings management	Nil (included in mill estimates)
Total Capital Estimate:	**$200.00**

An operating cost estimate to maintain water management systems is $0.20/t ore.

Costs of environmental monitoring are included in the Environment Operating budget.

The initial capital expenditure on the tailings dam would be incurred concurrently with mill construction.



COST ESTIMATE:	
Supply (dam, pipe, tanks) :	$80,000
Settling Pond:	$50,000
Waste Management:	$20,000
Reticulation:	$50,000
TOTAL:	$200,000

ALLEGIANCE MINING NL
Avebury Nickel Project
Scoping Study
WATER and WASTE MANAGEMENT

Newnham Exploration and Mining Services

Fig 7

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9. CONCENTRATES TRANSPORT (Fig 8)

The envisaged Avebury operation will produce approximately 19,400 dry tonnes per annum of nickel sulfide concentrate, which equates with approximately 22,000 wet tonnes per annum.

It is assumed these concentrates will be sold to an off-shore treatment facility on a FOB Burnie basis; ie, Allegiance would bear all costs of placing the concentrates in a ship at Burnie, and the buyer would bear all costs thereafter.

Several options for transporting concentrates to Burnie are available:

- road freight to Burnie
- road freight to Melba Flats rail head by one contractor, then rail freight with Tasrail to Burnie
- road freight to Melba Flats and rail to Burnie, all with Tasrail

At Burnie, Tasrail owns a large concentrates shed which services other west coast mines. They also own the only bulk ship loader in the port.

If the concentrate was taken by road to Burnie by contractors other than Tasrail, they would probably want to ship it out in containers.

Containers have advantages and disadvantages:

Advantages:

- avoids concentrate contamination
- avoids concentrate losses through three stages of handling

Disadvantages:

- higher sea freight charges which the buyer would not welcome
- more truck movements on highway

In terms of cost, there is not a great difference between the three transport options to Burnie. For simplicity of operation, the Tasrail

option has many advantages. It involves Tasrail picking up the concentrate at Avebury and placing it in a bulk ship at Burnie, for an estimated unit cost of $25/wet tonne. This equates with a transport cost of $1.80/t ore.

No capital expenditure is required from Allegiance apart from the provision of a truck loading hopper at Avebury.

In a final feasibility study, the transport options need to be investigated thoroughly.

For the purposes of this scoping report, a transport cost of $25/wet tonne of concentrate has been used, and is accounted for as an ex-mine cost.

661145

CONCENTRATE PRODUCTION

22,000 WET METRIC TONNES pa

in a loading bin at Avebury

OPTION A	OPTION B	OPTION C
contract load and road cart to Melba Flats (contractor)	load and road cart to Melba Flats (Tasrail)	load and road cart to Burnie (contractor)
MELBA FLATS DEPOT	MELBA FLATS DEPOT	
load and train haul to Burnie (Tasrail)	load and train haul to Burnie (Tasrail)	
BURNIE DEPOT (Tasrail)	BURNIE DEPOT (Tasrail)	BURNIE DEPOT (Tasrail)
ship load (Tasrail)	ship load (Tasrail)	ship load (Tasrail)
$27/wet tonne	$25/wet tonne	$27/wet tonne
	PREFERRED OPTION	

COST ESTIMATE
FOB BURNIE

$25/wet tonne = $ 550,000 pa

= $ 1.80/tonne ore

ALLEGIANCE MINING NL
Avebury Nickel Project
Scoping Study

CONCENTRATES TRANSPORT

Newnham Exploration and Mining Services

Fig 8

10. BONDS

The Tasmanian Government will require a bond prior to development of the Avebury project.

This bond is essentially an environmental bond to cover remedial and rehabilitation tasks not adequately addressed by the mine operator on closure. The amount of the bond varies and is negotiable.

Factors which influence it are:

- size and nature of operation
- area and type of land disturbed
- nature of waste products

Bonds can increase during the life of the operation if disturbance levels increase, but can also decrease if there is an effective program of on-going rehabilitation.

An initial bond of approximately $100,000 might be required at Avebury prior to commencement of decline development, and this may increase to approximately $500,000 prior to start of concentrate production. Part or all of the bond is refundable on closure if rehabilitation has been completed in accordance with an approved Mine Closure Plan.

The bond should be regarded as a capital item and, for this Scoping Study, a bond of $500,000 is assumed.

11. SCHEDULE and BUDGET

11.1 Schedule:

The infrastructure development schedule is strongly influenced by four principal factors:

- achieve an early start on underground mine development
- mill construction to be completed concurrently with first mine production
- most cost effective construction period is from October-March
- cost benefits are achieved if the schedule is compacted

This Scoping Study assumes infrastructure development would be undertaken in two summer campaigns:

First summer campaign:

- construct access road
- construct power transmission line and sub-station
- complete basic site works sufficient to facilitate mine development
- develop water supply and mine waste water management systems

Second summer campaign:

- complete site works
- construct mill
- construct site buildings (offices, workshops, etc)
- complete Stage 1 of Tailings Dam

This means there would be an 18-month construction schedule, from the decision to develop a mine to first concentrates production.

Major cost efficiencies would be achieved if this construction schedule

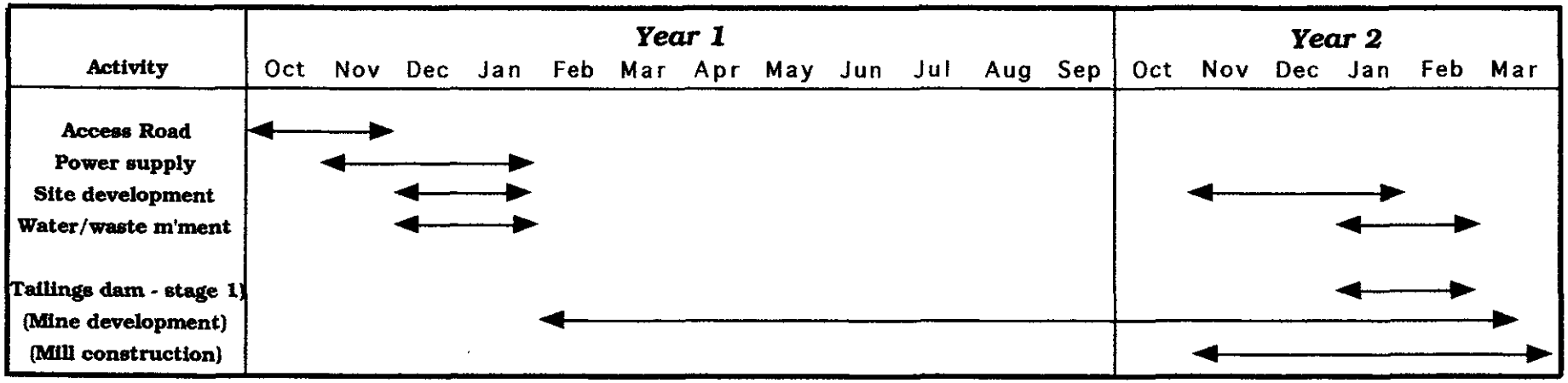
commenced in October.

The period between the two construction campaigns would be utilised for underground mine development work necessary to access the ore deposits. Waste rock generated during this period would be used for road and site development.

11.2 Budget:

The attached budget shows the first and second summer campaigns would total $2.46M and $1.84M respectively for a total $4.30M.

This includes a 15% provision for engineering management.



Schedule

Note: The schedule as shown is designed to take maximum advantage of seasonal factors

Activity	Year 1 Oct	Nov	Dec	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Year 2 Oct	Nov	Dec	Jan	Feb	Mar	Cap $'M	Oper $/t
Access road	0.15	0.20																	0.35	0.20
Power supply		0.20	0.25	0.25															0.70	
Site development			0.40	0.40										0.40	0.40	0.40			2.00	0.40
Water/waste m'ment			0.05	0.05												0.05	0.05		0.20	0.20
Bonds	0.10												0.40						0.50	
Management (15%)	0.04	0.06	0.10	0.10									0.06	0.06	0.06	0.07			0.55	
TOTALS	0.29	0.46	0.80	0.80									0.46	0.46	0.46	0.52	0.05		4.30	0.80

Budget

Note: The costs of tailings dam construction, mine development and mill construction allowed for elsewhere.

AVEBURY NICKEL PROJECT - SCOPING STUDY

Infrastructure Development - Schedule and Budget

66143

12. FEASIBILITY STUDY REQUIREMENTS

Completion of a full Feasibility Study will require an appropriate team of engineers to design, specify and cost all proposed infrastructure development.

A team of 5 persons would include a principal engineer, electrical and civil engineers, surveyor and drafting services.

A five (5) week schedule would represent 150 person/days at $1,000/day = $150,000.

Add to this travelling and other expenses, including a contingency requirement.

Total estimate:	$250,000

The scheduling of this component of the Feasibility Study should be left as late as possible to permit maximum time to acquire resource, mining, and environmental and milling data - all of which could influence infrastructure design.

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01-4526C (Vol 4 of 10)

MINERAL RESOURCES
FILE REF: EL28/88 PT5
16 FEB 2001
DOC. REF:
OFFICER | FOR ACTION | FOR INFO
See Vol 10 bks
RESUBMIT TO | DATE

ALLEGIANCE MINING NL

AVEBURY NICKEL PROJECT

MINERAL RESOURCE REPORT

MINERAL RESOURCES
14 FEB 2001
TASMANIA

NOVEMBER 2000

MICROFILMED
FICHE No 015502-13

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on behalf of

ALLEGIANCE MINING NL

01_4526 (c)

Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL*; Newnham Exploration and Mini
Newnham, L.A.
EL28/1988
Vol 4 of 10

IMPORTANT NOTES

This report has been prepared using information and data available to the author at the time of writing.

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Neither Michael V. McKeown nor McKeown Mining Pty Ltd nor their Associates hold shares or options in Allegiance Mining NL

MAP CONVENTIONS

Co-ordinates in this report are in AMG.

RLs in this report are MSL plus 2000m.

Cross sections are drawn looking west, that is, north to the right.

Longitudinal projections are drawn looking north, that is, east to the right.

EXECUTIVE SUMMARY

Allegiance Mining NL and Rio Tinto Exploration Pty Ltd jointly hold Exploration Licence 28/88, in two parts, south-west of Zeehan and at Trial Harbour. In accordance with a Joint Venture Agreement, there is an agreement to transfer the licence to Allegiance whereupon Allegiance will control 100% of the tenement and Rio Tinto will retain 2% Net Smelter Royalty on future production. EL 20/88 expires in December 2000 and application has been lodged to extend the licence beyond that date. The Avebury prospect occurs in that part of EL 28/88 south-west of Zeehan.

Early in 1988, Allegiance's first diamond drill hole, A001, intersected significant nickel mineralisation at the Avebury prospect. Allegiance has drilled 26 holes into three zones of mineralisation: North Avebury, Central Avebury and South Avebury and Mineral Resources have been identified at North Avebury and South Avebury.

Allegiance has drilled 26 diamond drill holes and five wedged holes, for a total length drilled of 11,144m; the former licence holder CRA Exploration Pty Ltd drilled three holes for a total drilled length of 796m.

The Avebury nickel sulphide mineralisation occurs within moderately to steeply dipping Cambrian serpentinite bodies lying within a sequence of Cambrian metamorphic rocks dominated by hornfels and chert. During the intrusion of Devonian granite, the serpentinites were partly converted to diopside-tremolite rocks along their margins, and nickel sulphide mineralisation was mobilised and concentrated in places within the serpentinites, particularly immediately adjacent to their contacts.

The North Avebury nickel mineralisation occurs near the western end of the northern contact of an east-west striking serpentinite body. The contact has been identified in drillholes over a strike length of about 500 metres and a vertical extent of just under 400 metres; the serpentinite appears to be terminated by a north-west striking fault at the western end but is open to the east and down dip.

The South Avebury nickel mineralisation occurs along both contacts of an east-west striking serpentinite. These contacts have been identified in drillholes over a strike length of about 300 metres and a vertical extent of just under 500 metres; the serpentinite appears to be terminated

by a fault at the western end but is open to the east and down dip. The South Avebury nickel mineralisation has been discovered near the western end of this serpentinite.

The zones of nickel mineralisation carry nickel sulphides and lesser arsenides and sulpharsenides; the dominant nickel bearing mineral is pentlandite but pyrrhotite also occurs and may be nickeliferous. The nickel mineralisation is coarse grained and petrological examination suggests that recovery of the nickel bearing sulphides by flotation should be feasible.

Core recoveries were generally 100%, with small losses in some broken zones but there was no systematic core loss in the nickel mineralisation. Core sampling and assay techniques have been subjected to comprehensive verification including duplicate nickel assaying by the NATA registered laboratories operated by Analabs Pty Ltd and Amdel Pty Ltd. Magnetite in the Avebury serpentinites significantly deforms the local magnetic field and Maxibor surveys of some holes were made, for others, a correction of downhole bearings made by camera surveys was necessary.

The outer boundary of the nickel mineralisation generally coincides with the sharp serpentinite contacts but the internal boundary of the nickel sulphide mineralisation is defined by the transition from high grade to background grade nickel mineralisation. The drill holes indicate along strike and down dip continuity of the mineralisation but the results of wedged holes and variography suggest a relatively high nugget effect and this has been taken into account in the resource estimate

The North Avebury nickel mineralisation strikes east west and dips north at about 70^{o} to 80^{o}. Known strike length is about 300 metres and the mineralisation extends from 2100m RL down to 1800m RL, the top being about 70 metres below the surface. The mineralisation ranges up to just over 30 metres in horizontal thickness, measured in a north-south direction. The South Avebury nickel mineralisation strikes east west and dips north at about 50^{o} to 60^{o}. Known strike length is about 150 metres and the mineralisation extends from 2060m RL down to 1920m RL, the inferred top of the mineralisation being about 75 metres below the surface. The mineralisation is in two lenses, a north lens and a south lens, from about 20 to 40 metres in true thickness.

At Central Avebury, nickel mineralisation occurs sporadically along the southern, faulted contact of a serpentinite but continuity of the mineralisation cannot be assumed and no resource estimate has been made for this zone.

No cost estimates are available for mining or milling of the Avebury mineralisation and, so, no parametric grade-tonnage estimates can be made. However, a grade-tonnage curve which shows tonnages of mineralisation above a particular cut-off grade, assuming that the deposit can be divided into equal sized blocks the size of the geological samples, that is into blocks of one cubic metre, has been made.

A cut-off grade of 0.75% Ni for the Avebury mineralisation has been estimated by calculating an equivalent nickel grade to an estimate of the Renison Bell cut-off grade, the operations at Renison Bell being similar in several significant ways to operations which can be envisaged at Avebury.

At Avebury, the Mineral Resource is:

AVEBURY NICKEL PROJECT

INDICATED MINERAL RESOURCE
at 0.75% Ni cut-off grade

North Avebury	990,000 tonnes	1.74% Ni

INFERRED MINERAL RESOURCE
at 0.75% Ni cut-off grade

South Avebury	2,110,000 tonnes	1.73% Ni
Total	3,100,000 tonnes	1.73% Ni

The nickel to cobalt ratio for the two zones of mineralisation is about 40 to 50: 1 and it is expected that this ratio will be maintained in nickel concentrates. The levels of the other base metals is low and no precious metals have been identified in, or reported in assays from, the Avebury nickel mineralisation.

By comparison with the resources and reserves at other operating nickel sulphide mines, the size and grade of the Avebury mineralisation suggest that the resource can be economically mined.

Further diamond drilling is recommended to convert the Mineral Resources to Ore Reserves. The length of drilling involved is 10,000 metres at an estimated cost of $1,250,000 and the program would take six months to complete.

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1 INTRODUCTION

Allegiance Mining NL (Allegiance) and Rio Tinto Exploration Pty Ltd (Rio Tinto) jointly hold Exploration Licence 28/88, a licence in two parts, south-west of Zeehan and at Trial Harbour. In accordance with a Joint Venture Agreement, there is an agreement to transfer the licence to Allegiance whereupon Allegiance will control 100% of the tenement and Rio Tinto will retain 2% Net Smelter Royalty on future production. EL 20/88 expires in December 2000 and application has been lodged to extend the licence beyond that date.

The Avebury prospect occurs in that part of EL 28/88 south-west of Zeehan (Figure 1). Allegiance has undertaken three drilling campaigns at Avebury: the summer of early 1998, the summer of early 1999, and the latest program which ran from November 1999 to October 2000.

Early in 1998, Allegiance's first diamond drill hole, A001, intersected significant nickel mineralisation at the Avebury prospect. Since then, Allegiance has drilled 26 holes into the Avebury mineralisation (Figure 2).

Three zones of nickel mineralisation have been identified: North Avebury, Central Avebury and South Avebury. Mineral Resources have been identified in two of these zones: North Avebury and South Avebury.

This Resource Report describes the resources which have been defined by the 26 diamond drill holes.

2 LOCAL GEOLOGY

A detailed description of the geology of the Avebury prospect is not presented here but the geological setting is described briefly so that the method of resource estimation can be understood.

The Avebury nickel sulphide mineralisation occurs within moderately to steeply dipping Cambrian serpentinite bodies lying within a sequence of Cambrian metamorphic rocks dominated by hornfels and chert. The serpentinites were derived from rocks associated with the McIvor Hill ultramafic complex; the other rocks were originally siltstones and volcaniclastics of the Crimson Creek Formation (Table 1). The serpentinites in the Avebury

area appear to be concordant in part with the enclosing Crimson Creek Formation. The whole sequence was metamorphosed and metasomatised during the intrusion of the Heemskirk granite at the time of the Devonian Tabberabberan Orogeny when the Crimson Creek Formation rocks were partly silicified, actinolitised and phlogopitised in part, partly converted to skarns in places, and the serpentinites were partly converted to diopside-tremolite rocks along their margins.

During the metamorphic and metasomatic events, nickel sulphide mineralisation was mobilised and concentrated within the serpentinite bodies, particularly immediately adjacent to their contacts (Figure 3). There are sporadic occurrences of nickel sulphide outside the serpentinites but the outer boundary of the nickel mineralisation generally coincides with the sharp serpentinite contacts. The internal boundary of the nickel sulphide mineralisation is defined by the transition from high grade to background grade nickel mineralisation (see 7.2 GEOLOGICAL INTERPRETATION).

Two large east-west striking serpentinite bodies have been identified at Avebury, a northern body and a southern body, and nickel mineralisation has been identified in both (Figures 7 to 17). Nickel mineralisation along the northern margin of the northern serpentinite is known as the North Avebury mineralisation, along the southern margin of the northern serpentinite as Central Avebury mineralisation, and within the southern serpentinite as South Avebury mineralisation.

The northern serpentinite has been identified in drillholes over a strike length of about 500 metres and a vertical extent of just under 400 metres; the serpentinite appears to be terminated by a north-west striking vertical fault at the western end but is open to the east and down dip (Figures 17 and 19). The North and Central Avebury zones of nickel mineralisation have been discovered near the western end of this serpentinite.

The southern serpentinite has been identified in drillholes over a strike length of about 300 metres and a vertical extent of just under 500 metres; the serpentinite appears to be terminated by a fault at the western end but is open to the east and down dip (Figures 17 and 27). The South Avebury nickel mineralisation has been discovered near the western end of this serpentinite.

TABLE 1

ALLEGIANCE MINING NL

AVEBURY NICKEL PROJECT - ROCK TYPES

Recentweathering products	
	clay
	claystone
	ironstone
Devonian	**metamorphic and metasomatic rocks resulting from Heemskirk granite intrusion**
	massive pyrrhotite zones replacing Cambrian limestone or dolomite
	skarns containing axinite and/or schorl
	actinolitised and phlogopitised hornfels and chert
	diopside-tremolite rock (after serpentinite) containing nickel mineralisation
Cambrian	**serpentinite (McIvor Hill Ultramafic Complex)**
	serpentinite
	pre-serpentinite (dunite)
Cambrian	**conglomerate/limestone association (Crimson Creek Formation?)**
	conglomerate
	limestone
	very altered sedimentary rocks
Cambrian	**chert/hornfels association (Crimson Creek Formation)**
	hornfels after volcaniclastic rocks
	chert after mudstone
	shale and siltstone

TABLE 2

ALLEGIANCE MINING NL
AVEBURY NICKEL PROJECT
A GEOLOGICAL HISTORY

erosion to present land surface

uplift

intrusion by Devonian Heemskirk granite
accompanied by
deposition of nickel sulphides, arsenides and sulpharsenides along margins of serpentinites
serpentinisation of Cambrian ultramafic rocks
metasomatism of Cambrian sedimentary rocks

Tabberabberan folding

intrusion? of Cambrian ultramafic rocks

deposition of Cambrian sedimentary rocks

3 MINERALOGY

The serpentinites at Avebury all contain some nickel and there is a concentration of nickel within, and marginal to, the serpentinite contacts. The zones of concentrated nickel mineralisation which have been identified carry nickel sulphides and lesser arsenides and sulpharsenides; the dominant nickel bearing mineral is pentlandite but pyrrhotite also occurs and may be nickeliferous.

The nickel mineralisation is coarse grained and even at low nickel grades, nickel sulphides can be seen in the core. Petrological examination of many specimens by Wally Fander (Appendix 2) suggests that recovery of the nickel bearing sulphides by flotation should be feasible.

Appendix 2 contains all the petrological descriptions which have been completed and also several photomicrographs which demonstrate the coarse grained nature of the nickel mineralisation.

The sulphides and sulpharsenides which have been identified in the zones of nickel mineralisation are listed in Table 3.

TABLE 3

ALLEGIANCE MINING NL
AVEBURY NICKEL PROJECT
SULPHIDE AND ARSENIDE MINERALS

pyrite	FeS_2
pyrrhotite	FeS
valleriite	$CuFeS_2(Mg,Al,Fe)(OH)_2$
mackinauwite	$(Fe,Ni)_9S_8$
pentlandite	(Fe,Ni)S
millerite	NiS
niccolite	NiAs
gersdorrfite	NiAsS
maucherite	Ni_3As_2 or Ni_4As_2

Based on petrological descriptions and assay data, four sulphide assemblages have been identified within the serpentinite bodies: three assemblages confined mainly to the nickel mineralisation, and a background assemblage throughout the serpentinite mass.

Pentlandite assemblage: apart from traces of pyrite and chalcopyrite, pentlandite is the dominant sulphide mineral present, for example A001, Central Avebury. Sulphur and nickel grades in this style of mineralisation are about equal, implying the presence of pentlandite, and the arsenic content is very low.

Pyrrhotite-pentlandite assemblage: pentlandite and pyrrhotite are the dominant sulphides present, for example A026, South Avebury. Sulphur grades exceed nickel grades, implying the presence of pyrrhotite as well as pentlandite, and arsenic content is low.

Pyrrhotite-pentlandite-arsenide assemblage: pentlandite and pyrrhotite are accompanied by various nickel arsenides and sulpharsenides such as niccolite, gersdorffite, valleriite and mackinauwite, for example A004, North Avebury. Arsenic grades can exceed 1% and, where this occurs, nickel grades can exceed sulphur grades, implying the presence of nickel arsenides and sulpharsenides.

Background pentlandite assemblage: pentlandite disseminations occur throughout the main serpentinite mass where nickel grades range from about 0.10% to 0.20%, and sulphur and nickel grades are about equal, implying the presence of pentlandite.

4 DRILLING PROGRAMS

At Avebury, Allegiance has drilled 26 diamond drill holes, numbered A001 to A026 and five wedged holes, for a total length drilled of 11,144m; the former licence holder CRA Exploration Pty Ltd drilled three holes, ZA1 to ZA3, for a total drilled length of 796m (Table 4).

All but three of these holes tested the mineralisation described in this report. (Figure 2).

Holes ZA1 to ZA3 and A001 to A006 were drilled by Diamond Drilling Tasmania Pty Ltd, holes A007 to A026 by Almac Drilling Pty Ltd.

TABLE 4

ALLEGIANCE MINING NL

AVEBURY NICKEL PROJECT

HOLES DRILLED

bhid	east	north	rl	length	collar brg	collar dip
	m	m	m	m	AMG	
ZA1	354,837.3	5,357,277.8	2,171.4	284.5	352	-65
ZA2	355,619.6	5,357,342.2	2,177.2	255.5	352	-48.5
ZA3	355,782.3	5,357,134.2	2,162.8	256.0	352	-55
A001	354,648.1	5,357,224.9	2,152.1	532.6	5	-47
A002	355,225.2	5,357,585.6	2,175.0	307.0	182	-52
A003	354,730.2	5,357,281.6	2,163.4	339.0	5	-48
A004	354,625.3	5,357,371.9	2,146.0	226.3	353	-49.5
A005	354,520.9	5,357,467.4	2,171.3	202.5	354	-47.5
A006	354,735.8	5,357,189.7	2,150.0	481.5	0	-43.5
A007	354,622.7	5,357,521.4	2,157.0	251.0	175	-45
A008	354,629.3	5,357,744.1	2,178.8	575.0	178	-46
A009	354,736.1	5,357,522.6	2,164.2	525.0	177	-45
A010	354,835.6	5,357,291.0	2,169.3	373.0	0	-46
A011	354,545.1	5,357,311.7	2,158.8	452.0	351	-50.5
A012	354,559.6	5,357,742.7	2,178.5	544.0	177	-59.5
A013	354,674.0	5,357,744.5	2,176.7	545.6	173	-58.5
A014	354,427.7	5,357,459.0	2,180.0	379.5	172	-45
A014A	*354,453.4*	*5,357,286.9*	*2,016.0*	40.9	wedged off A014 at 239.2m	
A015	354,907.5	5,357,752.5	2,168.0	467.0	174	-51.5
A016	354,316.4	5,357,457.4	2,181.2	355.5	175	-45
A017	354,431.5	5,357,362.2	2,179.4	187.0	174	-45
A018	354,718.7	5,357,599.0	2,166.4	396.9	193	-47
A018A	*354,686.7*	*5,357,474.0*	*2,032.3*	54.8	wedged off A018 at 186.2m	
A019	354,699.1	5,357,598.1	2,165.2	281.7	175	-60.5
A019A	*354,703.1*	*5,357,496.7*	*1,985.0*	72.3	wedged off A019 at 206.9m	
A020	354,607.7	5,357,623.3	2,167.0	465.9	189	-52
A021	354,657.2	5,357,750.6	2,177.5	507.6	179	-54
A022	354,629.0	5,357,743.0	2,179.0	172.9	178	-52
A023	354,569.4	5,357,742.6	2,179.5	524.6	174	-52
A024	354,465.7	5,357,753.7	2,174.5	613.0	180	-63
A024A	*354,452.0*	*5,357,586.2*	*1,869.5*	232.2	wedged off A024 at 348.5m	
A025	354,494.6	5,357,735.7	2,177.1	559.4	179	-50
A026	354,509.6	5,357,453.0	2,171.0	351.9	181	-45
A026A	*354,505.6*	*5,357,306.2*	*2,022.6*	124.9	wedged off A026 at 208.8m	
total				**11,938.5**		

5 RESOURCE ASSESSMENT CRITERIA

Drilling techniques: all holes considered in this resource report were diamond drill holes, core was NQ size, drilled using a standard inner tube on wireline, that is, not triple tube. All drill core is stored at the Allegiance Mining NL coreshed, Zeehan.

Logging: all core was lithologically logged in sufficient detail to support this Mineral Resource estimate and associated mining and metallurgical studies. Drill logs describe the rocks and mineralisation present and are supported by petrological descriptions by Wally Fander of Central Metallurgical Services (Appendix 2) and assays. All core was logged by the author with a second geological opinion provided by Lindsay Newnham of Newnham Exploration and Mining Services. Hard copies of drill logs are kept at the offices of Allegiance Mining NL at Zeehan and Newnham Exploration and Mining Services at Exeter; computer based copies are kept at the office of McKeown Mining Pty Ltd at Ridgley. Files of hard copies of assay data as received from analytical laboratories are kept at the offices of Allegiance Mining NL at Zeehan and Newnham Exploration and Mining Services at Exeter.

Drill sample recovery: Core recoveries were recorded for each drilling run (see drill logs) and were generally 100%, with small losses in some broken (fault?) zones. There was no systematic core loss in the nickel mineralisation which tends to occur associated with hard silicate rocks (see 2 LOCAL GEOLOGY) and, consequently, there can be no apparent relationship between sample recovery and nickel grade.

Sub-sampling techniques and sample preparation: these are described in Appendix 3. In essence, the core was sawn in half longitudinally, and half the core was crushed, and all or half the crushed material was pulverised for assay. All crushed and pulverised samples not consumed by the assay process were retrieved and are stored in Allegiance's Zeehan core shed. Some whole core of the nickel mineralisation in two wedged holes, A014 and A018, was used for metallurgical testwork and was consumed by the testwork process.

Quality of assay data and laboratory tests: these are described in Appendix 3. All nickel assays used for this resource estimate are total nickel assays determined by ICP following an acid leach. All assays were performed by NATA registered laboratories: Analabs Pty Ltd and Amdel Pty Ltd.

Verification of sampling and assaying: verification is described in Appendix 3. In addition, twinned holes, in the form of wedged holes have been drilled, the wedged intersections

generally being within about 5 metres of the original intersections. Of the five wedged holes, whole core from parts of three holes, A014A, A018A and A019A, was used for metallurgical testwork and yielded the following comparative results:

South Avebury	A014	254.7m to 267.3m	1.56% Ni	2.66% S
	A014A	254.7m to 264.8m	1.96% Ni	3.34% S
North Avebury	A018	214.4m to 217.4m	1.69% Ni	1.96%S
	A018A	213.9m to 217.0m	1.43% Ni	1.36% S
	A018	217.4m to 221.2m	1.23% Ni	1.12%S
	A018A	217.0m to 221.5m	1.29% Ni	0.97% S
	A019	241.8m to 245.0m	1.87% Ni	2.59%S
	A019A	240.5m to 244.4m	3.10% Ni	4.34% S
	A019	245.0m to 248.0m	3.52% Ni	4.00%S
	A019A	244.4m to 247.4m	3.50% Ni	4.24% S
	A019	248.0m to 251.0m	1.93% Ni	2.39% S
	A019A	247.4m to 250.4m	3.53% Ni	4.06% S

Core from a South Avebury intersection in one parent hole and its daughter, A024 and A024A, was split and assayed, the two South Avebury intersections being about 10 metres apart, with the following comparative results:

South Avebury	A024	548.1m to 549.1m	0.75% Ni	0.66% S
	A024A	548.2m to 549.4m	3.20% Ni	4.85%S
	A024	549.1m to 550.4m	1.21% Ni	1.50% S
	A024A	549.4m to 550.0m	1.68% Ni	2.10% S

The assays confirm the along strike and down dip continuity of the mineralisation but suggest a relatively high nugget effect and this has been taken into account in estimating both the tonnage and grade for this resource estimate (see 6 THE RESOURCE ESTIMATES).

Location of data points: The co-ordinates of the collars and the collar bearings of all drill holes drilled by Allegiance have been determined by theodolite traverse, most of the dips were also determined by theodolite traverse, a few by clinometer; this survey work was undertaken by Campbell-Smith, Phelps and Pedley Pty Ltd, licensed surveyors of Launceston.

Please note that all co-ordinates in this report are in the AMG system and RLs in this report are actual heights above MSL plus 2000m.

Magnetite in the Avebury serpentinites significantly deforms the local magnetic field. The largest variation detected occurs near the collars of A005 and A026 where there is a magnetic bearing error of 15^{o}. Drilling crews made downhole surveys of all holes using cameras but these surveys were subject to bearing errors caused by the presence of magnetite just described and, to a lesser extent, pyrrhotite and pentlandite. During the most recent drilling program, ten holes were surveyed downhole using a Maxibor, a downhole survey instrument which is not affected by surrounding magnetic materials. Not all the Maxibor surveys were successful and Maxibor surveys were not available for all holes and, so, for many holes, a method of correcting the downhole bearings by treating them as relative bearings was used; the method is described in Appendix 4.

The accuracy of the surveys is adequate for the class of Mineral Resource estimates made for this report (JORC Code, Appendix 5).

Data density and distribution: the data density and distribution is described in 6 THE RESOURCE ESTIMATES below.

Samples were not composited. The average length of samples used for the North Avebury grade estimate had an average length of 1.03m and a range from 0.4m to 1.8m with only one sample less than 0.5m long and only three samples greater than 1.5m long. Compositing over one metre lengths, a statistically obvious compositing length, with a 0.5m tolerance, would not have improved on the distribution of lengths and would inevitably have resulted in rejection of some sample data. Similarly, the average length of samples used for the South Avebury grade estimate had an average length of 1.03m and a range from 0.8m to 1.7m with only one sample greater than 1.4m long and compositing was not necessary. The effects of compositing over longer lengths should be reviewed later when sufficient data are available for the interpolation of grades into a block model.

Audits and reviews of sampling techniques and data: these reviews are described in Appendix 3.

Database integrity: all assay data used for estimates of grade in this report have been checked against the original laboratory assay sheets. The specific nickel assay data used for the resource estimates are attached (Appendix 6).

Geological interpretation: this is described in 6 THE RESOURCE ESTIMATES.

Estimation and modelling techniques: these are described in 6 THE RESOURCE ESTIMATES.

Cut-off grades or parameters: at the present time, no cost information is available and, so, no parametric cut-off grades have been calculated. A non-parametric cut-off grade has been applied and is discussed in 6.5.1 CUT OFF GRADE.

Mining factors or assumptions: the tonnage and grade of resources described in this report are such that there are reasonable prospects for eventual economic extraction (6.5.1 CUT OFF GRADE, 6.5.4 THE AVEBURY MINERAL RESOURCE) but, except for this assumption, the resources described in this report are Mineral Resources and, for this report, no account has been taken of mining factors. A scoping mining study is under way at present.

Metallurgical factors or assumptions: the grade of the nickel mineralisation, the apparently relatively coarse grained nickel sulphides observed in the drill core, and the petrological studies which have been undertaken (3 MINERALOGY), suggest that the nickel sulphides present at Avebury will be recoverable and that there are reasonable prospects for eventual economic extraction. The resources described in this report are Mineral Resources and no account has been taken of metallurgical recoveries; metallurgical testwork is under way at present.

Tonnage factors: the density of specimens of drill core has been determined by two methods. In the first method, the volume of the core is measured by displacement of water in a graduated cylinder and the air dried core is weighed; this method has been used for whole and split core. In the second method, the diameter and length of whole pieces of core are measured and the volume of the core is then calculated, and the air dried core is weighed. For both methods, the mass of the specimens has been measured using an electronic balance.

The densities of 36 samples of serpentinitic rocks and 11 samples of waste rocks have been determined (Figure 4). The densities of the serpentinitic specimens fall into three groups: dark serpentinite, nickel mineralisation and massive pentlandite.

The specimens in the dark serpentinite group are principally dark serpentinites with varying amounts of magnetite, the magnetite content being reflected by the magnetic susceptibility of the specimens. Within this group, there is a relation between magnetic susceptibility, measured using a Scintrex SM5 susceptibility meter, and density, and the approximate relation can be expressed as:

$$\text{core density} = 2.63 + 0.023 * \text{magnetic susceptibility}$$

The nickel mineralisation group is dominated by silicate rocks (after serpentinite), and mixed silicate rock (after serpentinite) and dark serpentinite. Petrological reports indicate that the silicate rock consists largely of diopside and tremolite. The specific gravity of this group averages about 3.2 gm per cubic cm (Figure 4a).

The third group, consisting of a single massive pentlandite specimen, had a specific gravity of about 4.8 gm per cubic cm (Figure 4a).

The specimens of waste rocks exhibit a much smaller range of density averaging about 2.9 gm per cubic cm (Figure 4b).

For the estimates in this report, the density of nickel mineralisation and waste have been discounted by about 3% to allow for voids in the rock mass; 3.0 tonnes per cubic metre has been used for the bulk density of nickel mineralisation, and 2.8 tonnes per cubic metre is recommended as the bulk density of waste.

Classification: this is explained in 6 THE RESOURCE ESTIMATES.

Audits and reviews: this resource report has been reviewed by Lindsay Newnham of Newnham Exploration and Mining Services. It is recommended that this resource estimate be audited by another independent Competent Person (JORC code, Appendix 5).

6 THE MINERAL RESOURCE ESTIMATES

6.1 THE GEOLOGICAL INTERPRETATION

The geological interpretation for this resource estimate can be seen in Figures 7 to 17. The interpretation was devised in the first instance from a detailed structure contour plan, drawn at 1:1000 scale, of the following features:

North Avebury serpentinite, northern contact
North Avebury mineralisation, southern contact
Central Avebury, northern contact
North Avebury serpentinite, southern contact
South Avebury serpentinite, northern contact
South Avebury mineralisation - north lens, southern contact
South Avebury mineralisation - south lens, northern contact
South Avebury serpentinite, southern contact
fault to the west of South Avebury

Ten, somewhat irregularly spaced, 1:1000 scale north-south cross sections from 354,445m E to 354,750mE were used during the geological interpretation showing drillhole traces, rock types as codes, and nickel assays as numbers.

The essence of the interpretation can be seen in the level plan at 2000m RL (Figure 17) and cross sections from 354,445m E to 354,750mE (Figures 7 to 16). When examining the cross sections, care should be taken to allow for the oblique angles between the drill holes and the cross sections (Figure 2) and the oblique angles between geological structures and the cross sections (Figure 17).

As has been described (2 LOCAL GEOLOGY), the nickel mineralisation occurs along the contacts of the serpentinite bodies. The occurrence of nickel mineralisation outside the serpentinites is sporadic and is possibly related to small fractures in the enclosing mass of country rock or narrow lenses of serpentinite outside the main serpentinite bodies. Whatever the origin of the extra-serpentinite nickel mineralisation, its continuity cannot be established at the present time and, so, the outer boundary of the nickel mineralisation was taken in all cases to be at the contact of serpentinite with country rock. In some cases, this meant the inclusion of relatively low grade material within the nickel mineralisation, for example, North Avebury in A004 (Figure 18). This means that the outer boundary of the mineralisation is in a consistent and definite geological position and is a hard boundary.

The identification of an inner boundary to the nickel mineralisation was less straightforward. However, examination of the assays available revealed a nickel grade drop-off at about 0.5% Ni in many holes (for example, Figure 18). Beyond, the 0.5% grade drop-off, nickel grades drop to the levels of the background nickel assemblage (see 3 MINERALOGY). In practice, the inner boundary was designated to occur at the beginning of a run of at least five assays below 0.5% Ni; this meant the inclusion of narrower zones than this within the mineralisation and, in A003, the presence of two zones of nickel mineralisation (cross section 354,750mE, Figure 16). This internal boundary to the nickel mineralisation is a soft boundary.

A check of the geostatistical and geological validity of the soft boundary was attempted. Correlation coefficients were calculated for sample pairs immediately on either side of this boundary, for sample pairs two samples away on either side of the boundary, sample pairs three samples away and so on with the following results:

pair	1	2	3	4
correlation coefficient	-0.08	0.28	-0.49	-0.33
number of pairs considered	14	12	8	7

For comparison, within the North Avebury mineralisation defined by the two boundaries described, the correlation coefficient for the grades of adjacent sample pairs was 0.58 indicating some correlation between adjacent grades, that is if the grade of a sample is high, it is likely that the grade of an adjacent sample is also high, but, nevertheless, implying a relatively high nugget effect. This conclusion is supported by the variogram derived from the nickel assays of the drill hole intersections in the North Avebury mineralisation which has a nugget effect which is nearly 50% of the total variance (Figure 6a).

The correlation coefficient of grades of samples immediately on either side of the outer boundary was 0.01, that is, there is no correlation between nickel grades across this boundary. This results is as expected across a hard geological boundary. The correlation coefficients between grades of sample pairs across the internal boundary ranged from -0.49 to 0.28 suggesting that this is a geostatistically (and geologically) significant boundary (Figure 5).

The two boundary types are similar to those of the Kambalda nickel sulphide mineralisation described by Border (1991), that is, tabular mineralised zones with one wall gradational, the other wall sharp.

6.2 THE GLOBAL RESOURCE ESTIMATES

6.2.1 NORTH AVEBURY GLOBAL RESOURCE

The geological interpretation of the North Avebury mineralisation is shown on sections 354,500mE to 354,750mE (Figures 10 to 16) and a level plan at 2000mRL (Figure 17). The distribution of intersections within the North Avebury mineralisation is shown on a longitudinal projection (Figure 19).

The North Avebury nickel mineralisation strikes east west and dips north at about 70^{o} to 80^{o}. Known strike length is about 300 metres and the mineralisation extends from 2100m RL down to 1800m RL, the top being about 70 metres below the surface. The mineralisation ranges up to just over 30 metres in horizontal thickness, measured in a north-south direction.

The perimeters of the nickel mineralisation from sections 354,530mE to 354,750mE were digitised and wireframed using Datamine software. The east-west and vertical extent of the wireframe is shown on the longitudinal projection (Figure 19).

The volume of this wireframe was 673,000 cubic metres and, at a bulk density of 3.0 tonnes per cubic metre, the North Avebury mineralisation contains 2.02 million tonnes. Two checks of the tonnage were made manually, a polygonal estimate and an isoline estimate: the polygonal estimate was 1.8 million tonnes, the isoline estimate 1.9 million tonnes (Appendix 7). Given the subjective nature of the two manual estimates, which are both based on the determination of horizontal north-south thicknesses for each drillhole intersection, and, for the isoline method, in manual contouring of horizontal thickness and nickel grade, the agreement is considered to be reasonable.

As has been explained (5 RESOURCE ASSESSMENT CRITERIA) it was not necessary to composite samples. The histogram of nickel assays, histogram of the log of nickel assays and the cumulative frequency plot of the log of nickel assays all suggest that the nickel grades of drill core samples for the North Avebury mineralisation are lognormally distributed, and appear to be almost all drawn from the same parent population, and contain no rogue outliers (Figures 20 to 22). The same can be concluded for the sulphur assays (Figures 23 to 25).

An omnidirectional, or composite downhole, variogram was calculated from the nickel grades of the samples in the North Avebury mineralisation (Figure 6a). The variogram displays a relatively high nugget effect equal to nearly 50% of the total variance, and a range of 15 metres. The relatively high nugget effect, short range of the variogram, and the spatially sparse

grade data suggest that, at this stage, grades interpolated into small blocks using inverse distance weighting or kriging would oversmooth the block grade estimates and, for this report, only a global grade estimate has been made. When more data are available, the reliability of alternative grade interpolation methods, including inverse distance weighting, kriging and multiple indicator kriging (MIK), should be investigated.

There were 204 samples in the North Avebury mineralisation and the arithmetic average grade was 1.05% Ni, 1.50% S. The length weighted average grade was 1.04% Ni, 1.50% S. Sichel's t estimate of the average nickel grade, which attempts to take into account the lognormality of the data, was 1.08% Ni (Sichel, 1966 but using Table 2.6 from Rendu, 1978). Two checks of the global grade were made as part of manual estimates, a polygonal estimate and an isoline estimate: grade from the polygonal estimate was 1.02% Ni, grade from the isoline estimate was 0.97% Ni (Appendix 7).

The arithmetic average grade has been used and the grade of the North Avebury nickel mineralisation is estimated to be 1.05% Ni, 1.50% S.

6.2.2 SOUTH AVEBURY GLOBAL RESOURCE

The geological interpretation of the South Avebury mineralisation is shown on sections 354,445mE to 354,560mE (Figures 7 to 12) and a level plan at 2000mRL (Figure 17). The distribution of intersections within the South Avebury mineralisation is shown on a longitudinal projection (Figure 27).

The South Avebury nickel mineralisation strikes east west and dips north at about 50^{O} to 60^{O}. Known strike length is about 150 metres and the mineralisation extends from 2060m RL down to 1920m RL, the currently inferred top of the mineralisation being about 75 metres below the surface. The mineralisation is in two lenses, a north lens and a south lens, from about 20 to 40 metres in true thickness.

The perimeters of the two lenses of nickel mineralisation from sections 354,410mE to 354,560mE were digitised wireframed using Datamine software. The east-west and vertical extent of the wireframe is shown on the longitudinal projection (Figure 27).

The volume of this wireframes was 1,232,000 cubic metres and, at a bulk density of 3.0 tonnes per cubic metre, the South Avebury mineralisation contains 3.70 million tonnes. A check of

the volume was made manually using a cross sectional method, the volume estimate being 1,270,000 cubic metres (Appendix 7).

As has been explained (5 RESOURCE ASSESSMENT CRITERIA) it was not necessary to composite samples. The histogram of nickel assays, histogram of the log of nickel assays and the cumulative frequency plot of the log of nickel assays all suggest that the nickel grades of drill core samples for the South Avebury mineralisation are lognormally distributed and contain no rogue outliers (Figures 28 to 30). There is evidence of two populations of nickel assays in the cumulative frequency plot of the logs of the nickel grades which shows a kink at about 1% Ni suggesting populations of nickel assays above and below that grade, and, probably, two corresponding generations of nickel mineralisation (Figure 30). Similar conclusions can be made about the distribution of sulphur assays (Figures 31 to 33).

An omnidirectional, or composite downhole, pairwise relative variogram was calculated from the nickel grades of the samples in the South Avebury mineralisation (Figure 6b). The variogram displays a relatively high nugget effect equal to about 40% of the total variance; the nugget effect confirms the poor correlation between nickel assays in A024 and A024A described above (5 RESOURCE ASSESSMENT CRITERIA). The variogram has a saw tooth shape, reflecting the presence of two populations of nickel assays in the data, and a range of about 20 metres. The relatively high nugget effect, short range of the variogram, spatially sparse grade data, and the possibility of mixed nickel grade populations suggest that, at this stage, grades interpolated into small blocks using inverse distance weighting or kriging would be unreliable and, for this report, only a global grade estimate has been made. When more data are available, the reliability of alternative grade interpolation methods should be investigated.

There were 100 samples in the South Avebury mineralisation and the arithmetic average grade was 1.17% Ni, 1.86% S. The length weighted average grade was 1.15% Ni, 1.85% S. Sichel's t estimate of the average nickel grade, which attempts to take into account the lognormality of the data, was 1.21% Ni (Sichel 1966 but using Table 2.6 from Rendu, 1978).

The arithmetic average grade has been used and the grade of the South Avebury nickel mineralisation is estimated to be 1.17% Ni, 1.85%S.

6.3 OTHER RESOURCES

6.3.1 CENTRAL AVEBURY

The Central Avebury nickel mineralisation occurs sporadically along the southern, faulted contact of the northern serpentinite (Figure 35); at the present time continuity of the mineralisation cannot be assumed and no resource estimate has been made.

The highest grade intersection made to date at Avebury occurs at Central Avebury: A001 intersected 0.4m downhole length at 24.8% Ni, 20.4% S. This mineralisation was part of a broader zone of 13.7m downhole length at 1.46% Ni. Hole A018 passed through Central Avebury about 10 metres away from A001 and intersected 28.0m downhole length at 0.95% Ni. The two different grade intersections imply a high nugget effect in the mineralisation but, interestingly, both carried similar nickel accumulations:

A001 13.7 * 1.46 = 20.0 % Ni-metres
A018 28.0 * 0.95 = 26.6 % Ni-metres

6.3.2 OTHER METALS PRESENT

The arithmetic average grades of all elements for which there is a full set of data were:

North Avebury	1.05% Ni	1.50% S			
	275ppm Co	106ppm Cu	11ppm Pb	137ppm Zn	582ppm As
South Avebury	1.17% Ni	1.86% S			
	248ppm Co	90ppm Cu	2ppm Pb	195ppm Zn	307ppm As

The nickel to cobalt ratio for the two zones of mineralisation is about 40 to 50: 1 and it is expected that this ratio will be maintained in nickel concentrates. The levels of the other base metals is low, but chalcopyrite and sphalerite are occasionally seen in drill core and these may report to nickel concentrate. Arsenic is present as nickel arsenides and sulpharsenides (3 MINERALOGY) and some arsenic may report to nickel concentrate.

No precious metals have been identified in, or reported in assays from, the Avebury nickel mineralisation.

6.3.3 MAGNETITE

Magnetite is a pervasive component of the serpentinite and the zones of nickel mineralisation, being generally less abundant in the mineralisation. For this report, no estimate of the magnetite content of the mineralisation has been made. Given the known relationship between magnetic susceptibility of the core and magnetite content as measured by Davis Tube (McKeown, 1999), estimates of the magnetite grade of the mineralised nickel zones are possible. Magnetite is a potentially valuable product and, when it is possible to create block models of the mineralisation, estimates of magnetite content could be made.

6.4 THE MINERAL RESOURCE ESTIMATES

6.4.1 CUT-OFF GRADE

Global Mineral Resources in the zones of nickel mineralisation were estimated within geological boundaries and will contain parts which are economically mineable, and parts which are not. Cut-off grades cannot be calculated from cost data because cost data are not yet available. However, an estimate of cut-off grade is required to make a realistic inventory of the mineralisation which may be economically extractable.

At the nearby Renison Bell tin mine, the cut-off grade can be estimated to be 1.19% Sn (Appendix 8). The operations at Renison Bell are similar in several significant ways to operations which can be envisaged at Avebury:

	Avebury	*Renison Bell*
location	10km west of Zeehan	15km north of Zeehan
annual production	250,000 to 500,000 tonnes	750,000 tonnes
access to mine	decline	decline
stoping methods	benching, open stoping	cut & fill, benching, open stoping
production drills	electric	electric
loaders	diesel	diesel
haulage	diesel trucks	diesel trucks, shaft
concentrator	flotation	flotation, gravity separation
concentrate transport	truck to Burnie, ship to overseas smelter	truck to Burnie, ship to overseas smelter

As a result of these similarities, a cut-off grade for the Avebury mineralisation has been estimated by calculating an equivalent nickel grade to the Renison Bell cut-off grade, making allowance for differences in metal prices, mill recoveries, and smelter charges:

tin price	5,236	$US per tonne
nickel price	7,604	$US per tonne
tin mill recovery	70	%
nickel mill recovery	90	%
tin smelter charges	10	%
nickel smelter charges	25	%
exchange rate	0.54	$US per $A
tin price	9,696	$A per tonne
nickel price	14,081	$A per tonne
tin price received	8,726	$A per tonne
nickel price received	10,560	$A per tonne
tin grade	**1.19**	**% Sn**
equivalent nickel grade	**0.76**	**% Ni**

The practical applicability of a cut-off grade will depend on the spatial continuity of blocks of mineralisation defined by the use of the cut-off. At North Avebury, the tendency of the nickel grade to increase towards the outer boundary of the mineralisation (for example, Figure 18) and the occurrence of several holes where the average grade far exceeds the cut-off grade (Figure 19) together with an envisaged mining method which allows for block selection even after sill development has been completed (for example, benching) all suggest that a cut-off grade can be applied successfully. At South Avebury, the application of a cut-off grade will depend upon the selection of a mining method which is conducive to its use, but again tendency of the nickel grade to be higher near the outer boundaries of the mineralisation suggests that a cut-off grade can be applied successfully.

For this resource estimate, the cut-off grade for the Avebury mineralisation has been set at 0.75% Ni.

6.4.2 NORTH AVEBURY MINERAL RESOURCE

No cost estimates are available for mining or milling the Avebury mineralisation and, so, no parametric grade-tonnage estimates can be made. However, a grade-tonnage curve based on Formery's chart (David, 1977), has been made. This method shows tonnages of mineralisation above a particular cut-off grade, assuming that the deposit can be divided into equal sized

blocks the size of the geological samples, that is, in this case, into blocks of one cubic metre. Because this method tells us nothing about the spatial location of the blocks, and the size of the blocks is smaller than a practical selective mining unit, **the grade-tonnage curve should not be used to estimate recoverable tonnages**, that is, the grade-tonnage curve should not be used to estimate *reserves* from *resources* (Figure 26):

cut-off % Ni	**tonnes millions**	**grade % Ni**
0.0	2.02	1.05
0.4	1.56	1.30
0.5	1.37	1.41
0.6	1.19	1.57
0.7	1.05	1.67
0.8	0.93	1.78
0.9	0.81	1.92
1.0	0.69	2.10

At a cut-off grade of 0.75% Ni, the North Avebury Mineral Resource is 0.99 million tonnes at 1.74% Ni.

A better estimate can be made when there are sufficient data to interpolate grades into a block model made up of blocks the size of a selective mining unit.

6.4.3 SOUTH AVEBURY MINERAL RESOURCE

The comments made above regarding the grade-tonnage curve for the North Avebury mineralisation also apply to South Avebury, and **the grade-tonnage curve should not be used to estimate recoverable tonnages**, that is, the grade-tonnage curve should not be used to estimate *reserves* from *resources* (Figure 34):

cut-off % Ni	tonnes millions	grade % Ni
0.0	3.70	1.17
0.4	2.85	1.43
0.5	2.66	1.50
0.6	2.44	1.58
0.7	2.26	1.65
0.8	1.96	1.81
0.9	1.78	1.88
1.0	1.55	2.01

At a cut-off grade of 0.75% Ni, the South Avebury Mineral Resource is 2.11 million tonnes at 1.73% Ni.

A better estimate can be made when there are sufficient data to interpolate grades into a block model made up of blocks the size of a selective mining unit.

6.4.4 TOTAL AVEBURY MINERAL RESOURCE

Drilling at North Avebury has suggested continuity of the nickel mineralisation and the North Avebury Mineral Resource has been classified as an Indicated Mineral Resource. The South Avebury mineralisation is in a similar geological framework, and the mineralisation has been intersected in ten drill holes, but there are few drill holes within the defined Mineral Resource (Figure 27) so the South Avebury Mineral Resource has been classified as an Inferred Mineral Resource:

AVEBURY NICKEL PROJECT

INDICATED MINERAL RESOURCE
at 0.75% Ni cut-off grade

North Avebury	990,000 tonnes	1.74% Ni

INFERRED MINERAL RESOURCE
at 0.75% Ni cut-off grade

South Avebury	2,110,000 tonnes	1.73% Ni
Total	3,100,000 tonnes	1.73% Ni

The size of this Resource, compared to that at other nickel sulphide mines, is shown in Figure 36 (after a graph in Severin (2000) and using extra information from Anonymous (2000) and Macdonald (2000)). The size and grade of the Avebury mineralisation suggest that the Resource can be economically mined.

7 RECOMMENDATIONS FOR FUTURE WORK

If a mine is to be developed at Avebury, then those parts of the Mineral Resource which are economically mineable must be identified and, so, converted to Ore Reserves. To achieve this, more closely spaced geological data are required and a program of drilling to gain that data is shown in Figures 37 and 38. The total length of drilling involved is 10,000 metres and a budget estimate for the cost of the drilling is $125 per metre, for a total budget of $1,250,000. This cost includes drill mobilisation and de-mobilisation, supervision of the program, surveying, core storage, lithological and geotechnical core logging, and core splitting and assaying. The per metre cost is slightly higher than has been experienced to date at Avebury to allow for Maxibor surveys of each hole and triple tube drilling through and adjacent to the nickel mineralisation for geotechnical purposes.

The most economical way to complete 10,000 metres of drilling would be to drill two ten hour shifts per day, seven days per week, using two drill rigs. Assuming an output of 35 metres per drill per day, this would take 5 months.

To expedite core logging and transfer of geological data to digital form, a core logging system based on bar codes is recommended. This would mean that the geological interpretation and resource and reserve estimation processes could be achieved within four weeks of the receipt of the last assay data.

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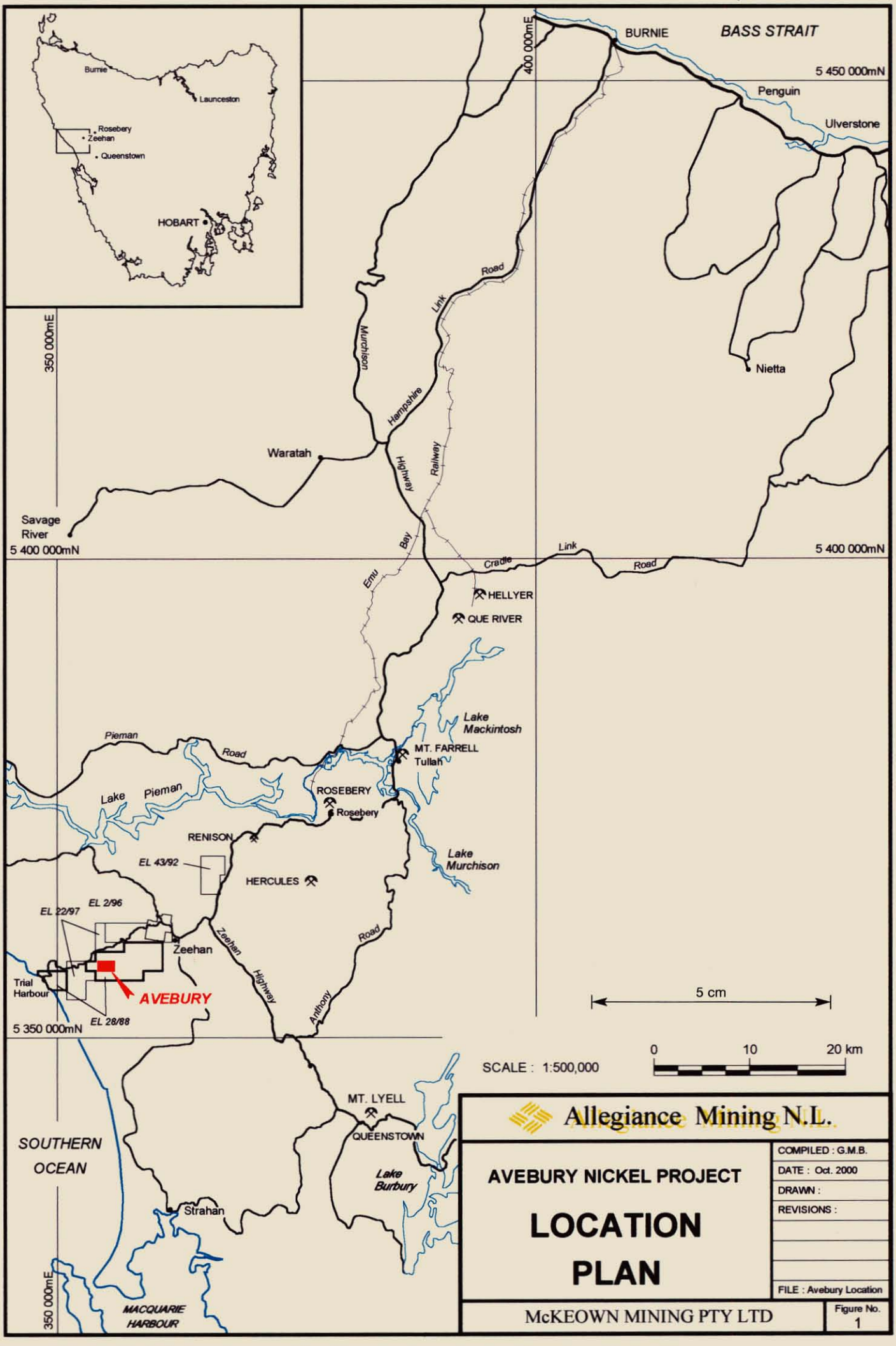
661184



Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

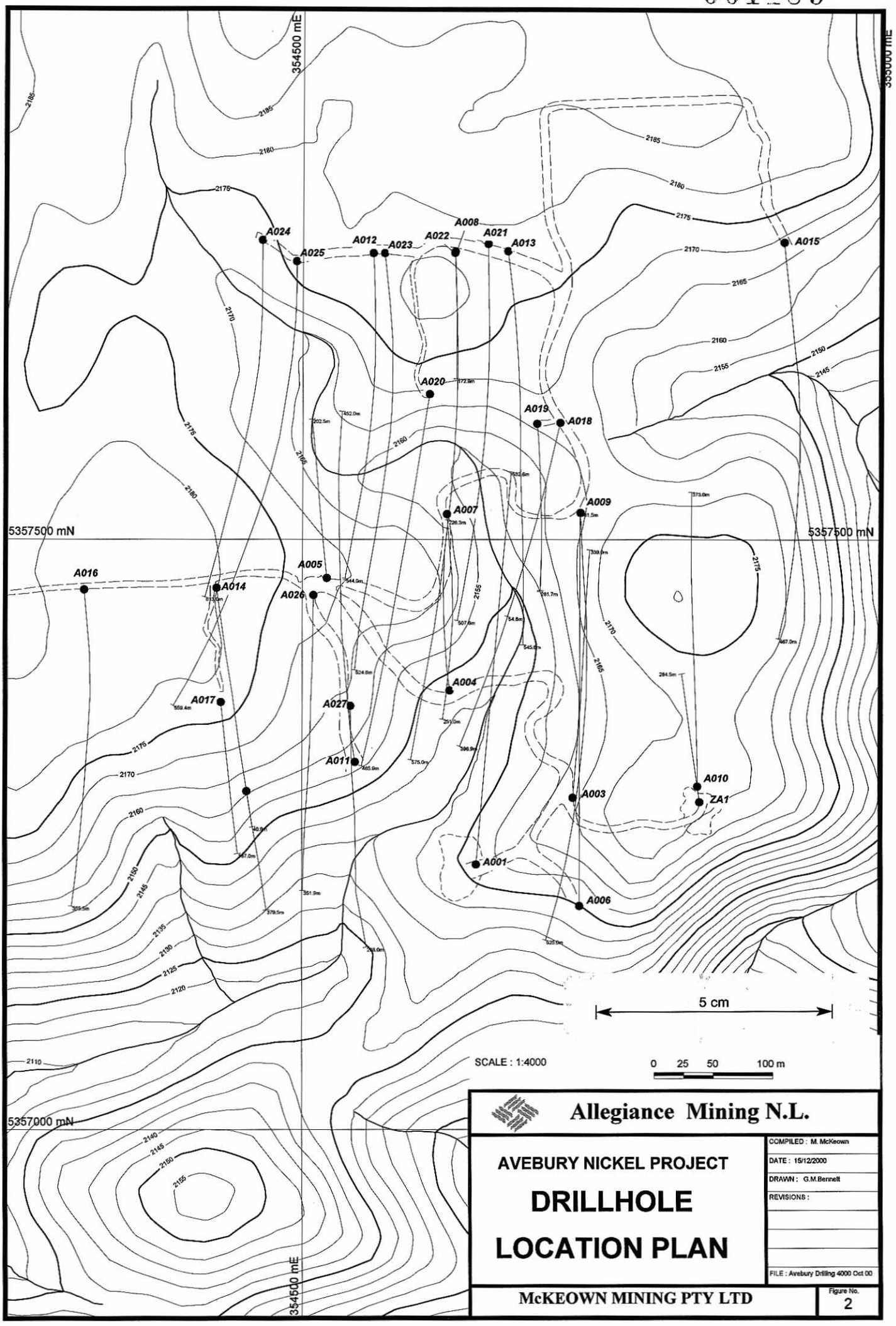
LOCATION PLAN

COMPILED : G.M.B.	
DATE : Oct. 2000	
DRAWN :	
REVISIONS :	
FILE : Avebury Location	

McKEOWN MINING PTY LTD

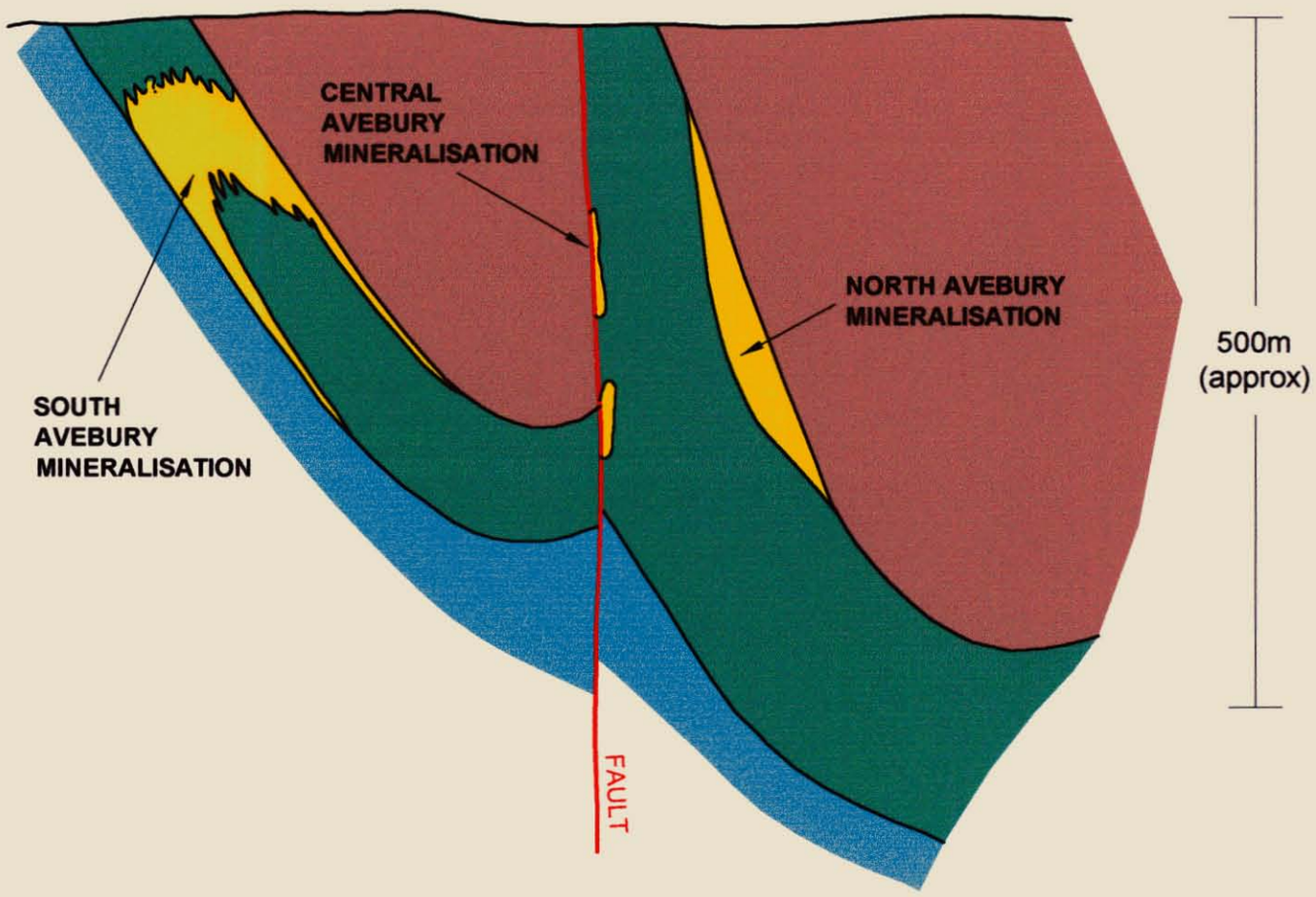
Figure No. 1

661185


354500 mE
354500 mE
5357500 mN
5357500 mN
5357000 mN
A024
A025
A012
A023
A022
A008
A021
A013
A015
A020
A019
A018
A009
A007
A016
A014
A005
A026
A017
A027
A004
A011
A010
ZA1
A003
A001
A006
5 cm
SCALE : 1:4000
0 25 50 100 m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
DRILLHOLE
LOCATION PLAN
COMPILED : M. McKeown
DATE : 15/12/2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : Avebury Drilling 4000 Oct 00
McKEOWN MINING PTY LTD
Figure No.
2

S

N



LEGEND

- Nickel mineralisation
- Serpentinite
- Hornfels and chert
- Chert, conglomerate, very altered rocks
- Fault

5 cm

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

GEOLOGICAL MODEL

SKETCH CROSS SECTION

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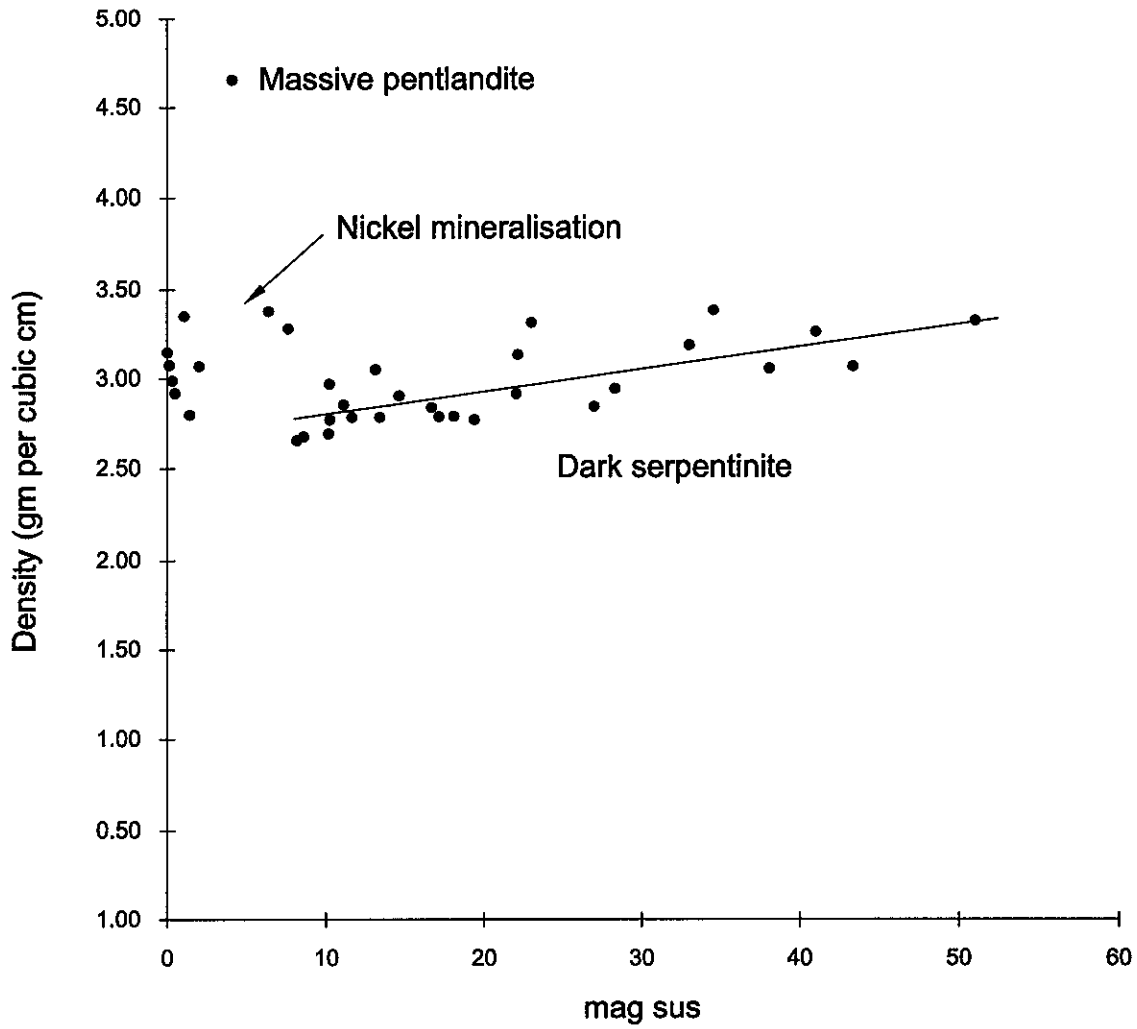
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : Geo Model SXS

McKEOWN MINING PTY LTD

Figure No. 3



(36 samples)

5 cm

Allegiance Mining NL

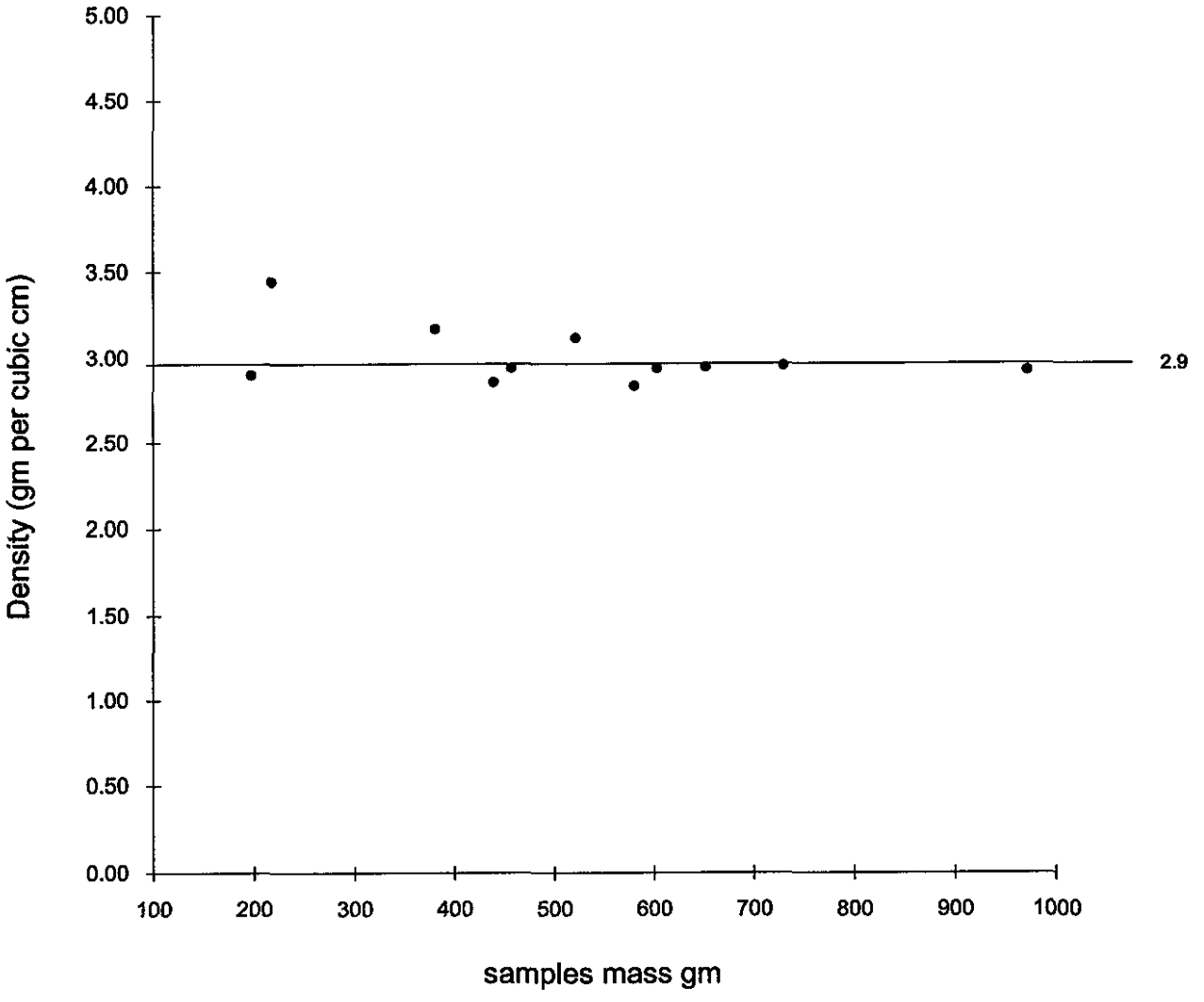
AVEBURY NICKEL PROJECT

DENSITIES OF SERPENTINITE SPECIMENS

COMPILED : M. McK
DATE : November 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : Serp Densities

McKEOWN MINING PTY LTD

Figure No. 4a



(11samples)

5 cm

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AVEBURY NICKEL PROJECT

DENSITIES OF WASTE SPECIMENS

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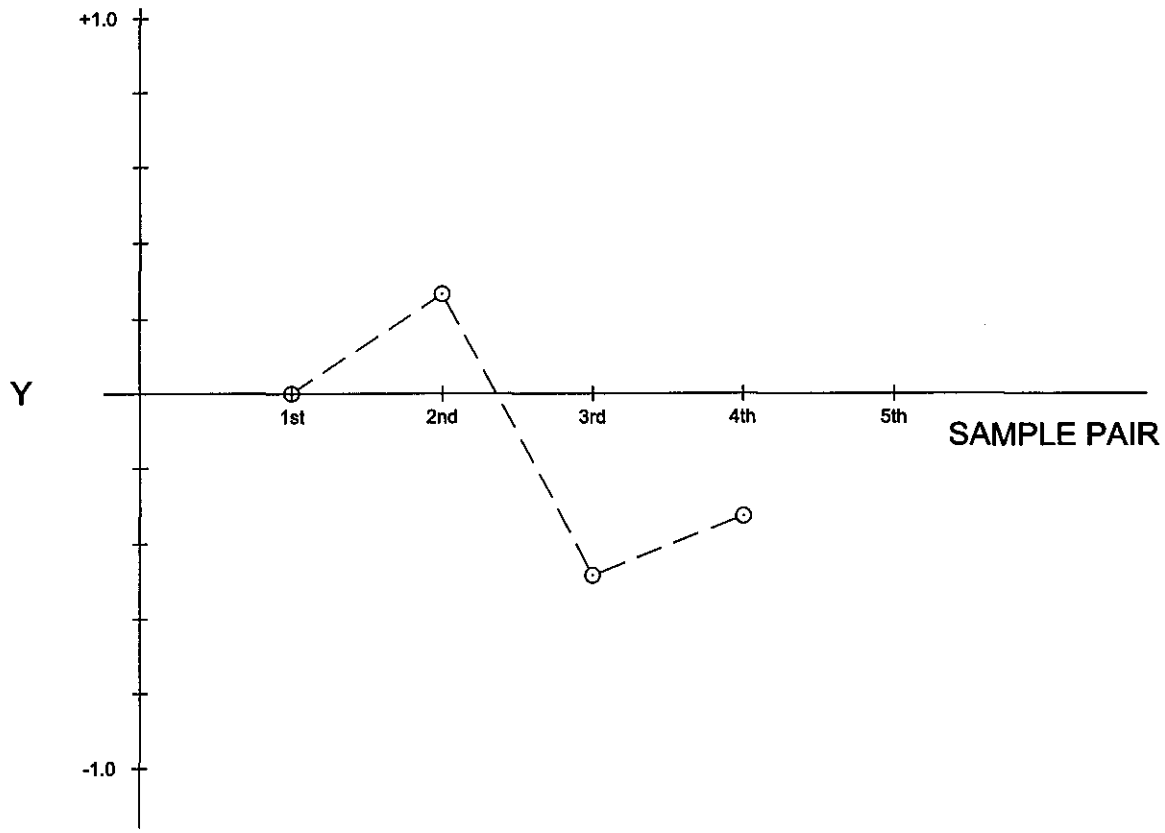
DRAWN : G.M.Bennett
REVISIONS :
FILE : Waste Gravities

McKEOWN MINING PTY LTD

Figure No. 4b



5 cm

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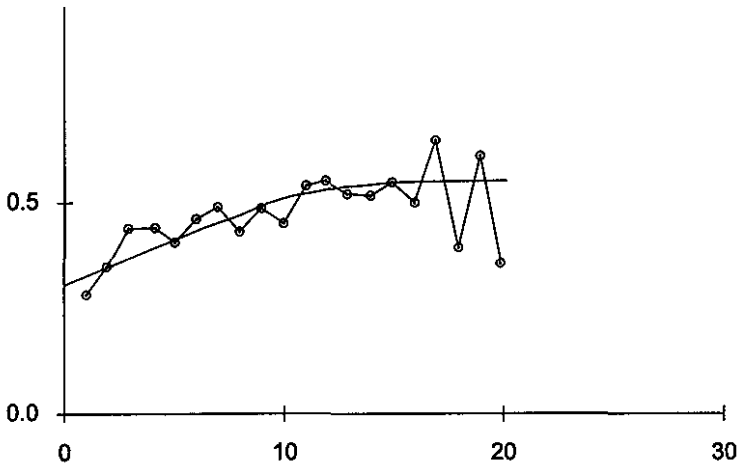
AVEBURY NICKEL PROJECT

NORTH AVEBURY CORRELATION COEFFICIENTS ACROSS SOFT BOUNDARY

COMPILED :	M. McK
DATE :	October 2000
DRAWN :	G.M.Bennett
REVISIONS :	
FILE :	CC Soft Boundary

McKEOWN MINING PTY LTD

Figure No. 5



Spherical: nugget 0.25, sill 0.55, range 15m

5 cm

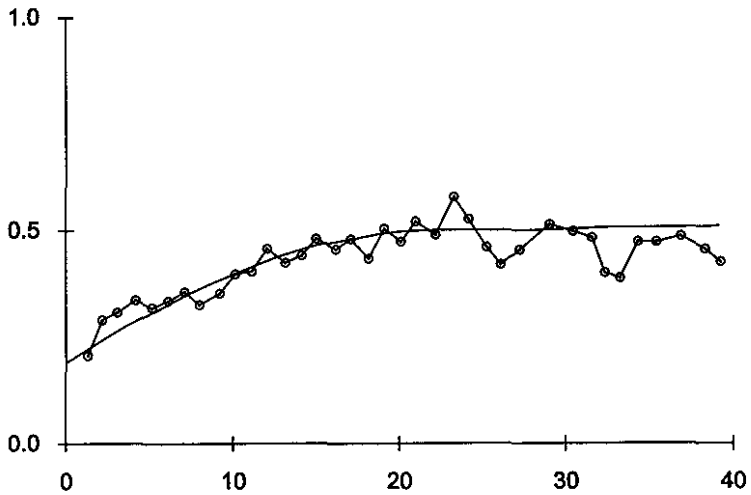
Allegiance Mining NL

AVEBURY NICKEL PROJECT
NORTH AVEBURY
MINERALISATION
VARIOGRAM

COMPILED : M. McK
DATE : November 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : NA Min V

McKEOWN MINING PTY LTD

Figure No. 6a



Spherical: nugget 0.20, sill 0.30, range 15m; sill 0.50, range 25m

5 cm

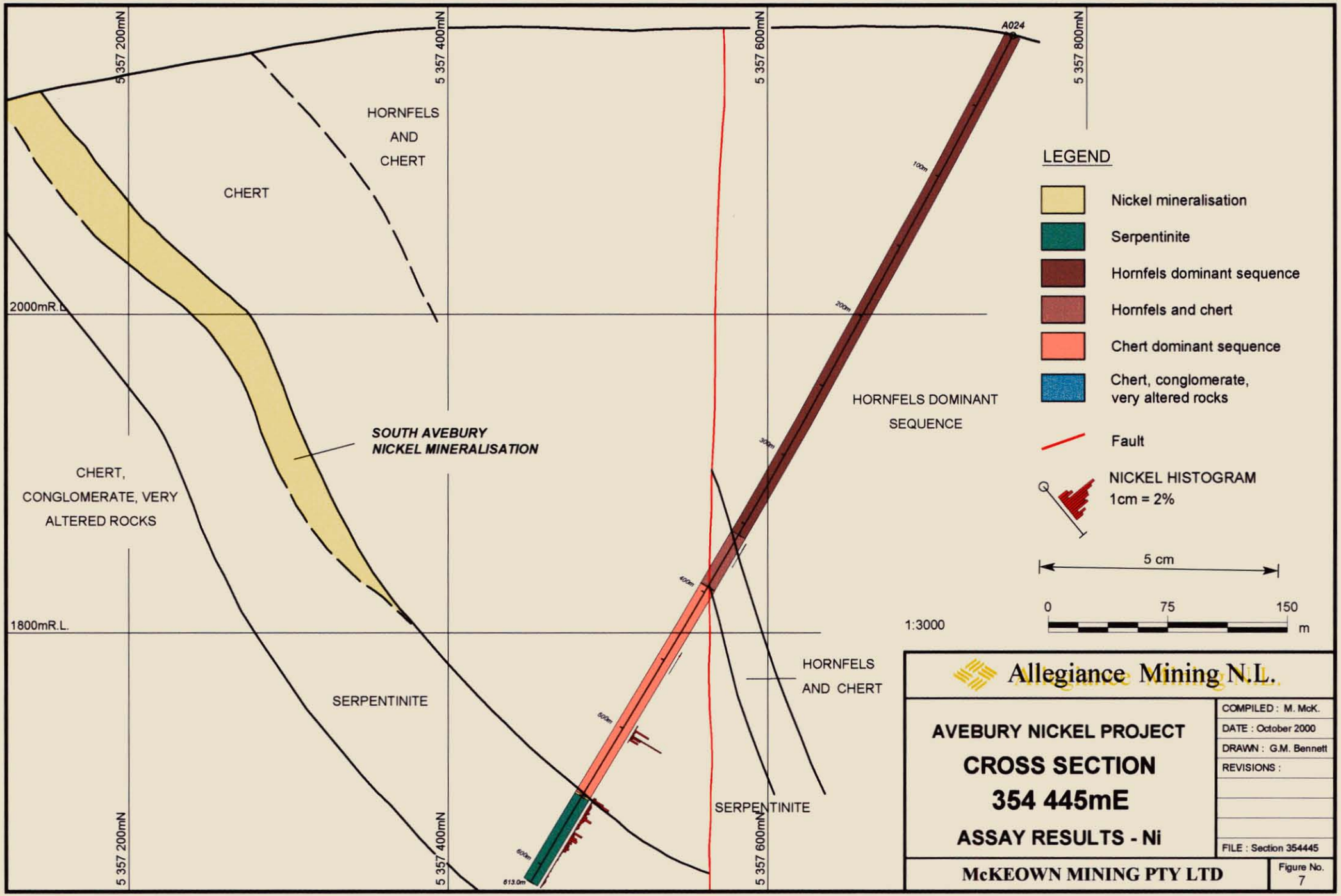
Allegiance Mining NL.

AVEBURY NICKEL PROJECT
SOUTH AVEBURY
MINERALISATION
PAIRWISE VARIOGRAM

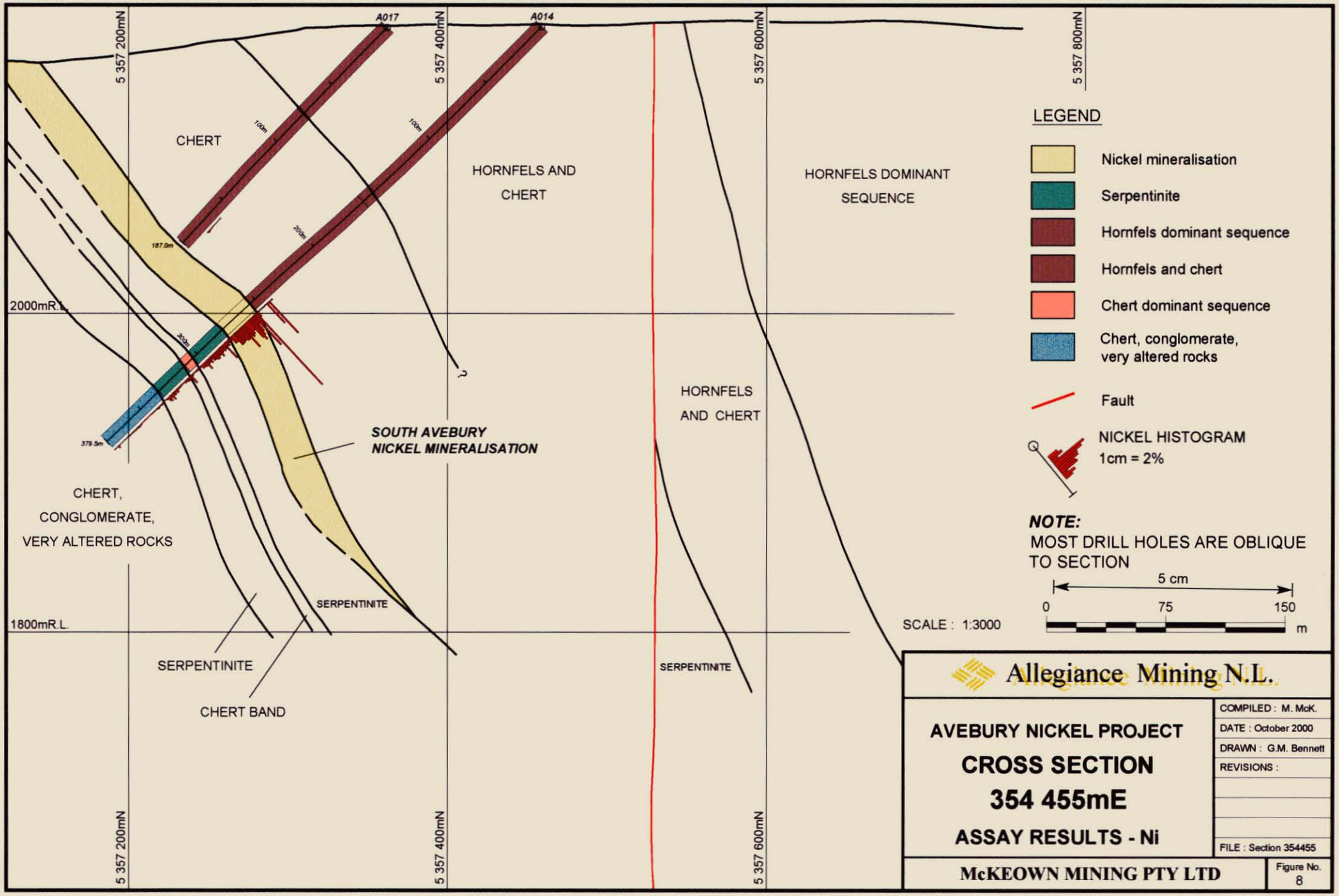
COMPILED : M. McK
DATE : November 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : SA Min PRV

McKEOWN MINING PTY LTD

Figure No. 6b


5 357 200mN
5 357 400mN
5 357 600mN
5 357 800mN
A024
HORNFELS
AND
CHERT
CHERT
2000mR.L.
1800mR.L.
SOUTH AVEBURY
NICKEL MINERALISATION
CHERT,
CONGLOMERATE, VERY
ALTERED ROCKS
SERPENTINITE
HORNFELS DOMINANT
SEQUENCE
HORNFELS
AND CHERT
SERPENTINITE
100m
200m
300m
400m
500m
600m
613.0m
LEGEND
Nickel mineralisation
Serpentinite
Hornfels dominant sequence
Hornfels and chert
Chert dominant sequence
Chert, conglomerate,
very altered rocks
Fault
NICKEL HISTOGRAM
1cm = 2%
5 cm
0
75
150
m
1:3000
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
CROSS SECTION
354 445mE
ASSAY RESULTS - Ni
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : Section 354445
McKEOWN MINING PTY LTD
Figure No.
7

69169


5 357 200mN
5 357 400mN
5 357 600mN
5 357 800mN
A017
A014
CHERT
HORNFELS AND CHERT
HORNFELS DOMINANT SEQUENCE
100m
100m
187.0m
200m
2000mR.L.
300m
379.5m
SOUTH AVEBURY NICKEL MINERALISATION
HORNFELS AND CHERT
CHERT, CONGLOMERATE, VERY ALTERED ROCKS
SERPENTINITE
1800mR.L.
SERPENTINITE
CHERT BAND
SERPENTINITE
LEGEND
Nickel mineralisation
Serpentinite
Hornfels dominant sequence
Hornfels and chert
Chert dominant sequence
Chert, conglomerate, very altered rocks
Fault
NICKEL HISTOGRAM
1cm = 2%
NOTE:
MOST DRILL HOLES ARE OBLIQUE TO SECTION
5 cm
0
75
150
m
SCALE : 1:3000
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
CROSS SECTION
354 455mE
ASSAY RESULTS - Ni
COMPILED : M. McK.
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DRAWN : G.M. Bennett
REVISIONS :
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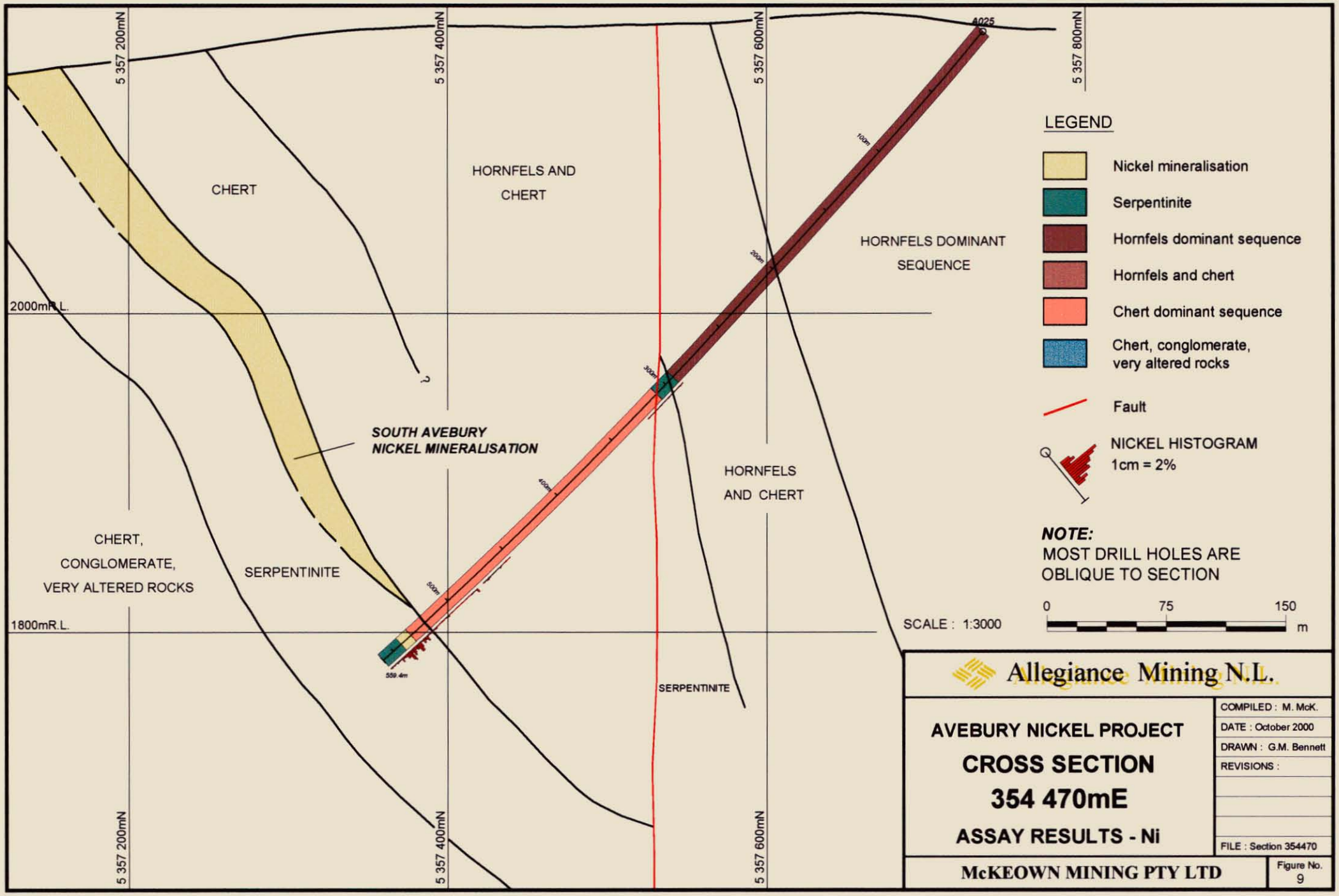
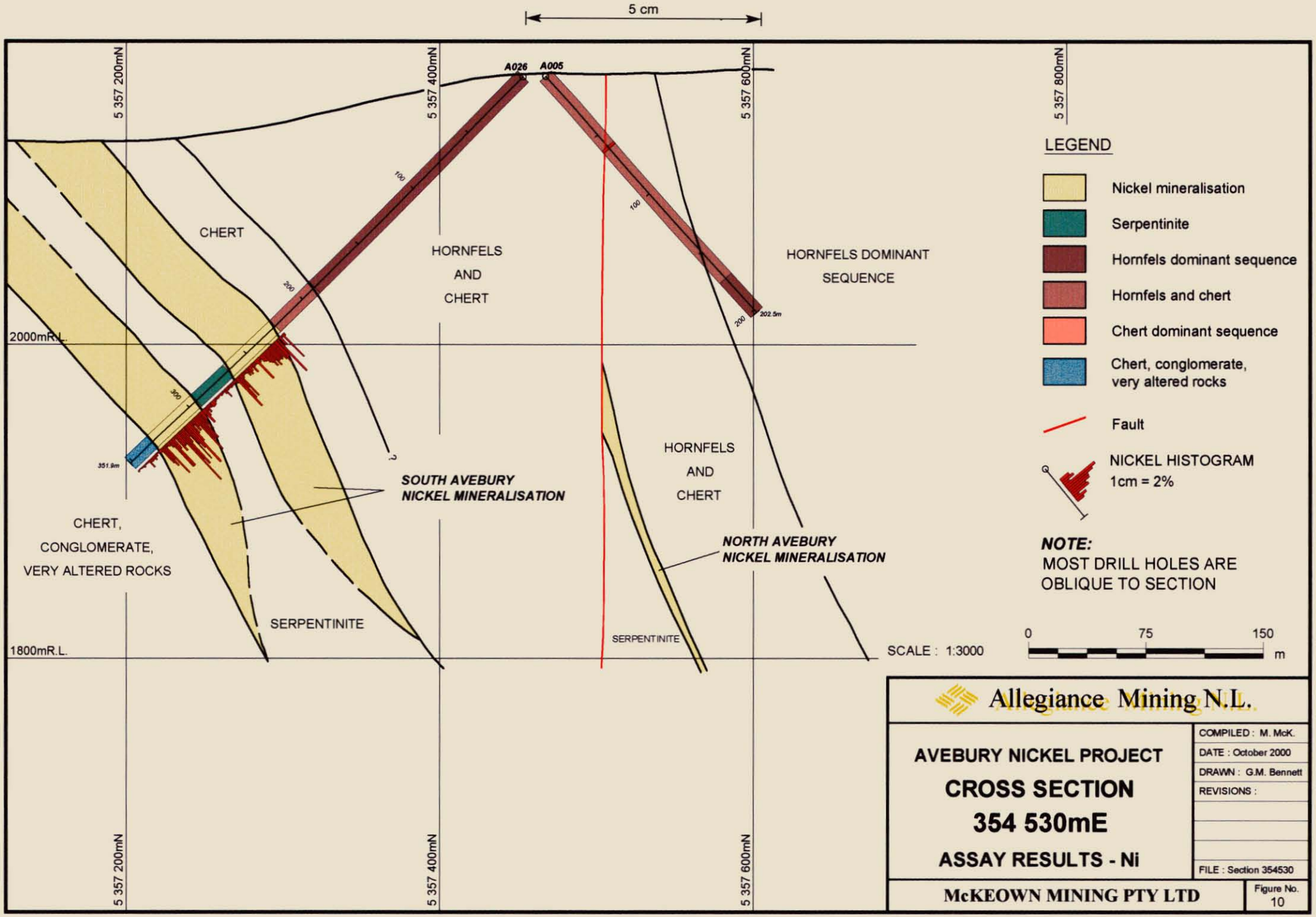
Figure No. 8

69143


5 cm
5 357 200mN
5 357 400mN
5 357 600mN
5 357 800mN
4025
CHERT
HORNFELS AND
CHERT
HORNFELS DOMINANT
SEQUENCE
2000mR.L.
SOUTH AVEBURY
NICKEL MINERALISATION
HORNFELS
AND CHERT
CHERT,
CONGLOMERATE,
VERY ALTERED ROCKS
SERPENTINITE
1800mR.L.
SERPENTINITE
559.4m
LEGEND
Nickel mineralisation
Serpentinite
Hornfels dominant sequence
Hornfels and chert
Chert dominant sequence
Chert, conglomerate,
very altered rocks
Fault
NICKEL HISTOGRAM
1cm = 2%
NOTE:
MOST DRILL HOLES ARE
OBLIQUE TO SECTION
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
CROSS SECTION
354 470mE
ASSAY RESULTS - Ni
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : Section 354470
McKEOWN MINING PTY LTD
Figure No.
9



Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

CROSS SECTION

354 530mE

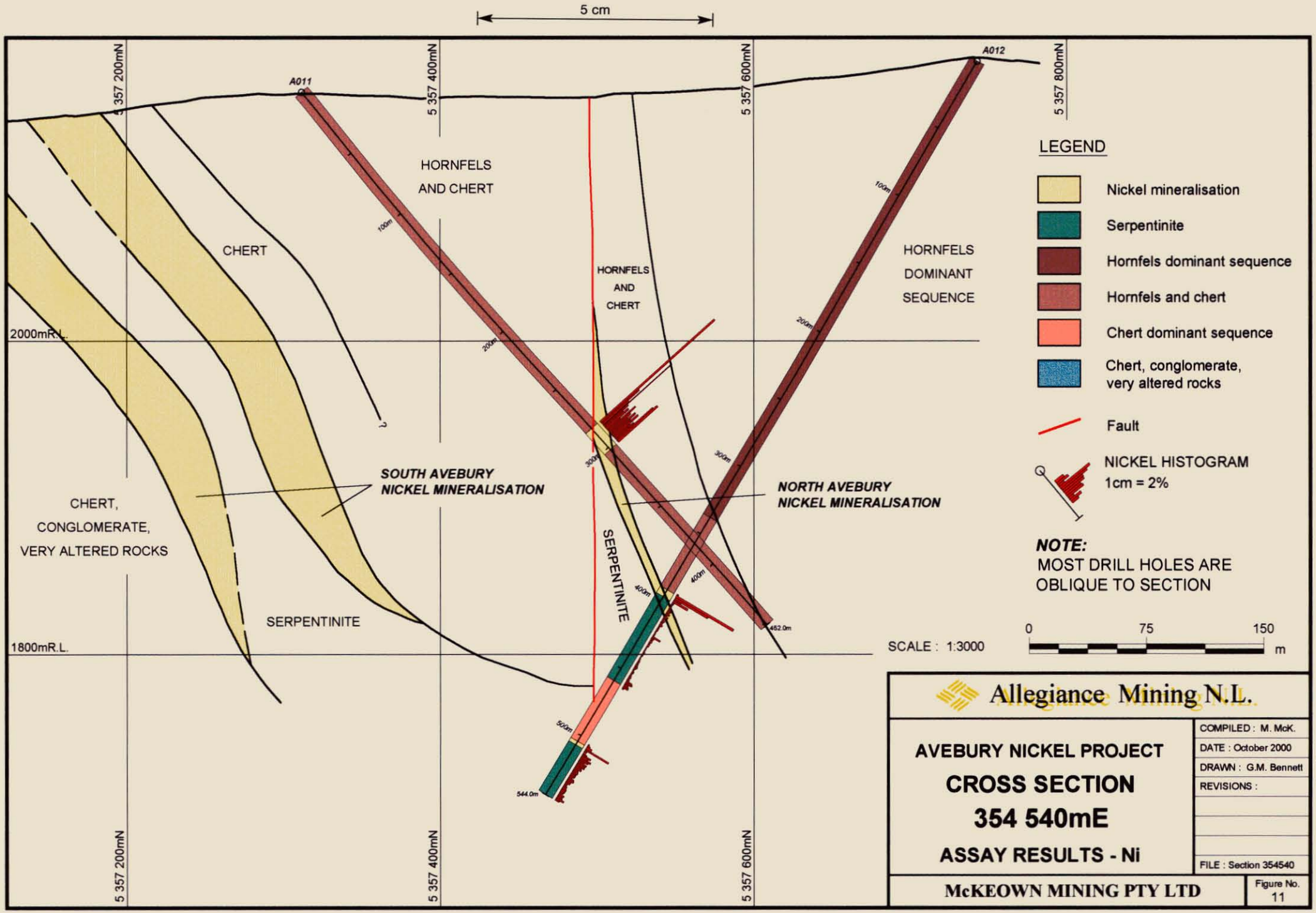
ASSAY RESULTS - Ni

COMPILED :	M. McK.
DATE :	October 2000
DRAWN :	G.M. Bennett
REVISIONS :	
FILE :	Section 354530

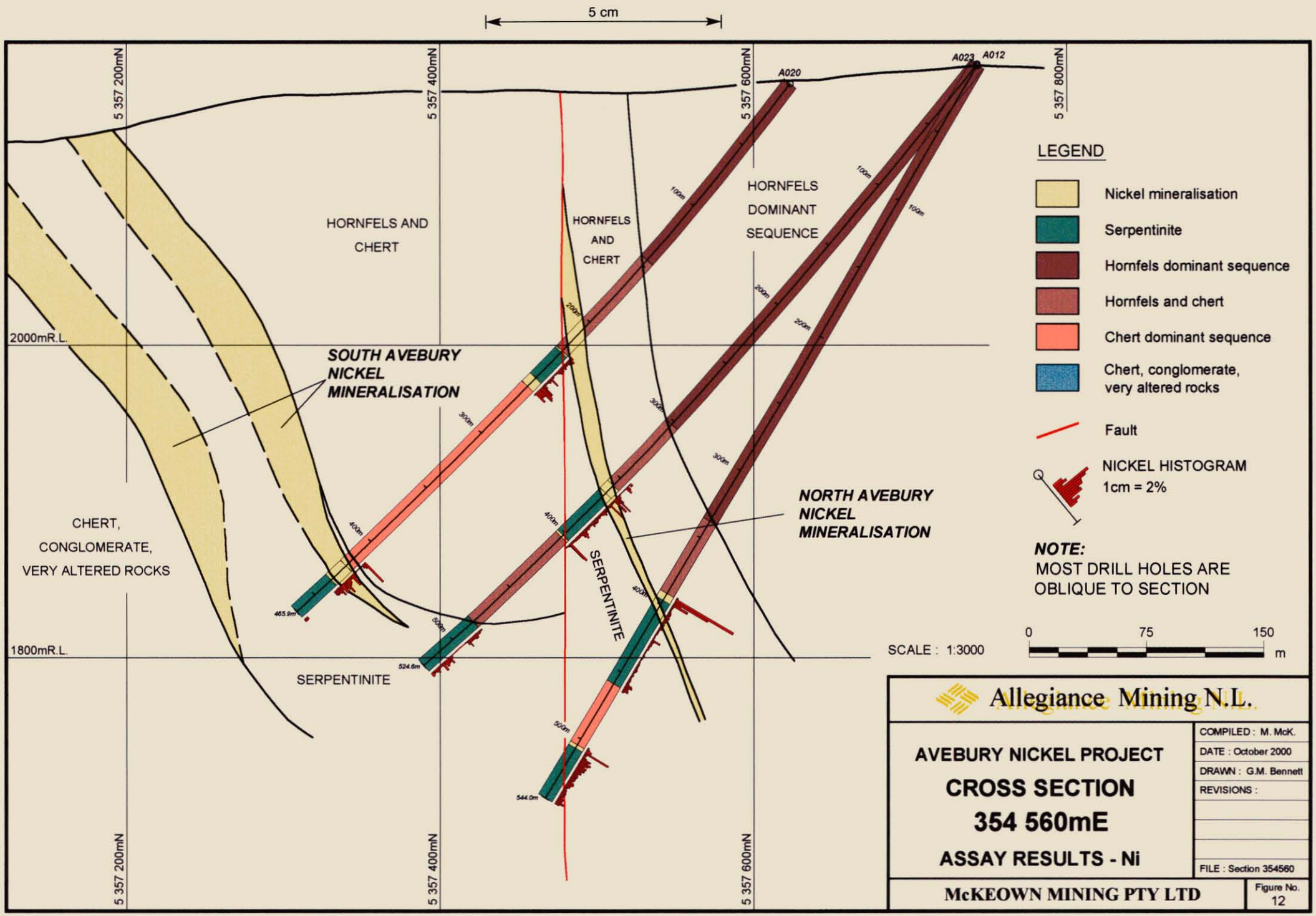
McKEOWN MINING PTY LTD

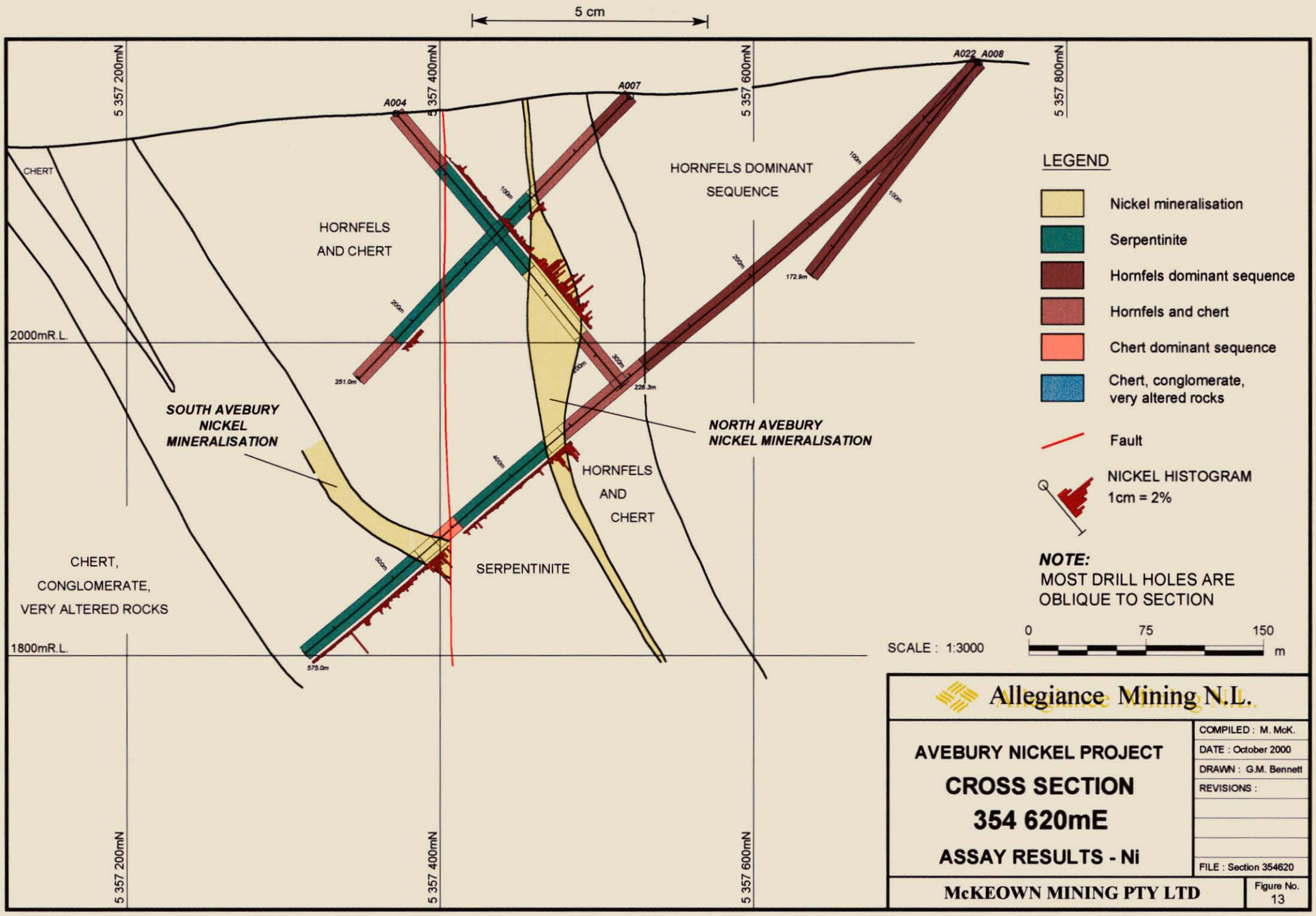
Figure No. 10

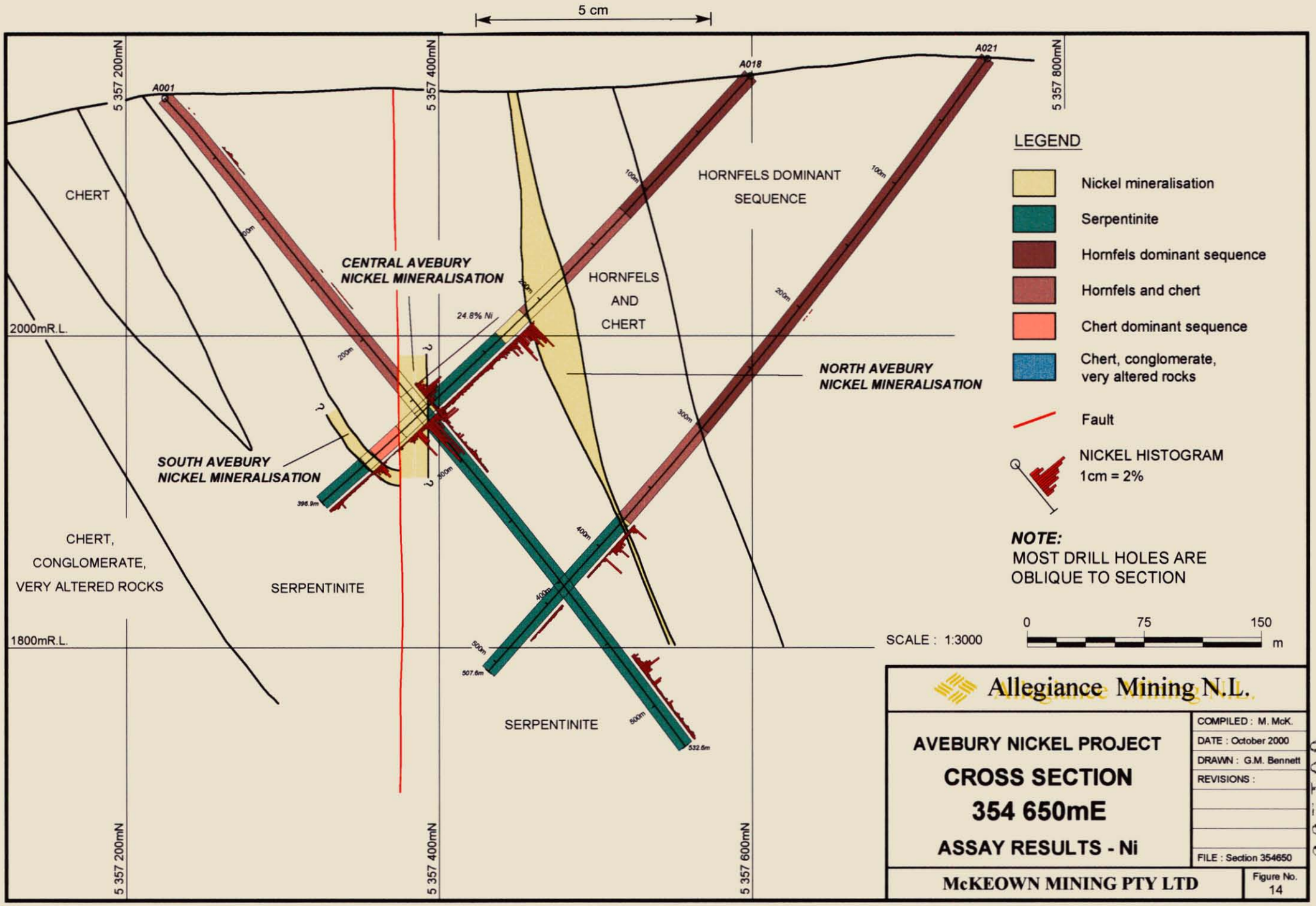
6993


5 cm
5 357 200mN
5 357 400mN
5 357 600mN
5 357 800mN
A011
A012
HORNFELS
AND CHERT
CHERT
HORNFELS
AND
CHERT
HORNFELS
DOMINANT
SEQUENCE
2000mR.L.
1800mR.L.
SOUTH AVEBURY
NICKEL MINERALISATION
NORTH AVEBURY
NICKEL MINERALISATION
CHERT,
CONGLOMERATE,
VERY ALTERED ROCKS
SERPENTINITE
SERPENTINITE
100m
200m
300m
400m
500m
452.0m
544.0m
LEGEND
Nickel mineralisation
Serpentinite
Hornfels dominant sequence
Hornfels and chert
Chert dominant sequence
Chert, conglomerate,
very altered rocks
Fault
NICKEL HISTOGRAM
1cm = 2%
NOTE:
MOST DRILL HOLES ARE
OBLIQUE TO SECTION
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
CROSS SECTION
354 540mE
ASSAY RESULTS - Ni
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : Section 354540
McKEOWN MINING PTY LTD
Figure No.
11

69-1369


5 cm
5 357 200mN
5 357 400mN
5 357 600mN
5 357 800mN
A020
A023
A012
2000mR.L.
1800mR.L.
HORNFELS AND CHERT
HORNFELS AND CHERT
HORNFELS DOMINANT SEQUENCE
SOUTH AVEBURY NICKEL MINERALISATION
NORTH AVEBURY NICKEL MINERALISATION
SERPENTINITE
CHERT, CONGLOMERATE, VERY ALTERED ROCKS
SERPENTINITE
100m
200m
300m
400m
500m
465.9m
524.6m
544.0m
LEGEND
Nickel mineralisation
Serpentinite
Hornfels dominant sequence
Hornfels and chert
Chert dominant sequence
Chert, conglomerate, very altered rocks
Fault
NICKEL HISTOGRAM
1cm = 2%
NOTE:
MOST DRILL HOLES ARE OBLIQUE TO SECTION
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
CROSS SECTION
354 560mE
ASSAY RESULTS - Ni
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : Section 354560
McKEOWN MINING PTY LTD
Figure No. 12


5 cm
5 357 200mN
5 357 400mN
5 357 600mN
5 357 800mN
A004
A007
A022
A008
CHERT
HORNFELS
AND CHERT
HORNFELS DOMINANT
SEQUENCE
2000mR.L.
1800mR.L.
SOUTH AVEBURY
NICKEL
MINERALISATION
NORTH AVEBURY
NICKEL MINERALISATION
HORNFELS
AND
CHERT
SERPENTINITE
CHERT,
CONGLOMERATE,
VERY ALTERED ROCKS
100m
200m
300m
400m
500m
251.0m
226.3m
172.9m
573.0m
LEGEND
Nickel mineralisation
Serpentinite
Hornfels dominant sequence
Hornfels and chert
Chert dominant sequence
Chert, conglomerate,
very altered rocks
Fault
NICKEL HISTOGRAM
1cm = 2%
NOTE:
MOST DRILL HOLES ARE
OBLIQUE TO SECTION
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
CROSS SECTION
354 620mE
ASSAY RESULTS - Ni
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : Section 354620
McKEOWN MINING PTY LTD
Figure No.
13



Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

CROSS SECTION

354 650mE

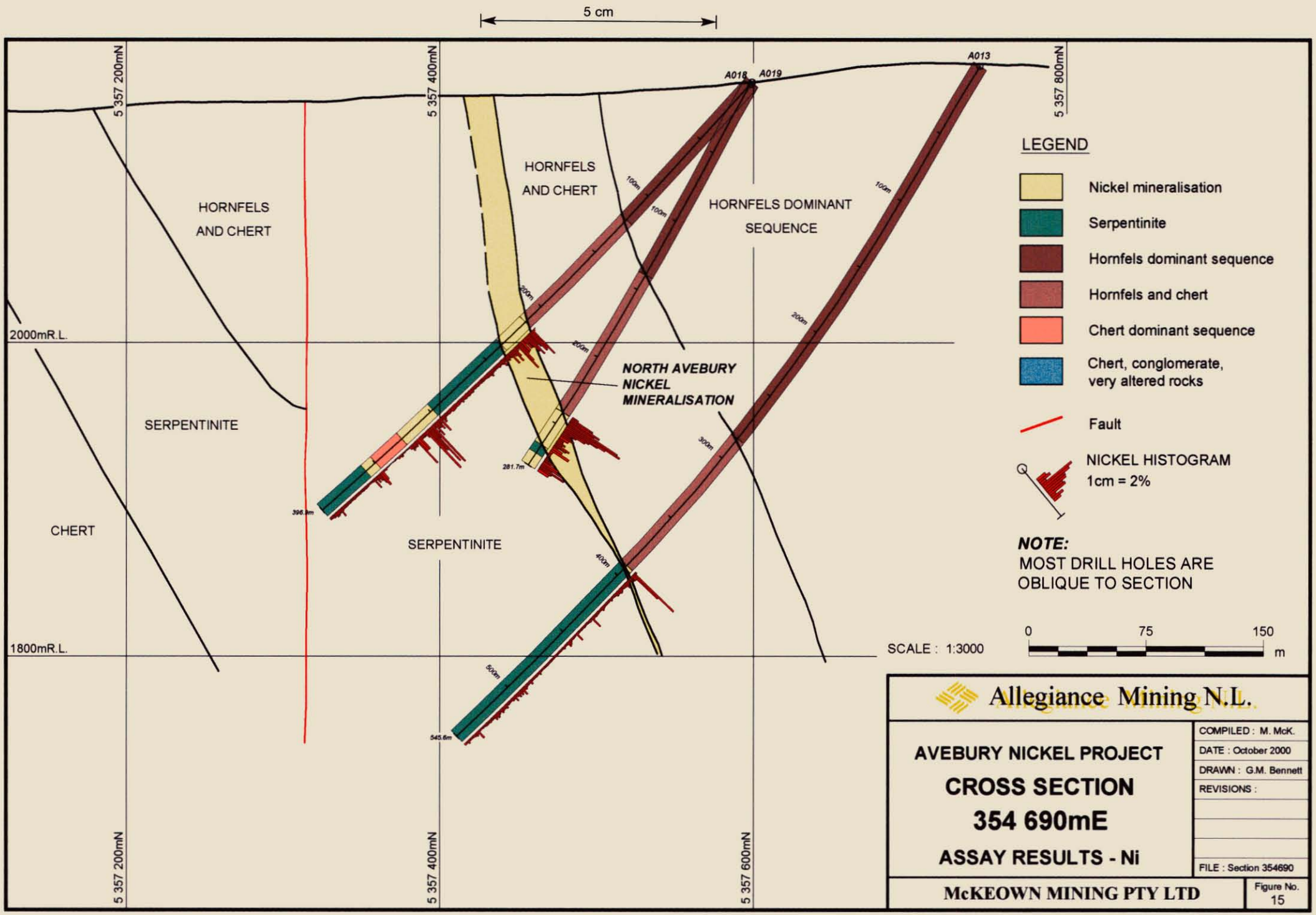
ASSAY RESULTS - Ni

COMPILED :	M. McK.
DATE :	October 2000
DRAWN :	G.M. Bennett
REVISIONS :	
FILE :	Section 354650

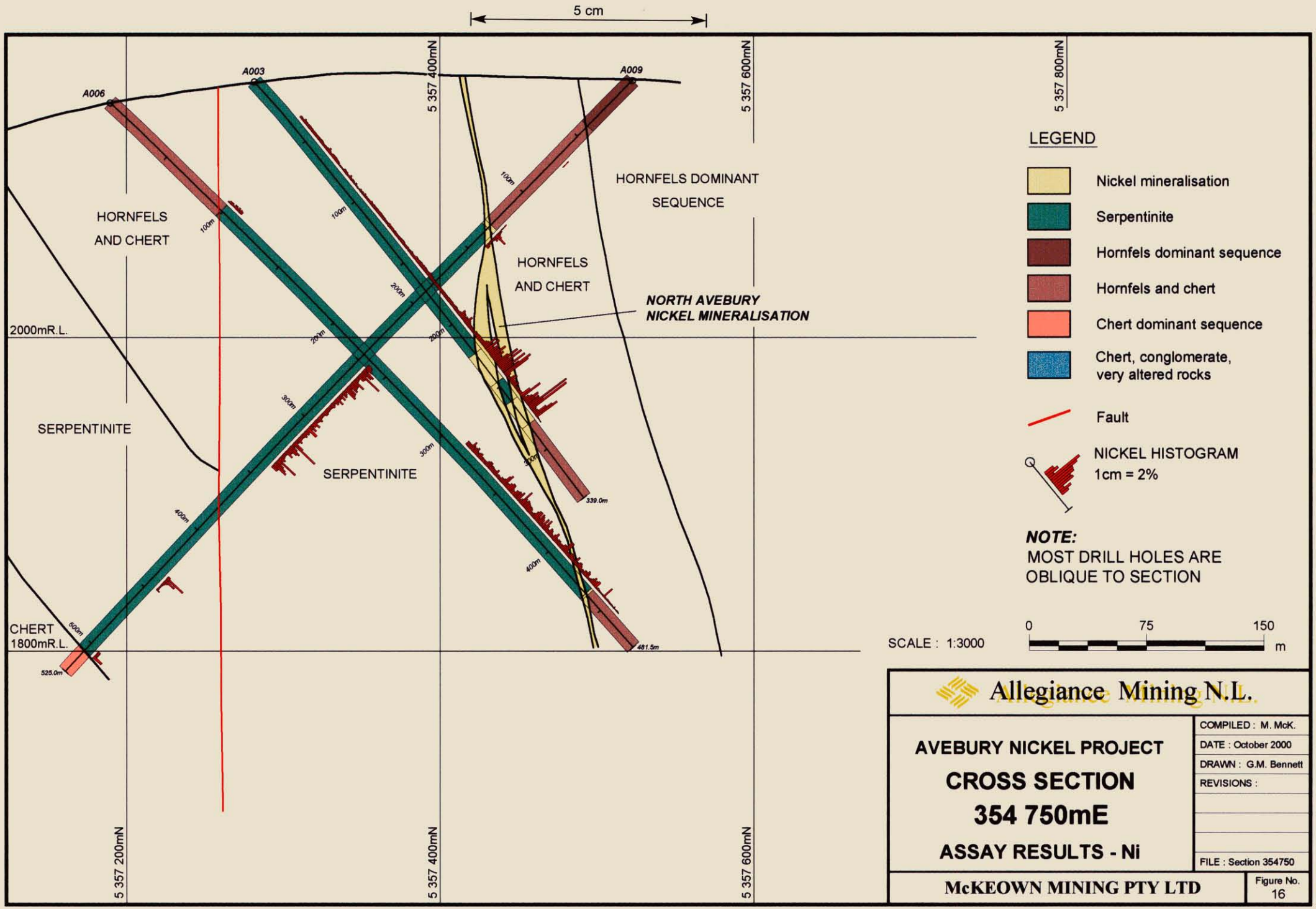
McKEOWN MINING PTY LTD

Figure No. 14

9361-1369


5 cm
5 357 200mN
5 357 400mN
5 357 800mN
5 357 600mN
A013
A018
A019
HORNFELS AND CHERT
HORNFELS AND CHERT
HORNFELS DOMINANT SEQUENCE
SERPENTINITE
SERPENTINITE
CHERT
NORTH AVEBURY NICKEL MINERALISATION
2000mR.L.
1800mR.L.
100m
200m
300m
400m
500m
396.9m
281.7m
545.6m
LEGEND
Nickel mineralisation
Serpentinite
Hornfels dominant sequence
Hornfels and chert
Chert dominant sequence
Chert, conglomerate, very altered rocks
Fault
NICKEL HISTOGRAM
1cm = 2%
NOTE:
MOST DRILL HOLES ARE OBLIQUE TO SECTION
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
CROSS SECTION
354 690mE
ASSAY RESULTS - Ni
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : Section 354690
McKEOWN MINING PTY LTD
Figure No. 15

002699



Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

CROSS SECTION

354 750mE

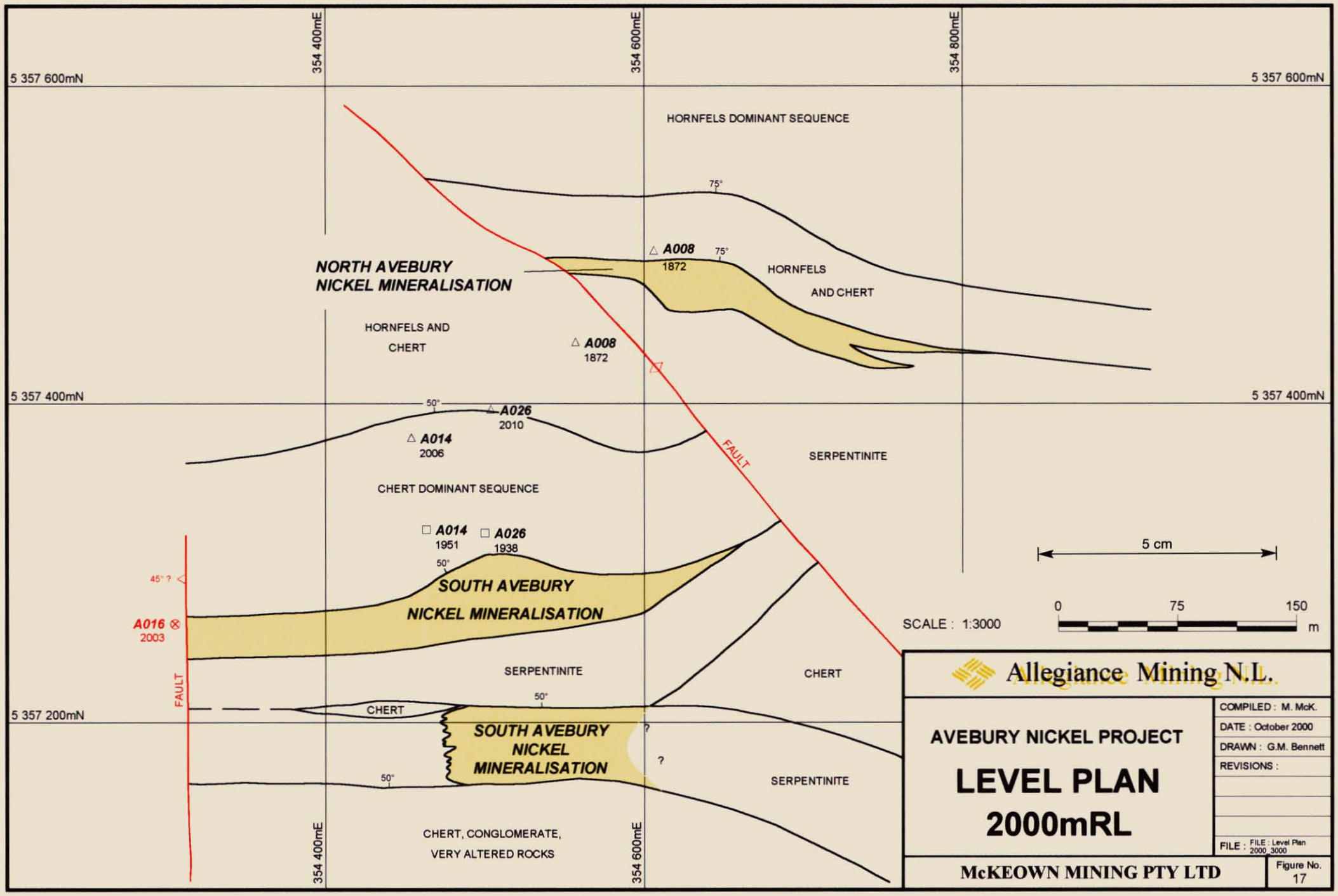
ASSAY RESULTS - Ni

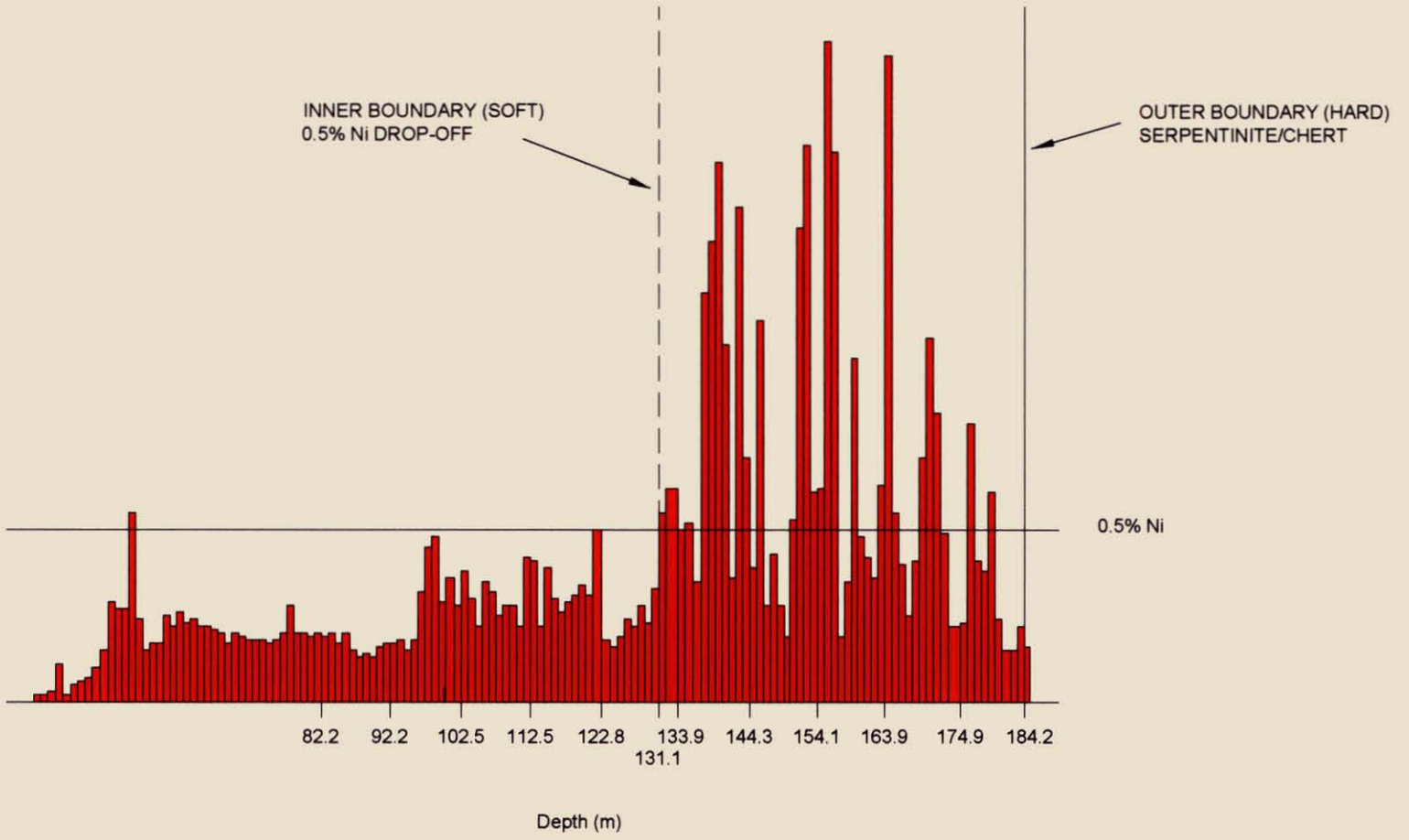
COMPILED :	M. McK.
DATE :	October 2000
DRAWN :	G.M. Bennett
REVISIONS :	
FILE :	Section 354750

McKEOWN MINING PTY LTD

Figure No. 16

T03699


5 357 600mN
354 400mE
354 600mE
354 800mE
HORNFELS DOMINANT SEQUENCE
75°
NORTH AVEBURY
NICKEL MINERALISATION
A008
1872
75°
HORNFELS
AND CHERT
HORNFELS AND
CHERT
A008
1872
5 357 400mN
50°
A026
2010
A014
2006
FAULT
SERPENTINITE
CHERT DOMINANT SEQUENCE
A014
1951
A026
1938
50°
45° ?
SOUTH AVEBURY
NICKEL MINERALISATION
A016
2003
FAULT
SERPENTINITE
CHERT
50°
5 357 200mN
CHERT
SOUTH AVEBURY
NICKEL
MINERALISATION
?
?
50°
SERPENTINITE
CHERT, CONGLOMERATE,
VERY ALTERED ROCKS
5 cm
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
LEVEL PLAN
2000mRL
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : FILE : Level Plan 2000_3000
McKEOWN MINING PTY LTD
Figure No.
17



0 0.5%

SCALE

5 cm

Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

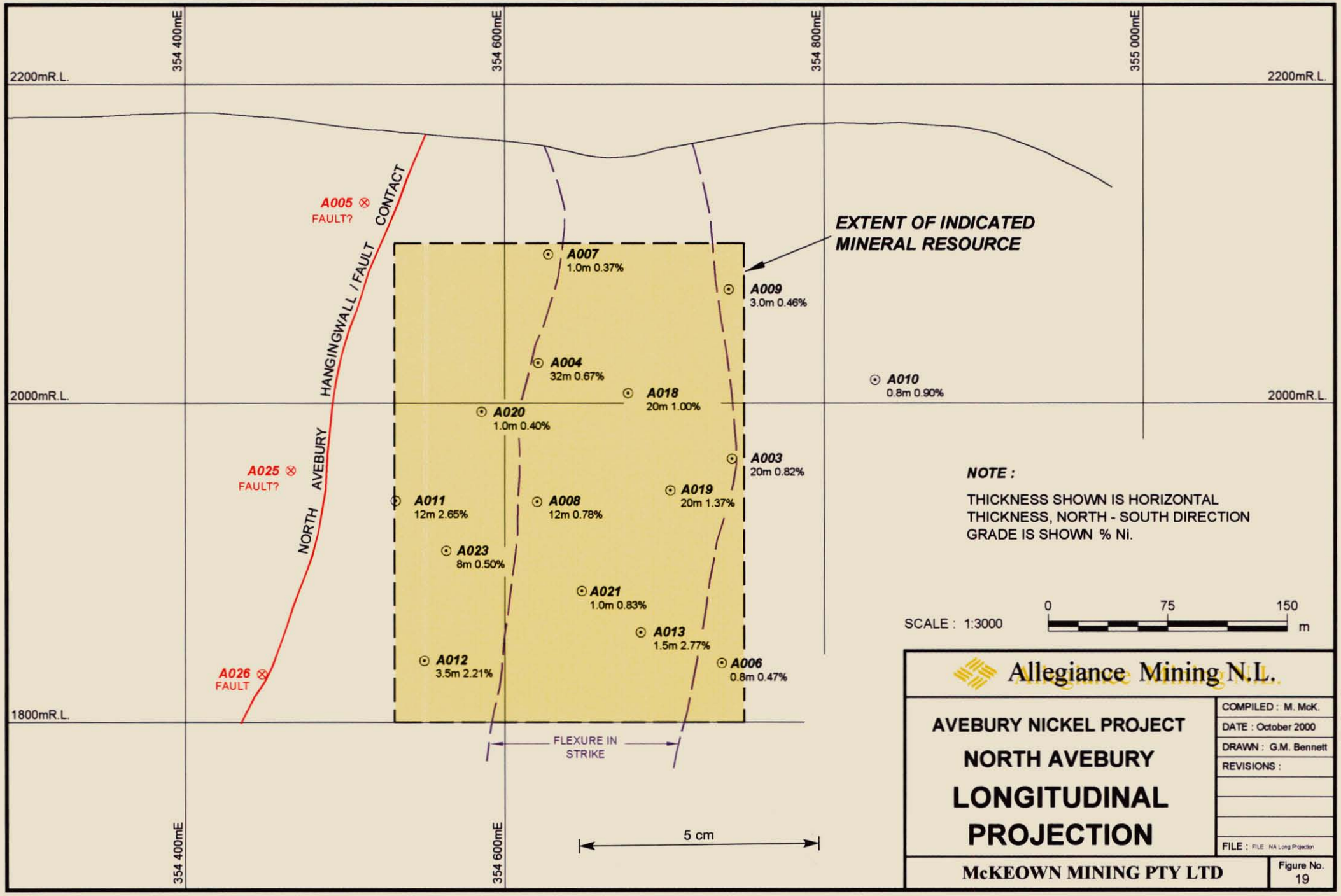
HISTOGRAM OF NICKEL ASSAYS IN A004

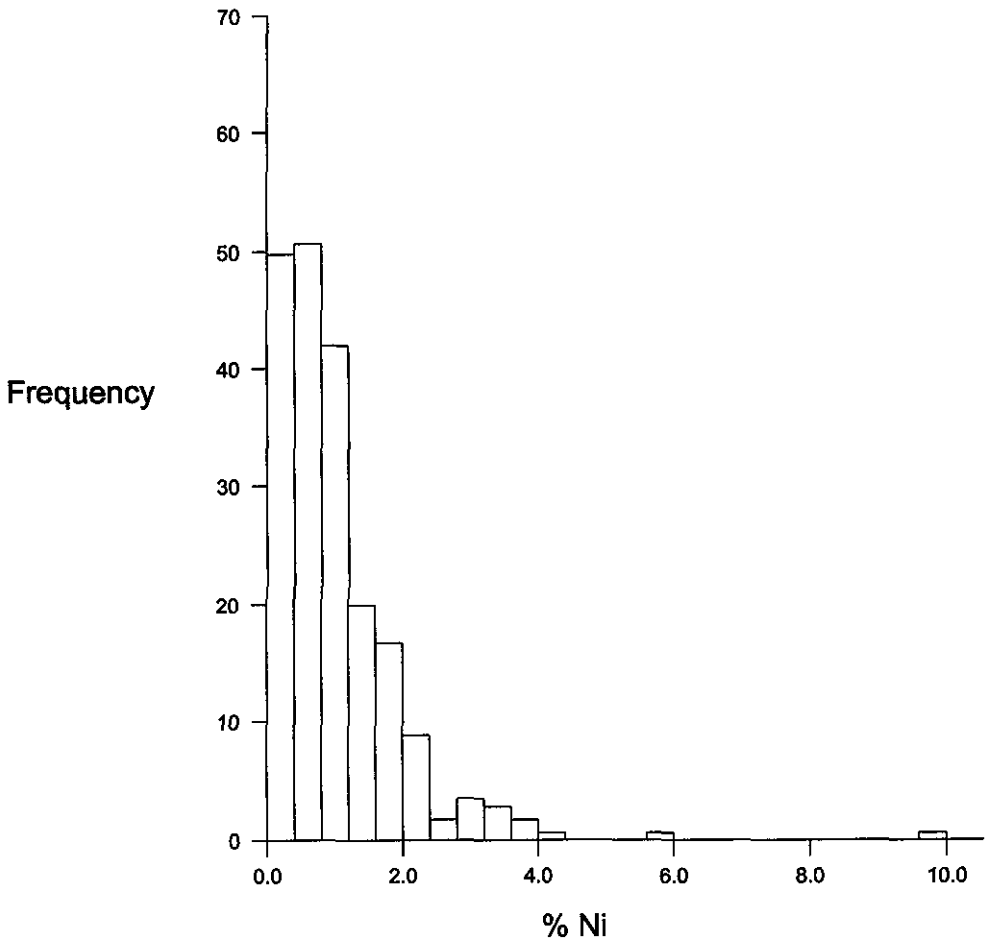
COMPILED : M. McK
DATE : December 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : A004 Histo Ni

McKEOWN MINING PTY LTD

Figure No. 18

69603


354 400mE
354 600mE
354 800mE
355 000mE
2200mR.L.
2000mR.L.
1800mR.L.
A005
FAULT?
CONTACT
HANGINGWALL / FAULT
NORTH AVEBURY
A025
FAULT?
A026
FAULT
EXTENT OF INDICATED MINERAL RESOURCE
A007
1.0m 0.37%
A009
3.0m 0.46%
A004
32m 0.67%
A018
20m 1.00%
A010
0.8m 0.90%
A020
1.0m 0.40%
A003
20m 0.82%
A011
12m 2.65%
A008
12m 0.78%
A019
20m 1.37%
A023
8m 0.50%
A021
1.0m 0.83%
A013
1.5m 2.77%
A012
3.5m 2.21%
A006
0.8m 0.47%
FLEXURE IN STRIKE
5 cm
NOTE :
THICKNESS SHOWN IS HORIZONTAL THICKNESS, NORTH - SOUTH DIRECTION
GRADE IS SHOWN % Ni.
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
NORTH AVEBURY
LONGITUDINAL PROJECTION
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : FILE NA Long Projection
McKEOWN MINING PTY LTD
Figure No. 19



Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

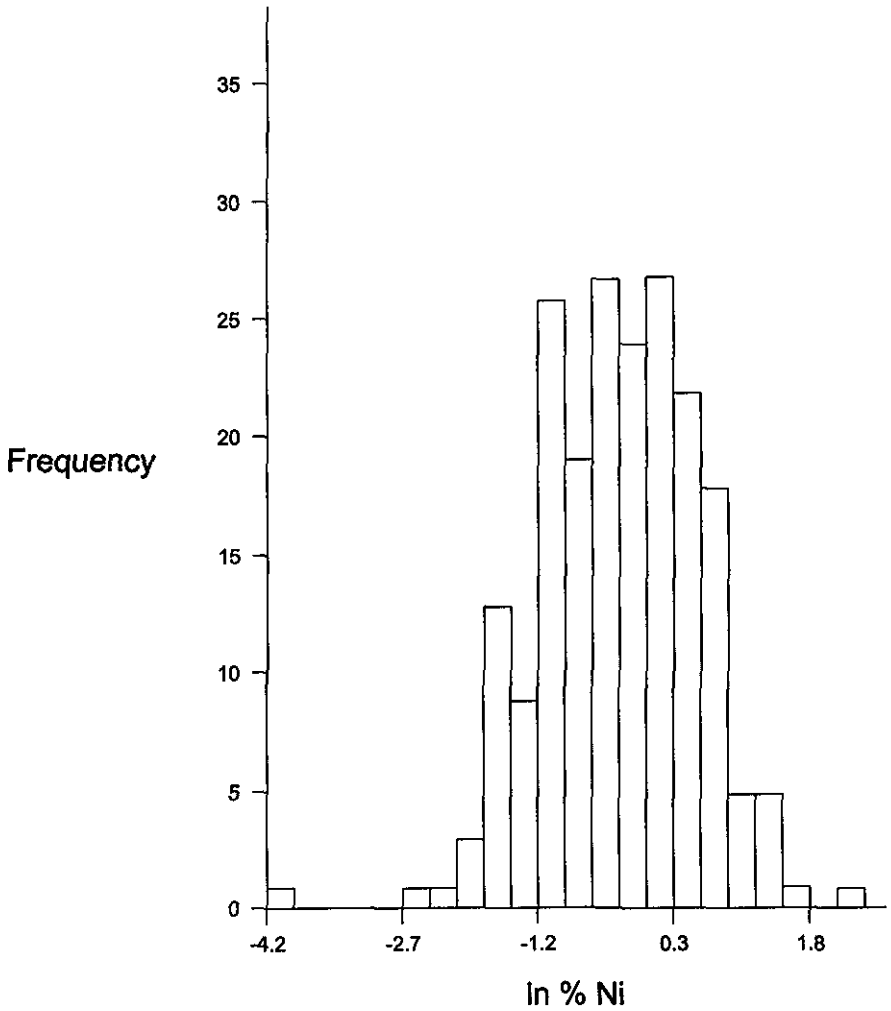
NORTH AVEBURY

HISTOGRAM OF NICKEL ASSAYS

COMPILED : M. McK
DATE : October 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : NA Histo Ni Assays

McKEOWN MINING PTY LTD

Figure No. 20



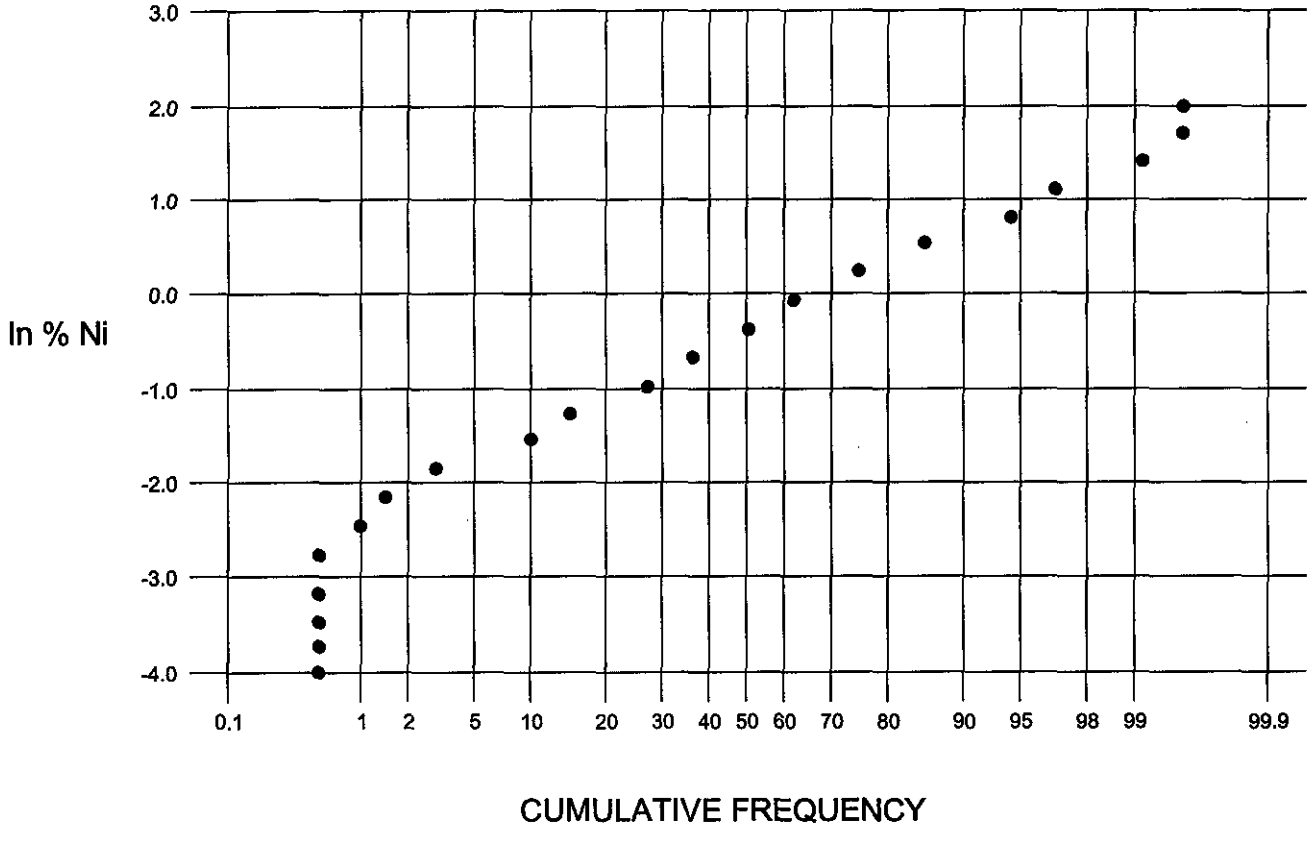
Allegiance Mining NL.

AVEBURY NICKEL PROJECT
NORTH AVEBURY
HISTOGRAM OF LOG OF NICKEL ASSAYS

COMPILED :	M. McK
DATE :	October 2000
DRAWN :	G.M.Bennett
REVISIONS :	
FILE :	NA Histo Log Ni Assays

McKEOWN MINING PTY LTD

Figure No. 21



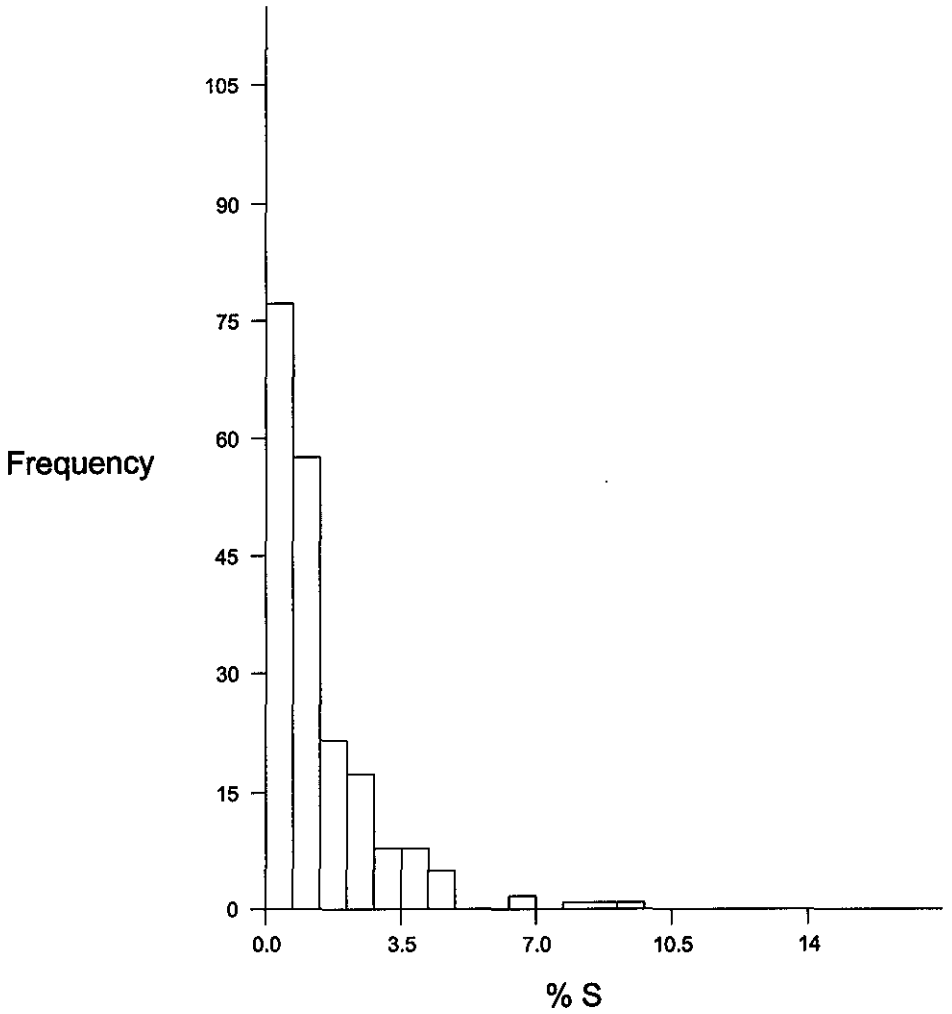
Allegiance Mining NL

AVEBURY NICKEL PROJECT
NORTH AVEBURY
CUMULATIVE FREQUENCY
PLOT OF LOG OF
NICKEL ASSAYS

COMPILED : M. McK
DATE : October 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : NA CF Log Ni

McKEOWN MINING PTY LTD

Figure No. 22



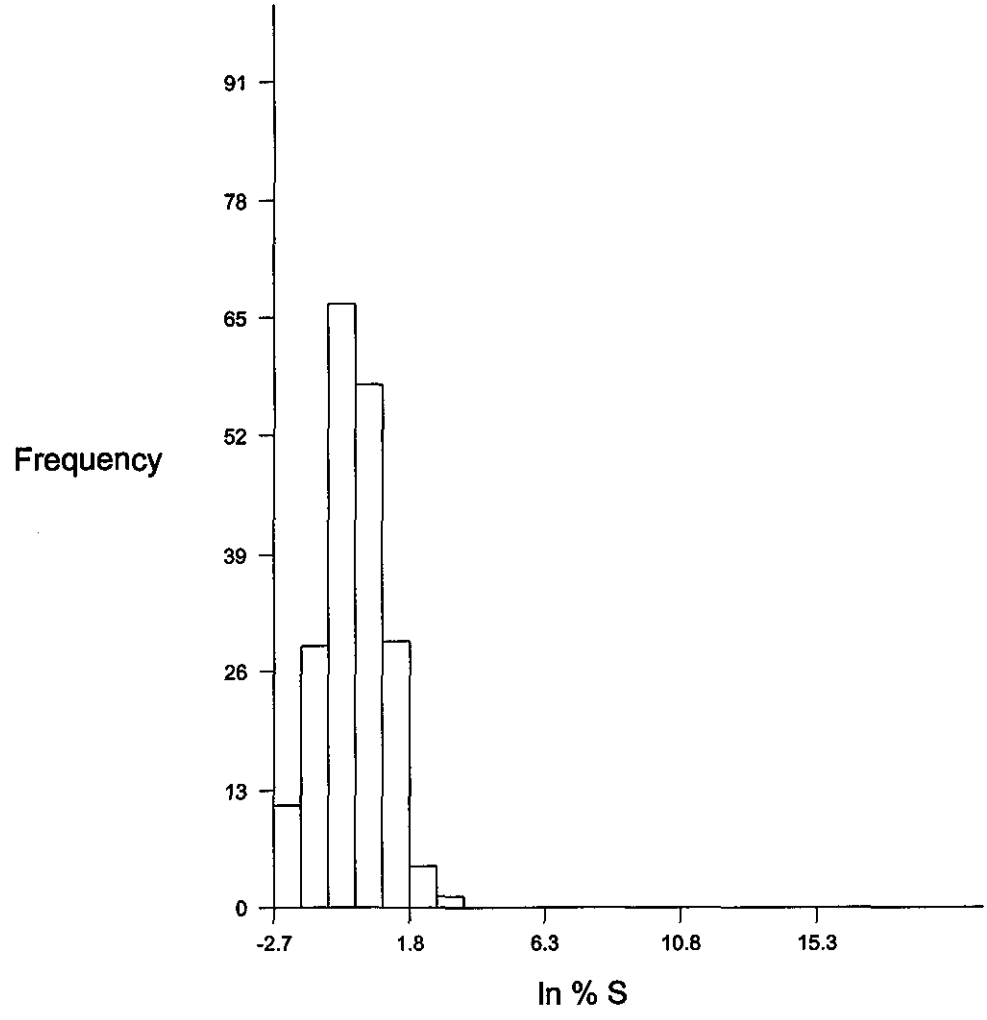
Allegiance Mining NL.

AVEBURY NICKEL PROJECT
NORTH AVEBURY
HISTOGRAM OF SULPHUR ASSAYS

COMPILED : M. McK
DATE : October 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : NA Histo S Assays

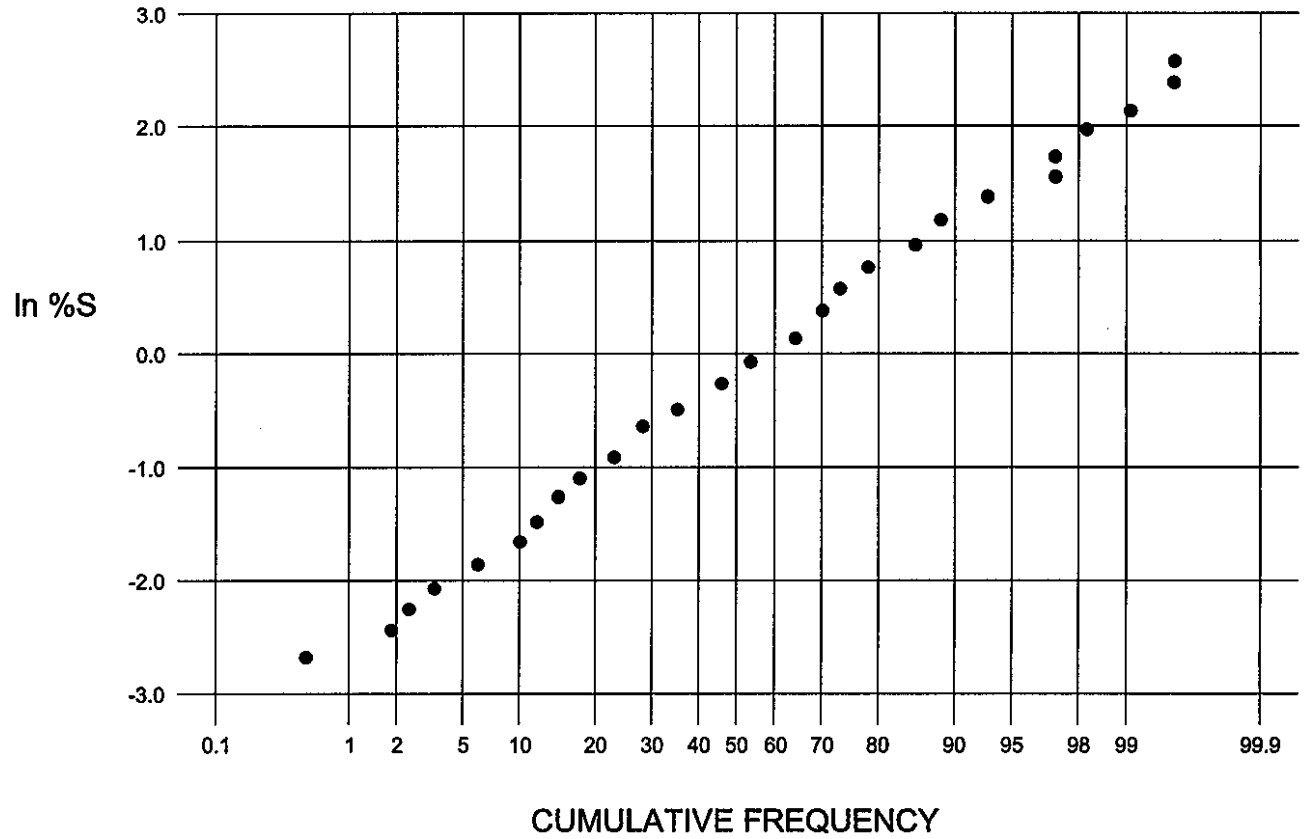
McKEOWN MINING PTY LTD

Figure No. 23



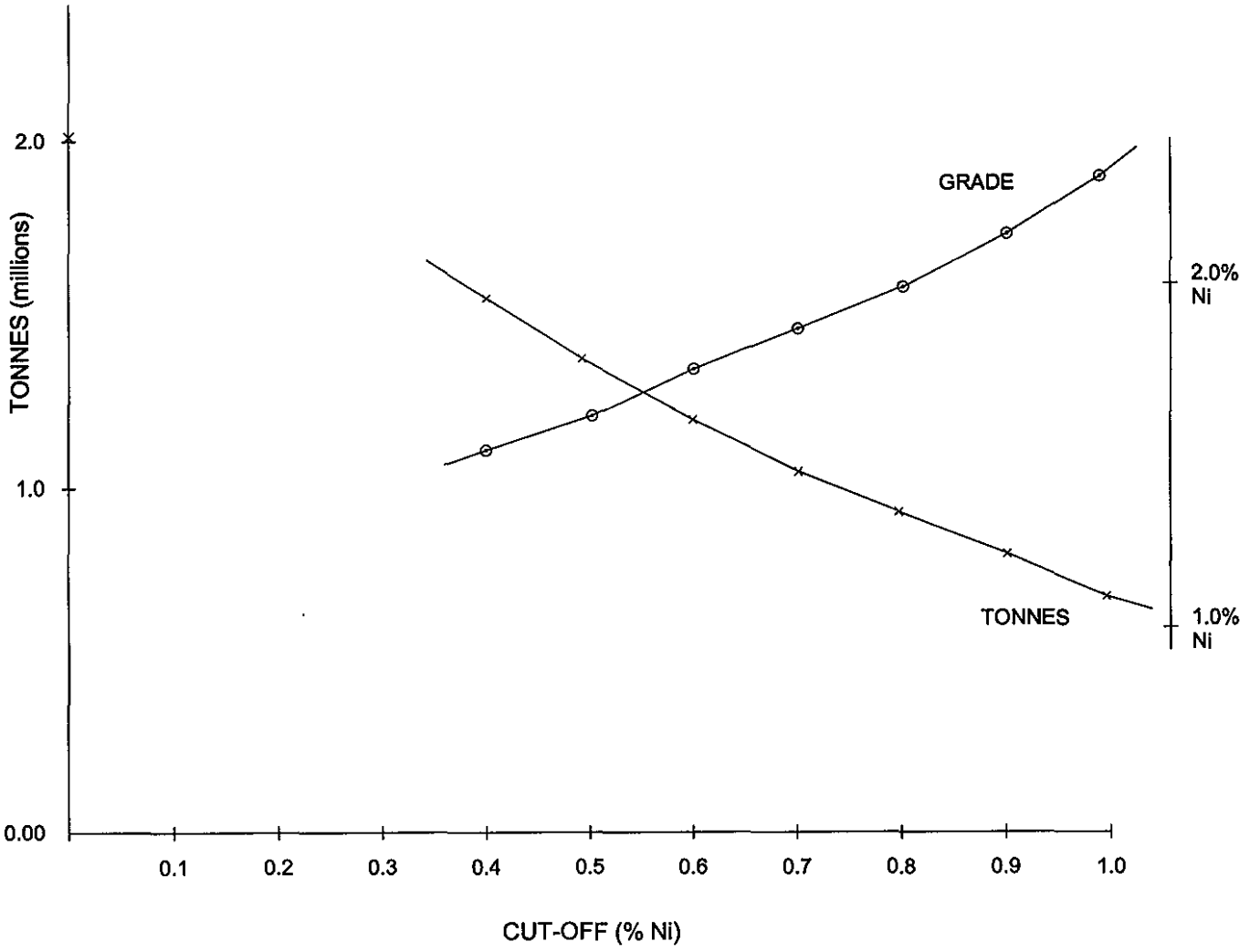
Allegiance Mining NL.

AVEBURY NICKEL PROJECT NORTH AVEBURY HISTOGRAM OF LOG OF SULPHUR ASSAYS	COMPILED : M. McK DATE : Octover 2000 DRAWN : G.M.Bennett REVISIONS : FILE : NA Histo Log S Assays
McKEOWN MINING PTY LTD	Figure No. 24



Allegiance Mining NL

AVEBURY NICKEL PROJECT NORTH AVEBURY CUMULATIVE FREQUENCY PLOT OF LOG OF SULPHUR ASSAYS	COMPILED : M. McK DATE : October 2000 DRAWN : G.M.Bennett REVISIONS : FILE : NA CF Log S
McKEOWN MINING PTY LTD	Figure No. 25



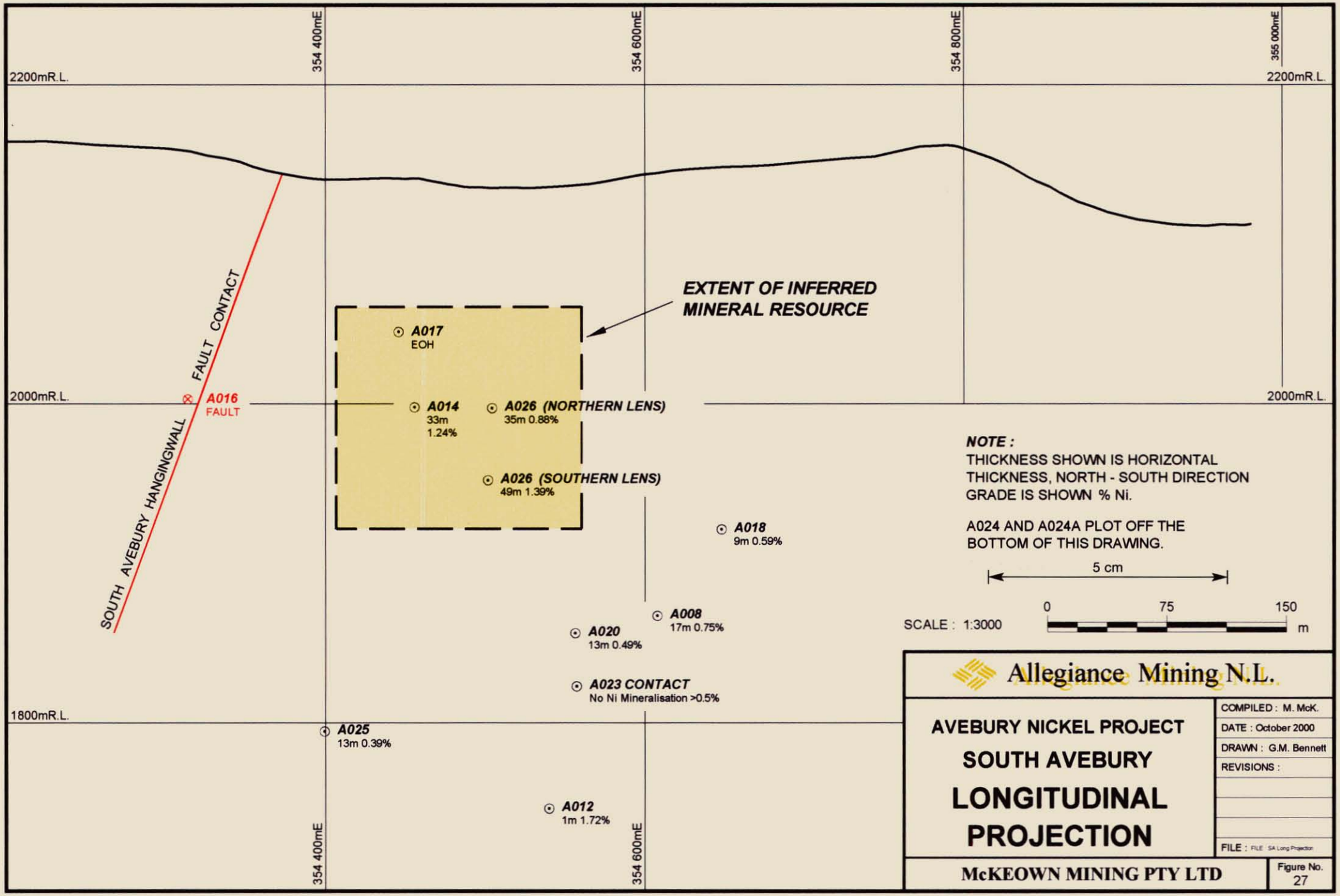
Allegiance Mining NL.

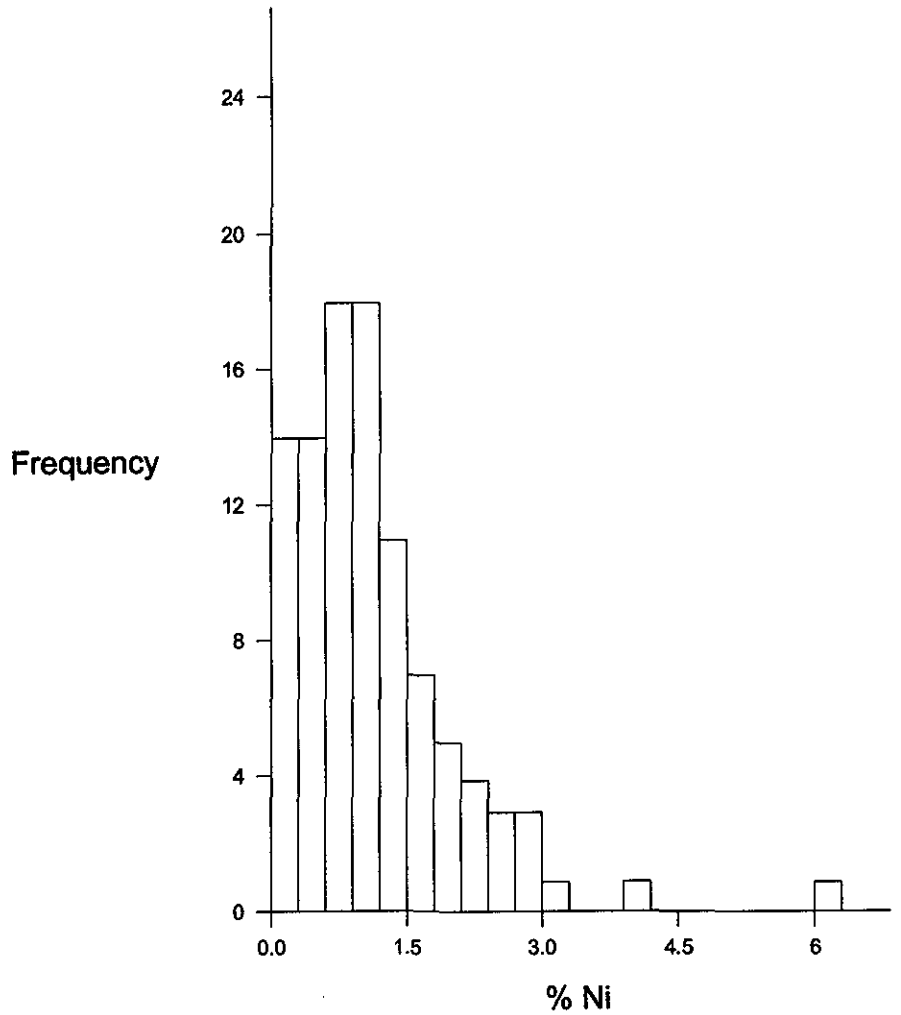
AVEBURY NICKEL PROJECT
NORTH AVEBURY
NON-PARAMETRIC
GRADE-TONNAGE CURVE

COMPILED : M. McK
DATE : November 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : NA Grade-Tonnage

McKEOWN MINING PTY LTD

Figure No. 26


354 400mE
354 600mE
354 800mE
355 000mE
2200mR.L.
2000mR.L.
1800mR.L.
EXTENT OF INFERRED MINERAL RESOURCE
A017
EOH
A014
33m
1.24%
A026 (NORTHERN LENS)
35m 0.88%
A026 (SOUTHERN LENS)
49m 1.39%
A016
FAULT
FAULT CONTACT
SOUTH AVEBURY HANGINGWALL
A018
9m 0.59%
A008
17m 0.75%
A020
13m 0.49%
A023 CONTACT
No Ni Mineralisation >0.5%
A025
13m 0.39%
A012
1m 1.72%
NOTE :
THICKNESS SHOWN IS HORIZONTAL THICKNESS, NORTH - SOUTH DIRECTION
GRADE IS SHOWN % Ni.
A024 AND A024A PLOT OFF THE BOTTOM OF THIS DRAWING.
5 cm
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
SOUTH AVEBURY
LONGITUDINAL PROJECTION
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : FILE : SA Long Projection
McKEOWN MINING PTY LTD
Figure No. 27



Allegiance Mining NL.

AVEBURY NICKEL PROJECT

SOUTH AVEBURY

HISTOGRAM OF NICKEL ASSAYS

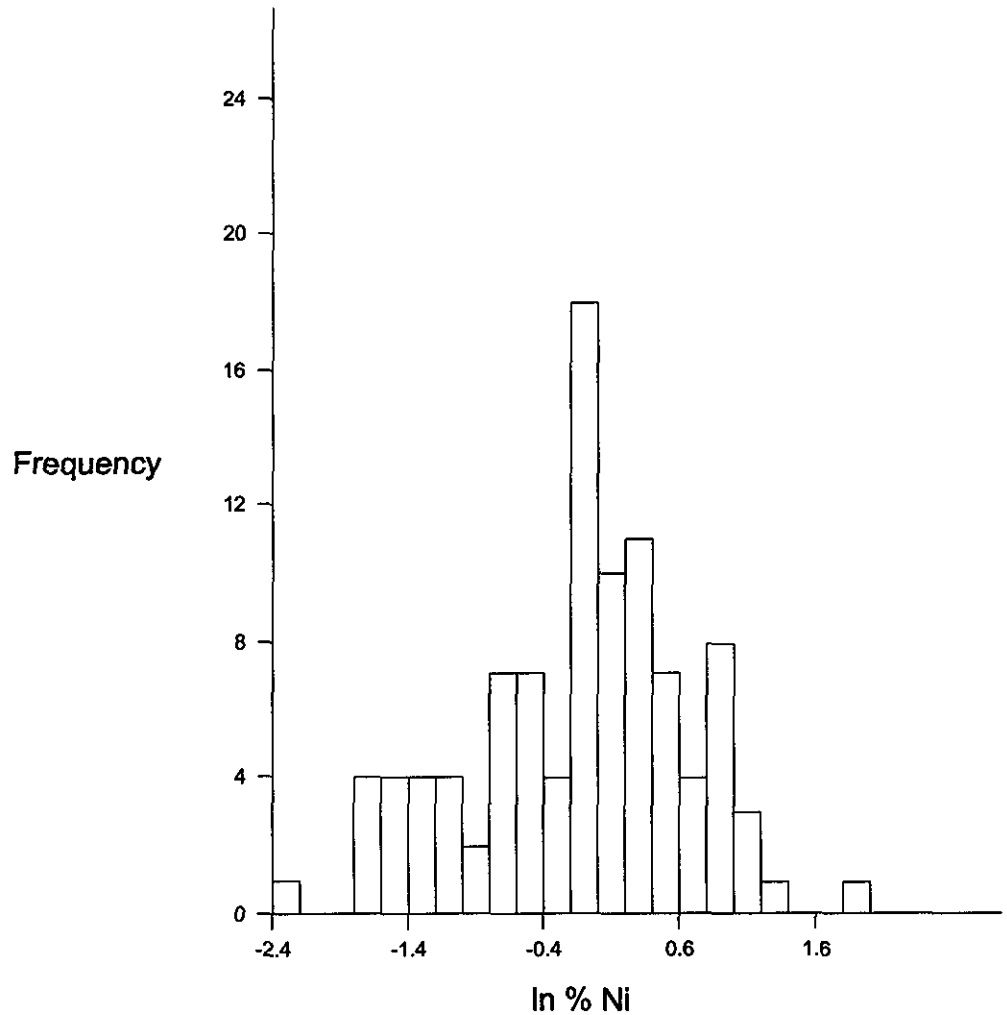
COMPILED : M. McK
DATE : October 2000
DRAWN : G.M.Bennett
REVISIONS :

FILE : SA Histo Ni Assays

McKEOWN MINING PTY LTD

Figure No. 28

661214



Allegiance Mining NL.

AVEBURY NICKEL PROJECT

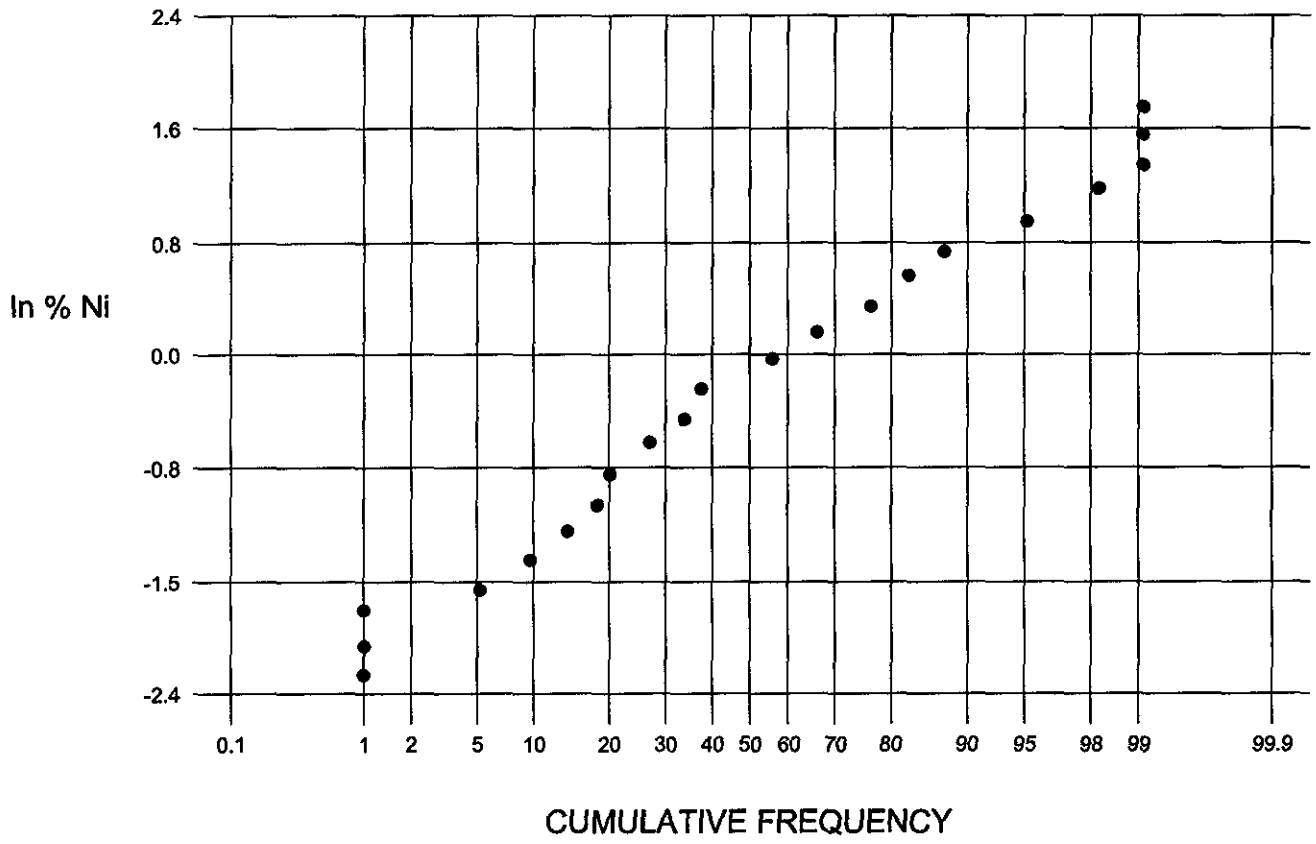
SOUTH AVEBURY

HISTOGRAM OF LOG OF NICKEL ASSAYS

COMPILED :	M. McK
DATE :	October 2000
DRAWN :	G.M.Bennett
REVISIONS :	
FILE :	SA Histo Log Ni Assays

McKEOWN MINING PTY LTD

Figure No. 29



Allegiance Mining N.L.

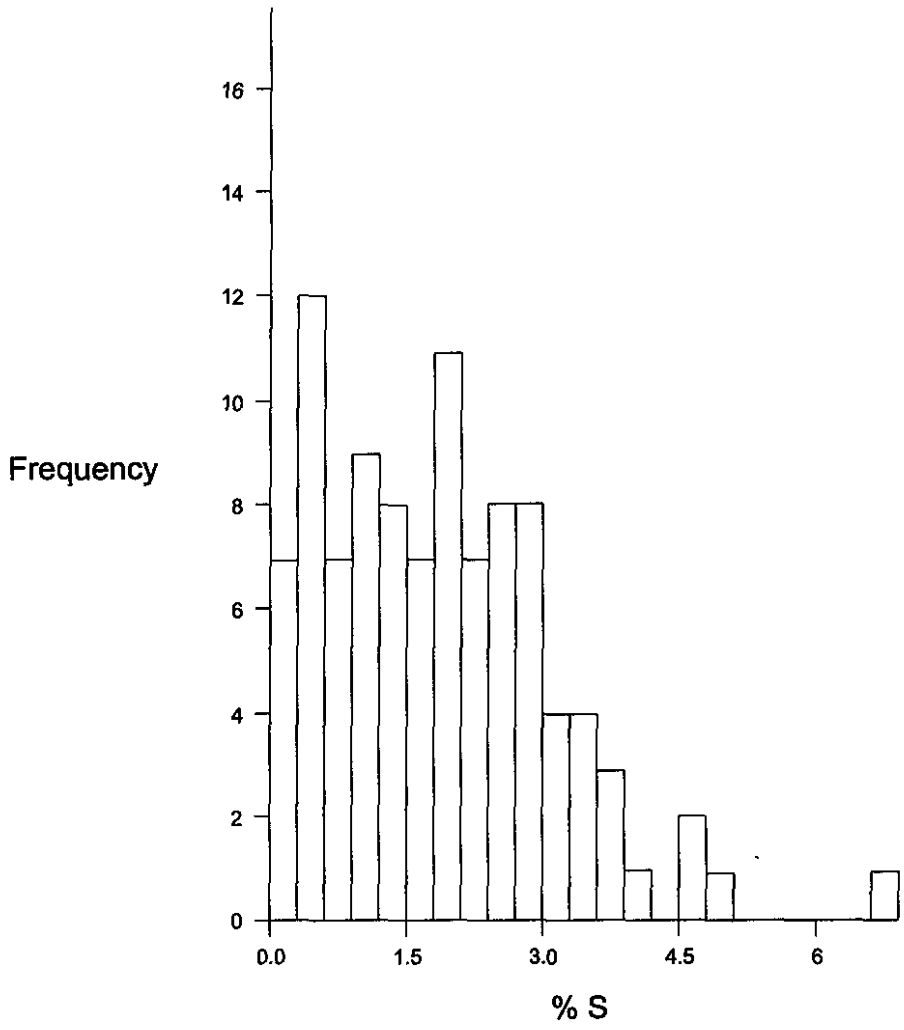
AVEBURY NICKEL PROJECT
SOUTH AVEBURY
CUMULATIVE FREQUENCY
PLOT OF LOG OF
NICKEL ASSAYS

COMPILED : M. McK
DATE : October 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : SA CF Log Ni

McKEOWN MINING PTY LTD

Figure No. 30

661216



Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

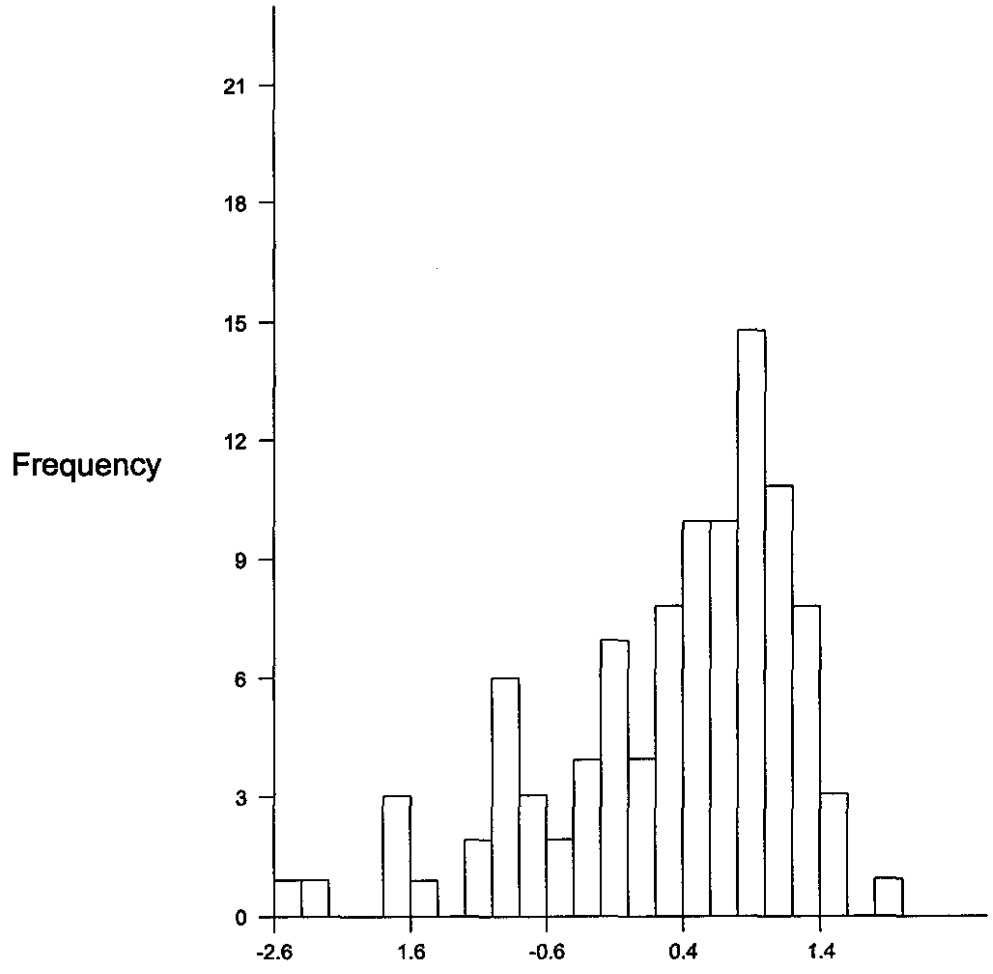
SOUTH AVEBURY

HISTOGRAM OF SULPHUR ASSAYS

COMPILED : M. McK	
DATE : October 2000	
DRAWN : G.M.Bennett	
REVISIONS :	
FILE : SA Histo S Assays	

McKEOWN MINING PTY LTD

Figure No. 31



Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

SOUTH AVEBURY

HISTOGRAM OF LOG OF SULPHUR ASSAYS

COMPILED : M. McK

DATE : October 2000

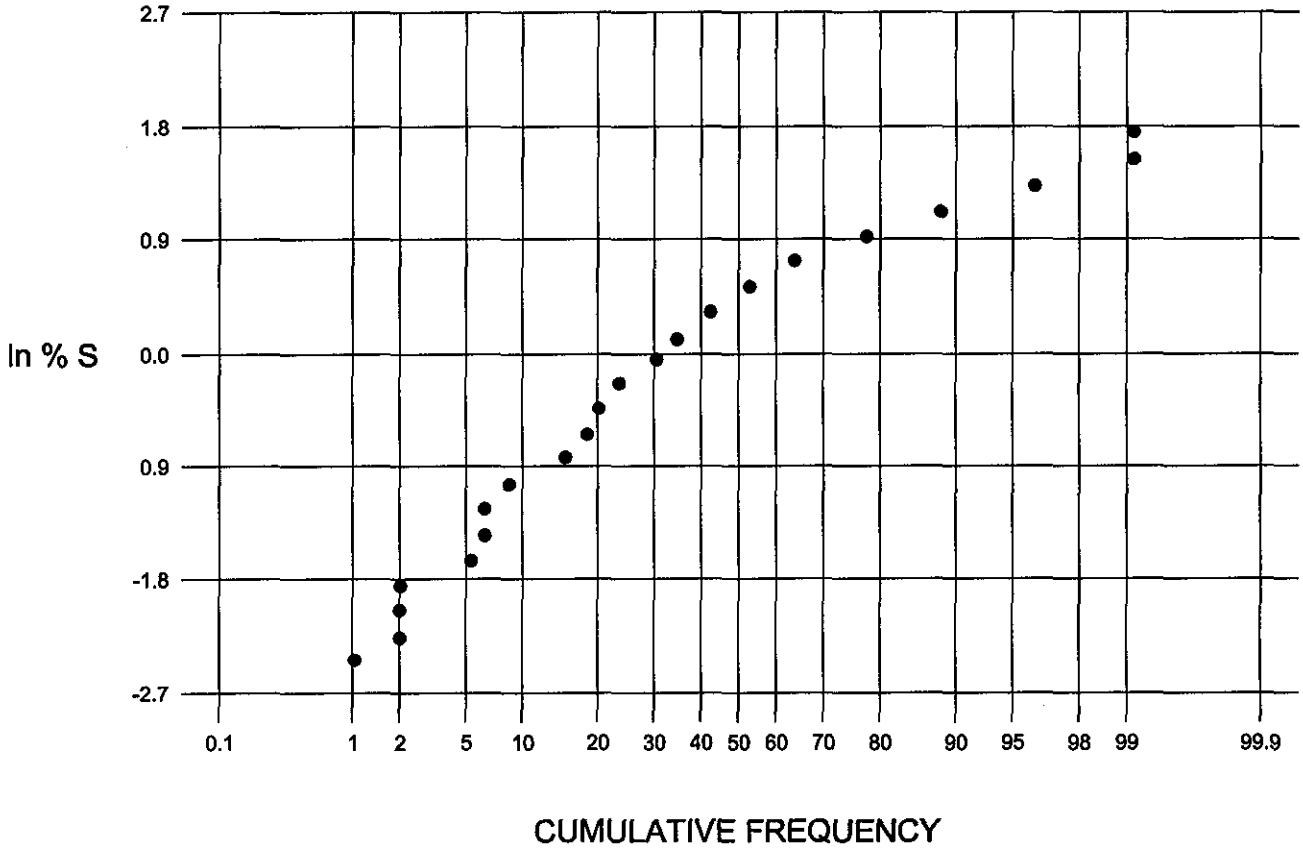
DRAWN : G.M.Bennett

REVISIONS :

FILE : SA Histo Log S Assays

McKEOWN MINING PTY LTD

Figure No. 32



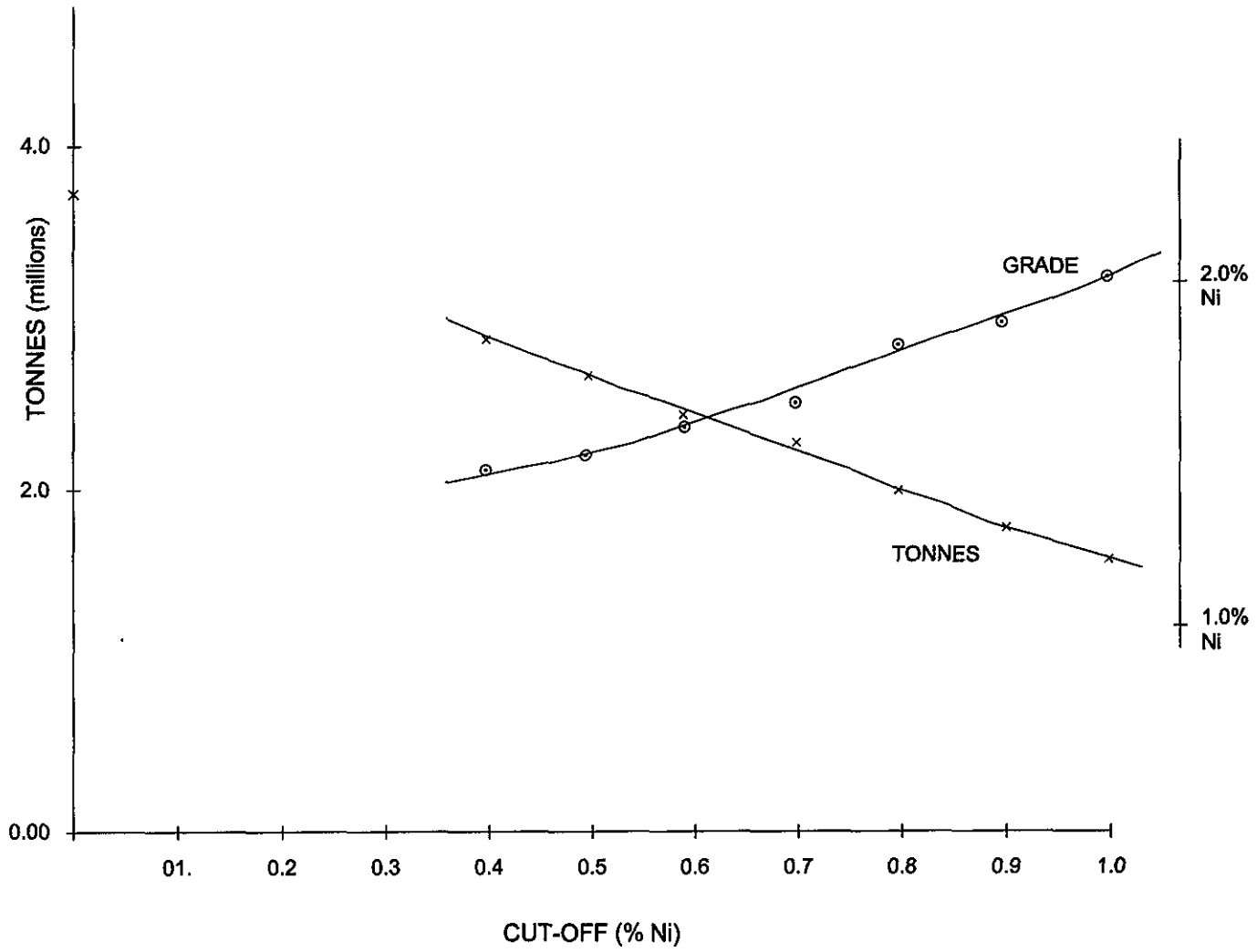
Allegiance Mining NL

AVEBURY NICKEL PROJECT
SOUTH AVEBURY
CUMULATIVE FREQUENCY
PLOT OF LOG OF
SULPHUR ASSAYS

COMPILED : M. McK
DATE : October 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : SA CF Log S

McKEOWN MINING PTY LTD

Figure No. 33



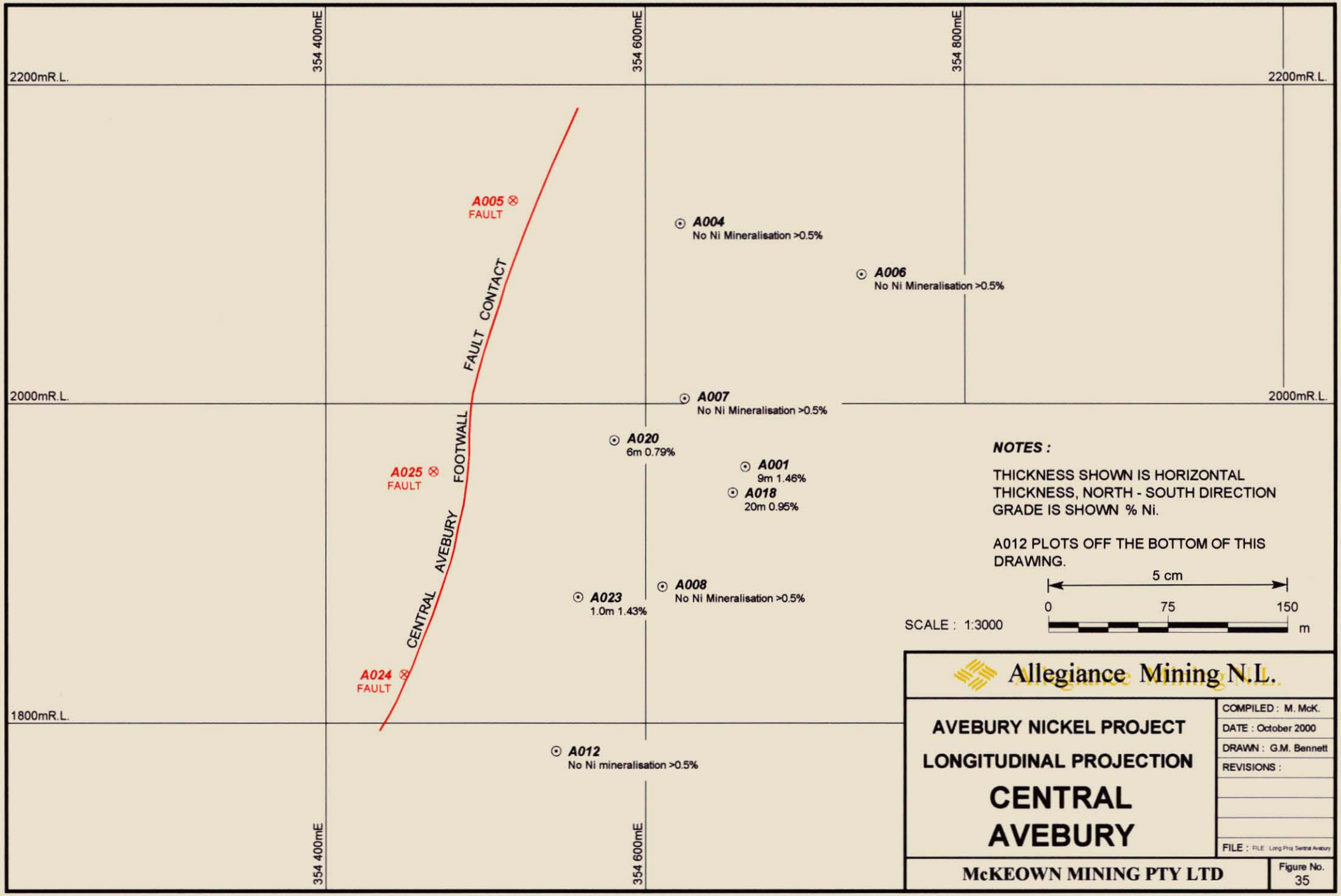
Allegiance Mining NL.

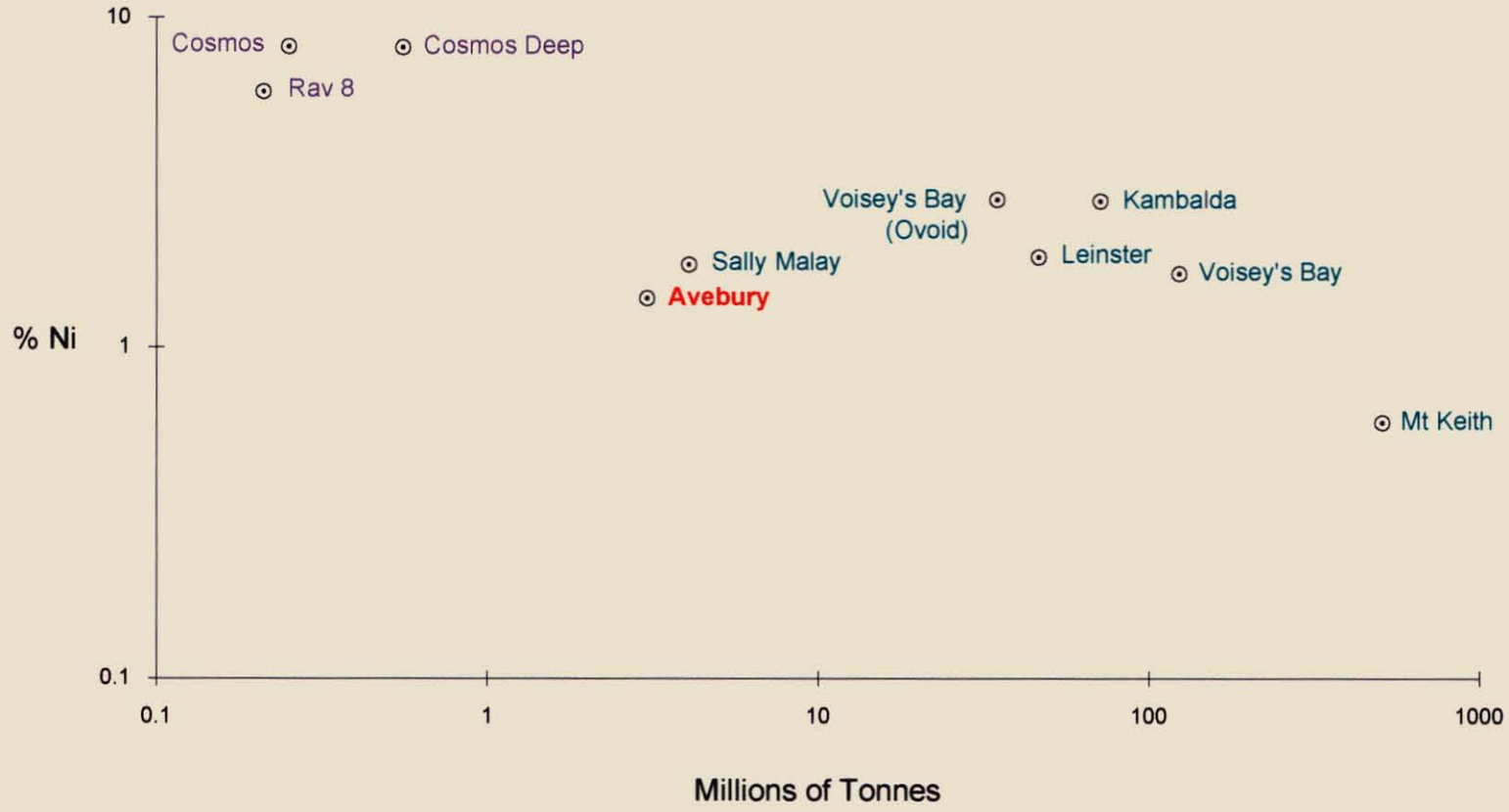
AVEBURY NICKEL PROJECT
SOUTH AVEBURY
NON-PARAMETRIC
GRADE-TONNAGE CURVE

COMPILED : M. McK
DATE : November 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : SA Grade-Tonnage

McKEOWN MINING PTY LTD

Figure No. 34


354 400mE
354 600mE
354 800mE
2200mR.L.
2000mR.L.
1800mR.L.
A005
FAULT
FAULT CONTACT
A004
No Ni Mineralisation >0.5%
A006
No Ni Mineralisation >0.5%
A007
No Ni Mineralisation >0.5%
A020
6m 0.79%
A001
9m 1.46%
A018
20m 0.95%
A025
FAULT
FOOTWALL
AVEBURY
CENTRAL
A008
No Ni Mineralisation >0.5%
A023
1.0m 1.43%
A024
FAULT
A012
No Ni mineralisation >0.5%
NOTES :
THICKNESS SHOWN IS HORIZONTAL THICKNESS, NORTH - SOUTH DIRECTION
GRADE IS SHOWN % Ni.
A012 PLOTS OFF THE BOTTOM OF THIS DRAWING.
5 cm
0
75
150
m
SCALE : 1:3000
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
LONGITUDINAL PROJECTION
CENTRAL AVEBURY
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : FILE: Long Proj Sentral Avebury
McKEOWN MINING PTY LTD
Figure No. 35



Allegiance Mining N.L.

AVEBURY NICKEL PROJECT

AVEBURY MINERAL RESOURCE COMPARED WITH RESOURCES AND RESERVES OF OTHER NICKEL SULPHIDE DEPOSITS

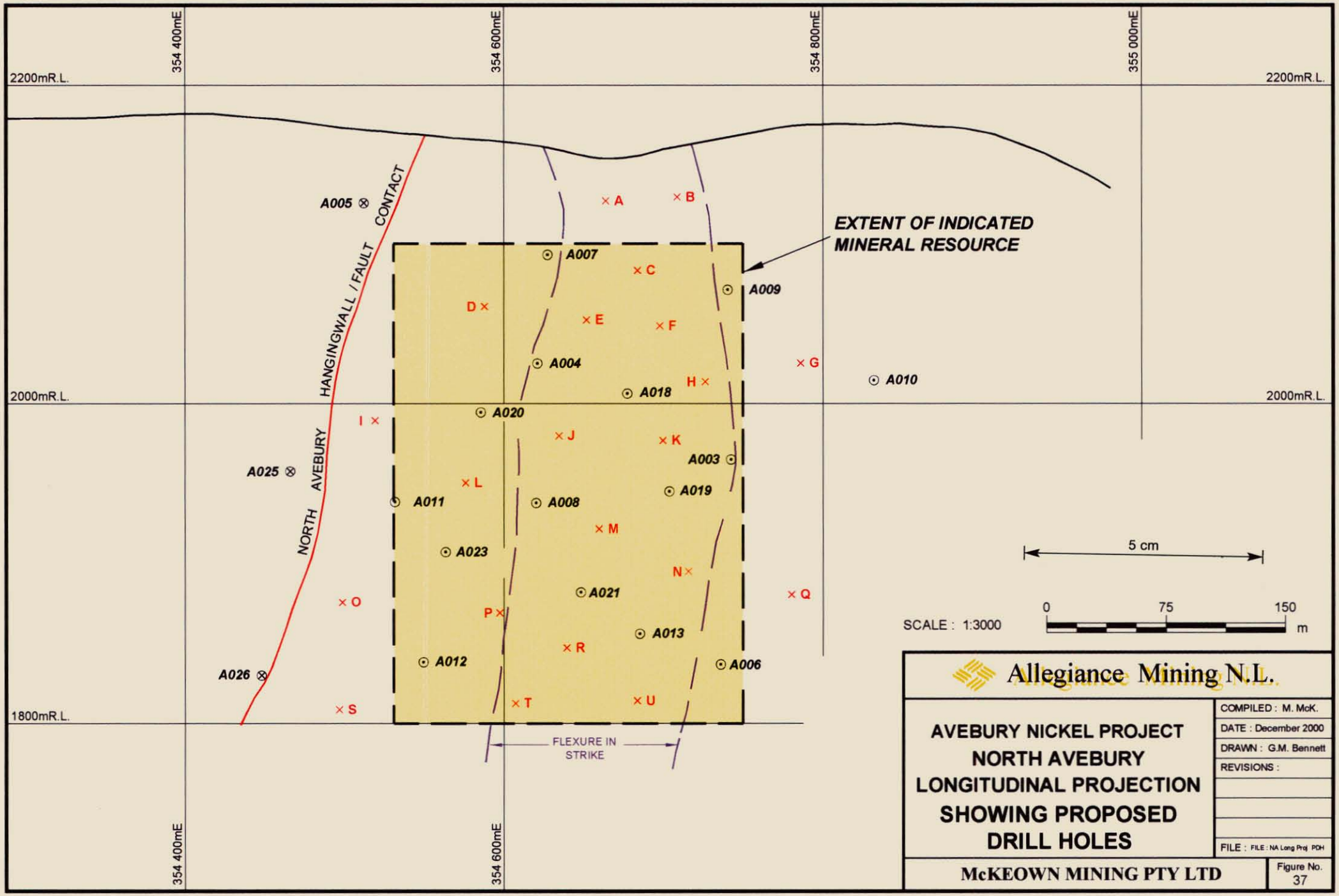
(See Text)

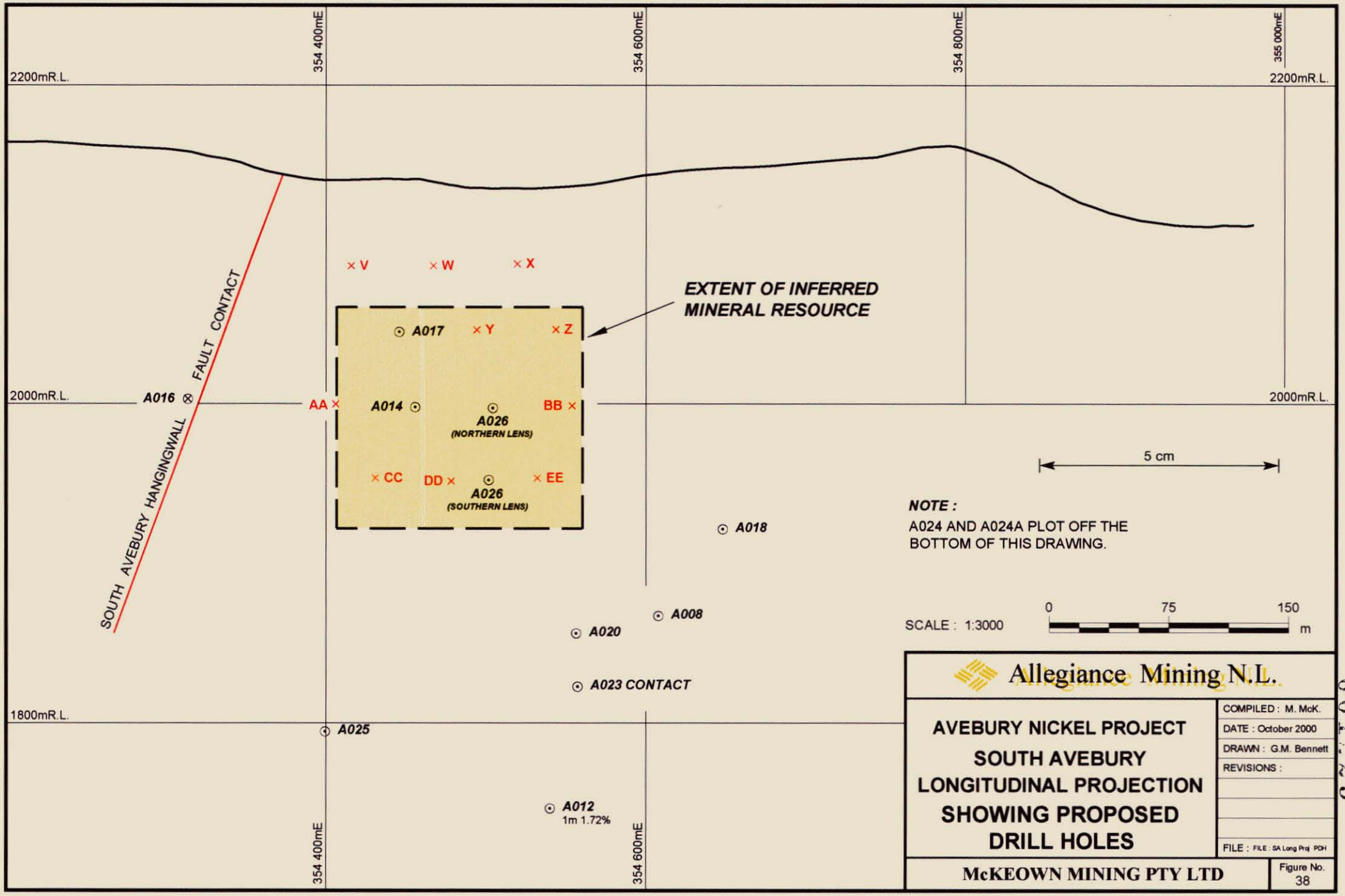
COMPILED : M. McK
DATE : December 2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : AMR Ni Comp

McKEOWN MINING PTY LTD

Figure No. 36

Allegiance Mining


354 400mE
354 600mE
354 800mE
355 000mE
2200mR.L.
2000mR.L.
1800mR.L.
HANGINGWALL / FAULT CONTACT
NORTH AVEBURY
EXTENT OF INDICATED MINERAL RESOURCE
A005
A007
A009
A004
A018
A010
A020
A025
A003
A011
A008
A019
A023
A021
A013
A012
A006
A026
A
B
C
D
E
F
G
H
I
J
K
L
M
N
O
P
Q
R
S
T
U
FLEXURE IN STRIKE
5 cm
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
NORTH AVEBURY
LONGITUDINAL PROJECTION
SHOWING PROPOSED
DRILL HOLES
COMPILED : M. McK.
DATE : December 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : FILE : NA Long Proj PDH
McKEOWN MINING PTY LTD
Figure No.
37


354 400mE
354 600mE
354 800mE
355 000mE
2200mR.L.
2000mR.L.
1800mR.L.
SOUTH AVEBURY HANGINGWALL FAULT CONTACT
EXTENT OF INFERRED MINERAL RESOURCE
A016
A017
A014
A026 (NORTHERN LENS)
A026 (SOUTHERN LENS)
A018
A008
A020
A023 CONTACT
A025
A012
1m 1.72%
V
W
X
Y
Z
AA
BB
CC
DD
EE
5 cm
NOTE :
A024 AND A024A PLOT OFF THE BOTTOM OF THIS DRAWING.
SCALE : 1:3000
0
75
150
m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
SOUTH AVEBURY
LONGITUDINAL PROJECTION
SHOWING PROPOSED
DRILL HOLES
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
REVISIONS :
FILE : FILE : SA Long Proj PDH
McKEOWN MINING PTY LTD
Figure No.
38

Appendix 1

Useful addresses

Consultant geologists:

McKeown Mining Pty Ltd
PO Box 50
Ridgley, Tas 7321

Contact: Mick McKeown, phone 03 6435 7560

Newnham Exploration and Mining Services
PO Box 183
Exeter, Tas 7275

Contact: Lindsay Newnham, phone 03 6394 3434

Petrological consultant:

Central Mineralogical Services
8 Bradshaw Ave
Crafers, SA 5152

Contact: Wally Fander, telephone 08 8370 9779

Diamond drilling contractors:

Almac Drilling Pty Ltd,
Quiggan St,
Zeehan, Tas 7469

Contact: Max Harvey, telephone 03 6471 6131

Diamond Drilling Tasmania Pty Ltd,
Fowler St,
Zeehan, Tas 7469

Contact: Peter Sharp, telephone 03 6471 6339

Survey consultant:

Campbell-Smith, Pedley and Phelps Pty Ltd,
60 Elphin Rd,
Launceston, Tas 7250

Contact: Ian Green, telephone 03 6331 4099

Analytical laboratories:

Analabs,
14 Thirkell St,
Cooee, Tas 7320

Contact: Rob Chapman telephone 03 6431 6837

Amdel Laboratories Ltd
Gate 3, Osman Place,
Thebarton, SA 5031

Contact: Andrew Gasiorowski, telephone 08 8416 5300

Appendix 2

Petrological descriptions

Central Mineralogical Services

8 Bradshaw Avenue, Crafers, S.A. 5152
Telephone (08) 8370 9779 Fax (08) 8370 9788
International: Telephone + 618 8370 9779 Fax + 618 8370 9788

16 March 1998

Mr L.A. Newnham
Newnham Mining & Exploration Services
PO Box 132
RIVERSIDE Tas 7250

REPORT CMS 98/3/3

YOUR REFERENCE: Letter M. McKeown

DATE RECEIVED: 10 March 1998

SAMPLE NO'S: A001 (17 cores)

SUBMITTED BY: M. McKeown

WORK REQUESTED: Petrology, Mineragraphy

H.W. Fander, M.Sc.

c.c. McKeown Mining Pty Ltd
PO Box 50
RIDGLEY Tas 7321

REPORT CMS 98/3/3

DDH A001 INTERSECTIONS

Seventeen drill core intersections were received for petrographic and mineragraphic study; thin and polished sections were prepared as appropriate and descriptions are appended.

The only serpentinites recognisable as such without doubt were the last three intersections; at ***289.9m***, some of the original dunite is preserved, and there is no evidence of gabbroic rocks.

Most of the other rocks are extensively or completely contact-metasomatised, so that the nature of the original rocks are not known or are speculative; normally skarns of this type would be formed from pre-existing carbonate rocks but derivation from strongly magnesian types such as serpentinites could be feasible. Chromium assays could assist in determining the origin of these metasomatised rocks.

The Ni mineralisation is apparently due solely to pentlandite, and the absence of significant amounts of other sulphides is metallurgically attractive. The mineralisation postdates serpentinisation and predates a later episode of minor shearing.

SAMPLE NO: A001/36.2m (T.S. 67633) Page 1

CLASSIFICATION: Tremolitised Limestone.

COMPOSITION: Mainly fine-grained calcite pigmented with ultrafine goethite. Extensive development of small, subrandom, replacive tremolite needles throughout. Carbonaceous partings.

FABRIC: Finely-crystalline; faint preferred orientation (bedding) augmented by some subparallel tremolite needles.

MINOR MINERALS: Semi-concordant sulphide veinlets, and crosscutting coarse (white) calcite veins with sulphides. Also small sulphide grains throughout.

INTERPRETATION/COMMENTS: This rock has been thoroughly metasomatised but, unusually, is not recrystallised, ie cannot be termed a marble in the metamorphic sense. The sulphides are not pyrite or pyrrhotite. Ni assay recommended.

SAMPLE NO: A001/48.8m (T.S. 67634)

CLASSIFICATION: Phlogopitised Subgreywacke.

COMPOSITION: Framework of matrix-supported sub-angular to subrounded grains of quartz, argillised feldspar, felsite, rhyolite, in a fine siliceous matrix with many small patches of fine replacive phlogopite.

FABRIC: Poorly to moderately sorted sand-sized grains, crudely bedded, with occasional silty-shaly intercalations. Indurated.

MINOR MINERALS: Quartz-sulphide veinlets; also fine sulphides throughout. Fine black pigmentation may be carbonaceous matter.

INTERPRETATION/COMMENTS: Volcanic components may be more plentiful but not recognisable because of alteration (?vitric material). Sulphides were mostly introduced with the replacive phlogopite, with later veining.

SAMPLE NO: A001/60.4m (T.S. 67635)

CLASSIFICATION: Diopside Hornfels.

COMPOSITION: Dominantly composed of random prismatic crystals of diopside, with interstitial patches of unusual green/brown chlorite in which sulphides preferentially occur.

FABRIC: Variable crystallinity, with some diopside crystals up to 12mm long. No relict textures to indicate nature of original rock.

MINOR MINERALS: Conspicuous sulphides (pyrrhotite and a trace of dark sphalerite). Trace of carbonate.

INTERPRETATION/COMMENTS: Nature of original rock not known, presumably represented by chlorite patches. Possibly ultramafic but this is speculative and other interpretations are possible.

SAMPLE NO: A001/73.8m (T.S. 67637)

CLASSIFICATION: Actinolite Hornfels.

COMPOSITION: Consists very largely of matted pale-green actinolite needles which have completely replaced the pre-existing rock. Patches of dark tourmaline (schorl) formed after the actinolite.

FABRIC: Relict textures indicate a fine-grained, bedded sediment. Some faulting/shearing occurred after actinolite formation and preceded tourmalinisation.

MINOR MINERALS: Ultrafine leucoxene/rutile occurs throughout, possibly inherited from the primary rock.

INTERPRETATION/COMMENTS: Thought to have been a semi-calcareous fine-grained sediment, thoroughly metasomatised, then sheared, followed by tourmalinisation.

SAMPLE NO: A001/88.6m (T.S. 67638)

CLASSIFICATION: Metachert.

COMPOSITION: Bands of microcrystalline quartz, with bands and lenses containing abundant fine apatite and ultrafine white leucoxene. Streaks and lenses of carbonate, partly replaced by actinolite.

FABRIC: Strong relict bedding, banding, and parallel lenses. No clastic textures recognisable.

MINOR MINERALS: Crosscutting veinlets of untwinned alkali feldspar and of quartz. Occasional grains of sphene.

INTERPRETATION/COMMENTS: Originally a silica-rich chemical sediment, mainly chert, with phosphatic lenses and bands.

SAMPLE NO: A001/121.0m (T.S. 67639)

CLASSIFICATION: Metasomatised Sediment.

COMPOSITION: Chiefly composed of alternating bands of fine-grained orange/amber phlogopite and bands and lenses of pale green, random actinolite needles, with relict cherty lenses and streaks.

FABRIC: General banding, due to replacement of beds of different composition by specific minerals. Minerals are fine-grained within bands.

MINOR MINERALS: Conspicuous sulphides associated with the actinolite rather than phlogopite. A few relatively large apatite crystals in phlogopite.

INTERPRETATION/COMMENTS: Originally a chemical sediment, perhaps correlatable with and broadly similar to **88.6m** though less cherty, more argillic. Needs a Ni assay.

SAMPLE NO: A001/150.9m (T.S. 67640)

CLASSIFICATION: Metasomatised Clastic Sediment (?Tuff).

COMPOSITION: Bands of clastic grains suspected to have been glassy, feldspathic and felsic volcanics, but now completely altered to quartz-sericite and variably replaced by fine actinolite. Parallel veins of K-feldspar/actinolite/epidote/quartz.

FABRIC: Original bedding and clastic (framework grain) textures have survived; very few clastic quartz grains.

MINOR MINERALS: Conspicuous semi-opaque finely granular leucoxene-sphene occurs throughout. Crosscutting chlorite-carbonate veins.

INTERPRETATION/COMMENTS: The inferred composition of the original rock suggests that it was a lithic-feldspathic tuff of felsic to intermediate derivation.

SAMPLE NO: A001/164.6m (T.S. 67641)

CLASSIFICATION: Axinite Rock.

COMPOSITION: Dominantly composed of large random bladed axinite crystals with characteristic habit, and minor ragged diopside patches, with patches of very dark Fe-rich sphalerite (marmatite).

FABRIC: Random, coarsely-crystalline, with some radiating-prismatic crystal groups. Sphalerite 50μ - 10mm+.

MINOR MINERALS: A few interstitial carbonate patches. Ultrafine leucoxene-sphene in places, probably inherited from original rock.

INTERPRETATION/COMMENTS: Origin of rock unknown but was presumably a carbonate which was metasomatised/pneumatolysed.

SAMPLE NO: A001/178.4m **(T.S. 67642)**

CLASSIFICATION: Axinite Skarn.

COMPOSITION: Mainly large prismatic axinite crystals, with minor interstitial quartz, actinolite, diopside. Poikiloblastic crystals of dark green tourmaline partly enclosing the other minerals.

FABRIC: Coarsely-crystalline with random fabric. No relict textures. Typical skarn fabric. Tourmaline was the last mineral to form.

MINOR MINERALS: A few calcite patches and chlorite aggregates. Conspicuous ultrafine leucoxene in parallel lines throughout the other minerals.

INTERPRETATION/COMMENTS: Similar to **164.6m** but with more diver composition.

SAMPLE NO: A001/227.5m **(T.S. 67643)**

CLASSIFICATION: Sheared, Altered Diopside Skarn.

COMPOSITION: Relict parallel bands of pale, prismatic diopside crystals with intercalated masses of fine replacive phlogopite (green) as dense matted flakes.

FABRIC: Banded structure probably due to parallel shears along which phlogopite alteration proceeded.

MINOR MINERALS: Occasional sphene crystals and small sulphide grains.

INTERPRETATION/COMMENTS: Appears to have been a virtually monomineralic diopside rock, sheared and retrograded. Originally a carbonate rock (?dolomite).

SAMPLE NO: A001/250.4m (T.S. 67467)

CLASSIFICATION: Steatitised Serpentinite.

COMPOSITION: Dark areas are serpentinite, consisting of antigorite/chrysotile with characteristic mesh textures, pigmented with ultrafine magnetite. There are large areas of replacive talc, with granular magnetite.

FABRIC: The mesh fabric is medium-grained and is typical of a dunite. The talc is clearly replacive.

MINOR MINERALS: Patches of pentlandite, associated with the granular magnetite. ***See separate description.***

INTERPRETATION/COMMENTS: The mineralisation appears to be associated with the magnetite and with the replacive talc. The serpentinite is of ultramafic origin (ie not gabbroic).

SAMPLE NO: A001/271.3m (T.S. 67648)

CLASSIFICATION: Serpentinite (Sheared).

COMPOSITION: Classical network fabric of antigorite units pigmented with ultrafine magnetite, and fine chrysotile veinlets and networks of magnetite veinlets.

FABRIC: Network textures are coarser - grained than at **250m**, indicating original grainsizes of 1-2mm for individual olivine crystals. Thin parallel shears postdating the sulphides.

MINOR MINERALS: Concordant and crosscutting pentlandite and magnetite veins. Chlorite in shears. Sulphides postdate the serpentinisation ***(see separate description).***

INTERPRETATION/COMMENTS: Originally a dunite or peridotite, correlatable with **250.4m** but coarser and not steatitised. Sulphides were introduced after serpentinisation and before shearing. There are two generations of magnetite, (a) ultrafine, from breakdown of olivine; (b) introduced, much coarser.

SAMPLE NO: A001/289.9m (T.S. 67649) Page 8

CLASSIFICATION: Tremolitised Serpentinite.

COMPOSITION: Dominantly composed of relatively large, stout prismatic crystals of tremolite, with irregular interstitial patches of pale chlorite and antigorite. *See below for a further description.*

FABRIC: Antigorite patches have shadowy serpentine network textures, and tremolite is clearly replacive.

MINOR MINERALS: Chromite grains with magnetite rims. Fine magnetite as small inclusions in tremolite. Scattered small pyrrhotite and pentlandite patches.

INTERPRETATION/COMMENTS: The distribution of the dark serpentinite patches in the core indicate very variable replacement by tremolite.
See below.

SAMPLE NO: A001/289.9m (T.S. 67649)A

CLASSIFICATION: Partly Altered Serpentinite.

COMPOSITION: Another section was prepared and consisted of only incipiently to partly serpentinised dunite (forsterite olivine) with olivine crystals up to 3mm. It has been partly replaced by coarse tremolite, and this in turn is partly replaced by pale green talc. In some places talc directly replaces serpentinite. Sulphides are associated mainly with tremolite. The occurrence of forsterite (the Mg end-member) is unusual, as olivine generally contains some Fe.

This rock shows a complex alteration history.

MINERAGRAPHY OF SELECTED SAMPLES

A001/61.7m (PS 67636)
The dominant sulphide is pyrrhotite as branching veinlike masses in a host of dense, dark chlorite. Some pyrrhotite is altered to pyrite, and there are thin, fine inclusions ("flames") of pentlandite sporadically distributed through the pyrrhotite. There are also small chalcopyrite inclusions.

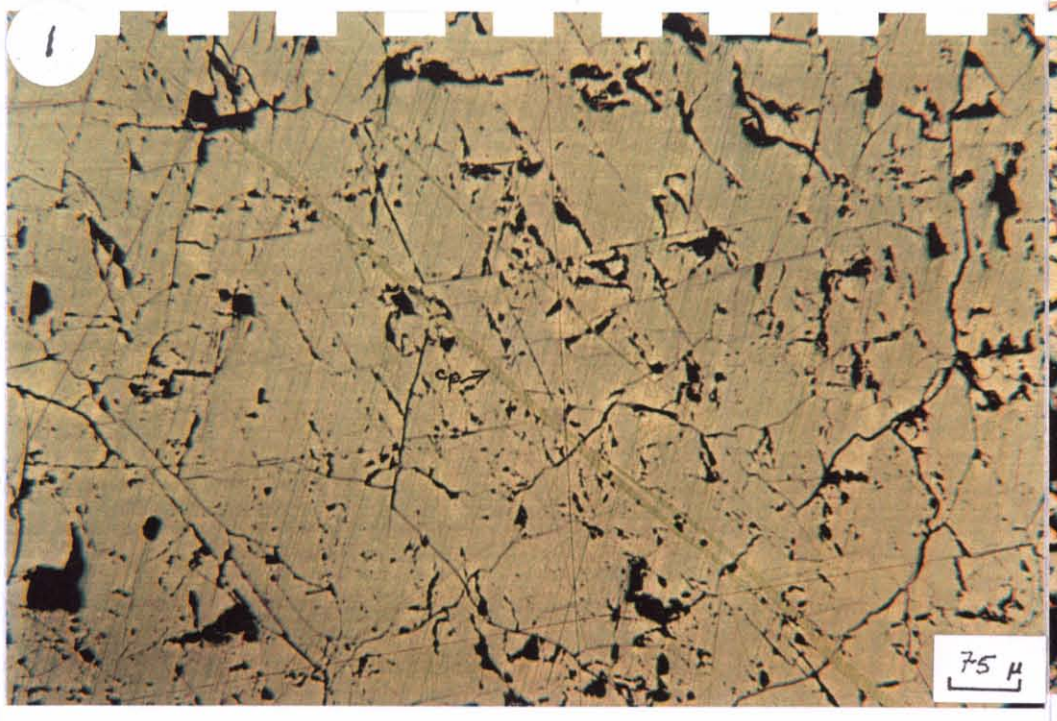
A001/244.2m (PS 67645)
This consists almost entirely of massive, coarsely-crystalline fresh **pentlandite**, cut by occasional pyrite veinlets and containing a few thin lamellae of chalcopyrite and pyrrhotite.
Photo 1; magnification = 135x

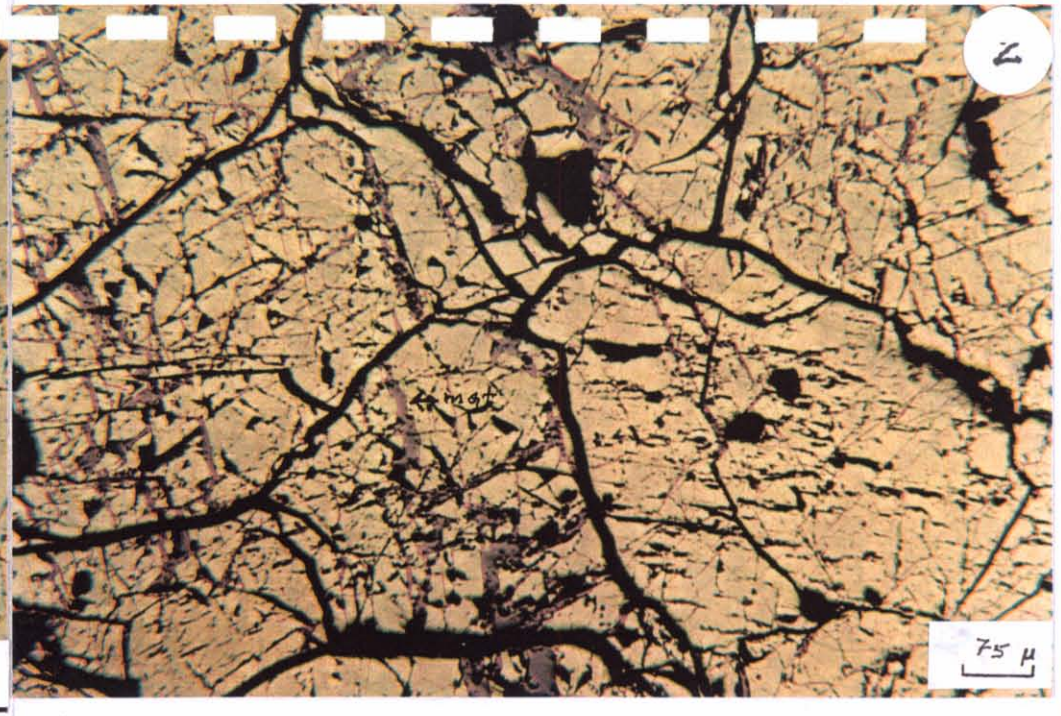
A001/250.4m (PS 67647)
The only sulphide detected was **pentlandite**, as coarsely-crystalline masses with good cleavage, cut by magnetite veinlets. There are associated masses of granular magnetite and fine magnetite along cleavage-planes.
Photo 2; magnification = 135x

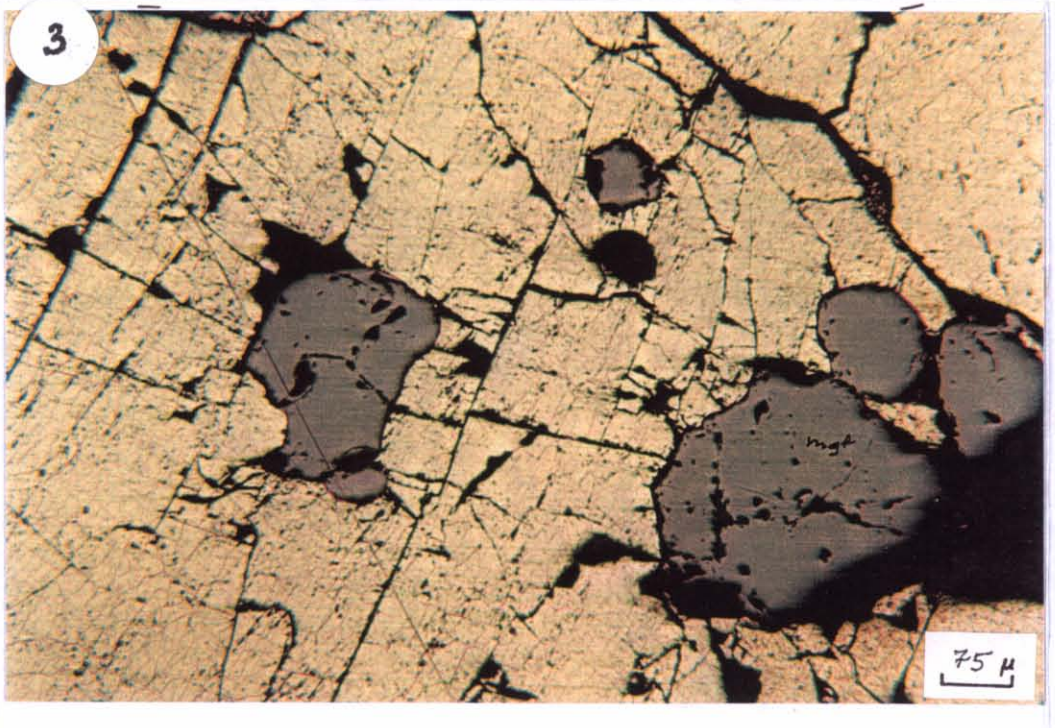
A001/271.3m (PS 67648)
Again, the only sulphide was **pentlandite**, as well-cleaved coarsely-crystalline masses; coarser granular magnetite is associated. There are traces of chalcopyrite in the magnetite.
Photo 3; magnification = 135x

In all these samples, the Co is contained in the pentlandite; there is a constant relationship between Co and Ni.

Metallurgically, the occurrence of only one sulphide is a very significant advantage, enabling high-grade Ni concentrates to be produced with the minimum of processing.







661229

661239

Central Mineralogical Services

8 Bradshaw Avenue, Crafers, S.A. 5152
Telephone (08) 8370 9779 Fax (08) 8370 9788
International: Telephone + 618 8370 9779 Fax + 618 8370 9788

2 March 1999

Mr M. McKeown
McKeown Mining Pty Ltd
PO Box 50
RIDGLEY Tas. 7321

REPORT CMS 99/2/9

YOUR REFERENCE: Letter 15.2.99

DATE RECEIVED: 17 February 1999

SAMPLE NO'S: A003

SUBMITTED BY: M. McKeown

WORK REQUESTED: Mineragraphy, photomicrographs

H.W. Fander, M.Sc.

REPORT CMS 99/2/9

NICKELIFEROUS DRILL CORE A003

Four core intersections were received for the purpose of determining the sulphide mineralogy. Polished sections were prepared and examined.

The mineralogy is very similar in all four intersections, differing only in extent rather than in kind.

Virtually the only sulphide present is fresh **pentlandite**; there are traces of associated chalcopyrite at ***259.9m***. The pentlandite is fresh but tends to be microfractured (along cleavage-planes) especially at ***231m***. It forms patches and veins in massive granular magnetite (often containing chromite), and is itself veined by a later generation of magnetite. Because of the microfracturing, the pentlandite would tend to overgrind (in metallurgical processing) and this needs to be carefully monitored.

Pentlandite grains from 10μ to 200μ occur in silicate gangue, and larger patches 500μ to several millimetres are common in the magnetite. At ***278.3m***, the pentlandite patches have thin envelopes of fine graphite. Pentlandite veinlets are from 3μ to 200μ wide in magnetite.

Evidently the pentlandite was emplaced after an early (?primary) magnetite-chromite phase and was succeeded by a later magnetite phase, perhaps related to serpentinisation.

REPORT CMS 99/2/9

PHOTOMICROGRAPHS

The photomicrographs were prepared to illustrate the various pentlandite occurrences in the mineralised intersections, and the relationships between the pentlandite (yellow) and magnetite (gangue is dark grey to black). All magnifications are the same - 135x.

1 and 2 A003/231m
Microfractured pentlandite (well-developed cubic cleavage) extensively veined by late magnetite.

3 A003/259.9m
Pentlandite patches and veins in massive primary (?) magnetite.

4 and 5 A003/272m
Massive coarse fresh pentlandite with minor magnetite (grey) and gangue (black).

6 A003/278.3m
Massive pentlandite with magnetite veinlets along cleavage-planes.

7 A003/278.3m
Pentlandite with magnetite veins, and primary magnetite enclosing chromite.

6613199

5

6.

7

69-C-199

661244

Central Mineralogical Services

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19 April 1999

Mr M. McKeown
McKeown Mining Pty Ltd
PO Box 50
RIDGLEY Tas. 7321

REPORT CMS 99/3/12

YOUR REFERENCE:	Fax 30 March 1999
DATE RECEIVED:	31 March 1999
SAMPLE NO'S:	A004
SUBMITTED BY:	M. McKeown
WORK REQUESTED:	Mineralogy

H.W. Fander, M.Sc.

REPORT CMS 99/3/12

DDH A004 INTERSECTIONS

Four drill core slabs were received for mineragraphic examination; polished sections were prepared, selecting the most sulphide-rich areas of the samples.

The mineralogy is more complex than in previous samples; several Ni-As phases were identified, and pyrrhotite is present in addition to pentlandite.

Pentlandite was the earliest sulphide mineral to form, and it is invariably extensively veined by magnetite. The Ni-As phases have partly replaced the pentlandite in places; in others, they occur independently (having perhaps completely replaced the pentlandite). In ***156.3m*** and ***170.2m*** there are unresolved difficulties in trying to reconcile As and Ni assays; the only As-bearing phases present are Ni compounds, and if the As assays are recalculated to the chief Ni-As phase (maucherite, with about 46% As), there is very little Ni left over for pentlandite. This suggests a sampling problem and indicates that the chief Ni-bearing phases in these two intersections are Ni-As minerals, (unless, elsewhere in these 1m lengths there are non-nickel As minerals such as arsenopyrite).

A004/121.4m

The only Ni sulphide present here is pentlandite, as scattered irregular to spheroidal patches up to 2 x 4mm in size and as sparse grains down to 10μ. The larger patches are laced with a network of very fine magnetite veins down to <5μ; there are also poorly-defined, minute dendritic patches of an unidentified phase, possibly mackinawite (FeS) or a related species. There would be some difficulty in eliminating either of these phases from the pentlandite but this is probably not essential in any case.

An earlier generation of magnetite occurs, often with chromite cores. Ultrafine graphite surrounds some of the pentlandite patches.

A004/139.2m

The two sulphides are pentlandite and pyrrhotite, as granular intergrowths generally on a <100μ scale, sometimes coarser; in these intergrowths the pentlandite is the major component. These composite patches are up to several millimetres across and are generally studded with 20μ - 200μ magnetite crystals, and some patches are thickly rimmed with magnetite. There are also granular magnetite clusters with small (5μ - 100μ) pentlandite inclusions. Shreds of pentlandite 5μ - 200μ are intergrown with silicate in an irregular fashion.

A004/156.3m

The main sulphide phases are pentlandite and pyrrhotite, with maucherite (Ni_3As_2), a trace of gersdorffite; rare, isolated small grains of native bismuth occur.

Pentlandite and pyrrhotite form coarsely-granular intergrowths on a 200μ - 600μ scale; the pentlandite is invariably full of magnetite veinlets along cleavage - planes.

Maucherite occurs as eutectoid intergrowths with pyrrhotite (on a 5μ - 30μ scale) in patches up to 1mm across; in places these eutectoids are seen to progressively replace pentlandite. Occasional patches of gersdorffite up to 200μ have invaded and partly replaced pentlandite and are finely intergrown with maucherite. Rare <30μ grains of native Bi are embedded in pyrrhotite.

A004/170.2m

The sulphides are pentlandite, pyrrhotite, maucherite and a trace of niccolite; no gersdorffite was seen. The pentlandite is full of magnetite veinlets and is coarsely intergrown with pyrrhotite. Maucherite and pyrrhotite occur as granular (50μ - 200μ) and eutectoid (10μ - 50μ) intergrowths replacing pentlandite - in some places the pentlandite has all but disappeared. Niccolite occurs as <50μ blebs in silicates and in pentlandite.

The pentlandite/pyrrhotite intergrowths occur as random patches and as veins up to 1mm wide.

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4 May 1999

Mr M McKeown
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REPORT CMS 99/4/9

YOUR REFERENCE: Fax 02/5/99

DATE RECEIVED: 30 April 1999

SAMPLE NO'S: A006

SUBMITTED BY: M. McKeown

WORK REQUESTED: Mineragraphy, Petrology

H.W. Fander.

H.W. Fander, M.Sc.

REPORT CMS 99/4/9

DRILLHOLE A006 INTERSECTIONS

Five slabs were received for examination, from 425.7m, 427.9m, 440.0 and 441.2m. The first three samples were examined in polished section, the last two in thin section.

A006/425.7m

The chief opaque mineral is magnetite, occurring as networks of granular crystals, containing patches of microbrecciated chromite.

There are traces of pyrrhotite, pyrite and ?pentlandite, as grains <1μ to 50μ in size, randomly scattered through the serpentinite.

The very dark colour of the rock is due to a pigmentation of ultrafine graphite; much of this is <2μ, as individual randomly orientated flakes within the serpentine minerals. Some graphite occurs as larger aggregates, up to 50μ, and occasionally as semi-continuous fine, veinlike bodies; rarely there are aggregates up to 120μ.

A006/427.9m

The mineralogy is similar to that of the previous intersection; there are more sulphides but less graphite. However, the graphite has a greater tendency to occur as subparallel wisps or semi-continuous veinlike bodies generally only a few microns wide.

The sulphides occur as small grains and occasional larger lenses (up to 2mm long) of intergrown pyrrhotite, pentlandite and ?valleriite (indistinguishable in polished section from mackinauwite); the pentlandite/?valleriite intergrowths are generally on an ultrafine scale (2-20μ).

A006/436.9m

In this intersection, much of the magnetite occurs as small, evenly-distributed grains with some clusters and very occasional embedded chromite.

Sulphides are common throughout and are generally composite patches, up to 1.5mm across, of finely intergrown pentlandite, ?valleriite, and pyrrhotite; the scale of these intergrowths is <50μ and is mostly in the 5-20μ range. In addition, ultrafine magnetite often occurs in the cleavage-planes of the pentlandite. Thus, this material is metallurgically problematical, especially as the valleriite (if confirmed as such) would be responsible for high levels of Mg in a nickel concentrate (the formula for valleriite is $[CuFeS_2]$ $[(Mg, Al, Fe)(OH)_2]$; for mackinauwite $(Fe, Ni)_9S_8$). A high Cu assay would indicate valleriite, since other Cu minerals are absent.

Traces of ultrafine graphite occur sporadically in the host-rock and especially along the margins of the sulphide patches.

A006/440m, 441.2m

These are similar rocks and may broadly be termed **silicified calc-silicate breccias.**

At 440m the rock consists of angular, small and large fragments of actinolite-quartz hornfelses and metaquartzite (?metachert) with minor actinolite patches, in a mass of crustiform, fine-to coarsely-crystalline quartz. There are small patches of crushed **chromite** crystals and occasional wisps of fine sulphides. Small aggregates of coarse diopside crystals occur.

At 441.2m the rock is made up of angular fragments of pale green coarsely-crystalline diopside, and fragments of fine-grained metaquartzite (?metachert), set in a mass of finely-to coarsely-crystalline and crustiform quartz. There are traces of fine sulphides; no chromite was definitely identified. No scheelite was detected in either sample.

Comments

Both rocks are thought to represent brecciated banded calc-silicates, perhaps originally chemical sediments (carbonate/chert units) contact-metamorphosed/metasomatised. The presence of chromite is intriguing; the Gordon Limestone contains detrital chromite (presumably at the base). However, they may have a different, ultimately ultramafic source.

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18 May 1999

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REPORT CMS 99/5/7

YOUR REFERENCE: Verbal Request

DATE RECEIVED: 13 May 1999

SAMPLE NO'S: A006

SUBMITTED BY: M. McKeown

WORK REQUESTED: Petrology

H.W. Fander, M.Sc.

REPORT CMS 99/5/7

DDH A006 INTERSECTIONS

Four intersections from A006, at 445.4m, 449.6m, 451.1m and 456.0m, were received for petrological study; thin sections were prepared and offcuts were examined under short- and long-wave U.V.

The rocks are essentially similar and may be described as a group.

The intersections are **calc-silicate breccias**. The original rocks consisted largely or wholly of fairly coarsely-crystalline pale diopside ranging into green hedenbergite, with sporadic patches of fibrous tremolite; the pale diopside fluoresces white/cream in short-wave UV, the tremolite is dull-pink/peach in long-wave UV (scheelite shows much more intense fluorescence). The darker hedenbergite does not fluoresce because of the quenching effect of the Fe.

The diopside-hedenbergite fragments are cemented by vein quartz. At 445.4m there are pinkish fragments of chert or metachert pigmented with ultrafine hematite (not unlike garnet in appearance in hand-specimen). In two of the intersections, prehnite has partly replaced the diopside, and at 449.6m and 451.1m there are traces of fragmented chromite. At 456.0m, traces of sulphides occur.

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15 June 1999

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REPORT CMS 99/6/3

YOUR REFERENCE: Letter 2.6.99

DATE RECEIVED: 4 June 1999

SAMPLE NO'S: A006 Intersections (15)

SUBMITTED BY: M. McKeown

WORK REQUESTED: Petrology, Mineragraphy

H.W. Fander, M.Sc.

REPORT CMS 99/6/3

DDH A006 (Avebury) INTERSECTIONS

Fifteen drill-core slabs were received for petrographic and mineragraphic study; thin sections and polished sections were prepared and examined, and are described in the attached sheets.

Summary

Most of the intersections are fine-to medium-grained serpentinites with chromite, probably originating as peridotites or dunites; some appear to be layered.

The upper and lower most intersections are diopside - magnetite rocks, probably representing pyrometasomatised serpentinites (though the intersection at 415.7m **may** have had a different origin).

The sulphides are apparently associated with a phase of magnetite formation/deposition which may be associated with the pyrometasomatic episode - this may have acted primarily as a mechanism for extracting and redistributing Ni from the original ultramafics, rather than being an actual source of Ni (the pyrometasomatic phase could have contributed S to the system).

SAMPLE NO: A006/116.5m **(T.S.** 68358) **Page 1**

CLASSIFICATION: Sepiolite-Magnetite Rock.

COMPOSITION: Consists mainly of finely fibrous sepiolite, with aggregates of small magnetite crystals with associated carbonate and ?recrystallised sepiolite with different texture.

FABRIC: Strongly radiating-fibrous textures in the massive sepiolite; individual crystals not recognisable. No relict textures to indicate original composition.

MINOR MINERALS: Crosscutting, late-stage carbonate veinlets. The introduced magnetite has a bronzy sheen.

INTERPRETATION/COMMENTS: Fibrous sepiolite is impossible to distinguish from finely-fibrous talc except by XRD. Origin of this rock was probably ultimately ultramafic.

SAMPLE NO: A006/122.0m **(T.S.** 68359)

CLASSIFICATION: Diopside-Magnetite Rock.

COMPOSITION: Chiefly small, random, well-formed diopside crystals which have largely replaced antigorite; stringers and clusters of small magnetite crystals. Interstitial antigorite partly replaced by fibrous carbonate.

FABRIC: Apart from isolated antigorite/carbonate patches showing network textures, there are no diagnostic features. The diopside is clearly replacive, with variable grainsizes and haphazard distribution.

MINOR MINERALS: Traces of sulphides, as scattered grains of ?pentlandite.

INTERPRETATION/COMMENTS: The composition of the various mineral phases (relict and replacive) indicates that the original rock was strongly magnesian and thus more likely to have been ultramafic. The magnetite was probably introduced (or possibly redistributed).

SAMPLE NO: A006/135.7m **(T.S. 68360)** **Page 2**

CLASSIFICATION: Diopside Rock.

COMPOSITION: Very largely composed of closely-packed small granular crystals of diopside, with scattered small antigorite patches (waxy yellow in hand-specimen), replacive carbonate patches and fine magnetite.

FABRIC: Antigorite patches have faint but definite fine-grained serpentinite textures. The diopside is featureless though some banding is suggested.

MINOR MINERALS: Scattered small grains of sulphides, possibly pentlandite.

INTERPRETATION/COMMENTS: Originally a fine-grained serpentinite which was thoroughly and almost completely contact-metasomatised, with a later phase of replacive carbonate.

SAMPLE NO: A006/141.5m **(T.S. 68361)**

CLASSIFICATION: Brecciated Serpentinite.

COMPOSITION: Fine-grained antigorite, with characteristic network textures, dusted with fine-grained secondary magnetite. Conspicuous, relatively coarse chromite surrounded by introduced magnetite and recrystallised antigorite.

FABRIC: Faint but definite preferred orientation in the overall fabric. Some shearing and wedging, with carbonate veining and recrystallisation of antigorite.

MINOR MINERALS: Traces of sulphides as ultrafine veinlets parallel with shearing; may be accompanied by very fine graphite.

INTERPRETATION/COMMENTS: Originally a medium-to fine-grained ultramafic rock but with coarser chromite. The introduction of magnetite preceded the shearing/brecciation event with which the sulphides and the ?graphite are associated.

SAMPLE NO: A006/142.6m **(T.S.** 68362) **Page 3**

CLASSIFICATION: Serpentinite.

COMPOSITION: Consists of antigorite with well-defined mesh or network fabric, variably impregnated with fine secondary magnetite; ovoid patches of clear, coarse recrystallised antigorite give the rock a troctolite-like appearance.

FABRIC: The mesh textures are quite fine, on a scale of 0.1 - 0.3mm. Recrystallised patches occur haphazardly and are up to 3mm across (occasionally larger). Minor fracturing and veining.

MINOR MINERALS: Patches of chromite surrounded by introduced magnetite and recrystallised antigorite. Carbonate veinlets in fractures.

INTERPRETATION/COMMENTS: The hand-specimen appearance is deceptive. However, troctolites contain abundant plagioclase and do not have the highly diagnostic network fabric.

SAMPLE NO: A006/154.2m **(T.S.** 68363)

CLASSIFICATION: Serpentinite.

COMPOSITION: Similar to **142.6m** but less "spotty". Antigorite with typical mesh fabric, dusted with fine secondary magnetite and recrystallised in places to coarser matted antigorite masses.

FABRIC: Good mesh fabric, variable from fine-to medium-grained; the recrystallised patches are up to 6mm across.

MINOR MINERALS: Occasional chromite grains; surrounded by magnetite. Scattered sulphide grains - possible pentlandite.

INTERPRETATION/COMMENTS: Inferred grainsize of the original ultramafic rock (?peridotite) suggests a minor intrusion or multiple intrusions.

SAMPLE NO: A006/176.7m **(T.S. 68364)** **Page 4**

CLASSIFICATION: Serpentinite.

COMPOSITION: Mainly consists of antigorite with fine mesh textures, pigmented with magnetite and variable amounts of ultrafine white leucoxene. Coarser, recrystallised antigorite as paler patches (without fine magnetite) and dark haloes (more magnetite).

FABRIC: Typical serpentinitic network textures. Weak preferred orientation, with parallel alignment of elongate recrystallised patches. Fine-to medium-grained.

MINOR MINERALS: Chromite patches with associated magnetite. Rare small sulphide grains.

INTERPRETATION/COMMENTS: Quite similar to the previous intersections. No evidence of gabbroic components.

SAMPLE NO: A006/211.5m **(T.S. 68365)**

CLASSIFICATION: Serpentinite.

COMPOSITION: Mainly antigorite with mesh textures; apparently coarser bands/zones alternating with finer material. Conspicuous, leucoxenic antigorite pseudomorphs after large ?pyroxene crystals.

FABRIC: Distinct compositional and textural banding. The ?pyroxene pseudomorphs tend to occur in parallel lenses/bands, with individuals up to 6mm.

MINOR MINERALS: Chromite grains, in thin stringers or bands. Rare, scattered sulphide grains. Crosscutting carbonate veinlets. Magnetite associated with chromite.

INTERPRETATION/COMMENTS: Relict features suggest a layered peridotite with pyroxene phenocrysts or accumulations.

SAMPLE NO: A006/259.7m **(T.S. 68366)** **Page 5**

CLASSIFICATION: Serpentinite.

COMPOSITION: Faintly banded due to variations in network textures and also the linear distribution of the chromite/magnetite crystals. Composed of fine antigorite with mesh textures, lenses and streaks of chromite-magnetite.

FABRIC: Random patches of recrystallised antigorite in an otherwise fairly uniform mass; these patches are fibrous-radiating and may be chrysotile rather than antigorite.

MINOR MINERALS: Some of the magnetite may have been introduced (perhaps during serpentinisation). Traces of fine sulphides.

INTERPRETATION/COMMENTS: Chromite is relatively conspicuous. The rock was probably a layered ultramafic type (peridotite-dunite).

SAMPLE NO: A006/304.5m **(T.S. 68367)**

CLASSIFICATION: Serpentinite.

COMPOSITION: Fine-grained antigorite with small mesh textures, pigmented with ultrafine magnetite, with scattered coarser magnetite, some with chromite cores. Parallel lenses of recrystallised antigorite.

FABRIC: Faint banding visible due to slight textural variations (coarser-finer) in the networks. Most individual textures are <0.1mm (ie "cells").

MINOR MINERALS: Scattered sulphides, possibly pentlandite. Occasional Mg-chlorite intergrown with coarser antigorite.

INTERPRETATION/COMMENTS: Very similar to **259.7m**.

SAMPLE NO: A006/333.1m **(T.S.** 68368) **Page 6**

CLASSIFICATION: Serpentinite with Sulphide Veins.

COMPOSITION: Fine antigorite mesh textures, pigmented with ultrafine magnetite. A few scattered coarser patches of recrystallised antigorite, especially associated with aggregates of magnetite crystals.

FABRIC: Fine-grained uniform network textures, with faint preferred orientation but otherwise featureless.

MINOR MINERALS: Sulphide veins up to 2mm wide, and random patches up to 3-4mm.

INTERPRETATION/COMMENTS: The sulphides postdate the serpentinisation but may be remobilised earlier material.

SAMPLE NO: A006/415.7m **(T.S.** 68372)

CLASSIFICATION: Diopside Rock with Magnetite.

COMPOSITION: A dense mass of matted prismatic diopside crystals with granular magnetite patches and veins (and associated sulphides). Younger veinlets of fibrous green actinolite.

FABRIC: No relict features. The diopside is uniform, randomly orientated, and has completely replaced pre-existing minerals.

MINOR MINERALS: Sulphides are associated and intergrown with the magnetite. Occasional veinlets of pale chlorite.

INTERPRETATION/COMMENTS: The nature of the original rock is not known - it may have been a carbonate rock (such as a dolostone) rather than a serpentinite. The sulphides are associated with the pyrometasomatic phase (diopside-magnetite).

POLISHED SECTION DESCRIPTIONS

333.1m
The sulphides occur in veins and consist of **pentlandite** with minor pyrrhotite, both extensively microfractured and penetrated by abundant magnetite veining. The chromite patches are also microfractured and magnetite-veined. No graphite was detected.

350.5m
Sulphides occur as finely fragmented patches almost always within magnetite, and comprise pyrrhotite and pentlandite; the occurrences are similar to those at **333.1m**.

Graphite occurs throughout the rock as small (<25μ) individual flakes and occasional groups of flakes in the antigorite networks, and also as thin rims around some sulphide patches. The sulphides have corroded margins.

372.8m
Pentlandite is the only sulphide present, largely confined to the magnetite, as microfractured masses veined by magnetite. Individual pentlandite crystals are up to 1mm in length, but inclusions in magnetite range from 1μ upwards. No graphite was seen.

413.3m
The sulphides are pentlandite finely intergrown with mackinawite, as grains down to 30μ in magnetite, and patches up to 1mm in gangue associated with magnetite; the pentlandite/mackinawite intergrowths are on a 1μ - 50μ scale. (**NB** Mackinawite is FeS in which up to 15% of the Fe can be replaced by Ni, Co, Cu, and may contain up to 9% Cu.)

There is abundant magnetite, and patches of fragmented chromite are embedded in granular magnetite.

415.7m
The main sulphide present is pyrrhotite, with associated minor pentlandite intergrown with mackinawite. The pentlandite usually occurs at the margins of pyrrhotite lenses as grains up to 300μ in size.

The sulphides are also intergrown with, and embedded in, magnetite masses.

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18 November 1999

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REPORT CMS 99/11/5

YOUR REFERENCE: Letter 14.11.99

DATE RECEIVED: 15 November 1999

SAMPLE NO'S: A011 285m, 289.5m

SUBMITTED BY: M. McKeown

WORK REQUESTED: Mineralogy

H.W. Fander, M.Sc.

CENTRAL MINERALOGICAL SERVICES

18 November 1999

SAMPLE REPORT (Mineralogy, Petrology, Ore Microscopy)

IDENTIFICATION

Job No CMS 99/11/5 **Date Received** 15.11.99

Reference Letter M. McKeown

Sample No. A011 285m, 289.5m

Nature of Specimen. DD core

DESCRIPTION **SECTION No.** 68730, 31

a. **Hand Specimen:**

Microscopic:

285m The dominant sulphides are fresh pyrrhotite and pentlandite with minor chalcopyrite and pyrite.

Pentlandite forms single crystals up to 6mm across, and also occurs as veinlike bodies 100μ - 400μ wide in pyrrhotite. There are also thin, platy, parallel inclusions up to 20μ wide and 100μ long, in pyrrhotite (?exsolution); there are occasional granular patches on a 50μ - 200μ scale but most of the pentlandite is >300μ.

Chalcopyrite forms lenses up to 1mm long, and much finer intergrowths with gangue, as well as granular intergrowths with pyrrhotite on a 100μ - 300μ scale.

289.5m

Disseminated sulphides as composite patches 50μ - 300μ and larger masses up to 20mm or more. They consist of pentlandite, pyrrhotite and pyrite, often with intergrown magnetite. The sulphide-sulphide and sulphide-magnetite intergrowths are on a much finer scale than at 285m, with pentlandite up to 700μ but mostly in the 50μ - 200μ range and inclusions in pyrrhotite 10μ - 50μ.

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(Mr M. McKeown)

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27 March 2000

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REPORT CMS 00/3/5

YOUR REFERENCE:	Fax 21.3.00
DATE RECEIVED:	22.3.00
SAMPLE NO'S:	As per report (A014)
SUBMITTED BY:	M. McKeown
WORK REQUESTED:	Mineragraphy, Petrology

H.W. Fander, M.Sc.

REPORT CMS 00/3/5

DRILL CORE A014 AVEBURY

Seven intersections from this hole were received for petrographic and mineragraphic examination; thin and polished sections were prepared and are detailed in the attached sheets.

Comments

The mineral assemblages in the first three intersections would normally be interpreted as metasomatised carbonate rocks in the absence of other evidence (eg relict igneous textures, presence of chromite) but the possibility of an Mg-rich ultramafic origin (via serpentinite) cannot be excluded.

In the last four intersections the evidence for an ultramafic origin is unequivocal. The intersection at **274.5m** is very dark and the serpentinite is thought to contain ultrafine graphite (?introduced).

The sulphides in their present form post-date the metasomatic event. The intersection at **246.6m** is of particular interest in containing well-formed dark green tourmaline crystals apparently replacing earlier, metasomatic silicates; this is also the rock in which all the pyrrhotite has been replaced by pyrite.

SAMPLE NO: **A014/244.3m** **(T.S.** 68885) **Page 1**

CLASSIFICATION: Mineralised Diopside-Talc Rock.

COMPOSITION: Zones of fine matted talc (green) and of prismatic fine diopside (white) form rims on larger masses of finely-intergrown talc and tremolite (grey areas).

FABRIC: Mostly uniform random intergrowths on a 0.1 – 0.2mm scale, with coarser diopside associated with the sulphides. No relict textures to suggest origin.

MINOR MINERALS: Sulphides are described in a separate section.

INTERPRETATION/COMMENTS: A metasomatic rock formed from a dolomitic carbonate or possibly from an Mg-rich ultramafic type (with the addition of Ca). Sulphides and diopside are closely associated, with sulphides the youngest deposited.

SAMPLE NO: **A014/246.6m** **(T.S.** 68886)

CLASSIFICATION: Altered Diopside-Talc Rock with Tourmaline

COMPOSITION: Originally composed of diopside and talc bands; the diopside bands are partly altered to matted actinolite needles passing into tremolite (away from sulphides).

FABRIC: Banded compositional fabric with coarser diopside bands rimmed with actinolite, and fine random-matted talc.

MINOR MINERALS: Well-developed stout prismatic green tourmaline crystals mainly in the sulphides apparently replacing diopside.

INTERPRETATION/COMMENTS: Similar to ***244.3m*** but with a further episode of metasomatism superimposed on the existing assemblage, in which actinolite developed, tourmaline formed, and pyrite replaced pyrrhotite.

CLASSIFICATION: Mineralised Tremolite Rock.

COMPOSITION: Entirely composed of tremolite and sulphides. The tremolite occurs as sheaves and matted masses of large and small needles, and the sulphides form irregular patches.

FABRIC: Random matted fabric, with tremolite needles up to 2-3mm. The sulphides incorporate tremolite and formed later. No relict textures.

MINOR MINERALS: - *(Sulphides described separately)*

INTERPRETATION/COMMENTS: Tremolite rocks are usually interpreted as metasomatised carbonate rocks in the absence of evidence indicating a different origin (eg relict serpentinite textures, anomalous Cn levels).

SAMPLE NO: **A014/261.5m** **(T.S.** 68888)

CLASSIFICATION: Mineralised Tremolite-Diopside Rock.
(with chromite)

COMPOSITION: *Relict patches of diopside are set in a mass of* matted tremolite needles. There are irregular patches of a greenish chlorite-like phase, possibly serpentinous, largely replaced by tremolite.

FABRIC: Fine to coarse needles. Matted fabric devoid of diagnostic relict textures.

MINOR MINERALS: Clusters and single grains of chromite with magnetite rims.

INTERPRETATION/COMMENTS: The presence of significant chromite supports an ultramafic origin for this rock (via serpentinite).

SAMPLE NO: **A014/267.7m**

CLASSIFICATION: Mineralised Serpentinite.

COMPOSITION: Typical fine-grained serpentinite, with antigorite forming fine network textures outlined and accentuated by ultrafine secondary magnetite. The antigorite is recrystallised to fine talc around sulphide patches.

FABRIC: Medium-to fine-grained fabric probably reflects grainsize of original ultramafic and suggests a minor intrusive or chilled contact.

MINOR MINERALS: Traces of carbonate associated with the sulphides. Thin chrysotile veinlets.

INTERPRETATION/COMMENTS: Network textures not sufficiently well-defined to deduce the composition of the original, fresh ultramafic rock. Mineralisation postdates serpentinisation.

SAMPLE NO: **A014/274.5m**

CLASSIFICATION: Mineralised Serpentinite.

COMPOSITION: This is a slightly coarser serpentinite with clearer textures. It consists of antigorite networks with abundant ultrafine magnetite (hence the dark colour of the rock); some recrystallisation to dark chlorite has taken place around sulphide patches.

FABRIC: More variable grainsizes. May have been porphyritic, with small olivine phenocrysts. Coarser acicular textures around sulphides.

MINOR MINERALS: Patches of altered ?tremolite needles associated with sulphides.

INTERPRETATION/COMMENTS: A slightly coarser version of **267.7m**

SAMPLE NO: **A014/281.9m** **Page 4**

CLASSIFICATION: Mineralised Tremolite Rock (with chromite).

COMPOSITION: Tremolite comprises the bulk of the rock as lenses of matted needles, as larger subradiating aggregates, and as subparallel bands. There are subparallel groups of magnetite crystals, with associated chromite.

FABRIC: The fabric has a definite preferred orientation; within larger tremolite lenses the textures are random or radiating. Some magnetite occurs as thin parallel bands.

MINOR MINERALS: A few patches of colourless chlorite and probable antigorite/serpentinite have survived.

INTERPRETATION/COMMENTS: Sufficient evidence of a serpentinitic-ultramafic origin. Fabric suggests an element of dynamic or regional metamorphism.

MINERAGRAPHY

High-sulphide patches were selected from each slab and were examined in polished section, to provide the maximum amount of information, of a purely qualitative nature.

Results

The sulphide mineralogy in all intersections is very similar, consisting chiefly of **pentlandite** and **pyrrhotite**; in the first three intersections, the pyrrhotite has been partly to entirely (***246.6m***) replaced by pyrite but the last four intersections contain no pyrite. There are sporadic traces of chalcopyrite, and the intersection of ***267.6m*** contains rare native Bi (<30μ) in pyrrhotite, with possible **rare** gersdorffite and millerite in pyrrhotite also.

The **pentlandite** is fresh and is intergrown with pyrrhotite on a 100μ to 1-2mm scale, and occurs in gangue silicates as 5μ - 50μ grains; where pyrite has replaced pyrrhotite, there are fine pentlandite-pyrite intergrowths on a 5μ - 50μ scale.

A rare textural feature is seen in 1mm patches of <20μ scale pentlandite/pyrrhotite intergrowths in a fine grid pattern.

Pyrrhotite is also fresh and commonly contains fine exsolution bodies ("flames") of pentlandite; it can be inferred that it will also contain some Ni in solid solution. This type of Ni occurrence will be difficult to recover.

The pentlandite shows good cleavage, and in the last three intersections it tends to be finely veined with magnetite, though not as extensively as in previous intersections (ie in other drill holes).

The sulphides were deposited after the host-rock silicates, and the conversion of pyrrhotite to pyrite must have occurred later, and may be related to shearing/fracturing.

There appear to be two generations of magnetite; an early one is associated with chromite, a later one veins the sulphides.

Most of the pentlandite is >200μ and occurs as simple intergrowths with other minerals; thus liberation should be good.

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8 August 2000

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REPORT CMS 00/7/8

YOUR REFERENCE:	Letter 20.6.00
DATE RECEIVED:	21.7.00
SAMPLE NO'S:	As per report
SUBMITTED BY:	M. McKeown
WORK REQUESTED:	Petrography, Mineragraphy

for **H.W. Fander, M.Sc.**

c.c. Allegiance Mining NL
C/- Newnham Mining & Exploration Services
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REPORT CMS 00/7/8

AVEBURY DRILL HOLES A018, A019

Twelve intersections (six from each drill hole) were received for petrographic and mineragraphic study; thin and polished sections were prepared and examined, and are briefly described in the accompanying summaries.

COMMENTS

1. The Rocks

The host rocks for the sulphide mineralisation are, or were, serpentinites of ultramafic (igneous) origin; all contain chromite, though this is generally embedded in magnetite and not visible in hand-specimen. The relict textural evidence suggests that the rocks were medium-to coarse-grained dunites or peridotite.

Some of the serpentinites have been modified by post-serpentinisation metasomatism to form a series of Ca/Mg and Mg silicates of broadly similar composition, including tremolite, talc, members of the chondrodite-humite-clinohumite group, and the closely related monticellite; it is not clear how the formation of these minerals relates to the sulphide mineralisation, but the occurrence of gersdorffite in ***A019/248.9m*** may be associated with this episode.

There are two generations of magnetite; the first has resulted from the serpentinisation process; the second (more extensive) is closely associated with the sulphides.

2. The Sulphides

The dominant sulphide and (almost) sole source of Ni is fresh pentlandite. Traces of others are very sporadic pyrrhotite, pyrite, chalcopyrite and gersdorffite. Since the distribution of the sulphides seems to be fairly random (on a small scale) it is unlikely that all the sulphides will be present in a single polished section of a particular intersection; thus for instance, although several intersections show significant As anomalies, only in one was gersdorffite detected. There are sampling problems on a larger scale too.

Pentlandite generally occurs dominantly as coarse crystals, up to 1-2mm forming compact patches up to 10mm+ and veins up to 2mm wide; also as 5μ - 100μ grains in magnetite and gangue and in other sizes *(see individual descriptions).*

Whilst most pentlandite is coarse grained (>100μ, mostly >200μ), some larger grains and patches are extensively microfractured (overgrinding!) or contain numerous ultrafine and coarser veins of magnetite (many too fine for liberation, thus reducing the grade).

However, amenability would seem to be good due to the simple mineralogy and textures, and good grades and recoveries should be achievable.

SUMMARY DESCRIPTIONS OF INTERSECTIONS

1. Petrography

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Sample	Brief Description
A018 214.9m	Mineralised serpentinite. Typical fine network textures except near sulphides where textures are fibrous. All antigorite and chrysotile, pigmented with ultrafine ?carbon. Apatite patches.
217.1m	Mineralised tremolite rock. Coarse prismatic and fine matted-fibrous tremolite. A few relict patches of unreplaced serpentinite. Patches of green Fe talc.
222.1m	Mineralised serpentinite. Coarse (large) network textures suggest an originally coarsely-crystalline ?peridotite or dunite.
225.1m	Mineralised serpentinite, very similar to ***217.1m***. Dark colour of these rocks believed to be due to ultrafine carbon within the antigorite and chrysotile.
227.6m	Mineralised serpentinite; medium-to fine-grained network textures with coarser fibrous patches. Possible breccia fabric.
231.7m	Mineralised serpentinite; medium-grained network textures. Thin parallel carbonate veinlets postdate sulphides. Chromite aggregates.
A019 245.2m	Mineralised serpentinite. Dark, carbon-pigmented networks of serpentine minerals. Patches of matted fibrous-radiating fresh and altered tremolite, and talc patches.
248.9m	Mineralised tremolite-humite rock. Large areas of granular humite or clinohumite (an Mg silicate with F,OH) incipiently serpentinised, and coarse fibrous tremolite.
253.3m	Mineralised serpentinite. Medium-to coarse-grained network textures with minor unaltered relict olivine, indicating a peridotite or dunite.
257.1m	Mineralised tremolite-monticellite rock, representing metasomatised serpentinite. Wide zone of pale matted-fibrous tremolite, a narrower zone of granular monticellite (or chondrodite), passing into serpentinite.
261.0m	Mineralised tremolite-humite rock. Wide zone of granular humite (pigmented with inherited carbon from original serpentinite) flanked by pale matted-fibrous tremolite.
266.1m	Mineralised coarse serpentinite with small replacive talc patches. Coarse network textures. Scattered chromite.

SUMMARY DESCRIPTIONS OF INTERSECTIONS
2. Mineragraphy

Sample	Brief Description
A018 214.9m	**Pentlandite** as 20μ - 1mm crystals forming masses up to 10mm. Also as <5μ - 100μ grains in gangue and veinlets in magnetite. Abundant magnetite; some encloses chromite. No other sulphides.
217.1m	**Pentlandite**, intergrown with magnetite, as 5μ - 500μ grains and as patches up to 2-3mm; most is >100μ. Traces of pyrrhotite intergrown with pentlandite on a <100μ scale. Chromite.
222.1m	**Pentlandite**, as granular intergrowths with magnetite and gangue on a 10μ - 300μ scale; some patches up to 1-2mm. Much is finely microfractured and may overgrind. Chromite.
225.1m	**Pentlandite** as patches up to 5mm with associated magnetite; also as 5μ - 100μ grains in magnetite. Most contains 1μ - 10μ veinlets and lamellae of magnetite and is microfractured. Chromite.
227.6m	**Pentlandite** as composite veins with magnetite (up to 2mm wide) in which pentlandite is 10μ - 500μ, mostly >100μ. Also as 10μ - 100μ grains in gangue. Rare small shreds of chalcopyrite. Chromite.
231.7m	**Pentlandite**, extensively microfractured and veined on an ultrafine scale with magnetite. Most is hosted by magnetite, as grains on a 10μ to 300μ scale (grains were originally much larger, up to 1-2mm).
A019 245.2m	**Pentlandite** as masses up to 10mm across composed of crystals 200μ - 3mm in size, with minor intergrown magnetite on a coarser scale. Also as <5μ - 100μ grains and 100μ - 1mm composites with magnetite in gangue.
248.9m	**Pentlandite** as granular intergrowths with magnetite on a 10μ - 300μ scale with patches up to 2-3mm of coarse crystals. In places rimmed and partly replaced by **gersdorffite** (30μ - 200μ scale). Chromite.
253.3m	**Pentlandite** as grains embedded in magnetite. Two size ranges - 100μ - 500μ, and 5μ - 100μ. A few patches partly to ?violarite. Most grains are >100μ. Chromite.

Sample	Brief Description
257.1m	**Pentlandite** is conspicuous, intergrown with gangue and with magnetite, as grains, angular patches 20μ - 1mm and larger masses up to 2mm with veinlike distribution; small grains are <5μ - 100μ. Chromite.
261.0m	**Pentlandite** as 5μ - 100μ grains, and veinlike masses up to 2mm wide in gangue. Also composite with magnetite as 5μ - 100μ grains and patches up to 1mm. Minor intergrown **pyrite**. Chromite.
266.1m	**Pentlandite**, as crystals 100μ - 2mm in magnetite, also as smaller grains 5μ - 100μ in magnetite, rarely in gangue. Rimmed and partly replaced by **bravoite** [(Fe, Ni, Co)S_2] 20μ - 1mm. Chromite.

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31 August 2000

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REPORT NO. CMS 00/8/6

YOUR REFERENCE:	Letter 24.8.00
DATE RECEIVED:	24 August 2000
SAMPLE NOS:	A024
SUBMITTED BY:	M. McKeown
WORK REQUESTED:	Petrology, Mineragraphy

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REPORT CMS 00/8/6

DDH A024 INTERSECTIONS

Eleven intersections were received for petrological and mineragraphic examination; thin sections were prepared of all samples, and polished sections were made of those samples with significant sulphides. The results of the examinations are attached.

Comments

The first four intersections comprise clearly recognisable though metasomatised sediments. They are mainly "black shales", i.e. pyritic, carbonaceous argillites in which most of the original pyrite now appears as pyrrhotite. A polymict conglomerate is also present, and a thin sandstone band occurs with a shale.

The following seven intersections consist largely or entirely of calc-silicates, especially diopside, with some talc (?retrograde metamorphism); the assemblage is characteristic of contact-metamorphism or pyrometsomatism of carbonate-rich rocks (e.g. dolostones). All contain detrital chromite. Two of the intersections contain mosaic quartz and are thought to have originated as cherts.

The sulphide mineralisation is simple (essentially pentlandite and pyrrhotite) and textures/grainsizes are generally coarse.

SAMPLE NO: A024/363.1m

CLASSIFICATION: Metasomatised Conglomerate.

COMPOSITION: Well-rounded pebbles of sandstone/quartzite, ?chert, felsic lavas and ?tuffs. Variably replaced by feldspars, epidote, sphene (from pre-existing TiO_2), phlogopite, actinolite. Matrix of the same assemblage.

FABRIC: Pebble structures well-defined and textures within pebbles also preserved in places, depending on susceptibility to alteration/replacement.

ACCESSORIES: Fine-grained sulphides (?pyrite) occur in more altered pebbles, and as veinlets.

INTERPRETATION/COMMENTS: Original nature of some pebbles is speculative because of severe alteration but nature of the whole rock is clear.

SAMPLE NO: A024/365.5m

CLASSIFICATION: Metasomatised Pyritic, Carbonaceous Argillite.

COMPOSITION; Scattered silt-sized quartz grains in finely-laminated clay/sericite, carbonaceous mater, ultrafine pyrite; much of the clay has been replaced by phlogopite, and there are veins and patches of matted actinolite needles.

FABRIC: Fine sedimentary fabric, with some later brecciation and veining, probably associated with the metasomatic event.

ACCESSORIES: Veinlets and patches of pyrrhotite (?altered pyrite).

INTERPRETATION/COMMENTS: Black shale facies, variably metasomatised.

SAMPLE NO: A024/366.6m

CLASSIFICATION: Brecciated, Metasomatised Argillite.

COMPOSITION: This intersection is similar to ***365.5m*** though more severely brecciated and metasomatised, with more conspicuous patches of pale grey-green matted actinolite. The rock is a carbonaceous, pyritic argillite.

FABRIC: Finely laminated; crenulated in places. Brecciated and veined.

ACCESSORIES: Fine pyrite occurs throughout. Veins and patches of pyrrhotite associated with metasomatism.

INTERPRETATION/COMMENTS: Correlatable with the previous intersection.

SAMPLE NO: A024/373.2m

CLASSIFICATION: Metasomatised Carbonaceous Argillite.

COMPOSITION: Mainly argillite with carbonaceous matter and pyrite; clays are variably phlogopitised. Sharp contact with phlogopitised argillaceous sandstone where clay matrix/cement has been replaced by fine phlogopite.

FABRIC: Finely laminated; with some crenulation and folding, minor brecciation.

ACCESSORIES: Scattered coarse-clastic quartz grains. Many quartz-phlogopite veinlets.

INTERPRETATION/COMMENTS: Same black shale sequence as the previous intersections, moderately metasomatised. The pyrrhotite in these rocks may be remobilised primary pyrite.

SAMPLE NO: A024/549.5m

CLASSIFICATION: Metasomatised Quartzite (?Chert)

COMPOSITION: Host rock of fine mosaic quartz typical of crystallised cherts, extensively veined and replaced by fine-and coarse-grained pale green diopside; patches of coarse talc flakes with associated sulphides.

FABRIC: Quartzite/chert is uniform, featureless, devoid of relict textures. Some talc and diopside is coarsely-crystalline.

ACCESSORIES: Isolated fragments of chromite embedded in the quartzite; they appear to be detrital. ***See separate description of sulphides***.

INTERPRETATION/COMMENTS: Metasomatism was intense; the original rock is thought to have been a chert of chemical origin. The sulphides are closely associated with the metasomatic assemblage.

SAMPLE NO: A024/561.7m

CLASSIFICATION: Metasomatic Rock with Sulphides

COMPOSITION: Consists of an assemblage of metasomatic minerals, mainly matted diopside needles with patches of talc and tremolite. The sulphides form irregular patches with inclusions of silicates (i.e. younger).

FABRIC: Random; no relict features. Chiefly large to small radiating-prismatic diopside. Talc is relatively coarse.

ACCESSORIES: Occasional grains of chromite. Sulphides are described separately.

INTERPRETATION/COMMENTS: Composition of original rock is not known but the present assemblage suggests that it may have been a carbonate rock (with detrital chromite).

SAMPLE NO: A024/574.6m

CLASSIFICATION: Diopside Rock with Sulphides

COMPOSITION: Consists almost entirely of finely-granular diopside, with scattered small patches of talc and tremolite, as well as small interstitial patches of sulphides.

FABRIC: Random, mostly granular and fine-grained, uniform and featureless. Relict textures or other diagnostic features are absent.

ACCESSORIES: A few detrital (?) grains of chromite. Sulphides are described separately.

INTERPRETATION/COMMENTS: Similar to ***561.7m***, and inferred to have been a carbonate rock.

SAMPLE NO: A024/587.0m

CLASSIFICATION: Diopside Rock with Sulphides

COMPOSITION: Almost entirely composed of medium-to fine-grained diopside with coarsely-crystalline patches and a few scattered needles of actinolite, with patches of sulphides and magnetite.

FABRIC: Random, uniform, featureless. No relict textures.

ACCESSORIES: Sulphides (described separately) are accompanied by magnetite.

INTERPRETATION/COMMENTS: Closely resembles the previous rock; similar inferred origin.

SAMPLE NO: A024/603.6m

CLASSIFICATION: Diopside Rock

COMPOSITION: Large and small interlocking plates of pale green diopside; the rock is virtually monomineralic.

FABRIC: Random fabric; diopside crystals up to 6mm+. No relict textures.

ACCESSORIES: Occasional small patches of fibrous tremolite. Scattered sulphide grains. Groups of small chromite splinters.

INTERPRETATION/COMMENTS: High grade contact metamorphism/metasomatism of a carbonate rock with detrital chromite.

SAMPLE NO: A024/609.3m

CLASSIFICATION: Metasomatised Quartzite (?Chert)

COMPOSITION: Quartzite consists of fine, clear mosaic quartz typical of chert. Extensively, randomly replaced by finely-to coarsely-granular diopside. Some diopside is bright green (probably chrome-diopside).

FABRIC: Probable breccia fabric, i.e. quartzite was brecciated, then metasomatised.

ACCESSORIES: Groups of small chromite splinters. A few small shreds of sulphides.

INTERPRETATION/COMMENTS: Very similar to the intersection of ***549.5m***; evidently some Ca from chromite was taken up by diopside in places to form chrome-diopside.

SAMPLE NO: A024/612.9m

CLASSIFICATION: Diopside Rock

COMPOSITION: Interlocking platy and prismatic crystals of diopside, and small interstitial sheaves of actinolite needles.

FABRIC: Random, medium-grained, featureless.

ACCESSORIES: Patches of sphalerite intergrown with diopside. Traces of fine sulphides.

INTERPRETATION/COMMENTS: Presumed pyrometasomatised carbonate rock.

MINERAGRAPHY OF SELECTED SAMPLES FROM A024

365.5m
Pyrrhotite only, as ultrafine grains and larger patches and veins.

549.5m
The major sulphide is pentlandite, with very minor pyrrhotite and a small trace of chalcopyrite.

The pentlandite mainly occurs as grains 200µ to 2mm or more, with veins up to 2mm wide; these are also <5µ - 100µ throughout the gangue; much of this is >20µ.

Pyrrhotite is intergrown with pentlandite, generally on a 20µ - 100µ scale; it represents only a minor metallurgical problem compared with the liberation of the pentlandite.

561.7m
The major sulphide is pyrrhotite, with associated minor pentlandite. The pyrrhotite is generally the host mineral, forming patches 500µ - 5mm in size containing grains, streaks and lenses of pentlandite 50µ - 300µ wide; in addition the pyrrhotite contains parallel exsolution bodies ("flames") of pentlandite 5µ - 20µ wide – these are numerous in places. Occasional pentlandite grains are up to 1mm in size. Finer pyrrhotite/pentlandite intergrowths, on a 20µ - 50µ scale, also occur.

574.6m
The sulphides are pyrrhotite and pentlandite, with a small trace of chalcopyrite. Pentlandite and pyrrhotite are generally intergrown, but on a coarse scale (200µ - 1mm) with some pentlandite grains up to 3mm; intergrowths on a 20µ - 100µ scale occur but are very minor only.

587.0m
Coarse intergrowths of pentlandite and pyrrhotite, generally on the scale of 200µ - 1mm with some pentlandite up to 2mm; 50-200µ pentlandite grains sparsely scattered through gangue. Chalcopyrite as occasional patches up to 600µ. A few crystals of ?gersdorffite 50µ - 150µ in gangue and embedded in pyrrhotite.

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2 October 2000

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REPORT NO. CMS 00/9/14

YOUR REFERENCE:	Fax 27.09.00
DATE RECEIVED:	28.09.00
SAMPLE NOS:	A025/293.7m, 315.5m
SUBMITTED BY:	M. McKeown
WORK REQUESTED:	Mineralogy

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661285

CENTRAL MINERALOGICAL SERVICES

SAMPLE REPORT (Mineralogy, Petrology, Ore Microscopy)

2 October 2000

IDENTIFICATION

Job No CMS 00/9/14 **Date Received** 28.09.00

Reference Fax M. McKeown 27.09.00

Sample No. A025 @ 293.7m and 315.5m

Nature of Specimen DD Core

DESCRIPTION **SECTION No.** 69116, 69117

a. Hand Specimen:

Microscopic:

293.7m

The major sulphides are pyrite and pyrrhotite, with minor chalcopyrite and a trace of sphalerite; no Ni minerals were identified and the Ni may be in solid solution in the pyrrhotite.

Whilst all the sulphides are associated and intergrown, it appears from textural relationships that pyrite is younger and has probably partly replaced pyrrhotite; it may thus also contain (inherited) Ni.

315.5m

The dominant sulphide is pyrrhotite, with traces of pyrite and chalcopyrite; no sphalerite was seen, indicating that the distribution of sulphides is erratic.

Ilmenite is conspicuous as well-formed crystals mostly embedded in pyrrhotite; most crystals are rimmed with sphene (result of contact metamorphism).

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(Mr M. McKeown)

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Appendix 3

Avebury Project - sampling and assay protocol

661287

TO File
FROM Mick McKeown
DATE 26 October 2000

SUBJECT AVEBURY PROJECT - SAMPLING AND ASSAY PROCEDURE

Sample handling from core tray to laboratory

After the drill core has been logged, the geologist identifies the core for splitting by marking the ends of each sample. The core is then split longitudinally by sawing; where the core is too friable for sawing, a sample equivalent to half the notional core is selected by hand. Each sample is placed in a calico bag. The bag is marked with the hole number and from and to depths; in addition, a cardboard tag is placed in each bag marked with the hole number and from and to depths.

Samples are transported to Analabs Pty Ltd laboratory in Burnie by staff of Newnham Exploration and Mining Services or McKeown Mining Pty Ltd.

Some samples are transported to Tasmanian Freight Services depot in Burnie or Devonport by staff of Newnham Exploration and Mining Services or McKeown Mining Pty Ltd, and then to the Amdel Pty Ltd laboratory in Adelaide by Tasmanian Freight Services.

Nomenclature

When a drill core sample is crushed and part of the crushed sample is retained, this is referred to as the *crushed reject*. When a crushed sample is pulverised and a smaller split is taken for assay, this is referred to as the *pulp*; the remaining pulverised sample is referred to as the *pulverised reject*. When Analabs takes a second sample of pulp for assay, this is referred to as the *second split*.

An assay is *accurate* if the assayed value is a true measurement of the true value. An assay is *precise* if repeated assays of the same sample result in the same value.

Sample size

Given the constraint of retaining half drill core for reference, the best sample which can be prepared is a fully pulverised sample of the half of the core which is sent to the laboratory. This was the procedure adopted for samples from holes A001 to A012 inclusive.

When hole A013 commenced it was becoming clear that some samples of drill core would soon be needed for metallurgical testwork. Consequently, to test the possibility of using smaller samples for assay, fifteen crushed samples from A013, between 390.m and 406.0m, were riffle split into two smaller samples both of which were fully pulverised and assayed, the samples being designated A and B:

bhid	from	to	Ni%	S%	Cu	Co	Pb	Zn
A013	390.9	391.9A	0.13	0.10	42	105	X	295
A013	390.9	391.9B	0.11	0.10	32	92	X	265
A013	391.9	392.9A	3.50	4.75	230	800	X	260
A013	391.9	392.9B	3.20	4.30	205	740	X	255
A013	392.9	393.9A	2.20	3.05	150	785	X	200
A013	392.9	393.9B	2.15	3.20	180	760	X	210
A013	393.9	395.1A	0.39	1.99	94	135	X	255
A013	393.9	395.1B	0.38	2.10	98	140	X	270
A013	395.1	396.0A	0.35	1.12	88	245	X	125
A013	395.1	396.0B	0.36	1.18	88	255	X	120
A013	396.0	397.0A	0.31	0.77	54	185	X	105
A013	396.0	397.0B	0.42	1.07	68	260	X	140
A013	397.0	398.0A	0.20	0.29	26	100	X	82
A013	397.0	398.0B	0.19	0.27	X	100	X	90
A013	398.0	399.0A	0.26	0.38	X	115	X	140
A013	398.0	399.0B	0.27	0.39	X	120	X	150
A013	399.0	400.0A	0.27	0.40	X	110	X	185
A013	399.0	400.0B	0.28	0.44	X	115	X	220
A013	400.0	401.0A	0.46	0.95	36	180	X	155
A013	400.0	401.0B	0.45	0.92	40	175	X	160
A013	401.0	402.0A	0.26	0.44	X	105	X	145
A013	401.0	402.0B	0.25	0.40	28	98	X	135
A013	402.0	403.0A	0.24	0.41	X	105	X	115
A013	402.0	403.0B	0.24	0.41	X	105	X	115
A013	403.0	404.0A	0.22	0.36	X	98	X	205
A013	403.0	404.0B	0.21	0.35	X	92	X	170
A013	404.0	405.0A	0.19	0.26	X	80	55	66
A013	404.0	405.0B	0.18	0.27	X	76	60	62
A013	405.0	406.0A	0.27	0.32	X	100	X	66
A013	405.0	406.0B	0.27	0.32	X	105	X	62

A plot of the nickel assays for samples A and B is attached (Figure 1). The results were thought consistent enough to reduce the crushed sample size to half of the

crushed half drill core. As a check, Gy's formula, with Francois-Bongarcon's correction (Francois-Bongarcon, 1999), was applied and confirmed the sample size as adequate (Attachment 1).

Subsequently, from hole A014 to hole A021 inclusive, the crushed core was riffle split and only one half of the crushed core was pulverised for assay. This procedure was adopted to preserve some crushed core for metallurgical testing. However, advice more recently received from John Butler, Metallurgical Consultant, suggests that crushed samples are too oxidised to be used in flotation testwork and, so, from hole A023 on, the whole half core has been pulverised.

Assay method

Samples from holes A001 and A002 were prepared and assayed by Amdel at the Adelaide, South Australia laboratory. Samples from all subsequent holes were prepared by Analabs at the Burnie, Tasmania laboratory and assayed at the Townsville, Queensland or Perth, Western Australia laboratories with some check assaying at Amdel. Both Analabs and Amdel have NATA registration.

When the first nickel assays were received for A003, the first hole of the 1999 program, the nickel assays did not tally with the proportion of nickel sulphide which could be seen in the core (McKeown, 1999) The assays had been determined by AAS by Analabs. Subsequent re-assaying of the samples by Amdel by ICP resulted in significantly higher nickel assays). Subsequent re-assaying of samples from A001 by ICP also revealed that the original assays of samples from this hole were also low, for example, from A001:

A001 243.9m to 254.6m	original AAS assays	1.15% Ni
	repeat ICP assays	1.64% Ni

A copy of part of McKeown (1999) is attached (Attachment 2).

All assaying is now by ICP, and intra-laboratory and inter-laboratory checks are routinely performed to ensure the validity of the method (see below).

Validation of assays

At Avebury, nickel occurs principally as pentlandite, to a lesser extent in pyrrhotite and, to a much lesser extent, as other nickel arsenide and nickel sulpharsenide minerals such as niccolite and gersdorffite. The various nickel bearing minerals are usually, at least partly, visible to the naked eye and the nickel and sulphur assays should reflect what can be seen in the core; this has not always been the case, for example, in A003 just described. This **visual check** is carried out by examining the core with the laboratory assay sheet in hand.

Analabs routinely assays **standards** (Attachment 3) and now reports them on the printed reports of each sample batch. The range of results reported from these standards is:

standard	**expected assay % Ni**	**range of assays % Ni**
SU1A	1.23	1.16 to 1.27
BM44	1.28	1.22 to 1.36
MHO	0.56	0.55 to 0.62
blank	<25ppm	<25ppm

The accuracy of nickel assays from Analabs can be inferred from these results. Plots of the nickel assays for these standards against date of reporting suggest that there is no systematic error in nickel assays over time (Figures 2 to 4).

Analabs also routinely undertakes **repeat assays** on individual pulps (for example, Attachment 4). A plot of repeat nickel assays against original nickel assays suggests that the precision of assaying is good (Figure 5). This is reinforced by the plot of nickel assays from **second splits** against original nickel assays (Figure 6).

As a further check, some pulps have been sent to Amdel for check assays; a plot of nickel assays from Amdel against nickel assays from Analabs suggests that accuracy and precision of nickel assays is maintained by both analysts (Figure 7).

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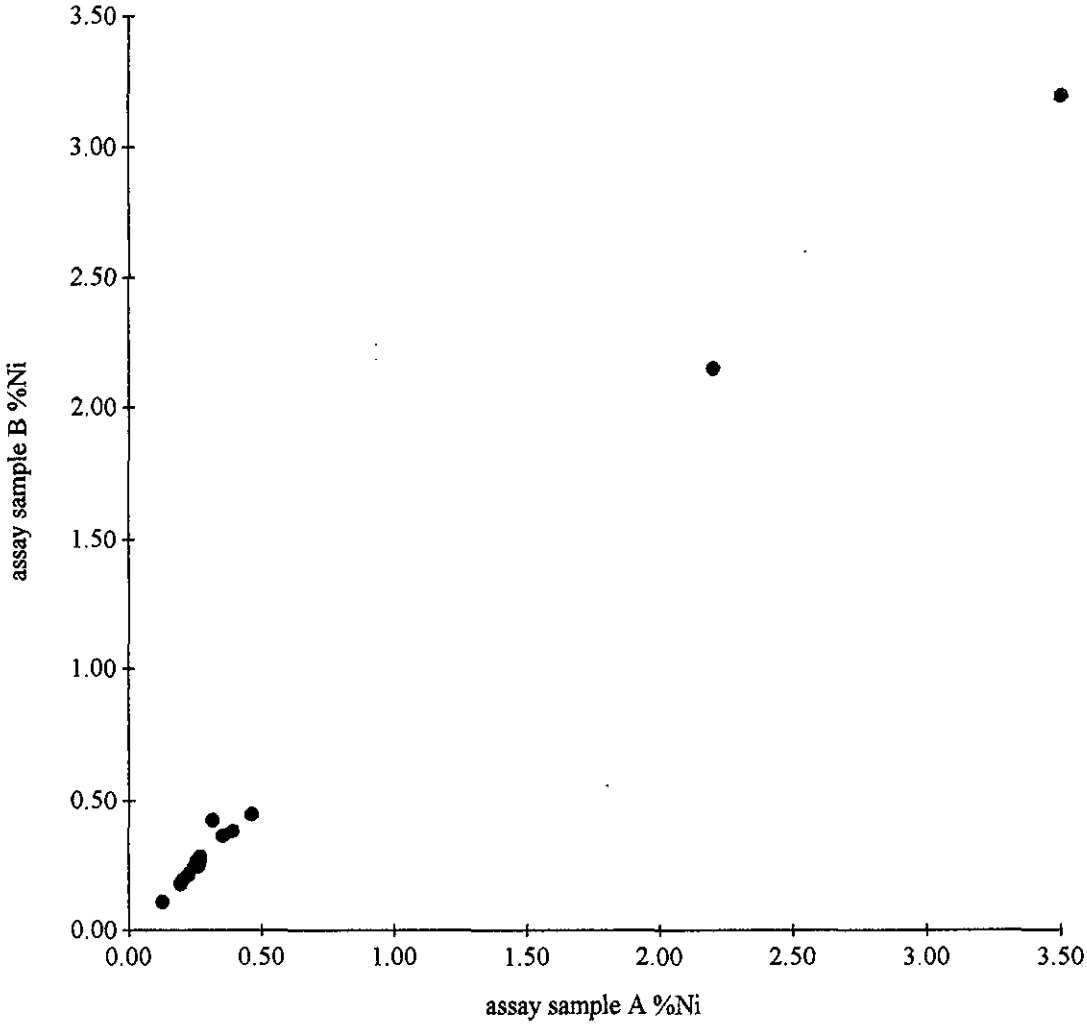
References

Francois-Bongarcon, D., 1999. The most common error in applying "Gy's formula" in the theory of mineral sampling.

McKeown, M.V., 1999. Results of diamond drilling on the Avebury grid EL28/88 by Allegiance Mining N.L., June 1999. McKeown Mining Pty Ltd.

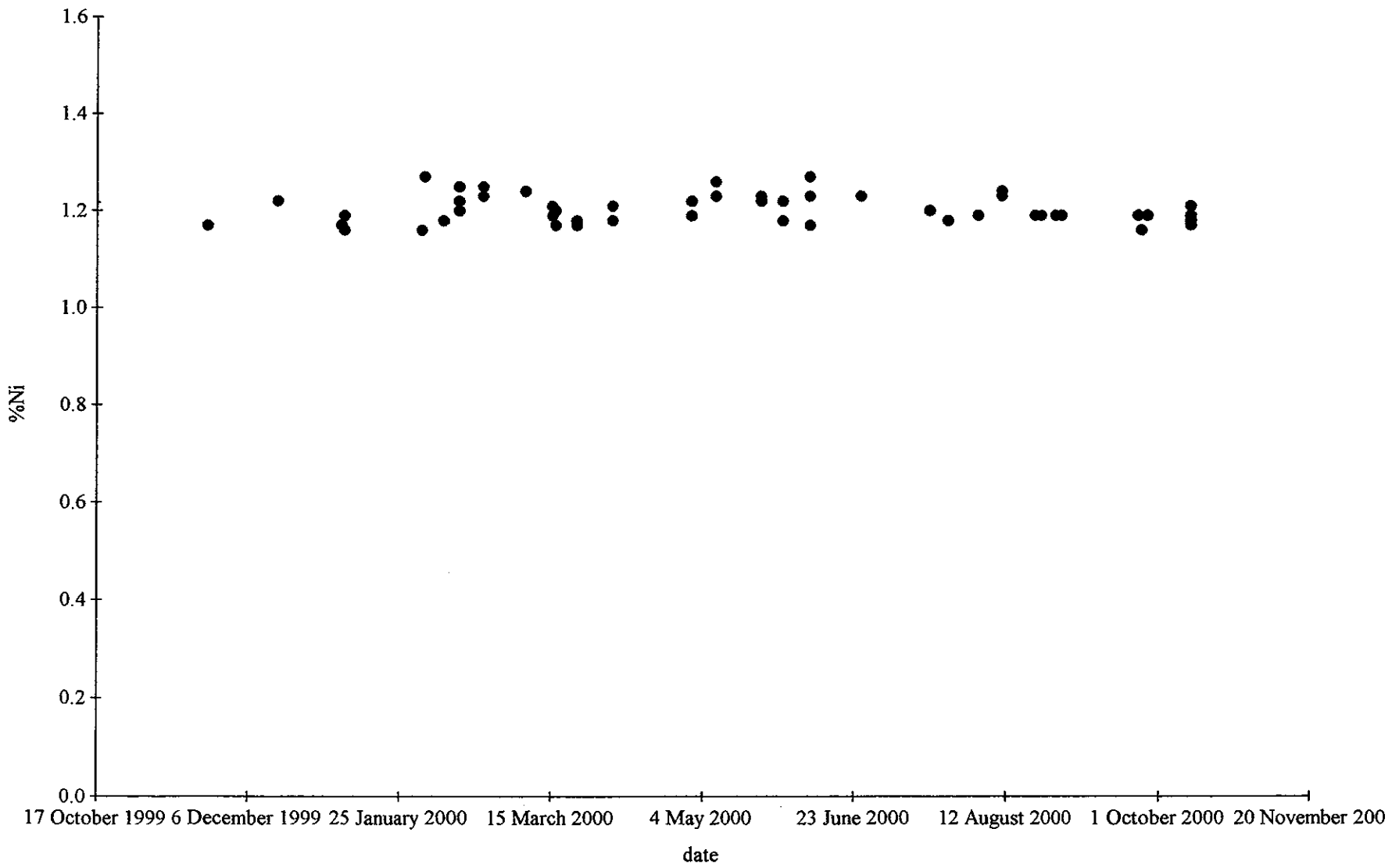
A013

assays of crushed splits



a013splits.wks

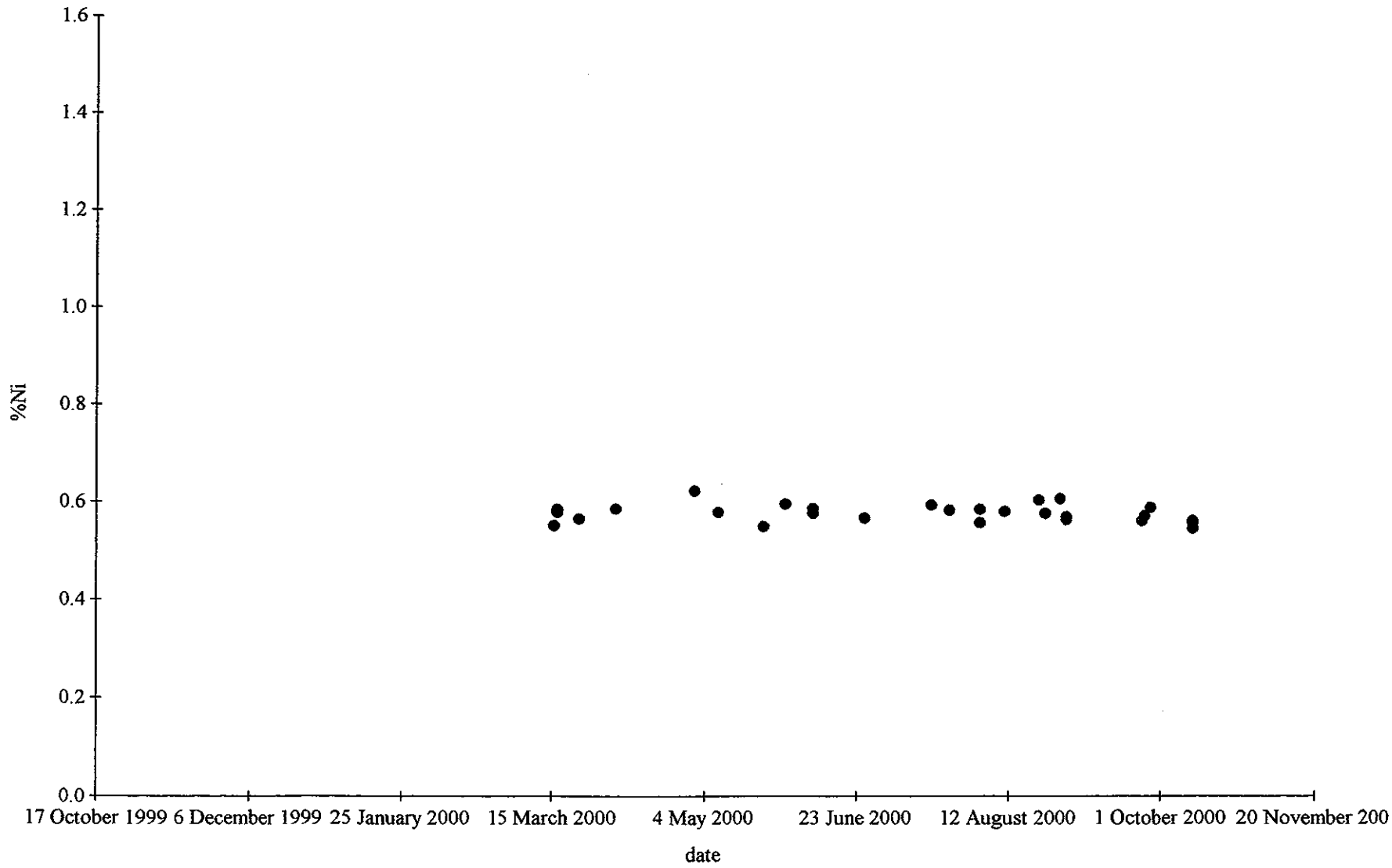
Figure 1



intralab.wks

Figure 2

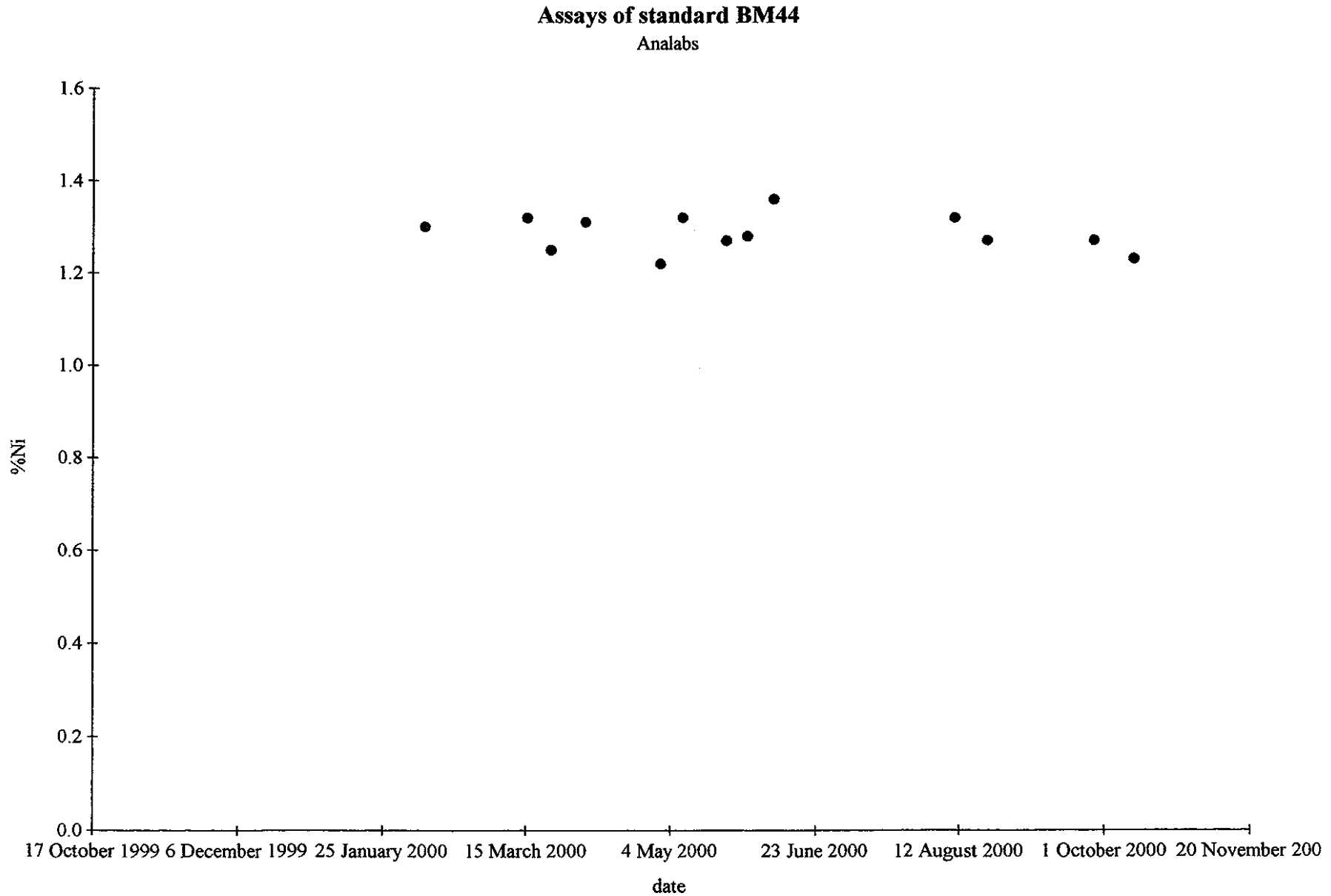
66393



intralab.wks

Figure 3

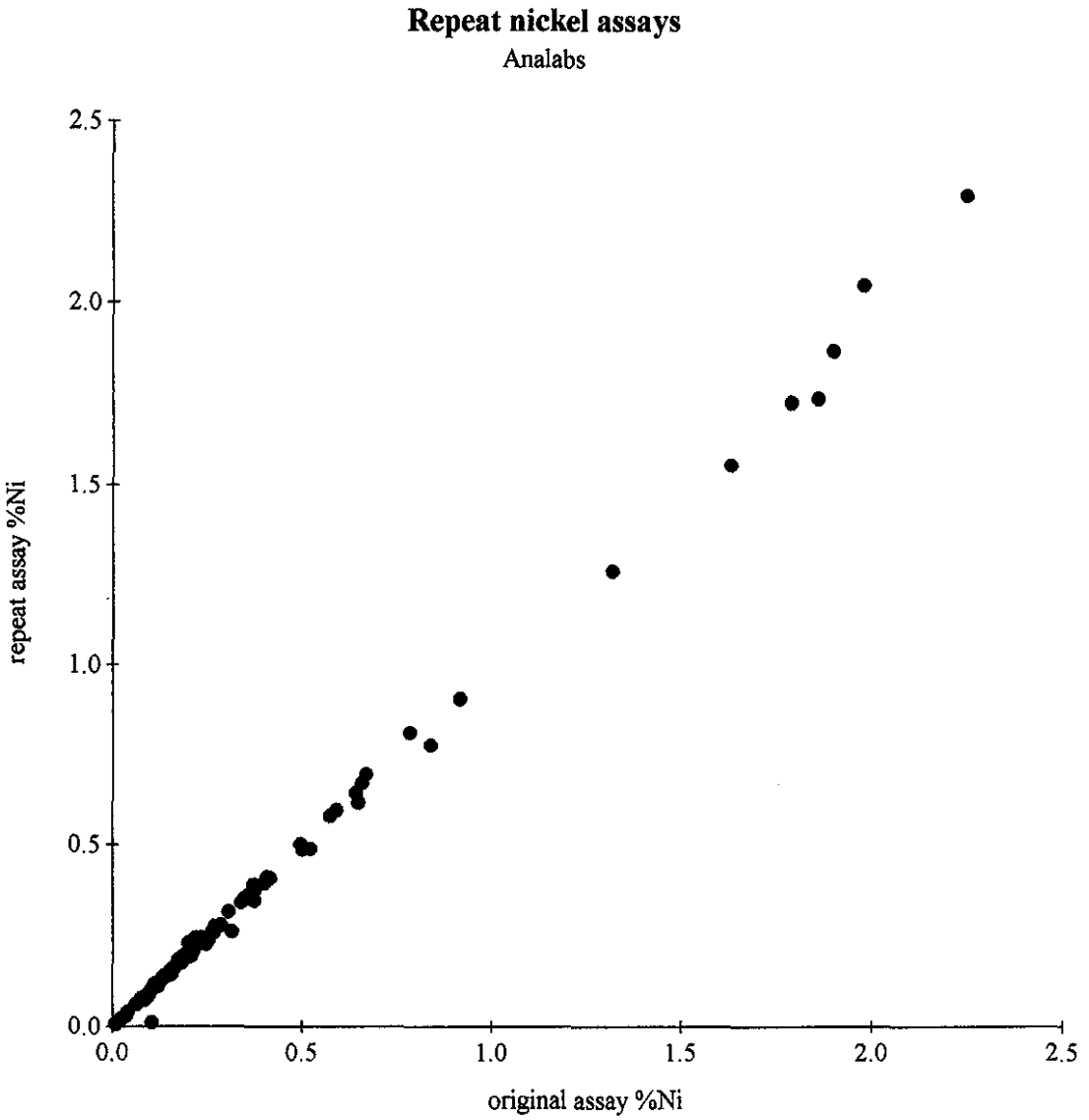
66193



intralab.wks

Figure 4

69

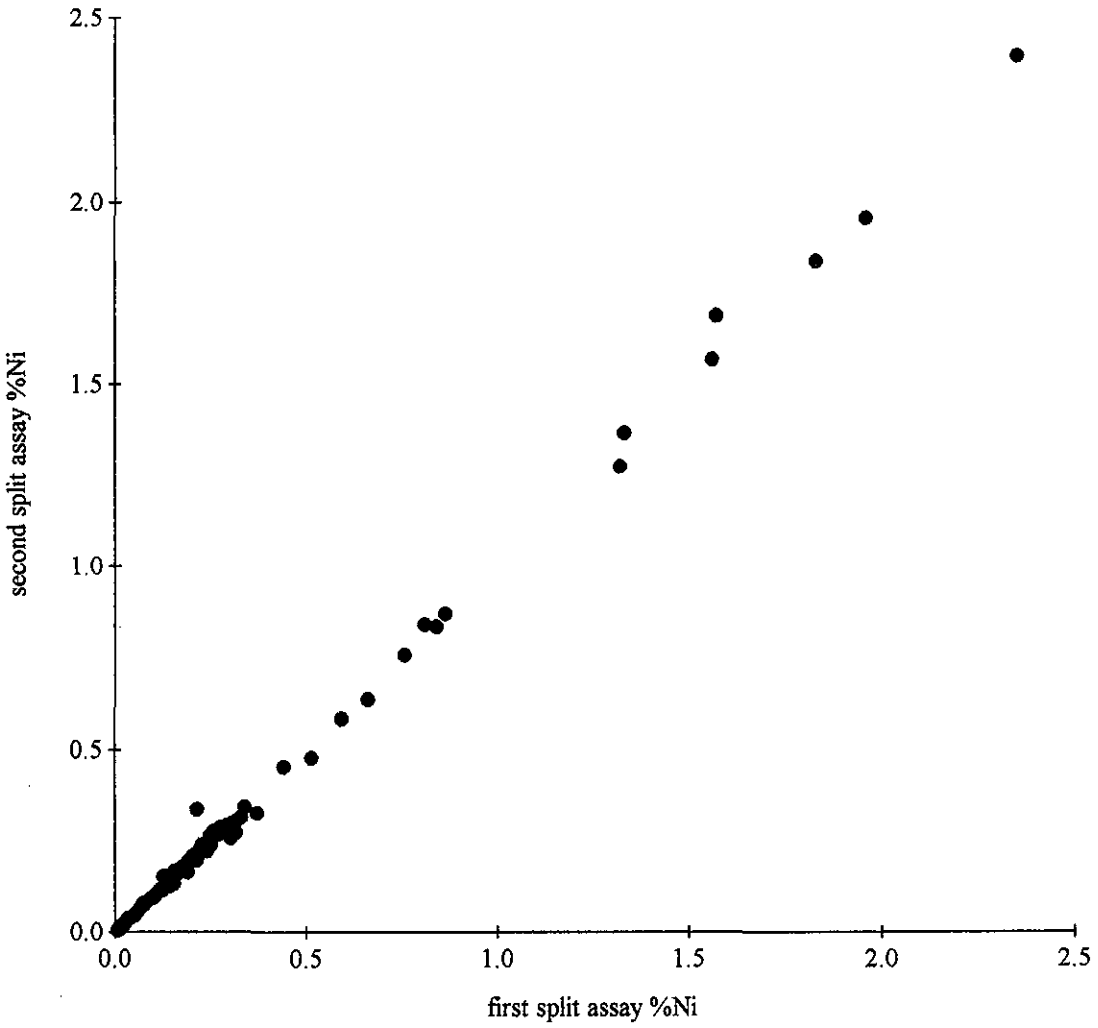


intralab.wks

Figure 5

Assays of second splits

Analabs

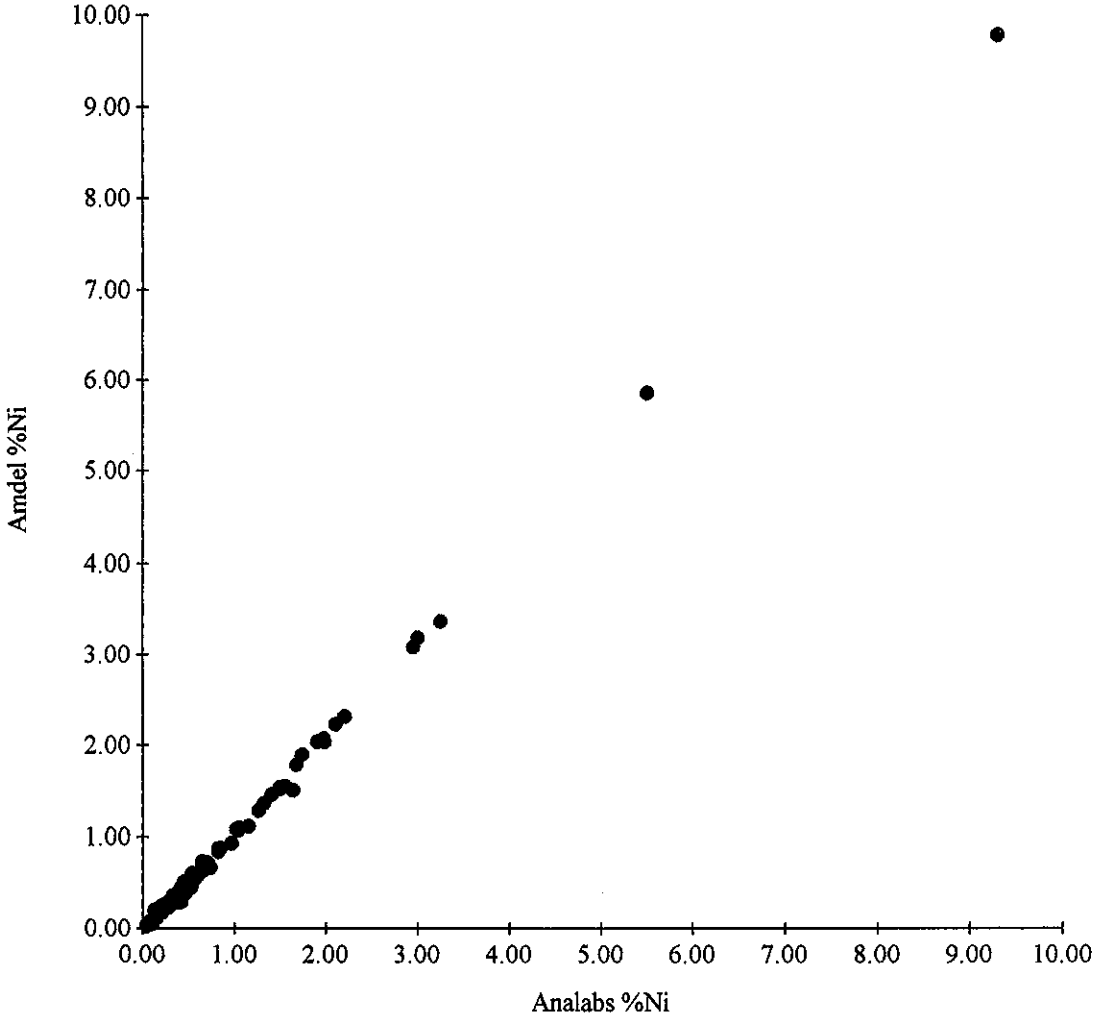


intralab.wks

Figure 6

Check assays of pulps

Analabs and Amdel



interlab.wks

Figure 7

Gy's formula for sample mass with the Francois-Bongarcon "correction"

units in Gy's formula are cgs (gm and cm)

			units
the mineral inputs			
density of the ore mineral		4.8	gm per cubic cm
density of the gangue		3	gm per cubic cm
metal grade		1.5	% metal
metal content of mineral		30	% metal
liberation diameter	dl	0.01	cm
the sample inputs			
mass being sampled		3	kg
the top size, the d95 diameter	dn	2	cm
the bottom size, d5 diameter		0.1	cm
the shape factor, flakes=0.2, spherical =0.5	f	0.25	
the statistical inputs			
the relative standard deviation required		1	%
"Francois-Bongarcon factor"	b	1.5	
calculated factors			
metal proportion by mass	-	0.050000	gm per gm
mineralogical factor	c	89.5	gm per cubic cm
liberation factor	lambda	0.000	1.000
grain size factor	g	0.25	
sampling constant		0.00	
actual standard deviation at grade quoted		0.015	+/- %Ni
fundamemtal sampling error	fse2	0.0001	
estimated sample mass			
minimum sample mass		158	gm

EXRACT FROM : RESULTS OF DIAMOND DRILLING ON THE AVEBURY GRID EL 28/88 BY ALLEGIANCE MINING NL, JUNE 1999

661300

1 INTRODUCTION

Allegiance Mining N.L. (Allegiance), in its own right or in Joint Venture with Rio Tinto Exploration Pty Ltd, has been exploring on three Exploration Licences in the Zeehan to Trial Harbour area:

EL28/88 in several parts, one of which lies immediately south-west of Zeehan, another at Trial Harbour

EL2/96 immediately to the north-west of Zeehan

EL22/97 in two parts, to the west and south-west of Zeehan.

Early in 1998, Allegiance completed two diamond drill holes, A001 and A002 at the Avebury prospect on EL28/88. A001 intersected significant nickel sulphide mineralisation, the zone exceeding 10 metres down-hole length at over 1% Ni (McKeown, 1998b). Allegiance considered that this result was sufficiently encouraging for more drilling to be undertaken and, in January 1999, the first of two planned holes was commenced.

A003, the first hole to be drilled this year, also intersected significant nickel mineralisation, including 57.3m at 0.82% Ni, and so three more holes were added to this year's program: A005, A006 and the deepening of A001. Table 1 lists the holes which have been drilled by Allegiance at the Avebury prospect. The total length drilled this year was 1465.6 metres.

2 1998 DRILLING PROGRAM

2.1 PROBLEMS WITH NICKEL ANALYSES

A003 intersected significant sulphide mineralisation but when the nickel analyses of apparently higher grade part of this hole were received the author considered that the nickel assays did not reflect the amount of sulphide visible in the core (McKeown, 1999a). There are four possible explanations for this:

- not all the sulphide is pentlandite,
- some of the nickel is occurring as a non-sulphide mineral,
- there was a problem with sample preparation,
- there was a problem with sample analyses.

A series of checks was initiated to resolve this apparent discrepancy.

ATTACHMENT 2

TABLE 1

ALLEGIANCE MINING NL

DIAMOND DRILLING AT THE AVEBURY PROSPECT

		collar co-ordinates					
year	**hole no**	**east**	**north**	**rl**	**brg**	**dip**	**final depth m**
1998	A001	354,648	5,357,225	152	005	-50	316.3
	A002	355,225	5,357,586	175	180	-50	307.0
1999	A001	354,648	5,357,225	152	005	-50	532.6
	A003	354,730	5,357,282	163	005	-50	339.0
	A004	354,625	5,357,372	146	353	-50	226.3
	A005	354,521	5,357,467	171	354	-50	202.5
	A006	354,736	5,357,190	150	000	-45	481.5

Notes: - all co-ordinates and bearings are AMG
- co-ordinates of all holes except A002 were based on trig survey; A002 was picked up using differential GPS

Petrological examination confirmed that the only sulphide present in A003 is pentlandite (Fander, 1999a).

Extensive re-assaying at three separate laboratories using different methods of nickel analysis resolved this problem (Appendix 3). The method decided upon for the nickel analyses for the 1999 drilling program is outlined in Table 2.

ATTACHMENT 2

TABLE 2

ALLEGIANCE MINING NL
ANALYTICAL PROCESS
1999 DRILLING PROGRAM

NQ drill core sawn in half longitudinally

whole sample crushed

whole sample pulverised (for samples less than 3kg), or
sample split in riffle, then pulverised (for samples more than 3kg)

small sample digested in hydrofluoric, perchloric, hydrochloric and nitric acids

Ni, As, S, analyses by ICP

2.2 UPDATE OF 1998 ASSAY RESULTS

A plot of nickel vs sulphur for the final nickel and sulphur assays for the part of the A003 intersection which was re-assayed indicates a nickel to sulphur ratio of about 1.0 (Figure 16). A similar plot for the 1998 assays for the A001 intersection indicates a nickel to sulphur ratio of about 0.67 (Figure 15). This suggested to the author that the 1998 nickel analyses of the A001 intersection were under-estimated (McKeown, 1999a). Consequently, pulps of relevant A001 samples were submitted for re-assay (Appendix 4) with the result that the nickel grade of the A001 intersection has been increased :

intersection	**length**	**analyses**	
A001 243.9m to 273.6m	29.7m	0.93% Ni	0.89% S
including			
A001 243.9m to 255.6m	10.7m	1.64% Ni	1.49% S
including			
A001 243.9m to 244.3m	0.4m	24.8% Ni	20.4% S

ATTACHMENT 2

Note that the nickel to sulphur ratio for these intersections is about 1.0; this ratio is in concordance with the composition of pentlandite, for which the Fe:Ni:S ratio is about 1:1:1.

3 1999 DRILLING PROGRAM

3.1 INTRODUCTION

As discussed above (see 1 INTRODUCTION), four diamond drill holes, totalling 1465.6 metres in length were drilled early this year. All holes were drilled using a Longyear 38 rig supplied by Diamond Drilling Tasmania Pty Ltd and access was by way of a track off the Zeehan to Trial Harbour Road. After the completion of the holes, down-hole geophysics was undertaken, so, to maintain physical integrity of the holes, they were lined to the bottom with plastic pipe.

In previous years, there were five holes drilled on the Avebury grid: holes ZA1 to ZA3 drilled by CRA Exploration Pty Ltd (CRAE) and A001 and A002 by Allegiance in 1998 (McKeown, 1998b). During this drilling program, the collars of all holes drilled so far were surveyed by staff of Campbell Smith, Phelps, Pedley Pty Ltd of Launceston (Table 1 and Appendix 6).

The locations of all the holes are shown in Figure 1, and cross-sections of holes A001 to A006, which have been drilled by Allegiance, in Figures 2, 3, 5, 6, 7 and 8.

The core was logged, and split and assayed where warranted. In addition, the magnetic susceptibility of the core was measured. Copies of drill logs (Appendix 1), final assay data (Appendix 5) and magnetic susceptibility data (Appendix 7) are attached.

3.2 NEW GEOLOGICAL MODEL

At the end of the 1998 drilling program, a geological model to explain the various nickel occurrences in the Zeehan to Trial Harbour area was proposed (Figure 4). The results of this drilling program and mapping along and around the drilling access track have led to changes in the model.

It now appears clear that, in the area on and around the Avebury grid, serpentinite is more extensive than has been indicated by previous mapping. The serpentinite is generally covered by deep clay, up to 15 or 20 metres thick, and, so, in general there is no serpentinite rock

ATTACHMENT 2

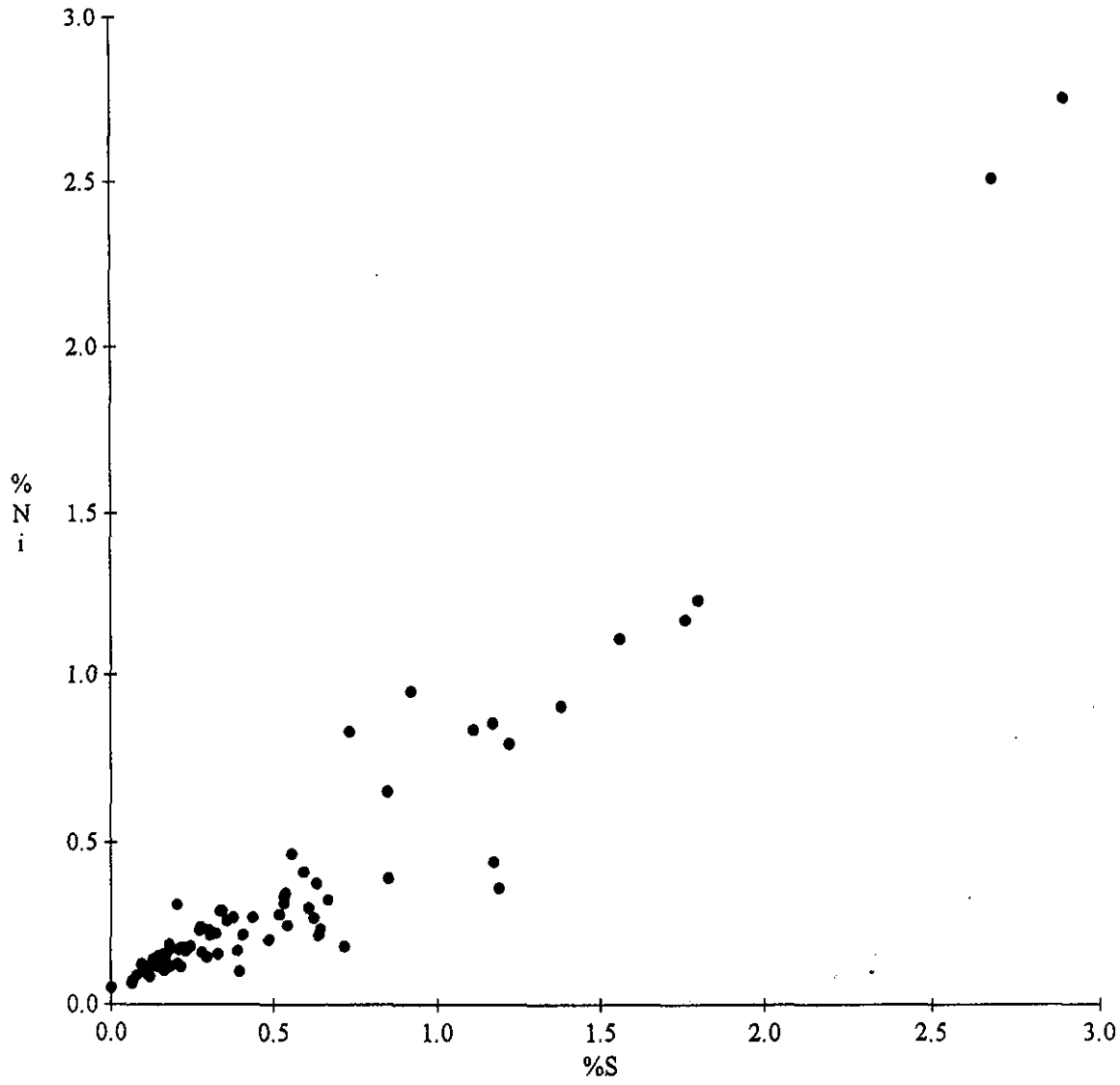


Figure 15 A003 - graph of % nickel vs % sulphur

ATTACHMENT 2

661305

SIGMA CHEMICALS
228 BALCATTA ROAD, BALCATTA W.A.
PHONE 345 2233 FAX 345 4012

CANADA CENTRE FOR MINERAL AND ENERGY TECHNOLOGY

NICKEL-COPPER-COBALT ORE SU-1a

CERTIFICATE OF ANALYSIS

	Recommended Values ± 95% Confidence Interval						
Ni	1.233	±	0.008%	Ag	4.3	± 0.3	µg/g
Cu	0.967	±	0.005%	Pt	0.41	± 0.06	µg/g
Co	0.041	±	0.001%	Pd	0.37	± 0.03	µg/g

WARNING

Bottles of SU-1a have been sealed under nitrogen in laminated foil pouches to provide long-term protection against oxidation during storage at CANMET. The recommended values for the certified elements pertain to the date of issue and the Canadian Certified Reference Materials Project cannot be responsible for changes occurring after receipt by the user. It is strongly recommended that opened bottles be stored under an inert gas in a desiccator or in a new heat-sealed foil pouch. Also the contents of the bottle should be exposed to air for the shortest time possible when taking subsamples. Unless these precautions are followed, the recommended values for SU-1a are potentially subject to change.

DESCRIPTION

SU-1a is intended as a replacement for SU-1 of which the stock is exhausted. The bulk material is a sample of feed to the Clarabelle mill of the International Nickel Company (Sudbury region) consisting of 27% chlorite, 15-19% of each of quartz, feldspar, mica and amphibole and less than 2% of each of calcite, siderite, sphalerite, pyrrhotite, pentlandite and chalcopyrite. It was dry-ground to minus 74 µm, blended, sampled systematically for analysis by X-ray fluorescence and chemical methods to demonstrate homogeneity suitable for use as a compositional reference material, and bottled in 200-g units.

ATTACHMENT 3

661206

CERTIFICATION

SU-1a was characterized by an inter-laboratory analysis method. The recommended values for nickel, copper, cobalt, platinum and palladium are the unweighted means of 80 to 242 accepted analytical determinations by 9 to 18 laboratories. The recommended value for silver is based on 100 accepted determinations by 10 laboratories of which the individual mean was calculated to lie within 2σ of the overall mean of those results by fire-assay methods.

Atomic absorption methods predominated in the determination of nickel, copper and cobalt. Colorimetric, X-ray fluorescence, gravimetric (for nickel) and electrolytic (for copper) methods were also employed.

Platinum and palladium were concentrated with a fire-assay technique and determined by atomic absorption, colorimetry, emission spectrography or neutron activation analysis.

The determination of silver was performed either by concentration with a fire-assay technique and atomic absorption or emission spectrographic finish or by acid dissolution and atomic absorption finish.

NON-CERTIFIED CONSTITUENTS

The concentrations of the following constituents are given for information only.

	Value
Au	0.15 µg/g
Rh	0.08 µg/g

INSTRUCTIONS FOR USE

The recommended values for SU-1a pertain to an "as is" basis.

LEGAL NOTICE

The Canadian Certified Reference Materials Project has prepared this reference material and statistically evaluated the analytical data of the interlaboratory certification program to the best of its ability. The Purchaser by receipt hereof releases and indemnifies the Canadian Certified Reference Materials Project from and against all liability and costs arising out of the use of this material and information.

REFERENCE

The preparation and certification procedures used for SU-1a are described in CANMET Report 80-9 "SU-1a: A Certified Nickel-Copper-Cobalt Reference Ore" which is available free of charge on application to:

Coordinator, CCRMP

CANMET

555 Booth Street

Ottawa, Ontario K1A 0G1

Canada

This Certificate of Analysis is available in French on request to the Coordinator, CCRMP.

ATTACHMENT 3

WINDARA LT

661207

GANNET HOLDINGS PTY. LTD
A.C.N. 009 214 466
P.O. Box 329, South Perth,
Western Australia, 6151.

Telephone: (08) 9367 1164
Fax: (08) 9368 1636

Recommended Values.

	Ni ppm.	Cu ppm.	Co ppm.	Fe ppm.	Mg ppm.	Cr ppm.
BM-44	12800	680	440	173000	28870	1740
[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]

Order No. 312800

Date 19-Jan-98

Reference Certificate No. U64BM.xls

Adrian Knowles.

Adrian Knowles.

FACTORY
43, Frederic Street, Naval Base, Western Australia. 6165.
Telephone: (08) 9410 2356 Fax: (08) 9437 1212

ATTACHMENT 3

661208

GANNET HOLDINGS PTY. LTD.
A.C.N. 009 214 466
P.O. Box 329, South Perth,
Western Australia, 6151.

Telephone: (08) 9367 1164
Fax: (08) 9368 1636

Recommended Values.

	Ni ppm.	Cu ppm.	Co ppm.	Fe ppm.	Mg ppm.	Cr ppm.
[redacted]						
MHO	5610	350	230	86400	27950	1280

Order No. 312800

Date 19-Jan-98

Reference Certificate No. U64BM.xls

Adrian Knowles.

Adrian Knowles.

FACTORY
43, Frederic Street, Naval Base, Western Australia. 6165.
Telephone: (08) 9410 2356 Fax: (08) 9437 1212

ATTACHMENT 3

Appendix 4

Avebury Project - downhole surveys

TO File
FROM Mick McKeown
DATE 12 October 2000

SUBJECT AVEBURY NICKEL PROJECT - DOWNHOLE SURVEYS

Error in magnetic bearings at Avebury

Magnetite in the Avebury serpentinites significantly deforms the local magnetic field. The largest variation detected, so far, occurs near the collars of A005 and A026 where there is a bearing error of 15°. Magnetic errors have been determined by comparing theodolite bearings of drill hole collars with magnetic bearings from downhole camera surveys made close to collars, and theodolite bearings between known points with magnetic bearings read by Suunto compass.

The error in magnetic variation resulted in some early drill holes, which were aligned assuming the accuracy of the magnetic compass, being drilled slightly skew to the planned bearing, for example, A014 which was planned on a collar bearing of 180° AMG was set up at 172° AMG . Other holes were set up by determining true north by the location of the sun at true noon, for example, A008, planned at 180° AMG actually drilled at 178° AMG, or by alignment along a line established by theodolite traverse, for example, A025, planned at 180° AMG, actually drilled at 179° AMG.

In the future, it is recommended that all holes be aligned along lines established by theodolite traverse.

Downhole surveys

Drilling crews made downhole surveys of all holes using downhole cameras owned by the drilling contractors. These camera surveys are subject to bearing errors caused by the presence of magnetite and pyrrhotite and, probably but to a lesser extent, pentlandite in the rocks at Avebury. During the most recent drilling program, ten holes were surveyed downhole using a Maxibor, a downhole survey instrument which is not affected by surrounding magnetic materials (Table 1). The Maxibor was supplied by, and operated by staff from, Diamond Drilling Tasmania Pty Ltd.

661211

TABLE 1

Down-hole survey methods

A001	camera	
A002	camera	
A003	camera	
A004	camera	
A005	camera	
A006	camera	
A007	camera	
A008	camera	Maxibor
A009	camera	
A010	camera	Maxibor
A011	camera	
A012	camera	Maxibor
A013	camera	Maxibor
A014	camera	Maxibor
A014A	camera	
A015	camera	
A016	camera	
A017	camera	
A018	camera	Maxibor
A019	camera	
A019	camera	Maxibor
A019A	camera	
A020	camera	Maxibor
A021	camera	Maxibor
A022	camera	
A023	camera	Maxibor
A024	camera	
A024A	camera	
A025	camera	
A026	camera	
A026A	camera	

Downhole dips from the camera surveys and the Maxibor surveys were compared (Figures 1 to 10); there were seven holes where the dips from both survey methods were in at least reasonable agreement, that is within about 3^{o} of each other (Table 2).

TABLE 2

hole no	dip agreement	maximum dip disagreement	operator
A008	good	1.7^{o}	S
A010	good	0.1^{o}	S
A012	poor	3.3^{o}	P
A013	poor	14.0^{o}	P
A014	good	1.0^{o}	S
A018	good	1.2^{o}	W
A019A/A019A	reasonable	2.1^{o}	W
A020	reasonable	3.1^{o}	W
A021	good	1.7^{o}	W
A023	poor	9.2^{o}	M

The reliability of the Maxibor surveys seems to rely on the Maxibor operator, all of the apparently good and reasonable surveys were carried out by two particular operators.

Also, a comparison of the downhole camera bearings and the Maxibor bearings was made. For all camera surveys, downhole bearings as read were converted to relative bearings. For each hole, the collar bearing as determined by theodolite survey was compared to the downhole camera bearing closest to the collar, the difference between the two bearings being applied to all the downhole bearings, for example, for A008 the difference applied to all bearings was -4^{o}:

bhid	at	bearing as read magnetic	bearing as read AMG	bearing relative AMG
	m			
A008	0	-	178	178
A008	50	169	182	178
A008	100	173	186	182
A008	150	174	187	183
A008	200	179	192	188
A008	250	178	191	187
A008	300	184	197	193
A008	350	188	201	197
A008	400	119	132	128
A008	454	177	190	186
A008	498	186	199	195
A008	560	163	176	172

After this correction was applied, obviously erratic bearings, such as that at 400m, were interpolated or extrapolated to fit the general trend of the bearing change for the hole; the general trend of the bearing change is best determined from a plot of bearing against downhole depth (see bearing plot for A008):

bhid	at	bearing as read magnetic	bearing as read AMG	bearing relative AMG	bearing corrected AMG		
	m						
A008	0	-	178	178	178		
A008	50	169	182	178	178		
A008	100	173	186	182	182		
A008	150	174	187	183	183		
A008	200	179	192	188	188	pyrrhotite	
A008	250	178	191	187	187		
A008	300	184	197	193	193		
A008	350	188	201	197	197		
A008	400	119	132	128	196	magnetite	interpolated
A008	454	177	190	186	196	magnetite	interpolated
A008	498	186	199	195	195		
A008	560	163	176	172	195	magnetite	extrapolated

After the correction was made, there was reasonable agreement between the corrected downhole relative camera bearings and the Maxibor bearings:

bhid	at	bearing as read magnetic	bearing as read AMG	bearing relative AMG	bearing corrected AMG	bearing Maxibor AMG
	m					
A008	0	-	178	178	178	178
A008	50	169	182	178	178	178
A008	100	173	186	182	182	179
A008	150	174	187	183	183	181
A008	200	179	192	188	188	183
A008	250	178	191	187	187	184
A008	300	184	197	193	193	186
A008	350	188	201	197	197	187
A008	400	119	132	128	196	188
A008	454	177	190	186	196	190
A008	498	186	199	195	195	190
A008	560	163	176	172	195	190

The downhole survey data and information for all holes is attached.

Recommended protocol for dealing with downhole surveys

As a result of the investigations so far, the following protocol is now recommended for dealing with the existing downhole surveys.

always use the collar bearing and dip determined by the surveyor

These data are the most reliable survey data we have.

always use the down-hole dips from the camera surveys

It is difficult to see how the dip apparatus in the downhole camera can be drastically inaccurate, whereas, the dips from the Maxibor are apparently subject to operator error, probably when the operator incorrectly assembles the Maxibor instrument.

if a Maxibor survey is available and if the dips from the Maxibor survey are in good agreement with dips from the camera, use the Maxibor survey bearings

If the Maxibor is working properly, the method that the Maxibor determines bearings will be unaffected by magnetic effects in the surrounding rocks and, so, the Maxibor bearings should be used. If the maximum dip difference between the two methods at each survey depth is less than 3^{o}, use the Maxibor bearing results.

if a Maxibor survey is available but if the dips from the Maxibor survey are not in good agreement with dips from the downhole camera, or if a Maxibor survey is unavailable, use corrected relative camera bearings

The procedure to be followed has been outlined in the example, based on hole A008, given above.

1. Compare the first reliable camera bearing with the collar bearing as supplied by the surveyor; that is, the first reliable camera bearing is the first bearing which is not obviously affected by magnetic effects caused by mineralisation, casing or equipment on the surface. Determine the bearing correction constant.

2 Apply the correction constant to all the as-read camera bearings.

3. Plot a graph of relative camera bearings and corrected camera bearings.

4. For bearings which are extreme to the trend of bearing change, determine and note the cause; this will usually be the presence of magnetic mineralisation or casing.

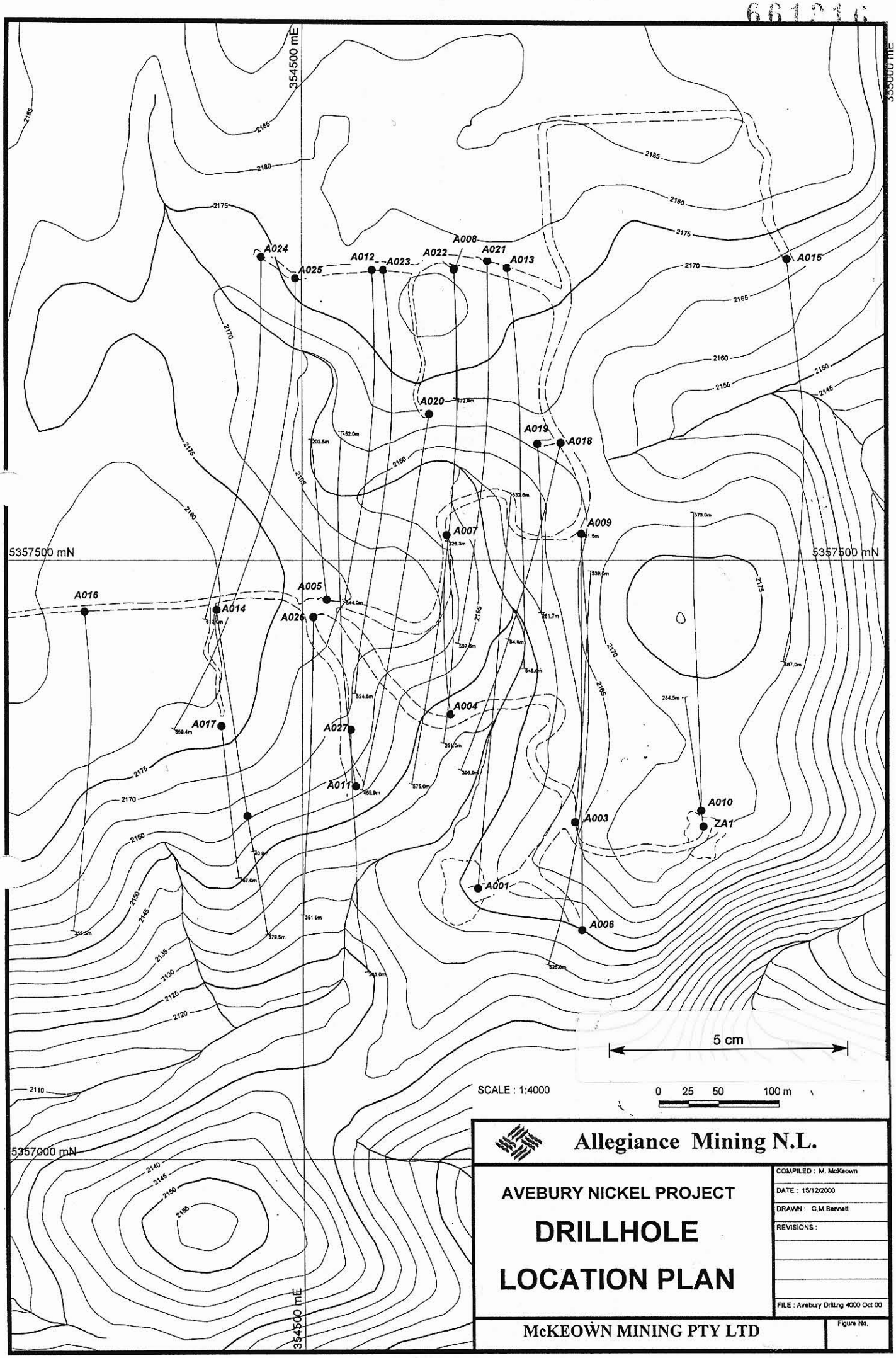
5. If there are sufficient good corrected bearings, then, for the bearings which do not fit the trend of bearing change, interpolate or extrapolate a bearing and note that this has been done.

if there is no reliable Maxibor survey and if there are insufficient good corrected relative camera bearings, then the bearings for the hole should be determined by analogy to the bearings in nearby holes of similar orientation

Note which nearby hole has been used as the analogous model.

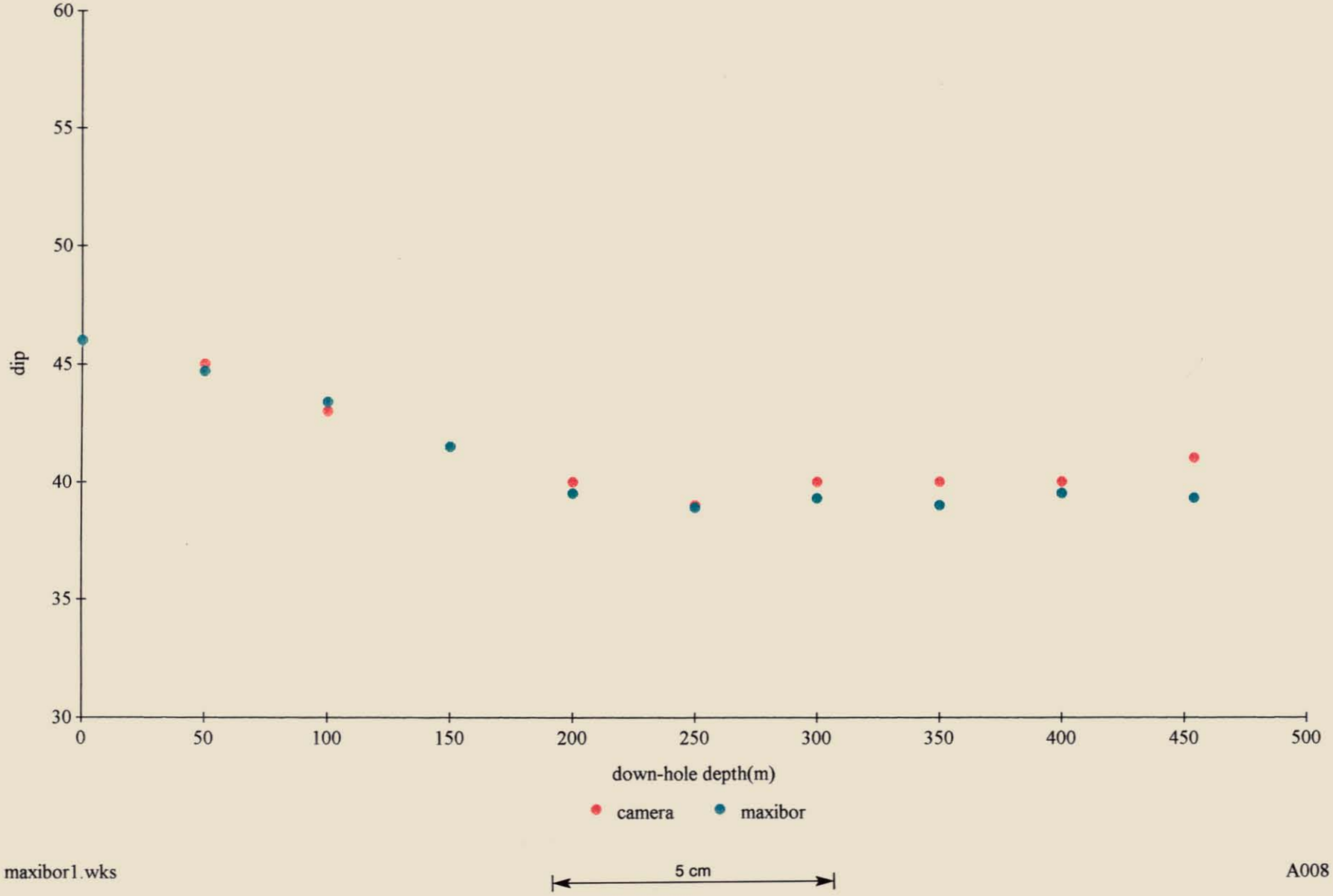
Conclusion

The downhole bearings which have been used for each hole are listed on the cover page of the drill log. A plot of the plan traces of all drill holes reveals a reasonably consistent variation in deviation which is dependent on whether the holes are drilled to the north or to the south (plan attached). Holes drilled to the north appear to tend to maintain collar bearing; holes drilled to the south tend to maintain collar bearing, and dip, to about 200 to 250 metres downhole depth and then deviate to the west, that is in the direction of bit rotation.


354500 mE
355000 mE
5357500 mN
5357000 mN
A024
A025
A012
A023
A022
A008
A021
A013
A015
A020
A019
A018
A009
A007
A016
A014
A005
A026
A017
A027
A004
A011
A010
ZA1
A003
A001
A006
5 cm
SCALE : 1:4000
0 25 50 100 m
Allegiance Mining N.L.
AVEBURY NICKEL PROJECT
DRILLHOLE
LOCATION PLAN
COMPILED : M. McKeown
DATE : 15/12/2000
DRAWN : G.M.Bennett
REVISIONS :
FILE : Avebury Drilling 4000 Oct 00
McKEOWN MINING PTY LTD
Figure No.

A008 camera and Maxibor dips

(Maxibor operator: S)



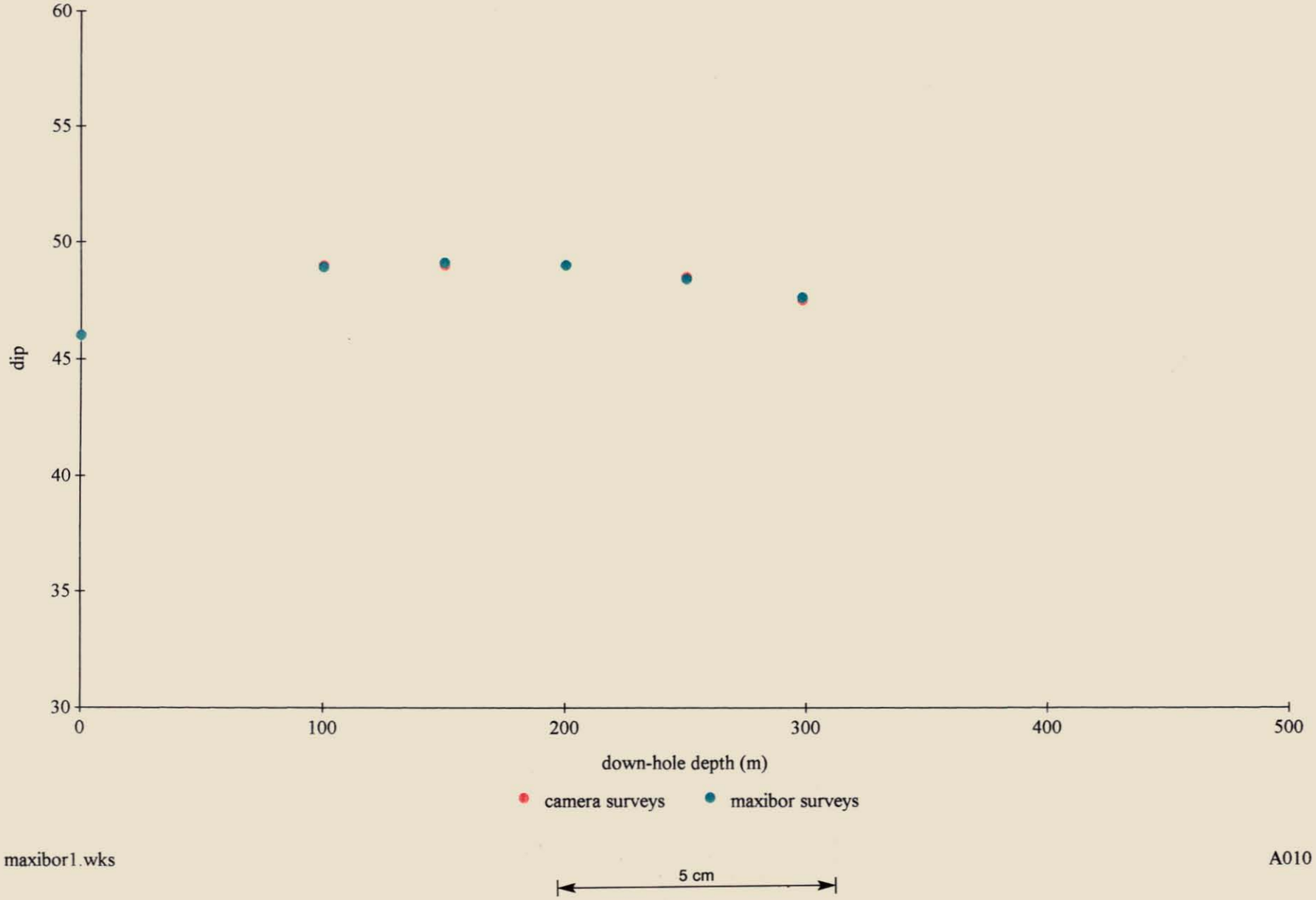
maxibor1.wks

5 cm

A008

A010 camera and Maxibor dips

(Maxibor operator: S)



maxibor1.wks

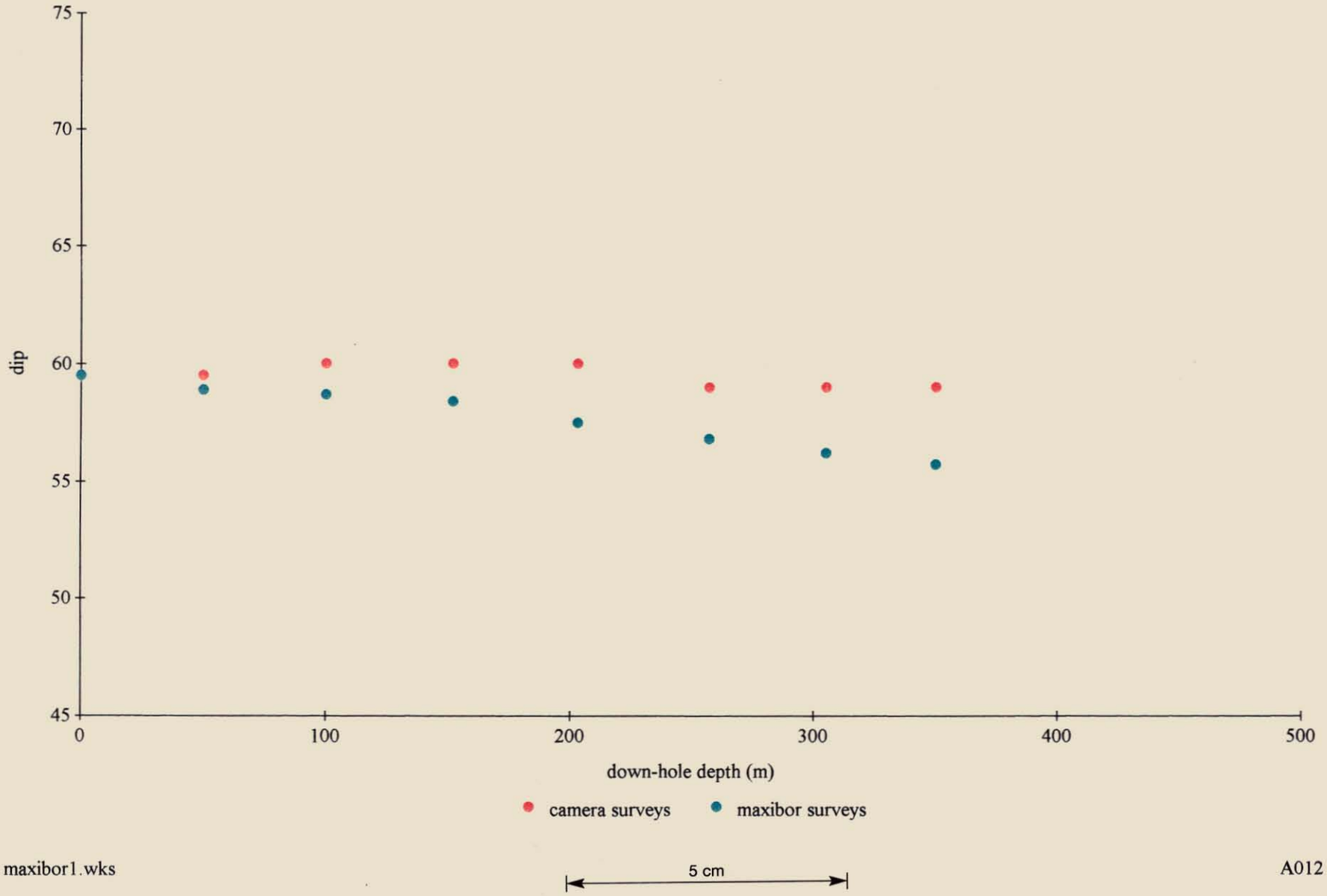
5 cm

A010

S 6913 6

A012 camera and Maxibor dips

(Maxibor operator: S)



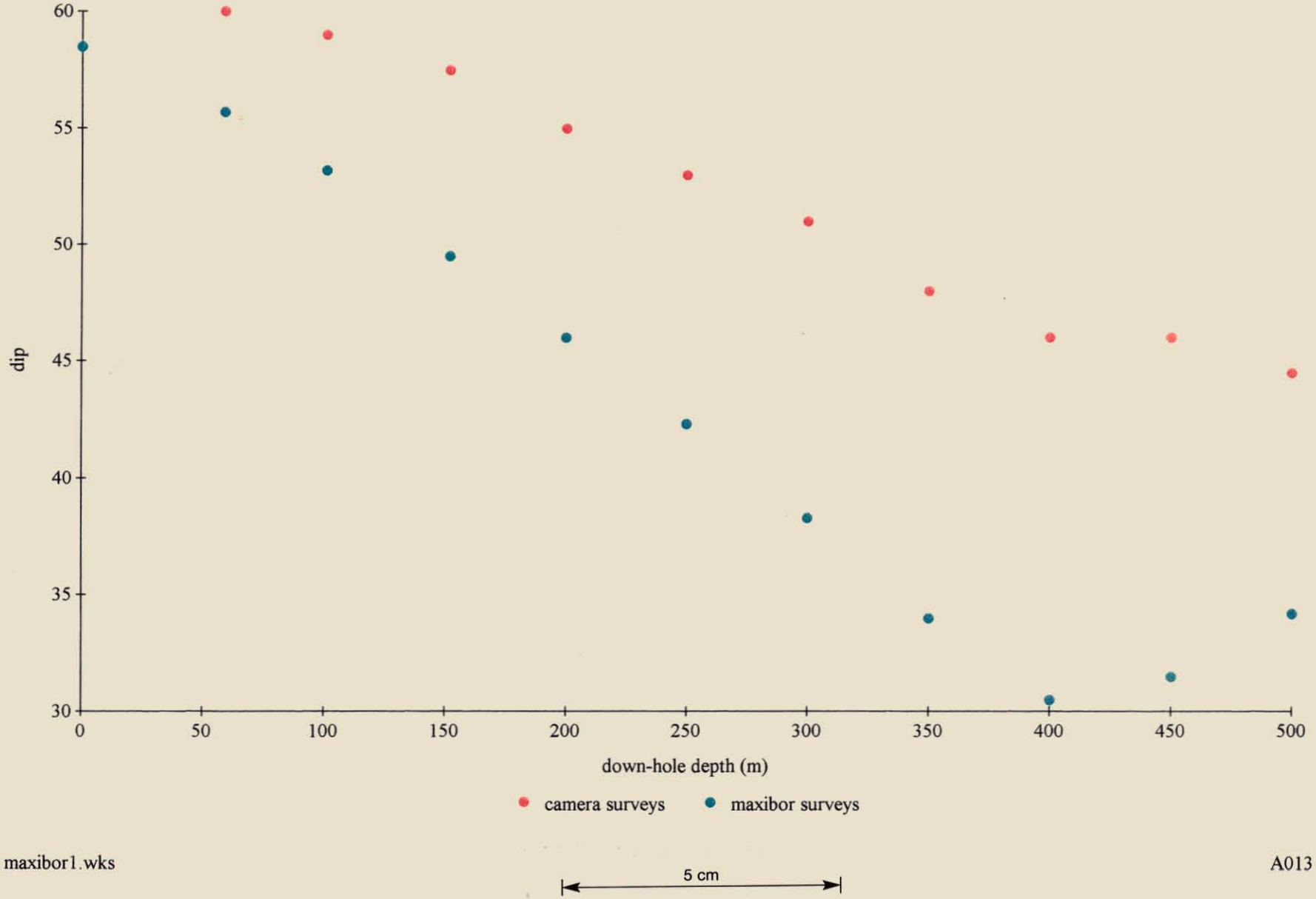
maxibor1.wks

A012

6699

A013 camera and Maxibor dips

(Maxibor operator: P)



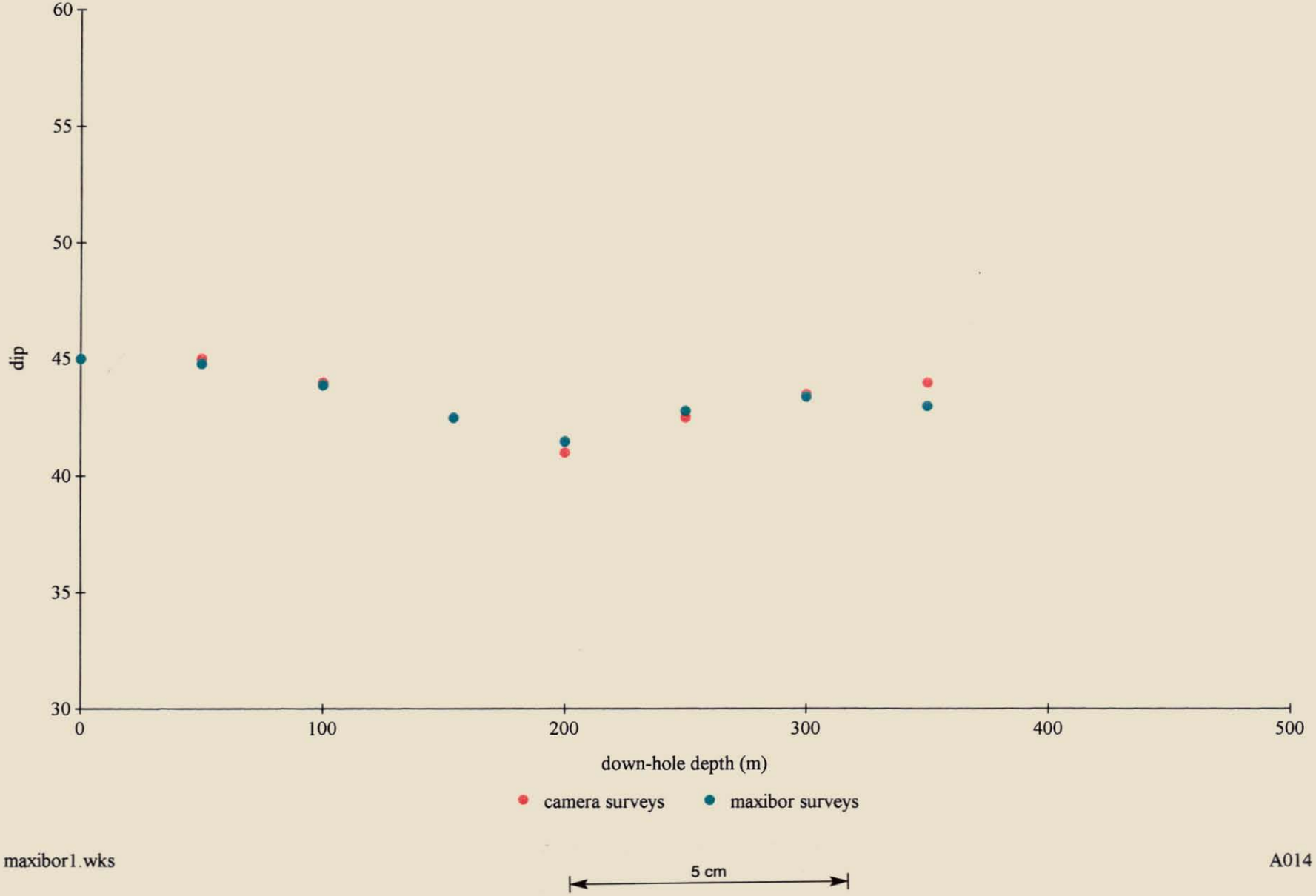
maxibor1.wks

A013

66230

A014 camera and Maxibor dips

(Maxibor operator: S)



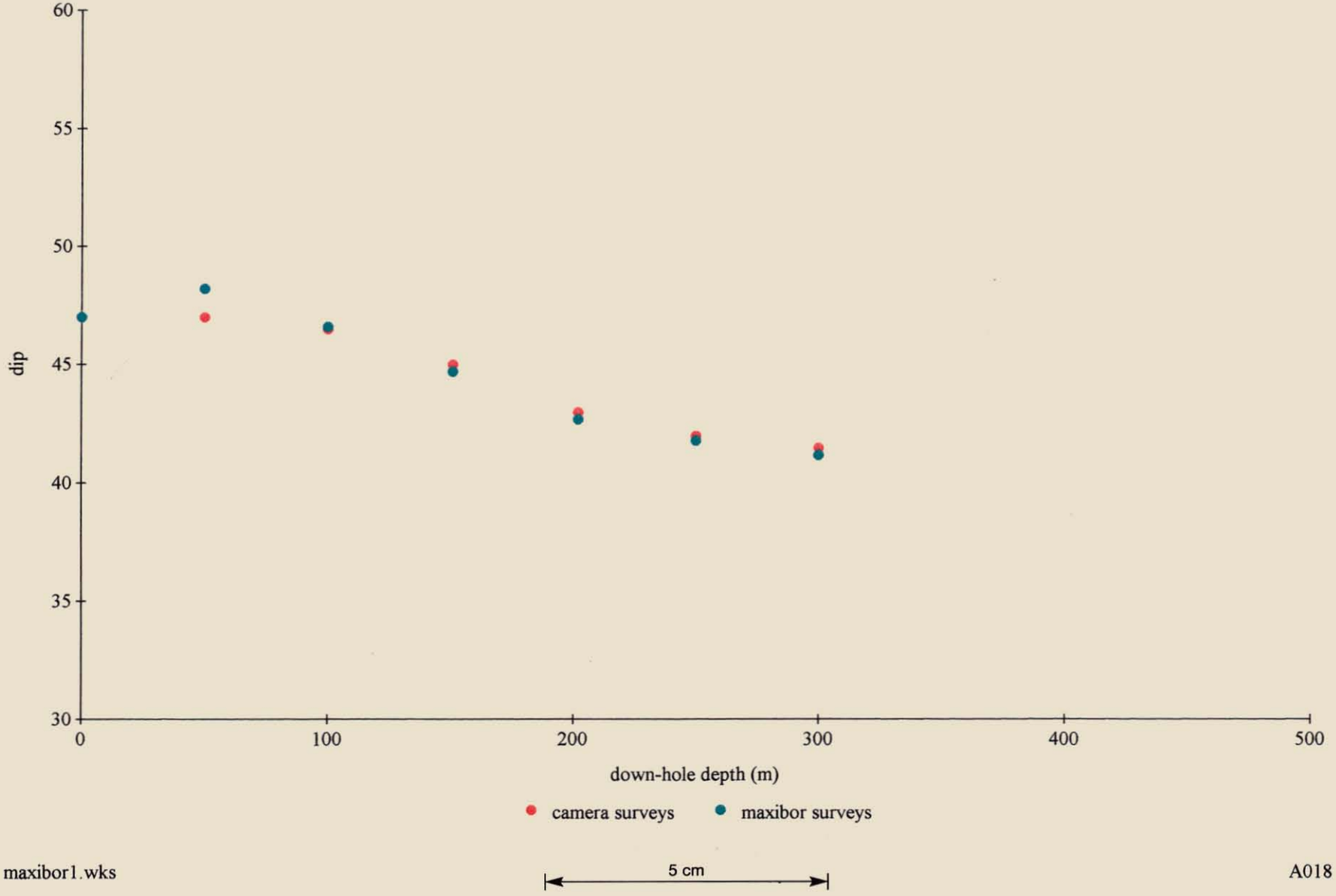
maxibor1.wks

A014

66[illegible]

A018 camera and Maxibor dips

(Maxibor operator: W)



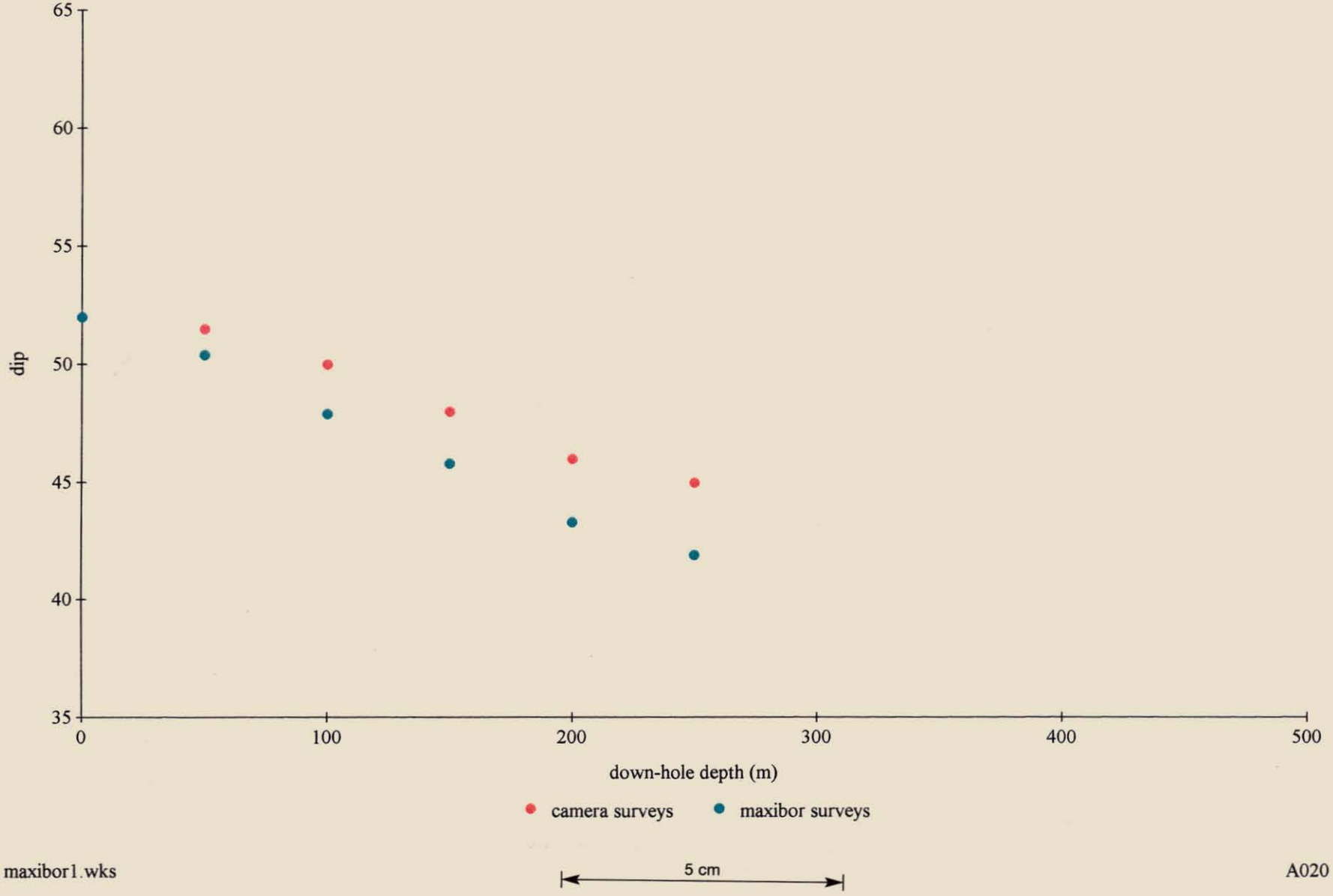
maxibor1.wks

A018

66223709

A020 camera and Maxibor dips

(Maxibor operator: W)



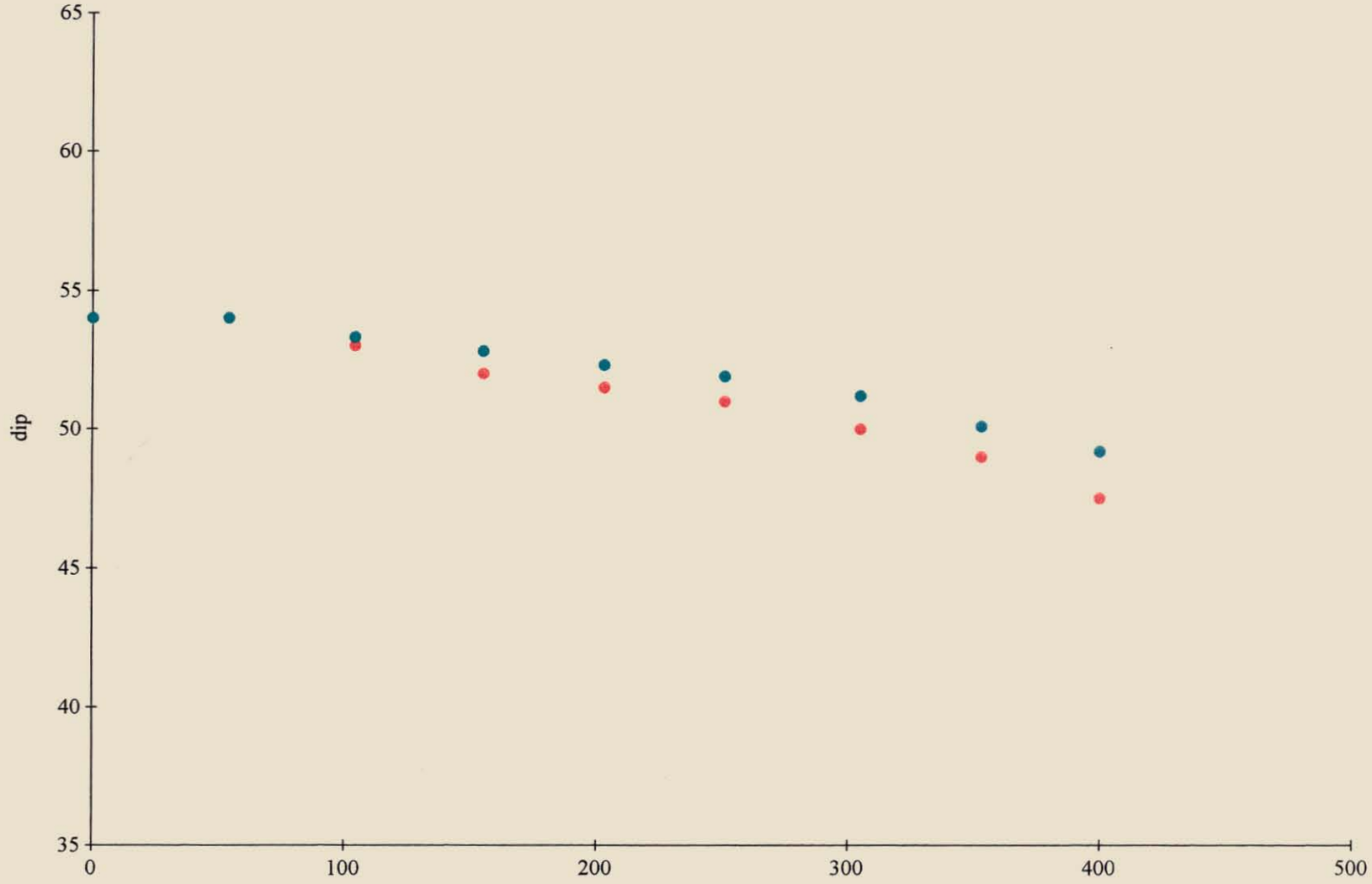
maxibor1.wks

A020

69123

A021 camera and Maxibor dips

(Maxibor operator: W)



maxibor2.wks

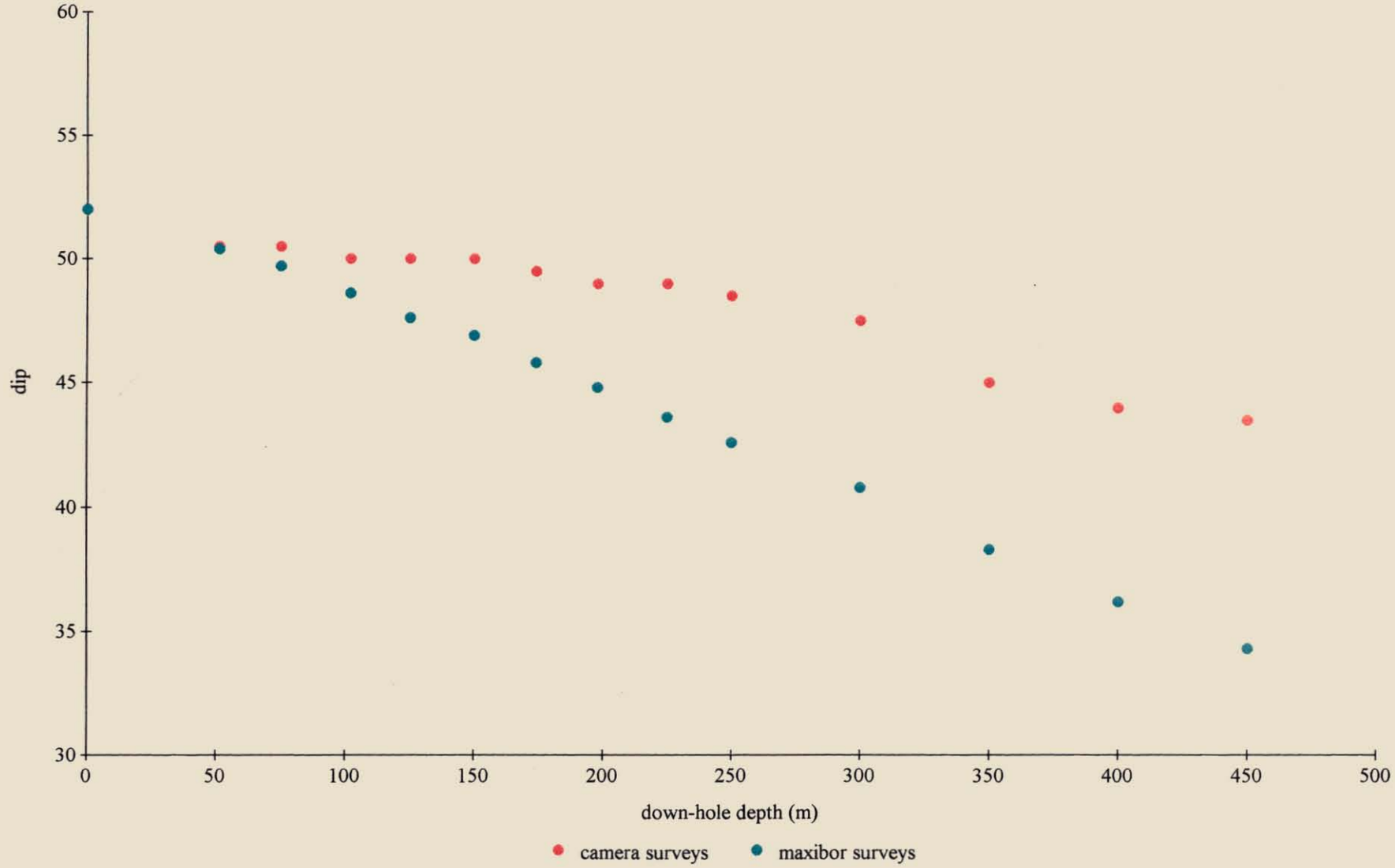
5 cm

A021

A2 2 6 9 6 1 3 4

A023 camera and Maxibor dips

(Maxibor operator: M)

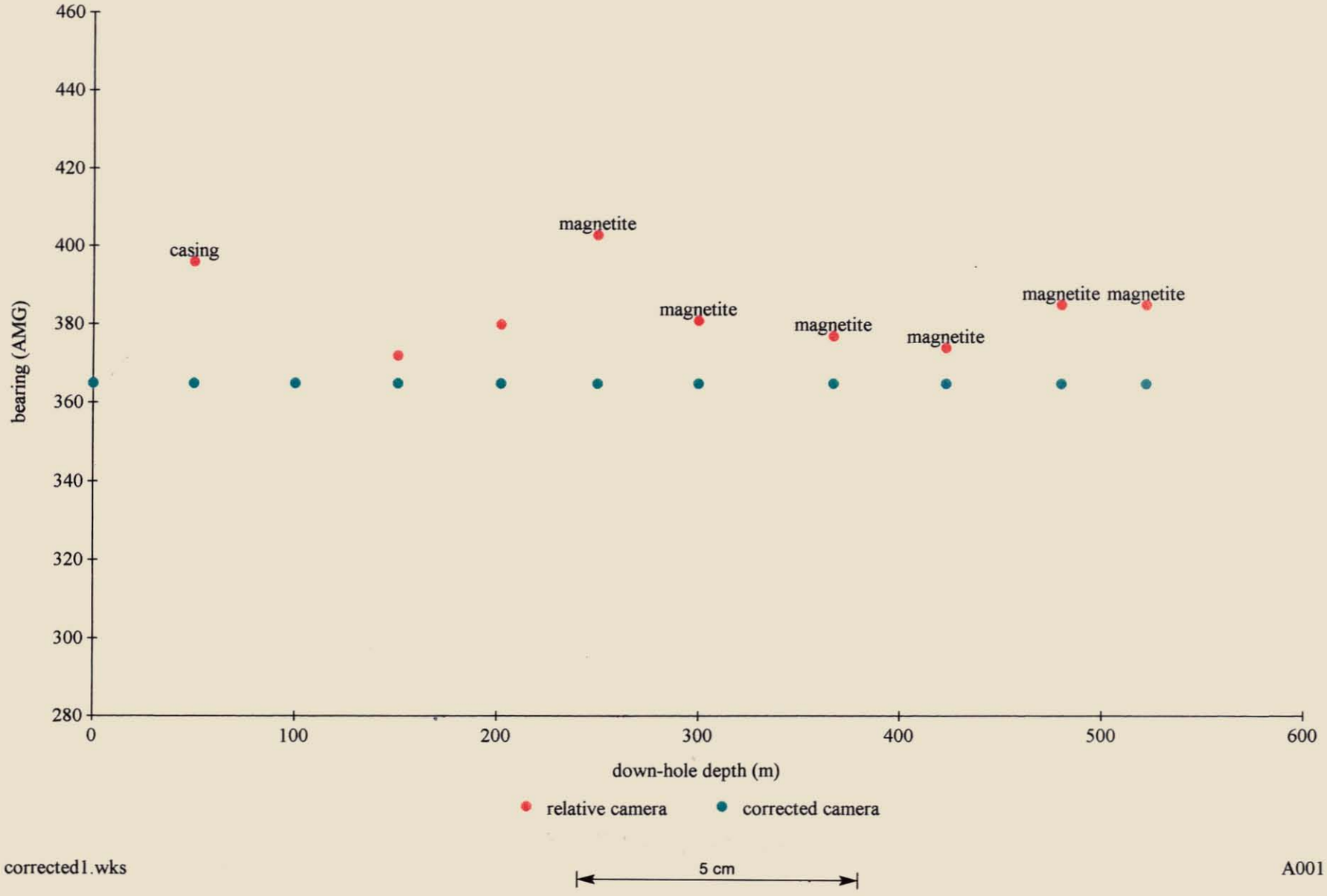


maxibor2.wks

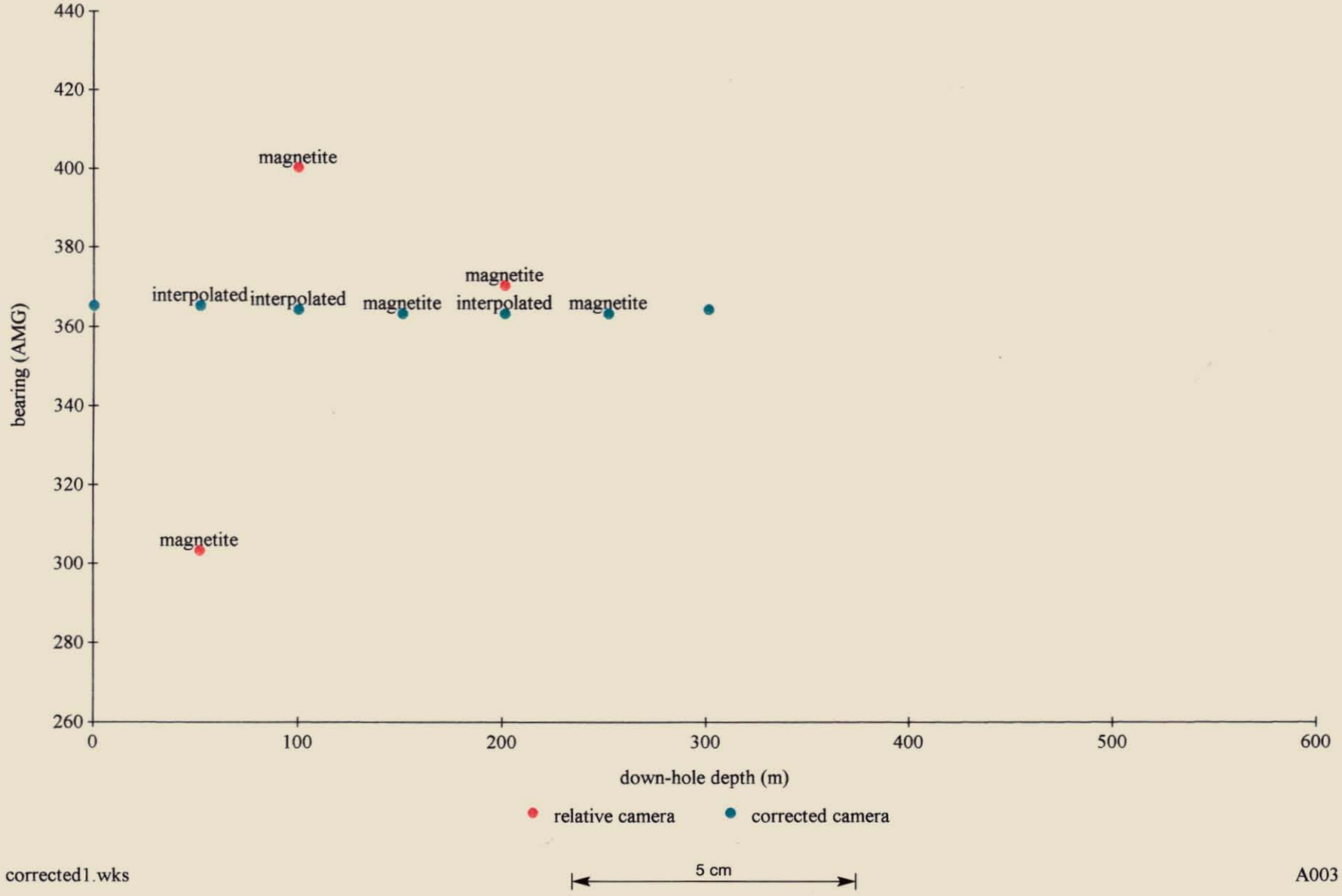
5 cm

A023

69199


A001 down-hole bearings
relative and corrected camera
460
440
420
400
380
360
340
320
300
280
bearing (AMG)
casing
magnetite
magnetite
magnetite
magnetite
magnetite
magnetite
0
100
200
300
400
500
600
down-hole depth (m)
relative camera
corrected camera
corrected1.wks
5 cm
A001

66226



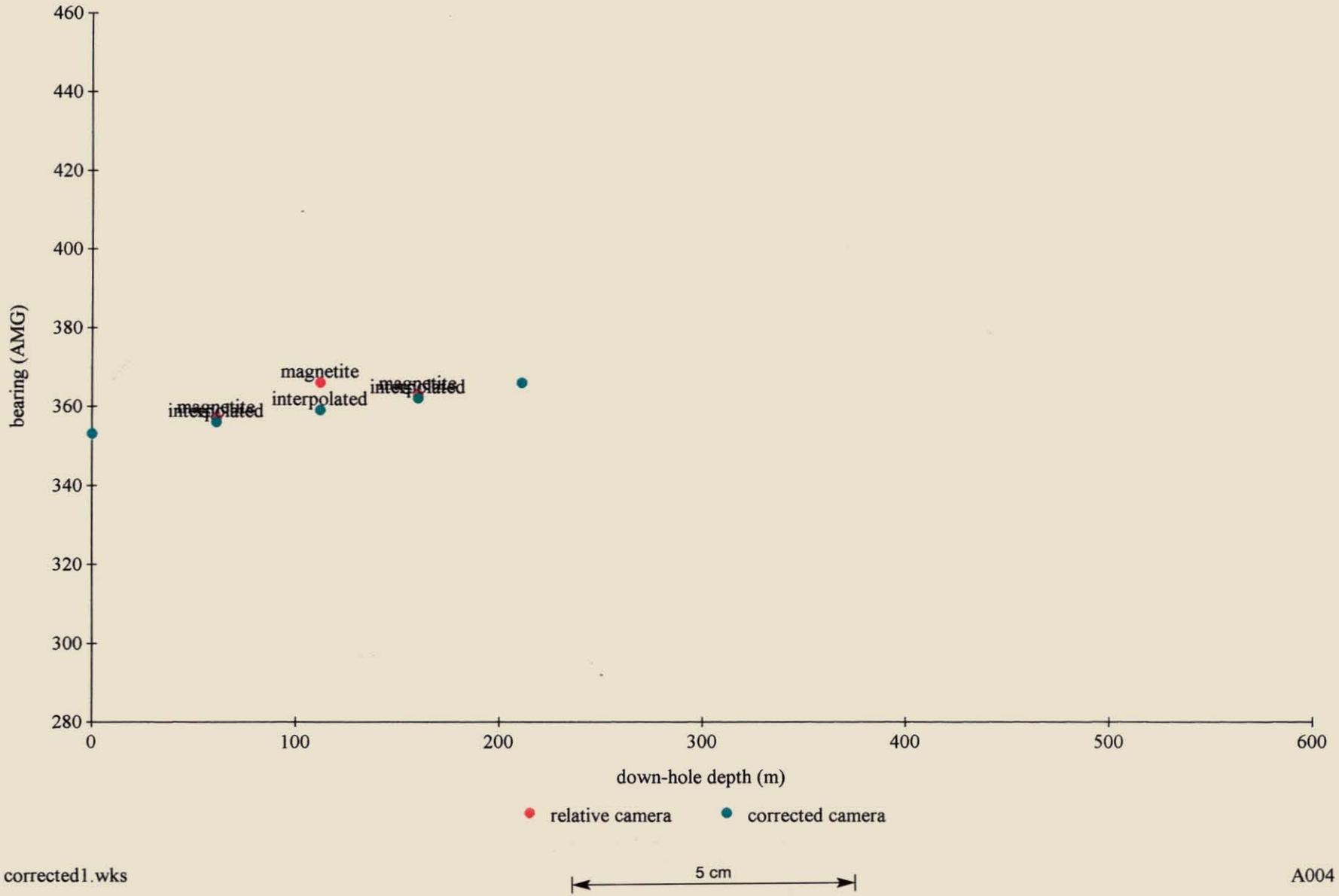
corrected1.wks

A003

69

A004 down-hole bearings

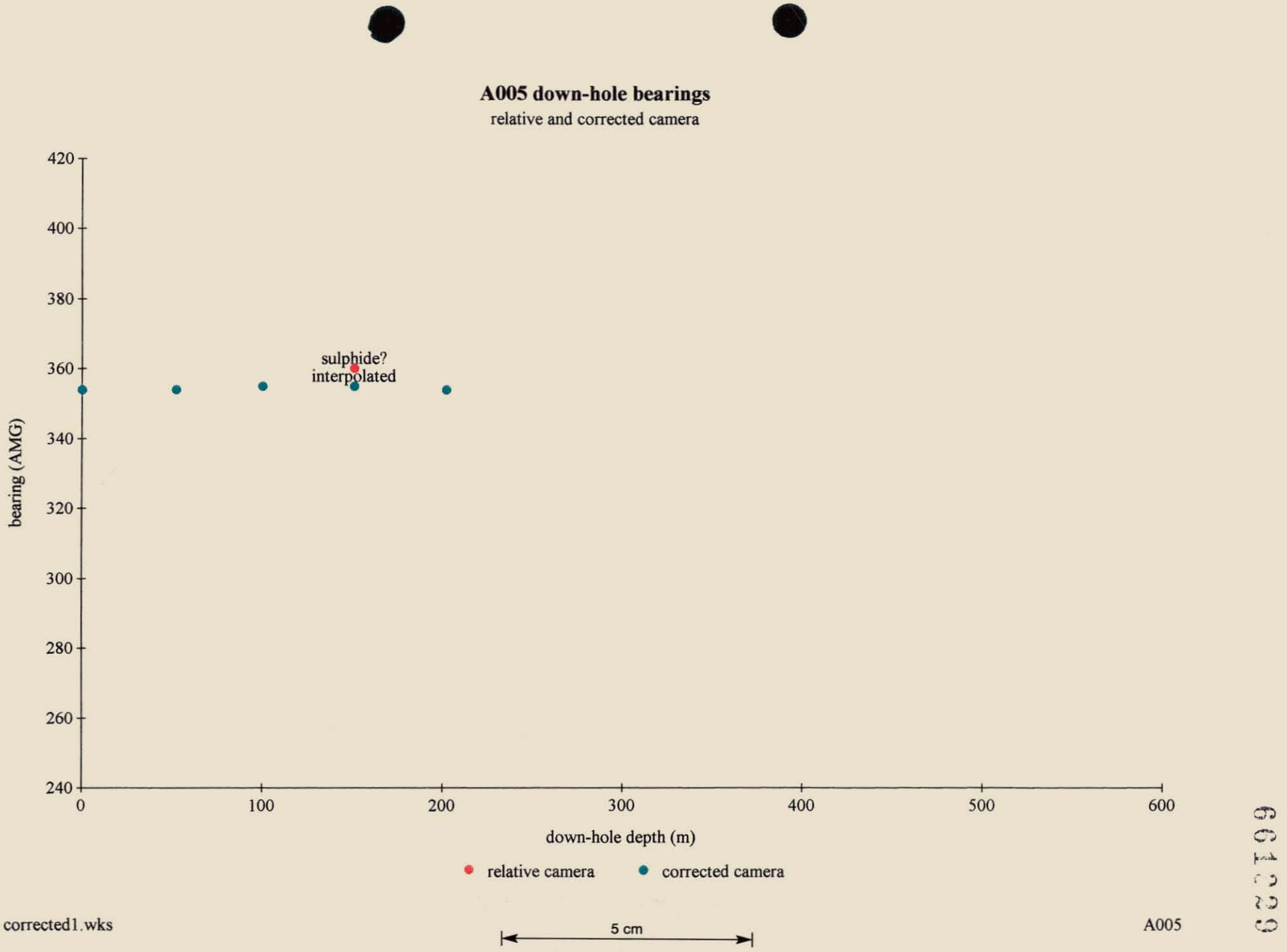
relative and corrected camera

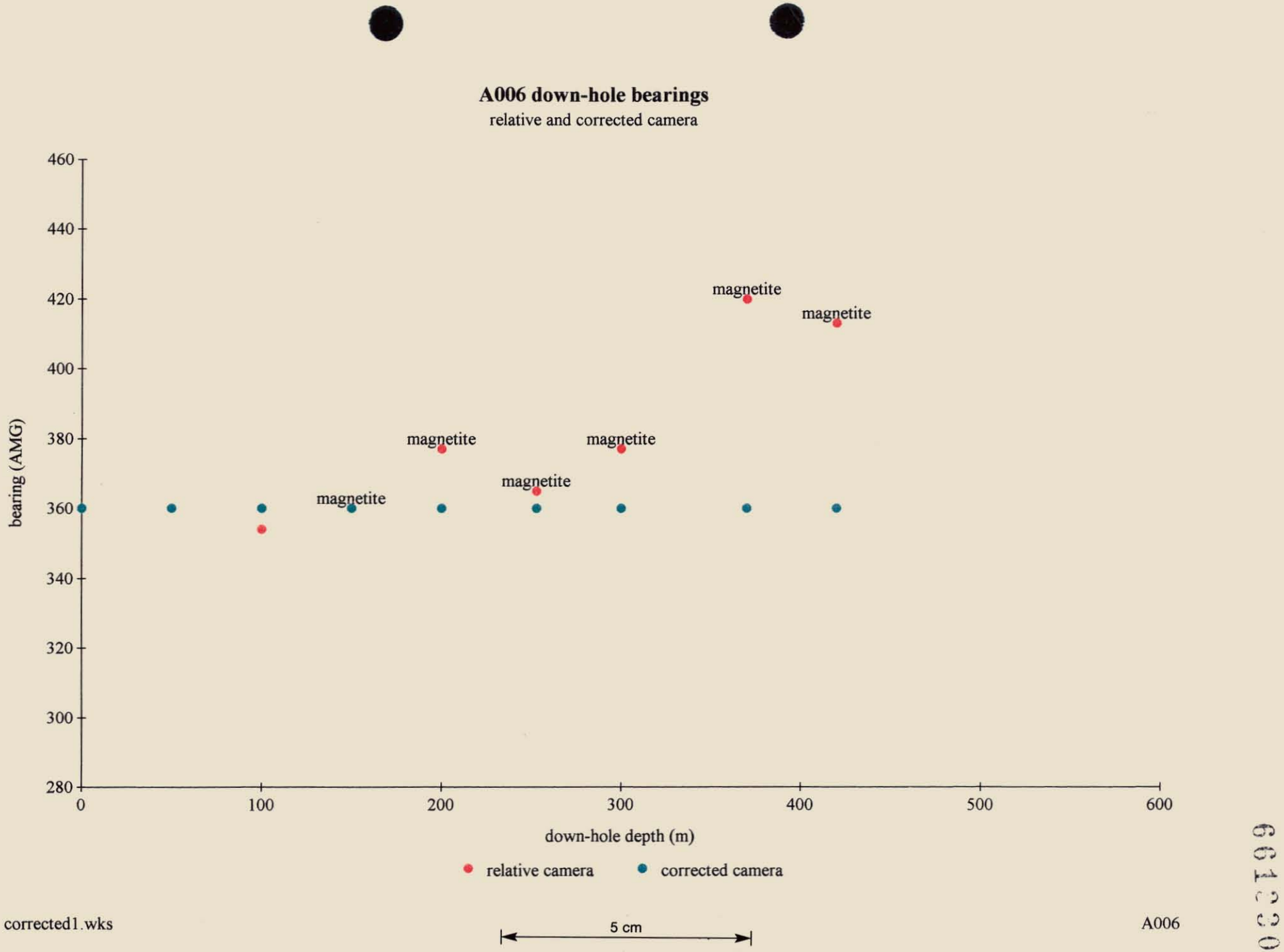


corrected1.wks

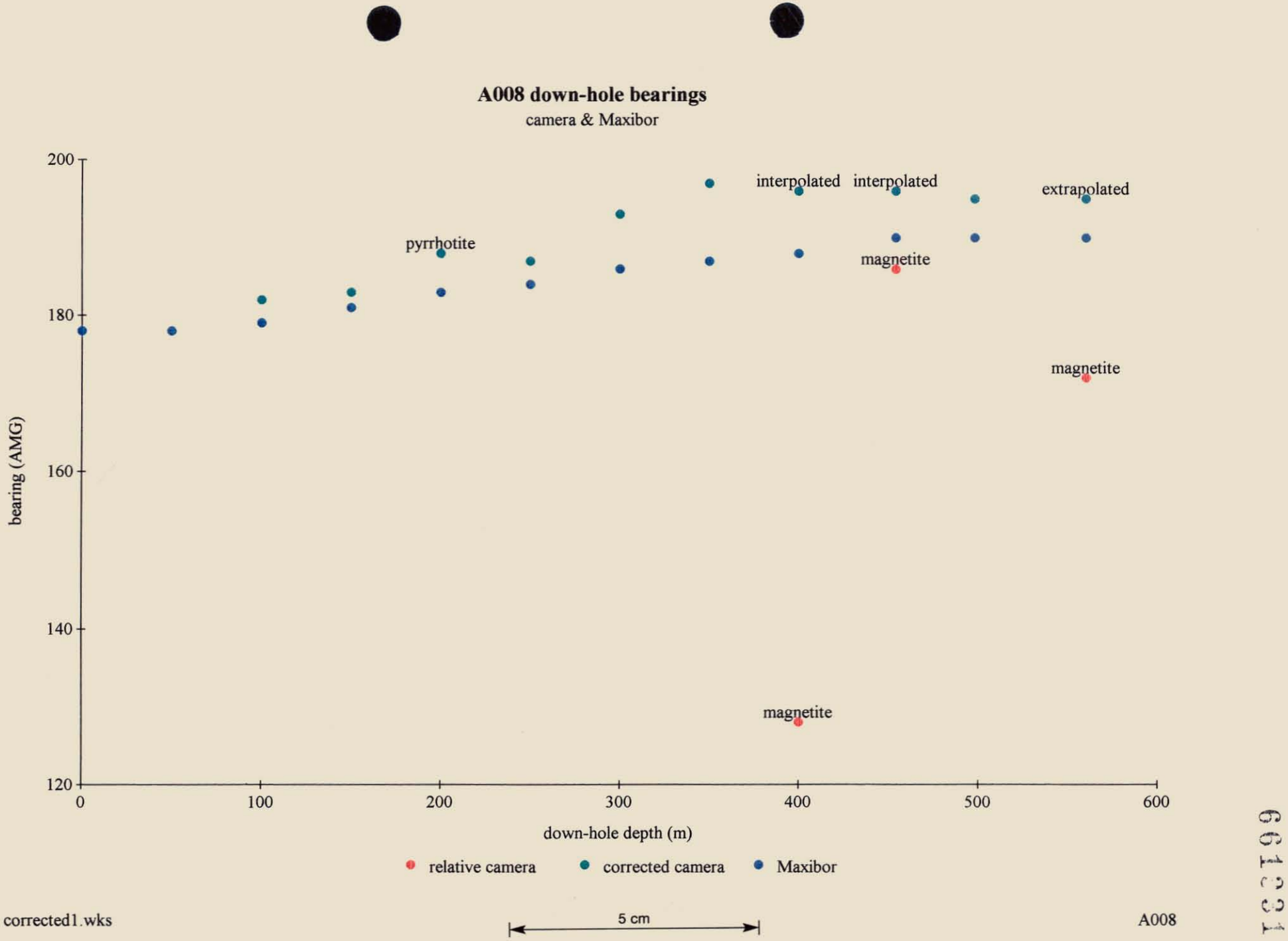
A004

69139

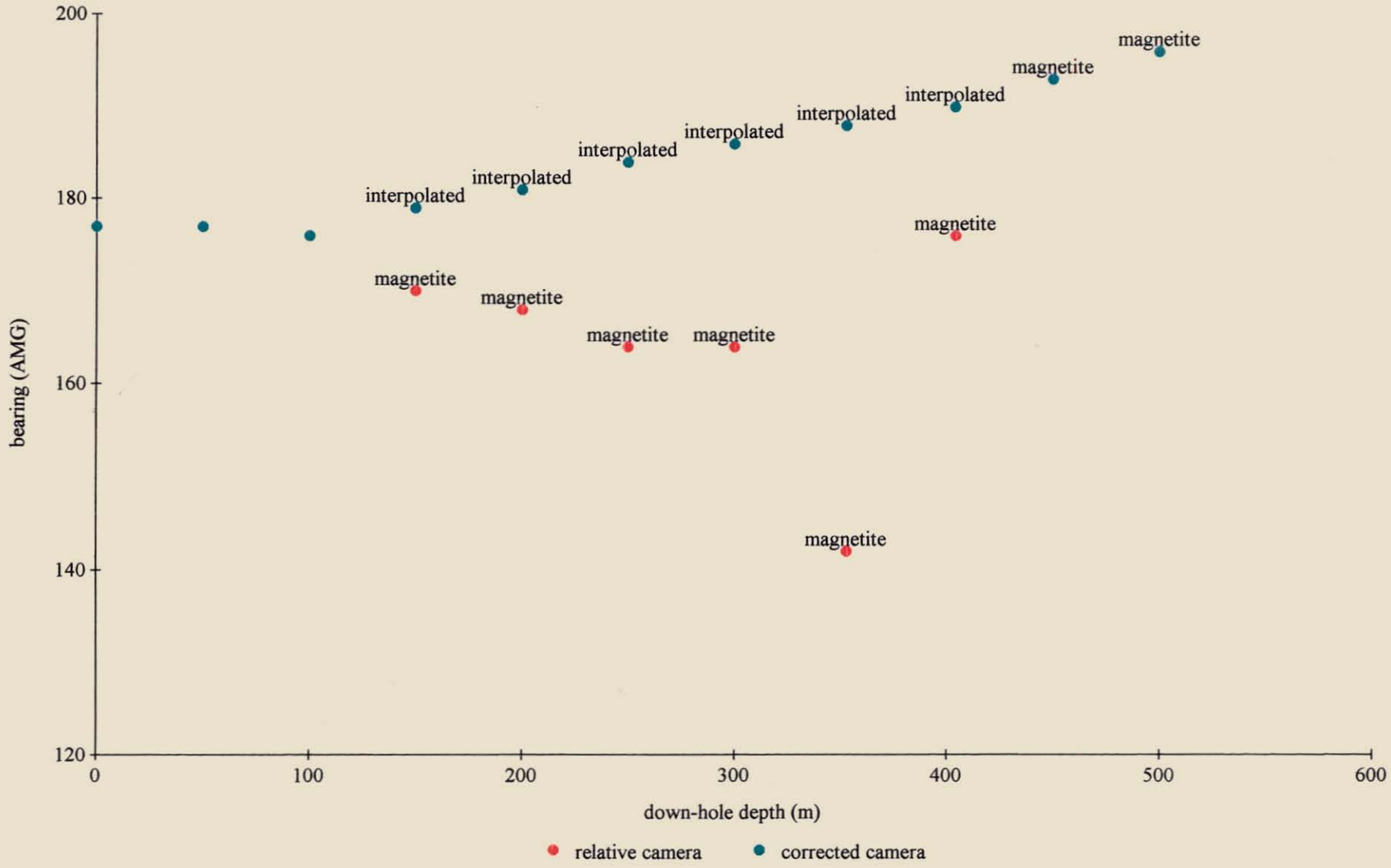

A005 down-hole bearings
relative and corrected camera
420
400
380
360
340
320
300
280
260
240
bearing (AMG)
sulphide?
interpolated
0
100
200
300
400
500
600
down-hole depth (m)
relative camera
corrected camera
corrected1.wks
5 cm
A005


A006 down-hole bearings
relative and corrected camera
bearing (AMG)
460
440
420
400
380
360
340
320
300
280
0
100
200
300
400
500
600
down-hole depth (m)
magnetite
magnetite
magnetite
magnetite
magnetite
relative camera
corrected camera
5 cm
corrected1.wks
A006

66220

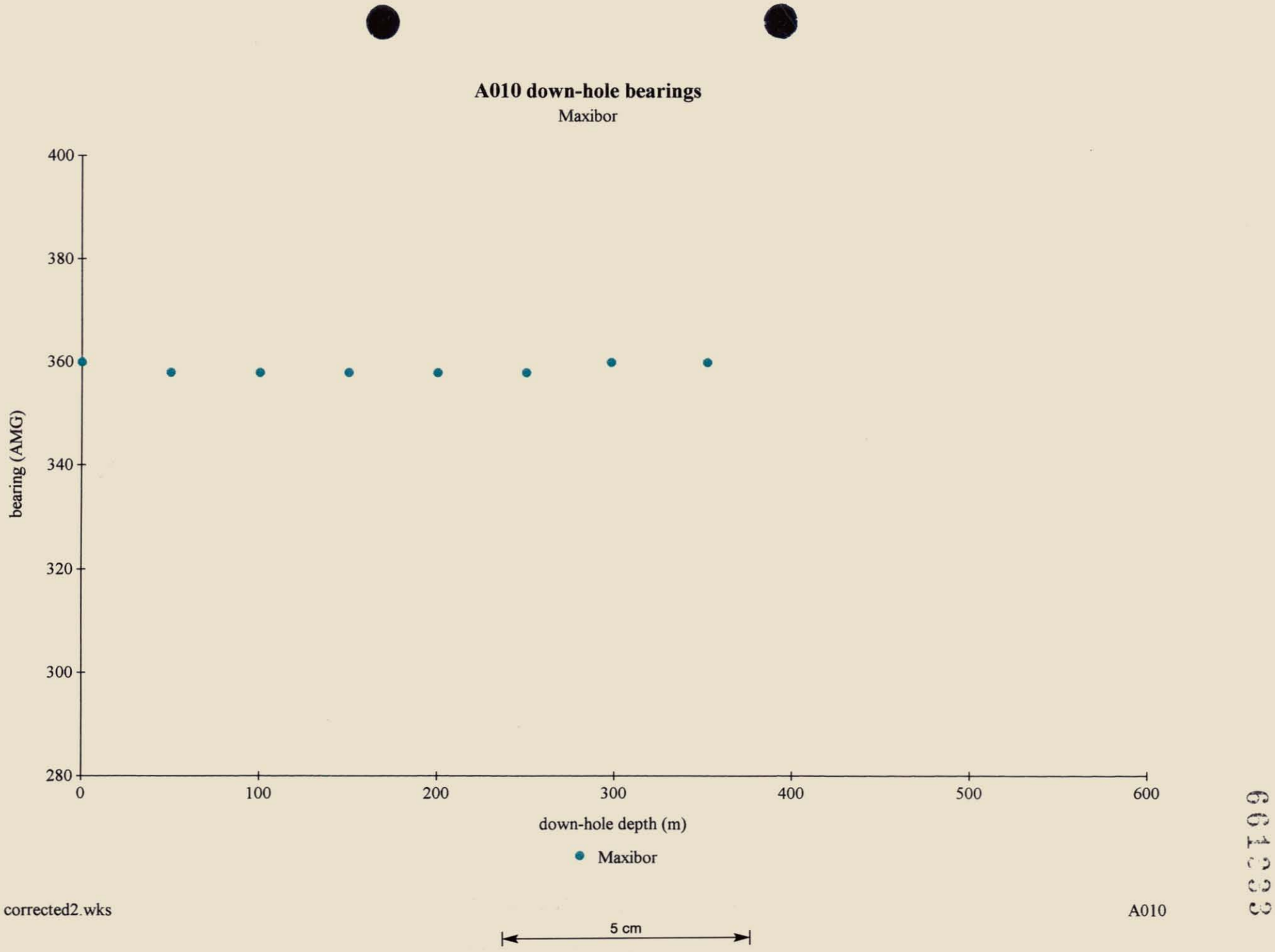

A008 down-hole bearings
camera & Maxibor
200
180
160
140
120
bearing (AMG)
0
100
200
300
400
500
600
down-hole depth (m)
interpolated
interpolated
extrapolated
pyrrhotite
magnetite
magnetite
magnetite
relative camera
corrected camera
Maxibor
corrected1.wks
5 cm
A008

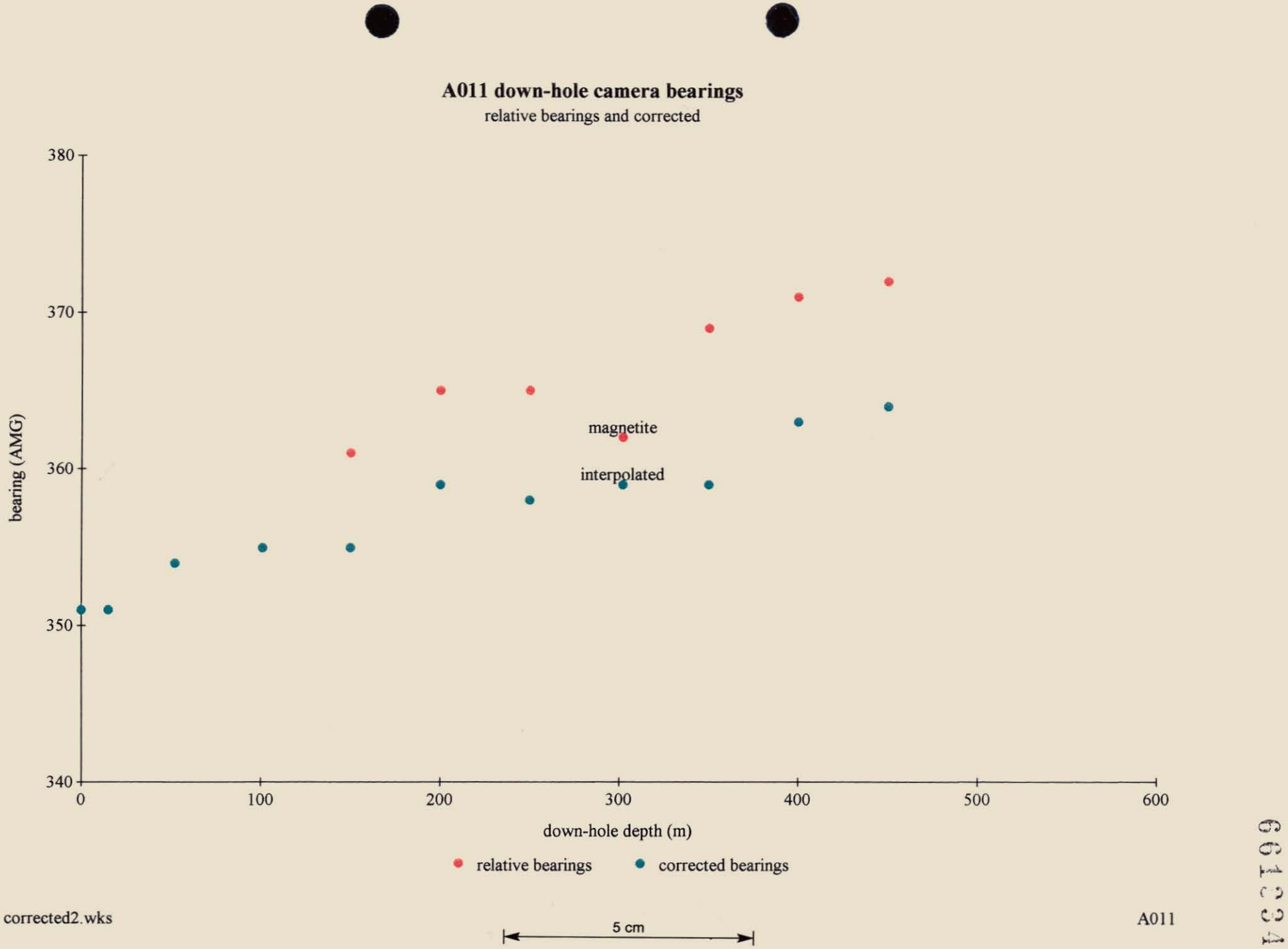
6613139

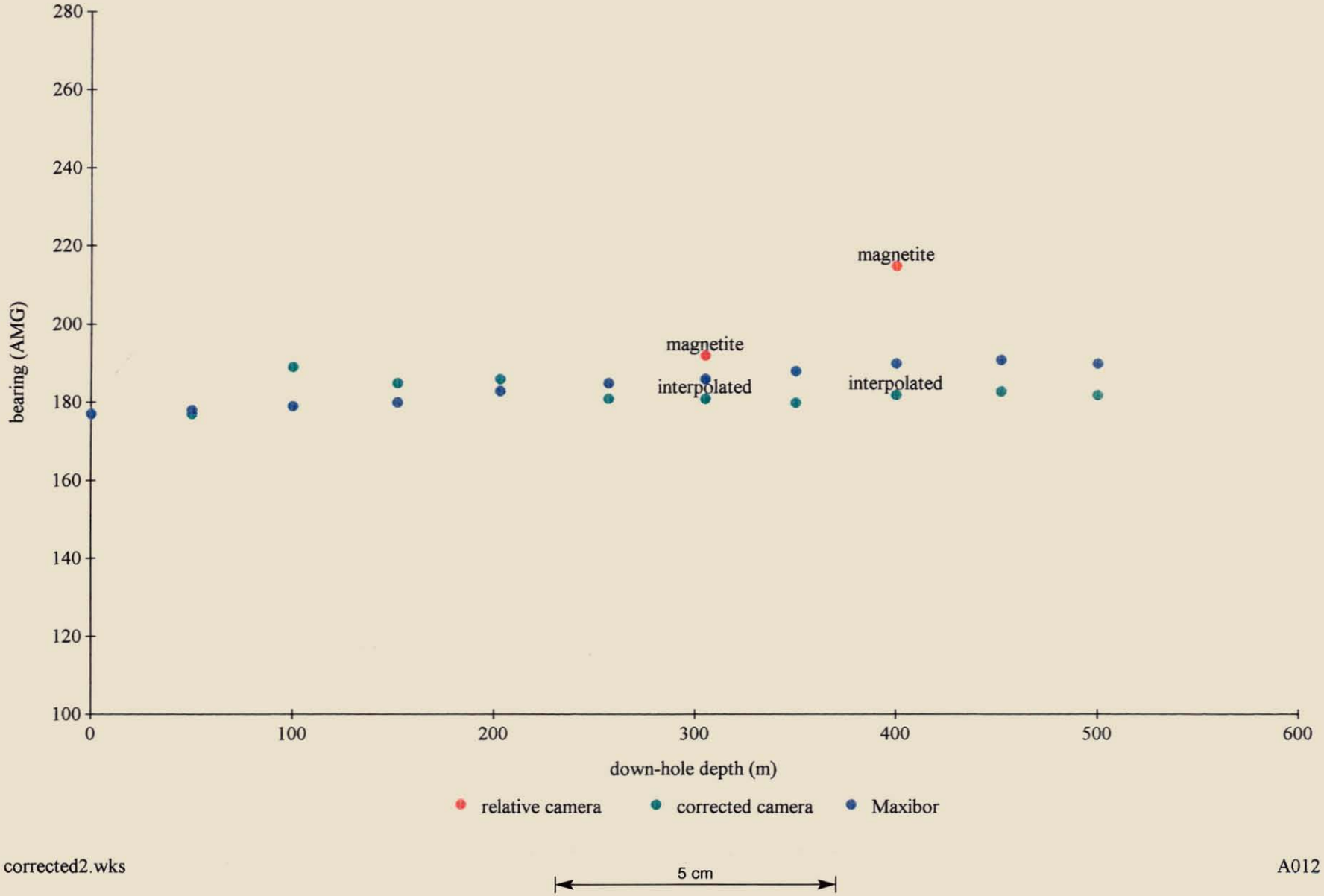


corrected2.wks

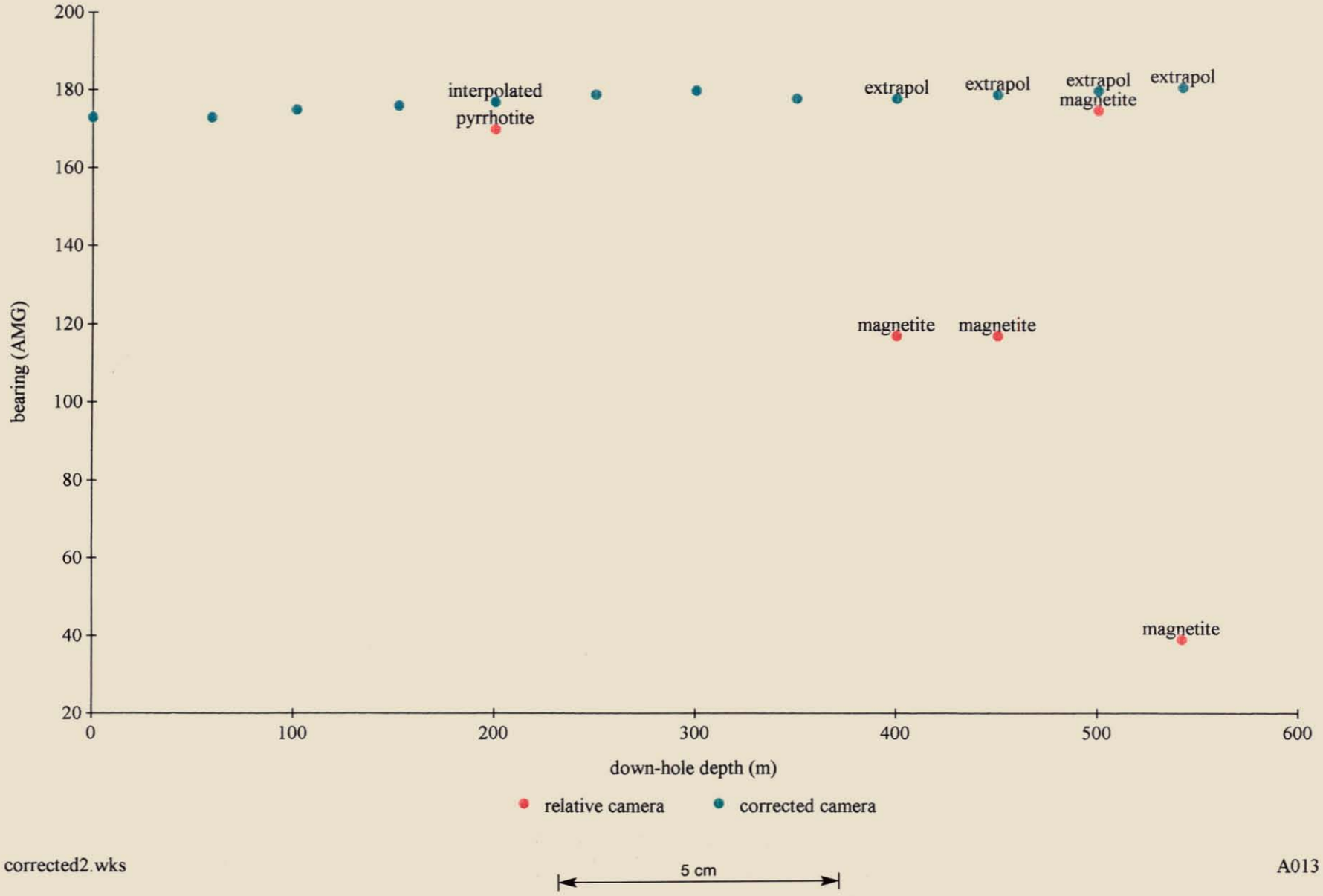
A009


A010 down-hole bearings
Maxibor
400
380
360
340
320
300
280
bearing (AMG)
0
100
200
300
400
500
600
down-hole depth (m)
Maxibor
corrected2.wks
A010
5 cm


A011 down-hole camera bearings
relative bearings and corrected
380
370
360
350
340
bearing (AMG)
0
100
200
300
400
500
600
down-hole depth (m)
magnetite
interpolated
relative bearings
corrected bearings
corrected2.wks
5 cm
A011


A012 down-hole bearings
Camera and Maxibor
bearing (AMG)
280
260
240
220
200
180
160
140
120
100
0
100
200
300
400
500
600
down-hole depth (m)
magnetite
magnetite
interpolated
interpolated
relative camera
corrected camera
Maxibor
corrected2.wks
5 cm
A012

669239



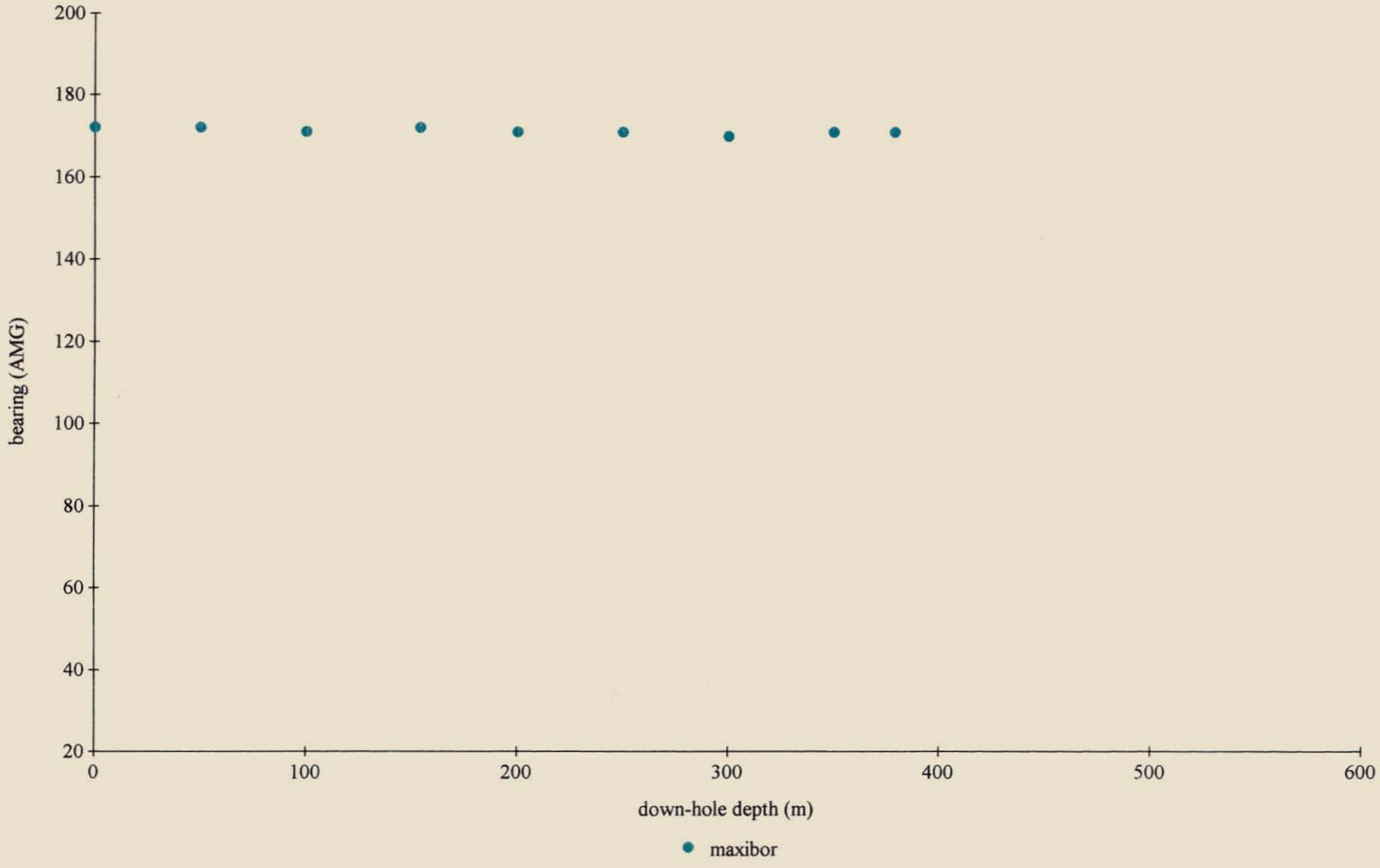
corrected2.wks

A013

66236

A014 down-hole bearings

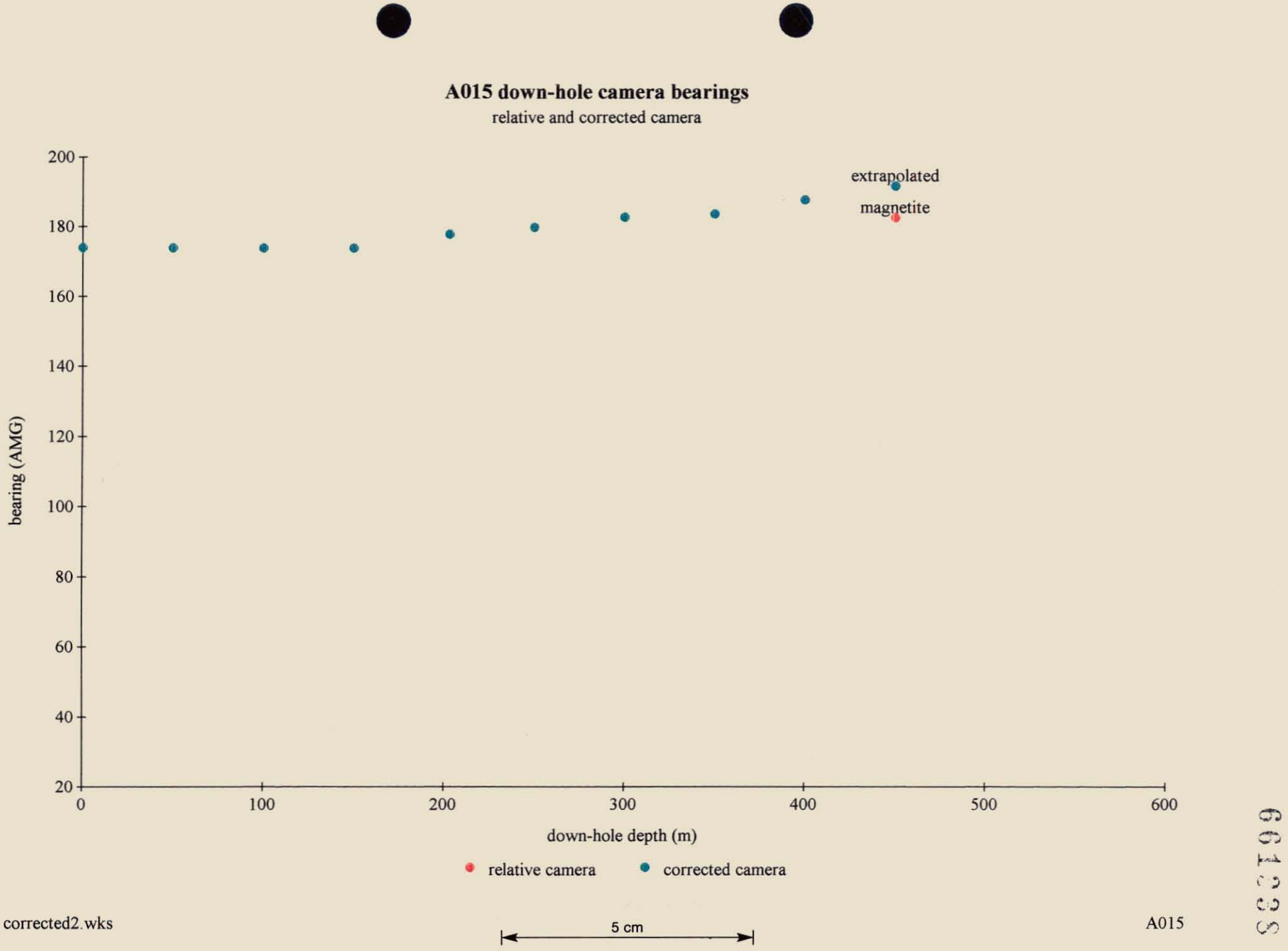
Maxibor



corrected2.wks

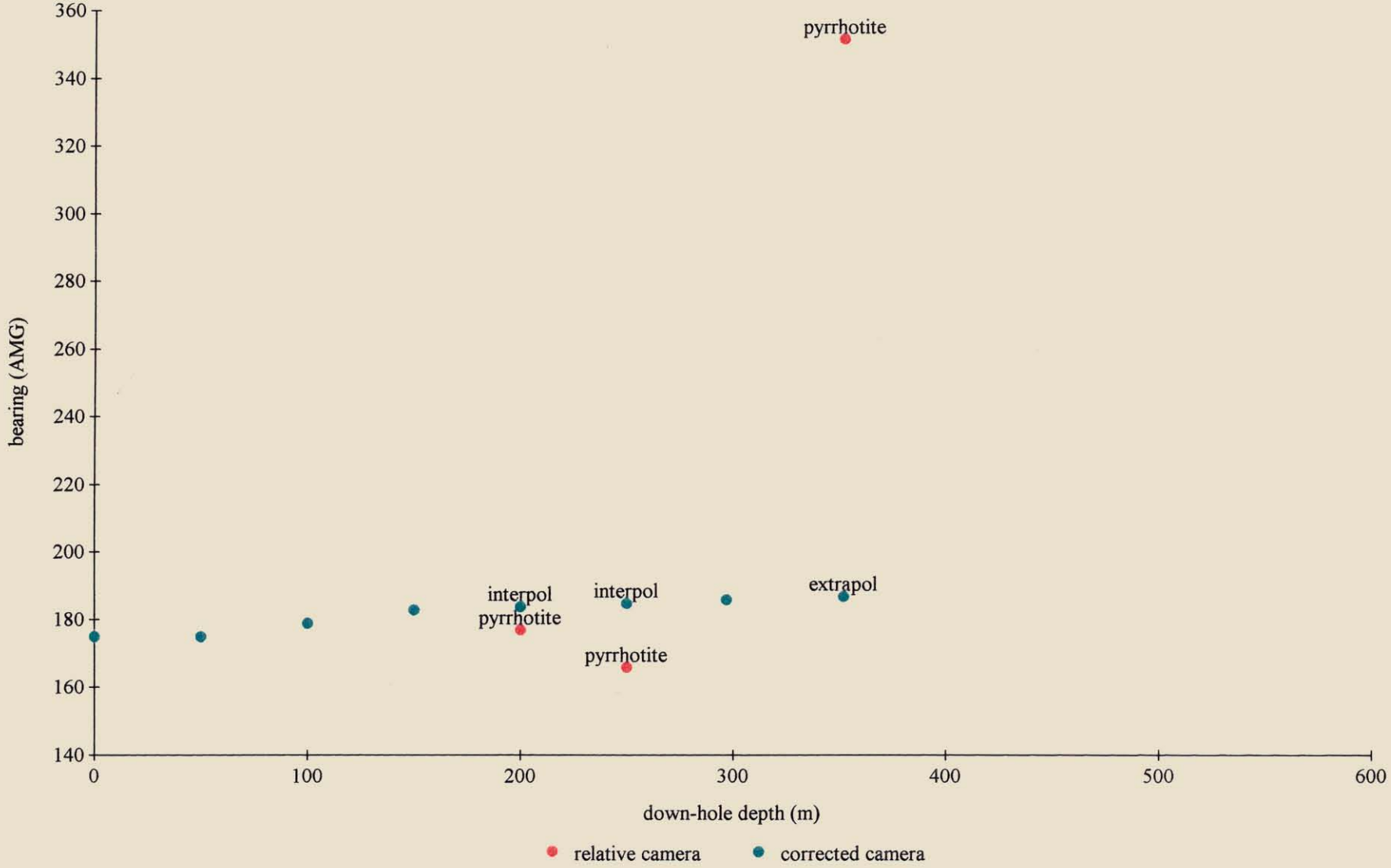
A014

66139


A015 down-hole camera bearings
relative and corrected camera
200
180
160
140
120
100
80
60
40
20
bearing (AMG)
extrapolated
magnetite
0
100
200
300
400
500
600
down-hole depth (m)
relative camera
corrected camera
corrected2.wks
5 cm
A015

A016 down-hole camera bearings

relative and corrected camera



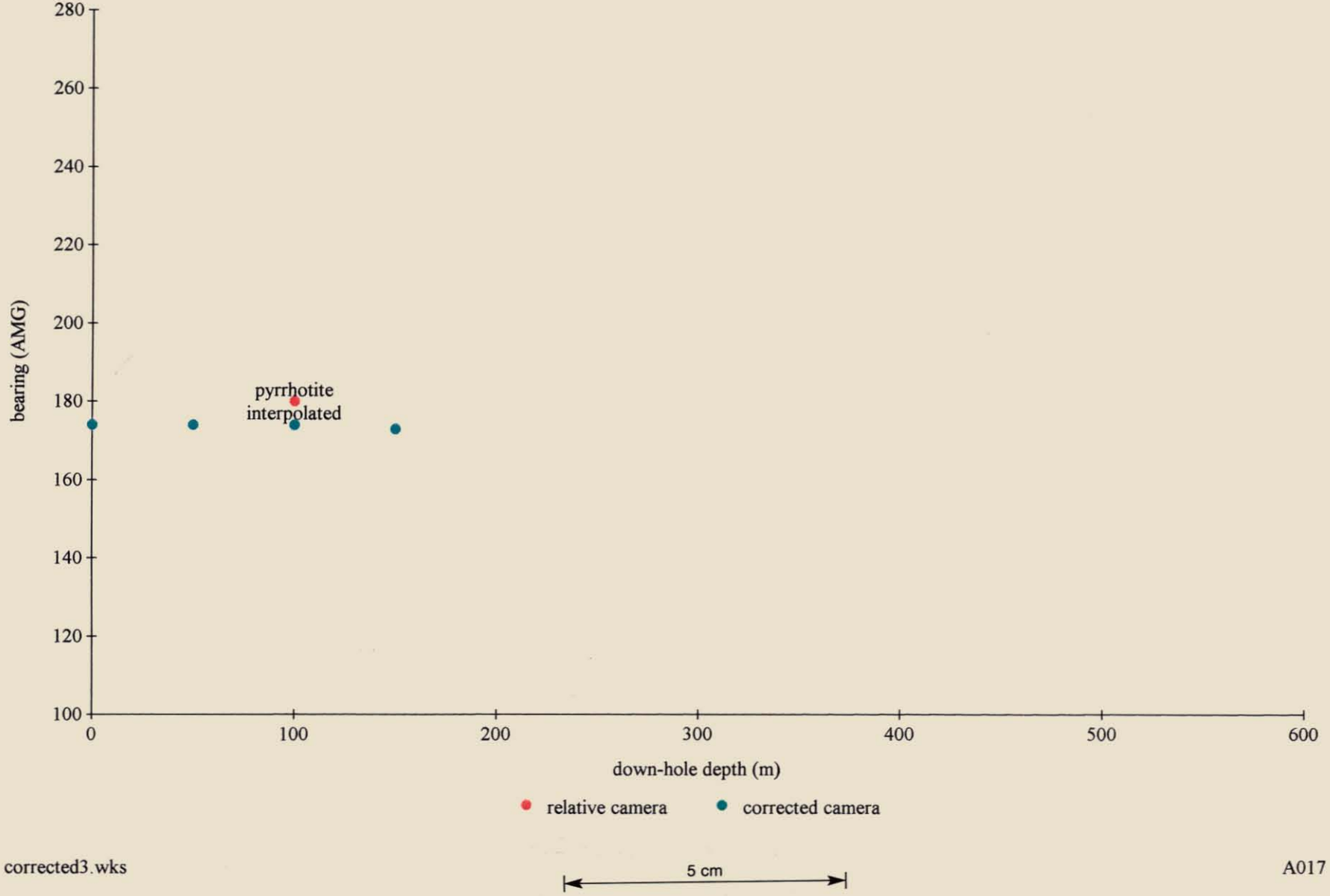
corrected2.wks

A016

69 2 69

A017 down-hole camera bearings

relative and corrected camera



corrected3.wks

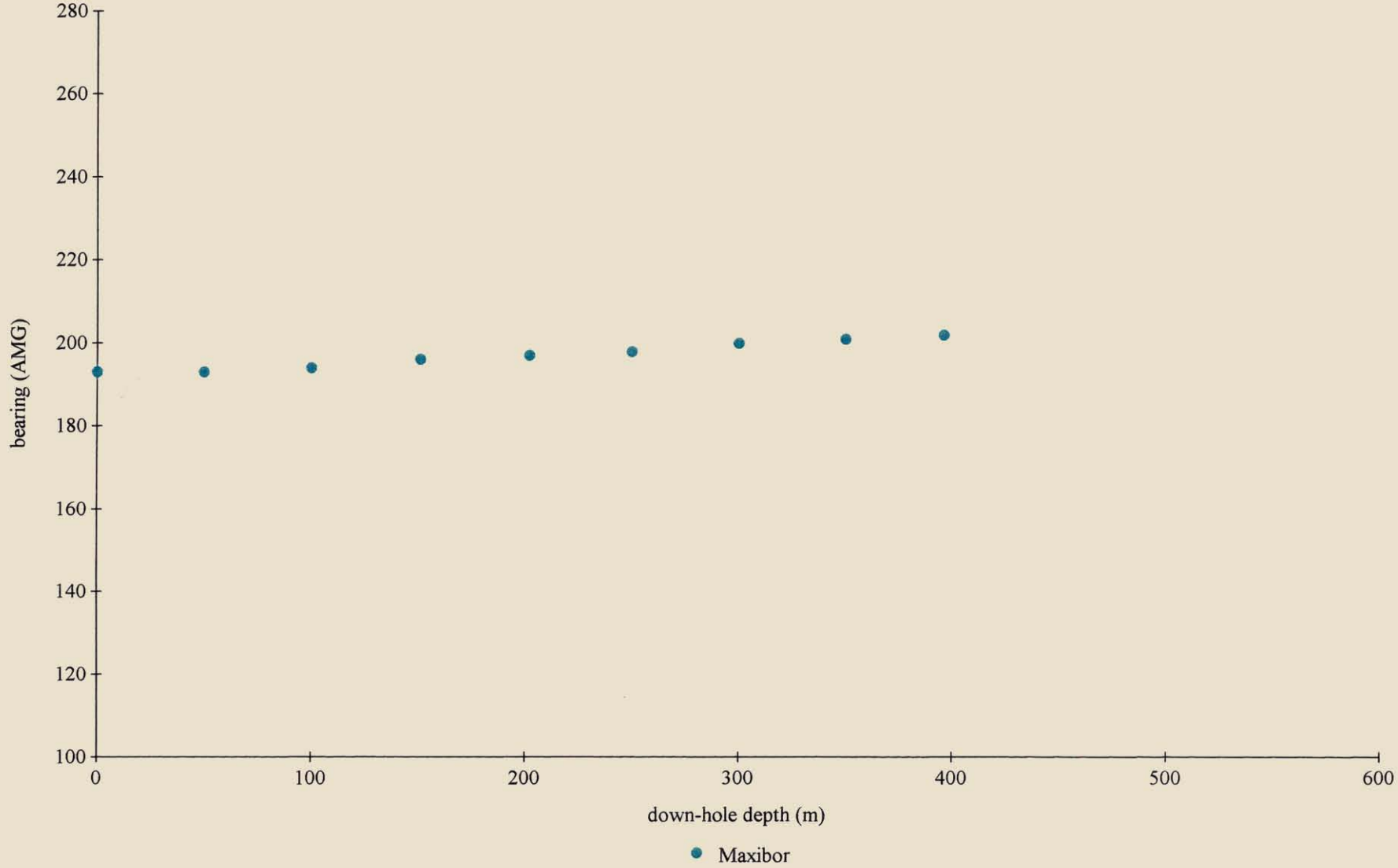
5 cm

A017

0 6913 9

A018 down-hole bearings

Maxibor



corrected3.wks

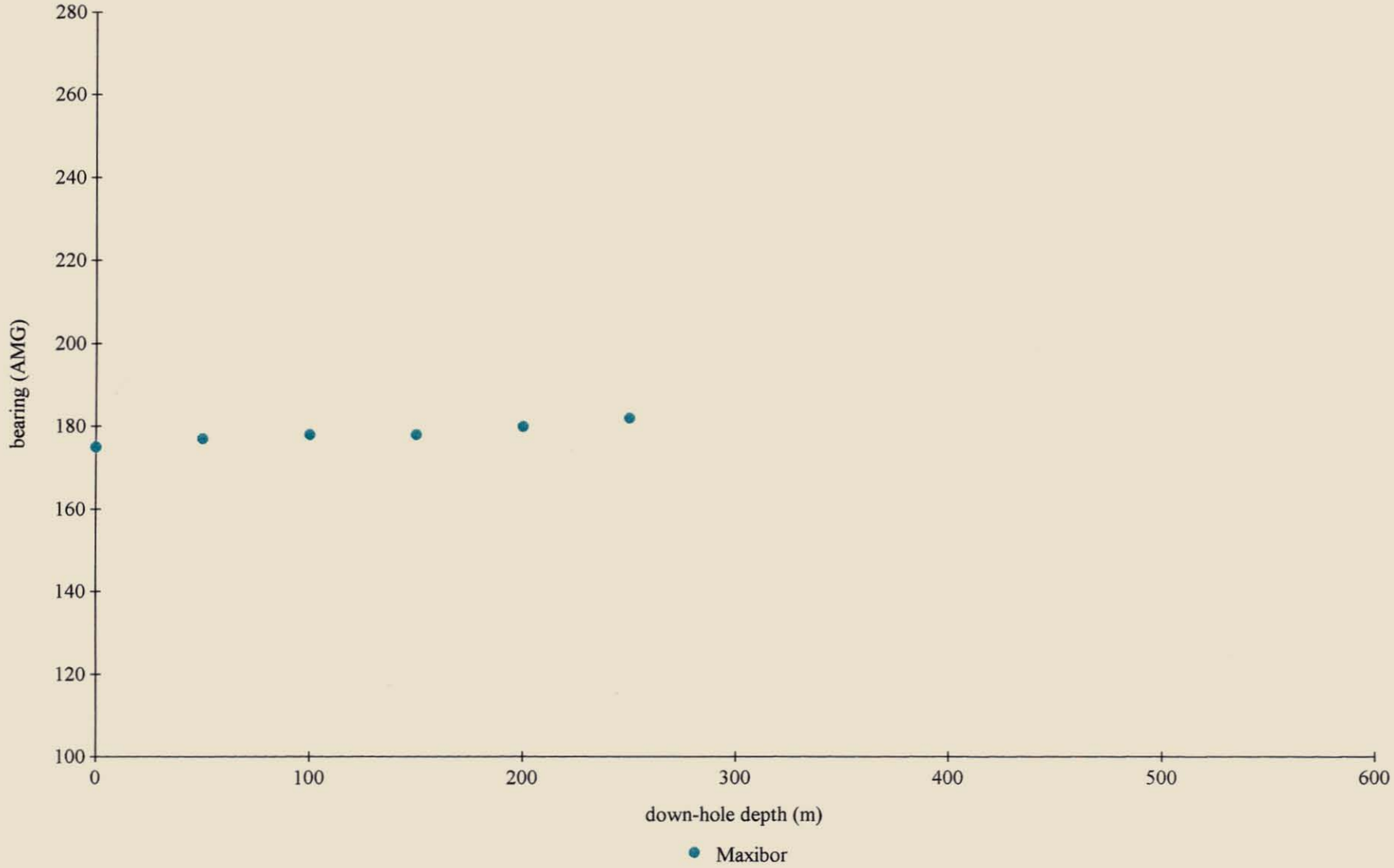
5 cm

A018

1173769

A019 down-hole bearings

Maxibor



corrected3.wks

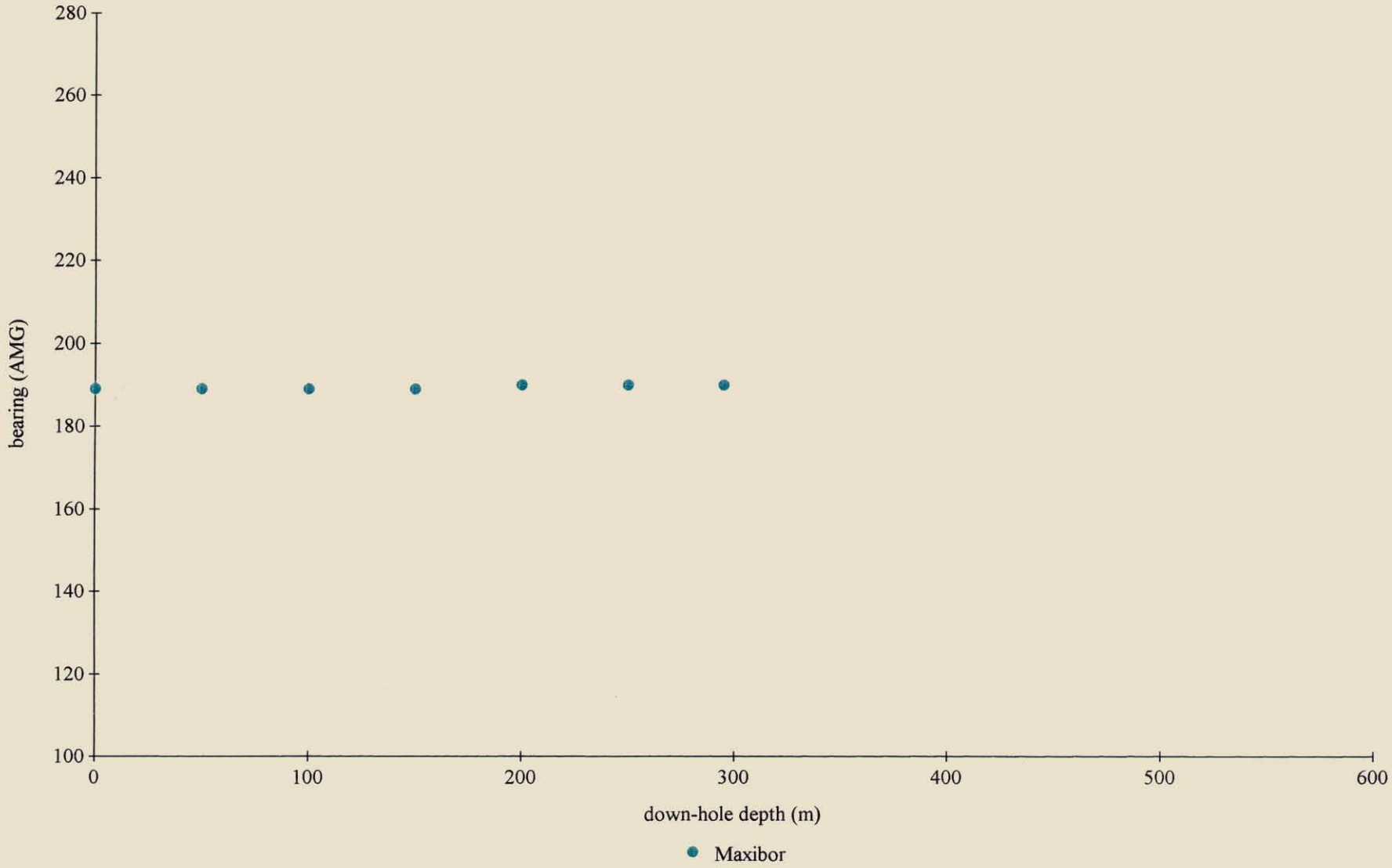
5 cm

A019

6612769

A020 down-hole bearings

Maxibor

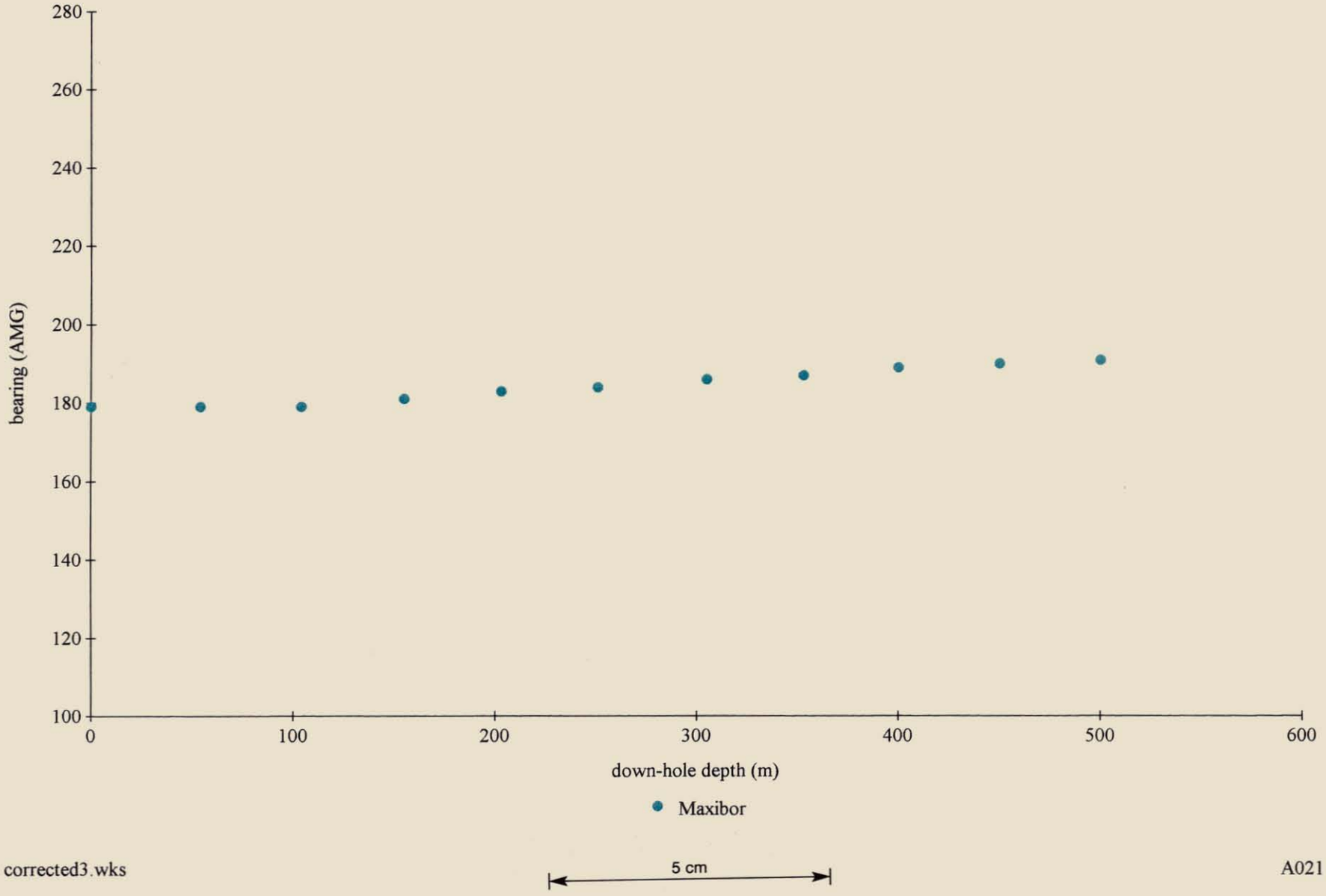


corrected3.wks

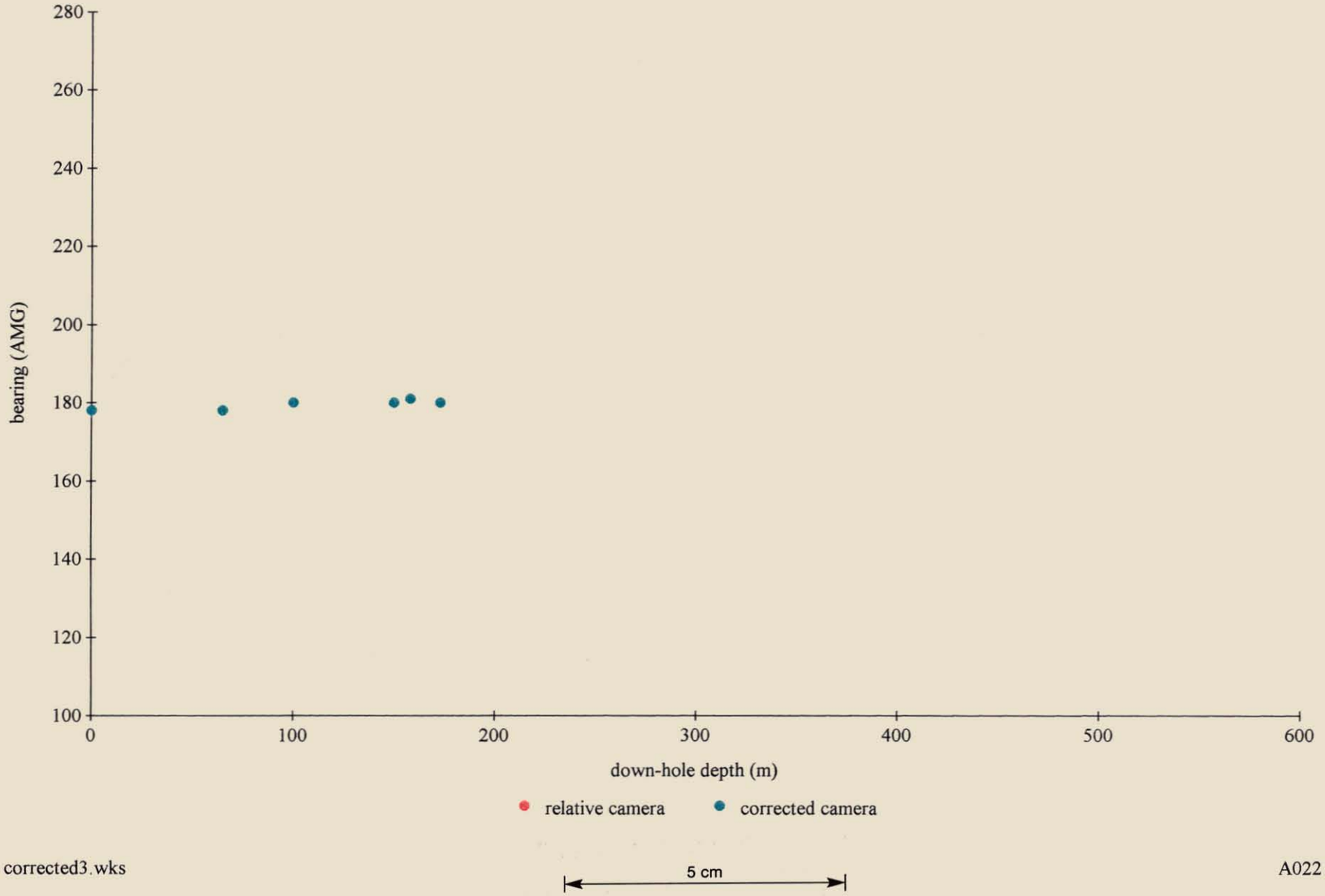
5 cm

A020

CL 6199


A021 down-hole bearings
Maxibor
bearing (AMG)
280
260
240
220
200
180
160
140
120
100
0
100
200
300
400
500
600
down-hole depth (m)
Maxibor
corrected3.wks
5 cm
A021

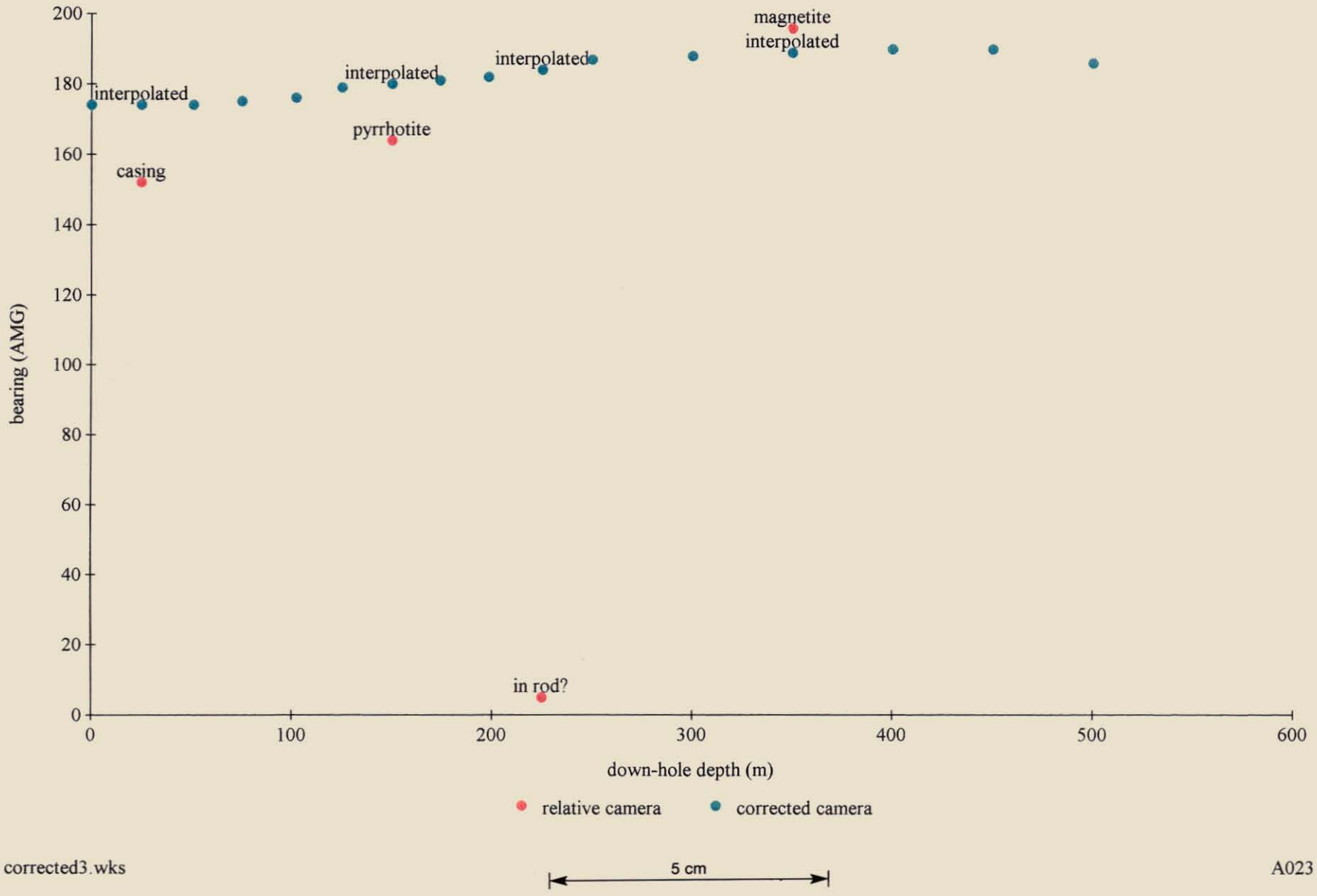
69 F 3 4



corrected3.wks

A022

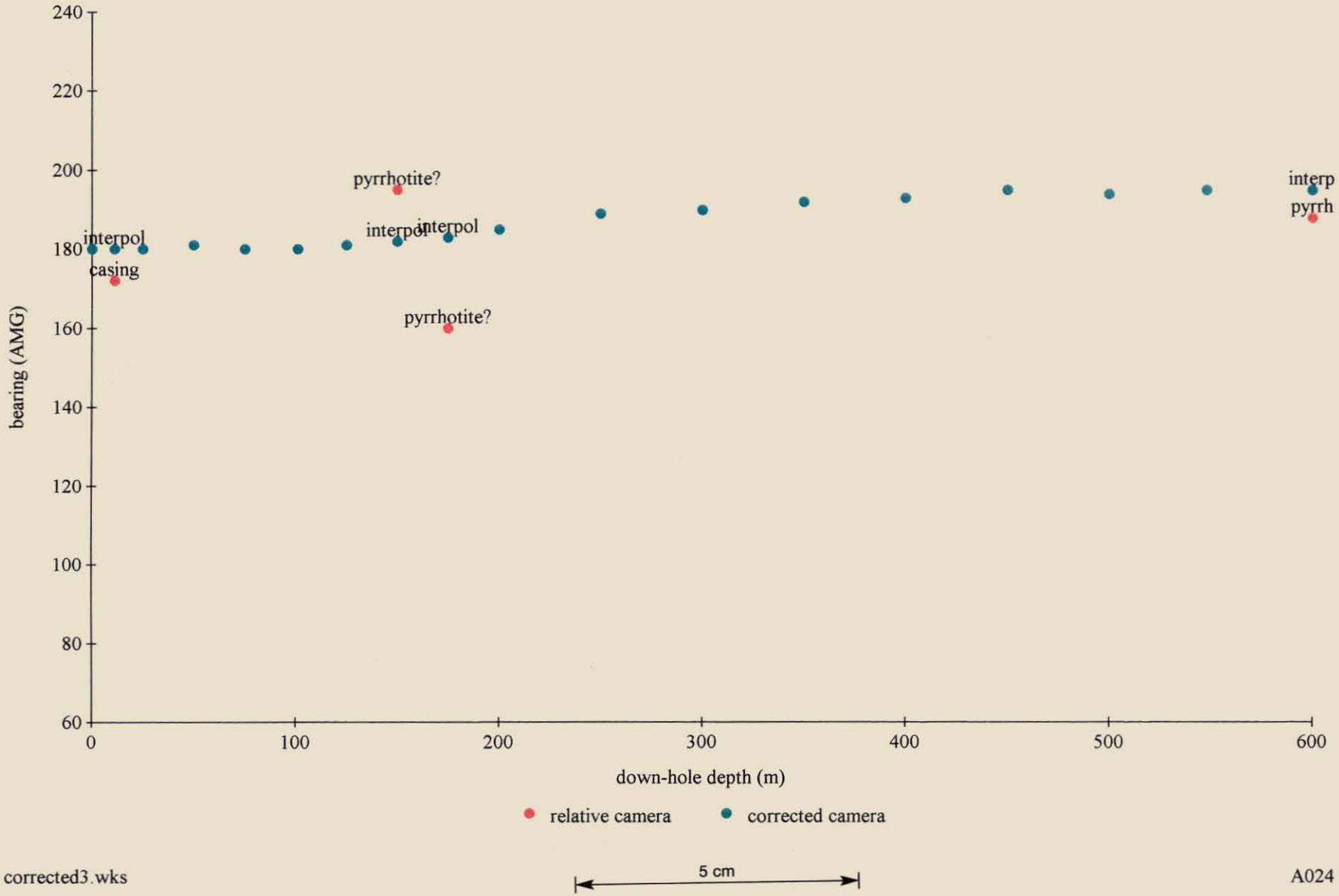
69139



corrected3.wks

A023

6012346


A024 down-hole bearings
relative and corrected camera
240
220
200
180
160
140
120
100
80
60
bearing (AMG)
0
100
200
300
400
500
600
down-hole depth (m)
pyrrhotite?
interpol
interpol
interpol
casing
pyrrhotite?
interp
pyrrh
relative camera
corrected camera
5 cm
corrected3.wks
A024

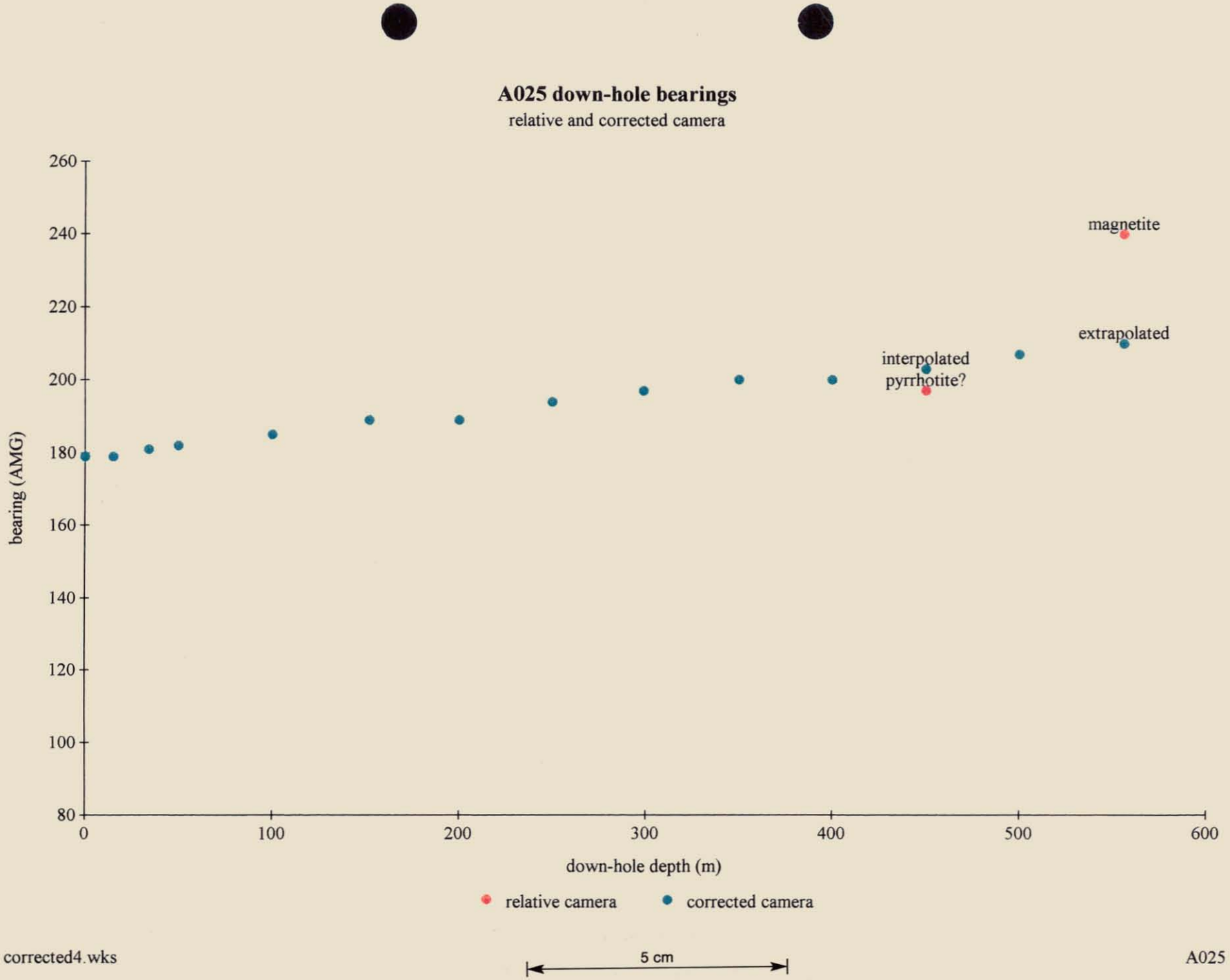
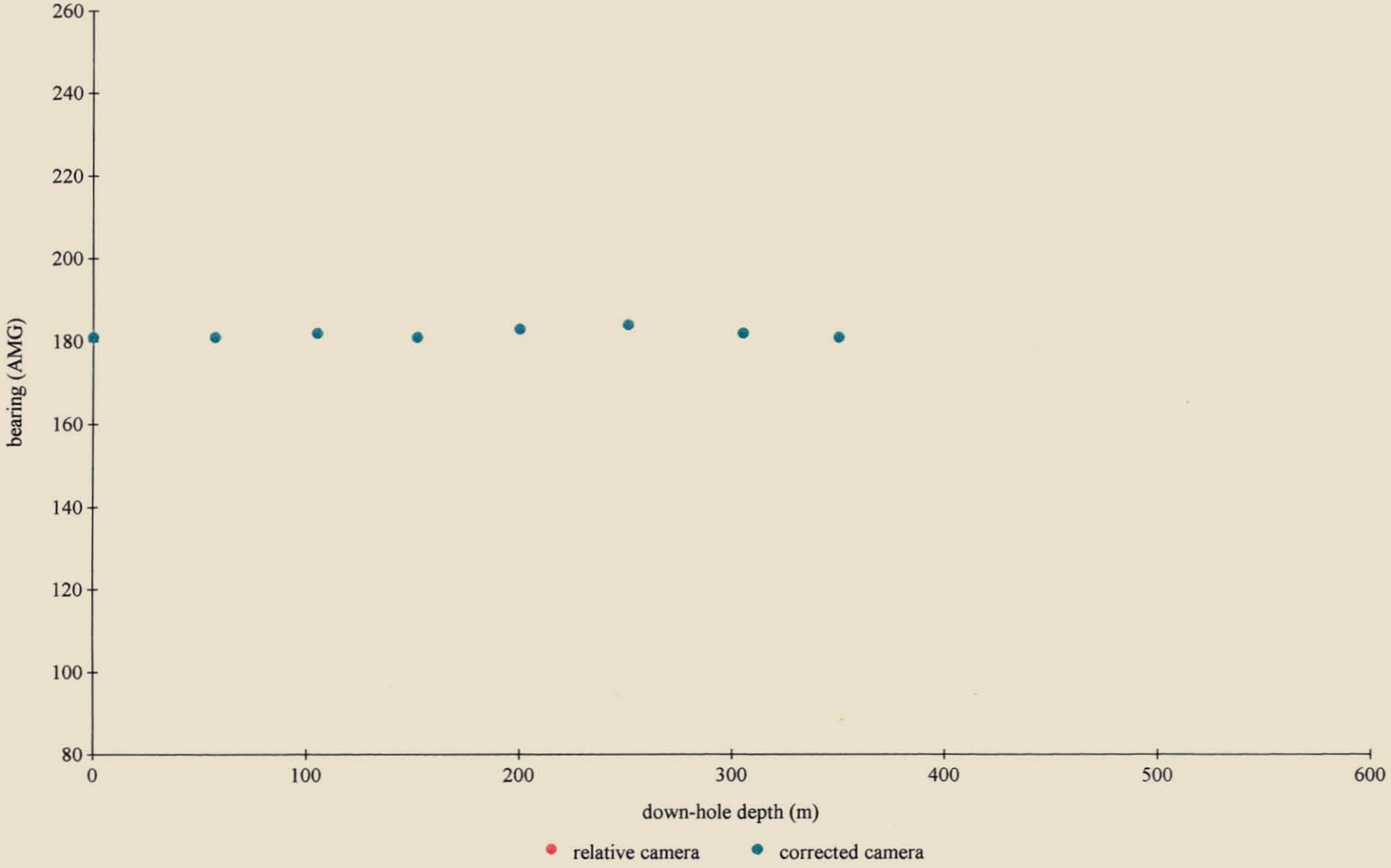

A025 down-hole bearings
relative and corrected camera
bearing (AMG)
260
240
220
200
180
160
140
120
100
80
0
100
200
300
400
500
600
down-hole depth (m)
magnetite
extrapolated
interpolated
pyrrhotite?
relative camera
corrected camera
corrected4.wks
5 cm
A025

Fig 6.6

A026 down-hole bearings

relative and corrected camera



corrected4.wks

5 cm

A026

6 FEB 99

Appendix 5

JORC Code

661251

Australasian Code for Reporting of Mineral Resources and Ore Reserves

(The JORC Code)

Prepared by the Joint Ore Reserves Committee of
The Australasian Institute of Mining and Metallurgy,
Australian Institute of Geoscientists and Minerals
Council of Australia (JORC)

Effective September 1999

MINERALS COUNCIL OF AUSTRALIA

FOREWORD

1. The *Australasian Code for Reporting of Mineral Resources and Ore Reserves* (the 'JORC Code' or 'the Code') sets out minimum standards, recommendations and guidelines for Public Reporting of exploration results, Mineral Resources and Ore Reserves in Australasia. It has been drawn up by the Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, the Australian Institute of Geoscientists and the Minerals Council of Australia. The Joint Ore Reserves Committee was established in 1971 and published a number of reports which made recommendations on the classification and Public Reporting of Ore Reserves prior to the first release of the JORC Code in 1989.

2. In this edition of the JORC Code, the guidelines, which were previously separated from the Code, have been placed after the respective Code clauses to provide improved assistance and guidance to readers. These guidelines are indented and are in a different, smaller type face. They do not form part of the Code but should be considered persuasive when interpreting the Code. The same indented and reduced type face formatting has been applied to Appendix 1 - '*The JORC Code and Australasian Stock Exchanges*', and to Table 1 - '*Check List of Assessment and Reporting Criteria*' to emphasise that both these sections are guidelines, and that the latter is not a mandatory list of assessment and reporting criteria. Also in this edition of the Code, the first or a particularly significant mention, after Clause 2, of terms which are defined in the Code have been marked with a superscript '[D10]', and the corresponding definitions have been highlighted in bold type. For example, Competent Person[D10] means that this term is defined in Clause 10.

3. The Code has been adopted by The Australasian Institute of Mining and Metallurgy and the Australian Institute of Geoscientists and is therefore binding on members of those organisations. It is supported by the Minerals Council of Australia and the Securities Institute of Australia as a contribution to best practice. The Australian Stock Exchange and New Zealand Stock Exchange listing rules each incorporate the Code. See Appendix 1.

4. The main principles governing the operation and application of the JORC Code are transparency, materiality and competence. 'Transparency' requires that the reader of a Public Report[D5] is provided with sufficient information, the presentation of which is clear and unambiguous, to understand the report and is not misled. 'Materiality' requires that a Public Report contains all the relevant information which investors and their professional advisers would reasonably require, and reasonably expect to find in the report, for the purpose of making a reasoned and balanced judgement regarding the mineralisation being reported. 'Competence' requires that the Public Report is based on work which is the responsibility of a suitably qualified and experienced person who is subject to an enforceable professional code of ethics.

5. The Code is a required minimum standard for Public Reporting. The committee also recommends its adoption as a minimum standard for other reporting. **Reference in the Code to a Public Report or Public Reporting is to a report or reporting on exploration results, Mineral Resources[D20] or Ore Reserves[D29], prepared for the purpose of informing investors or potential investors and their advisers. This includes a report or reporting prepared to satisfy regulatory requirements.** Companies are encouraged to provide information which is as comprehensive as possible in their Public Reports.

   Public Reports include, but are not limited to: company Annual Reports, quarterly reports and other reports to the Australian or New Zealand Stock Exchanges or required by corporations law. It is recommended that the Code apply to the following reports if they have been prepared for the purpose described in Clause 5: environmental statements; Information Memoranda; Expert Reports and technical papers in respect of reporting on exploration results, Mineral Resources or Ore Reserves.

   The term 'regulatory requirements' as used in Clause 5 is not intended to cover reports by companies to government agencies which may be required for State Government or Federal Government inventory or planning purposes. If reports prepared for such purposes subsequently become available to the public, they would not normally be regarded as Public Reports in terms of the JORC Code (refer also to the guidelines to Clauses 20 and 37).

   It is recognised that there may be situations where a Competent Person[D10] prepares documentation for internal company purposes or similar non-public purposes that does not comply with the JORC

Note: Code is in normal typeface, guidelines are in indented italics, definitions are in bold.

Code. In such circumstances, the documentation should include a statement that it does not comply with the JORC Code. This will minimise the likelihood of non-complying documentation being used as a basis for Public Reports, since Clause 8 requires Public Reports to fairly reflect Mineral Resource and/or Ore Reserve estimates and supporting documentation prepared by a Competent Person (refer to Clause 8, and also to Appendix 1 in respect of stock exchange requirements on Public Reporting).

While every effort has been made within the Code and Guidelines to cover most situations likely to be encountered in the Public Reporting of exploration results, Mineral Resources and Ore Reserves, there will inevitably be occasions when doubt exists as to the appropriate procedure to follow. In such cases, users of the Code and those compiling reports under the Code should be guided by its intent, which is to provide a minimum standard for Public Reporting and to ensure that such reporting contains all information which investors and their professional advisers would reasonably require, and reasonably expect to find in the report, for the purpose of making a reasoned and balanced judgement regarding the mineralisation being reported.

6. The Code is applicable to all solid minerals, including diamonds, other gemstones and coal, for which Public Reporting of exploration results, Mineral Resources and Ore Reserves is required by the Australian and New Zealand Stock Exchanges.

7. The Joint Committee recognises that further review of the Code will be required from time to time.

COMPETENCE AND RESPONSIBILITY

8. A Public Report concerning a company's Mineral Resources and/or Ore Reserves is the responsibility of the company acting through its Board of Directors. Any such report must be based on, and fairly reflect, the Mineral Resource and/or Ore Reserve estimates and supporting documentation prepared by a Competent Person[D10] or Persons.

   In compiling Mineral Resource and/or Ore Reserve information in a Public Report, a company may need to edit the documentation prepared by the Competent Persons. Where such editing takes place, the Competent Persons must give their consent in writing to the company to the inclusion in the Public Report of the matters based on their information in the form and context in which it appears in the Public Report.

   Refer to Appendix 1 for information on stock exchange requirements to name the Competent Person(s).

9. Documentation detailing Mineral Resource and Ore Reserve estimates from which a Public Report on Mineral Resources and Ore Reserves is prepared, must be prepared by or under the direction of, and signed by, a Competent Person or Persons.

10. **A 'Competent Person' is a person who is a Member or Fellow of The Australasian Institute of Mining and Metallurgy and/or the Australian Institute of Geoscientists with a minimum of five years experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which that person is undertaking. If the Competent Person is estimating, or supervising the estimation of Mineral Resources, the relevant experience must be in the estimation, assessment and evaluation of Mineral Resources. If the Competent Person is estimating, or supervising the estimation of Ore Reserves, the relevant experience must be in the estimation, assessment, evaluation and economic extraction of Ore Reserves.**

    The key qualifier in the definition of a Competent Person is the word 'relevant'. Determination of what constitutes relevant experience can be a difficult area and common sense has to be exercised. For example, in estimating Mineral Resources for vein gold mineralisation, experience in a high-nugget, vein-type mineralisation such as tin, uranium etc. will probably be relevant whereas experience in (say) massive base metal deposits may not be. As a second example, for a person to qualify as a Competent Person in the estimation of Ore Reserves for alluvial gold deposits, he or she would need to have considerable (probably at least five years) experience in the evaluation and economic extraction of this type of mineralisation, due to the characteristics of gold in alluvial systems, the particle sizing of the host sediment, and the low grades involved. Experience with placer deposits containing minerals other than gold may not necessarily provide appropriate relevant experience.

    The key word 'relevant' also means that it is not always necessary for a person to have five years experience in each and every type of deposit in order to act as a Competent Person if that person has relevant experience in other deposit types. For example, a person with (say) 20 years experience

in Mineral Resource estimation in a variety of metalliferous hard-rock deposit types may not require five years specific experience in (say) porphyry copper deposits in order to act as a Competent Person. Relevant experience in the other deposit types could count towards the required experience in relation to porphyry copper deposits.

In addition to experience in the style of mineralisation, a Competent Person preparing or taking responsibility for Mineral Resource estimates should have sufficient experience in the sampling and assaying techniques relevant to the deposit under consideration to be aware of problems which could affect the reliability of the data. Some appreciation of extraction and processing techniques applicable to that deposit type would also be important.

As a general guide, persons being called upon to act as Competent Persons should be clearly satisfied in their own minds that they could face their peers and demonstrate competence in the commodity, type of deposit and situation under consideration. If doubt exists, the person should either seek opinions from other colleagues or should decline to act as a Competent Person.

Estimation of Mineral Resources is often a team effort (for example, involving one person or team collecting the data and another person or team preparing the Mineral Resource estimate). Within this team, geologists usually occupy the pivotal role. Estimation of Ore Reserves is almost always a team effort involving a number of technical disciplines, and within this team, mining engineers usually occupy the pivotal role. Documentation for a Mineral Resource or Ore Reserve estimate must be compiled by, or under the supervision of, a Competent Person or Persons, whether a geologist, mining engineer or member of another discipline. However, it is recommended that, where there is a clear division of responsibilities within a team, each Competent Person should accept responsibility for his or her particular contribution. For example, one Competent Person could accept responsibility for the collection of Mineral Resource data, another for the Mineral Resource estimation process, another for the mining study, and the project leader could accept responsibility for the overall document. It is important that the Competent Person accepting overall responsibility for a Mineral Resource or Ore Reserve estimate and supporting documentation which has been prepared in whole or in part by others is satisfied that the work of the other contributors is acceptable.

If the Competent Person is a Member or Fellow of The Australasian Institute of Mining and Metallurgy ('The AusIMM'), he or she is answerable to The AusIMM Ethics Committee if a complaint is made in respect of his or her professional work. If the Competent Person is a Member or Fellow of the Australian Institute of Geoscientists ('AIG'), the matter will be dealt with by the Ethics and Standards Committee of the AIG Council, if a complaint is made in respect of his or her professional work.

When an Australian listed or New Zealand listed company with overseas interests wishes to report an overseas Mineral Resource or Ore Reserve estimate prepared by a person who is not a member of The AusIMM or of the AIG, it is necessary for the company to nominate a Competent Person or Persons to take responsibility for the Mineral Resource or Ore Reserve estimate. The Competent Person or Persons undertaking this activity should appreciate that they are accepting full responsibility for the estimate and supporting documentation under ASX or NZSX listing rules and should not treat the procedure merely as a 'rubber-stamping' exercise.

11. For Public Reports dealing with diamond or other gemstone mineralisation, it is also a requirement of this Code that, if a valuation of a parcel of diamonds or gemstones is reported, the person(s) or organisations valuing the parcel must be named in the report and their professional valuation experience, competency and independence stated.

REPORTING TERMINOLOGY

12. Public Reports dealing with Mineral Resources and/ or Ore Reserves must only use the terms set out in Figure 1.

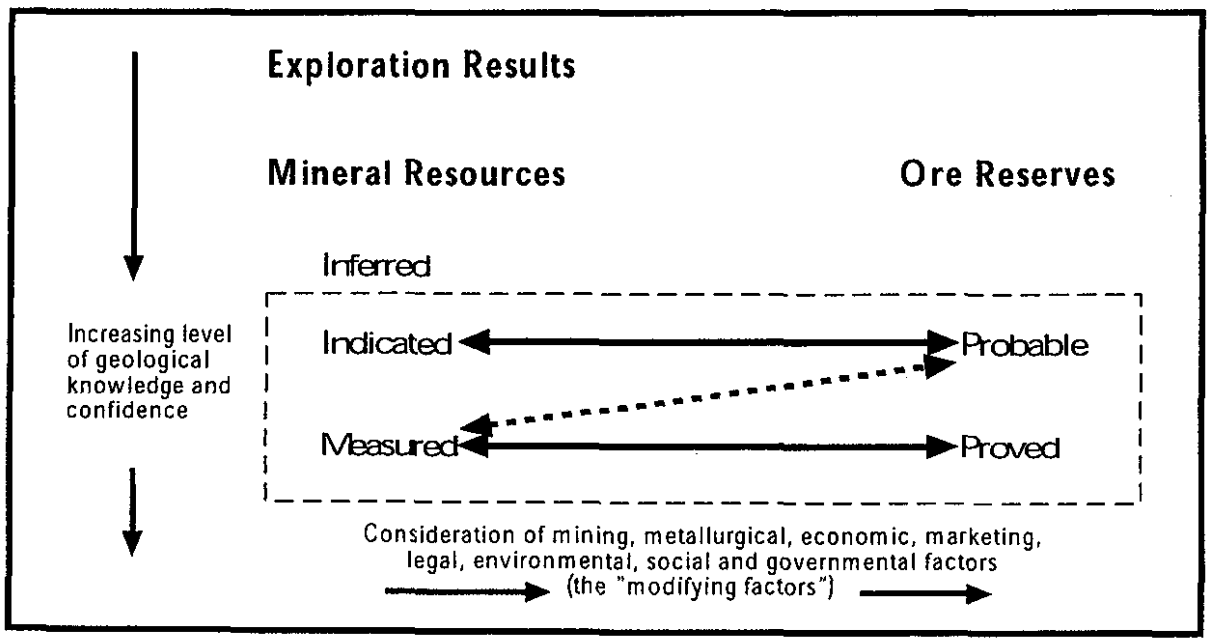
Figure 1 sets out the framework for classifying tonnage and grade estimates so as to reflect different levels of geological confidence and different degrees of technical and economic evaluation. Mineral Resources can be estimated mainly by a geologist on the basis of geoscientific information with some input from other disciplines. Ore Reserves, which are a modified sub-set of the Indicated[D22] and Measured Mineral Resources[D23] (shown within the dashed outline in Figure 1), require consideration of those factors affecting extraction, including mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors, and should in most instances be estimated with input from a range of disciplines.

Note: Code is in normal typeface, guidelines are in indented italics, definitions are in bold.

In certain situations, Measured Mineral Resources could convert to Probable Ore Reserves[030] because of uncertainties associated with the modifying factors which are taken into account in the conversion from Mineral Resources to Ore Reserves. This relationship is shown by the broken arrow in Figure 1 (although the trend of the broken arrow includes a vertical component, it does not, in this instance, imply a reduction in the level of geological knowledge or confidence). In such a situation these modifying factors should be fully explained.

Refer also to the guidelines to Clause 32.

Figure 1. General Relationship between Exploration Results, Mineral Resources and Ore Reserves.



REPORTING – GENERAL

13. Public Reports concerning a company's Mineral Resources or Ore Reserves should include a description of the style and nature of mineralisation.

14. A company must disclose relevant information concerning the status and characteristics of a mineral deposit which could materially influence the economic value of that deposit. A company must promptly report any material changes in its Mineral Resources or Ore Reserves.

15. Companies must review and publicly report on their Mineral Resources and Ore Reserves annually.

16. Throughout the Code, where appropriate, 'quality' may be substituted for 'grade' and 'volume' may be substituted for 'tonnage'.

REPORTING OF EXPLORATION RESULTS

17. A company may choose, or be required under stock exchange listing rules, to report exploration results. If a company reports exploration results in relation to mineralisation not classified as a Mineral Resource or an Ore Reserve, then estimates of tonnage and average grade must not be assigned to the mineralisation.

    Where descriptions of exploration targets or exploration potential are given in Public Reports, any tonnage/grade figures mentioned must be clearly order-of-magnitude and conceptual in nature and expressed so as not to misrepresent them as an estimate of Mineral Resources or Ore Reserves.

18. Public Reports of exploration results relating to mineralisation not classified as Mineral Resources or Ore Reserves must contain sufficient information to allow a considered and balanced judgement of the significance of the results. This must include relevant information such as sampling intervals and methods, sample locations, assay data, laboratory analyses, data aggregation methods plus information on any of the other criteria listed in Table 1 that are material to an assessment. The reporting of exploration sampling or

geophysical results must not be presented so as to unreasonably imply that potentially economic mineralisation has been discovered.

Table 1 is a check list and guideline to which those preparing reports on exploration results, Mineral Resources and Ore Reserves should refer. The check list is not prescriptive and, as always, relevance and materiality are overriding principles which determine what information should be publicly reported. Reporting of isolated assays without placing them in perspective is unacceptable.

19. Public Reports dealing with diamonds require the following additions:

- Reports of diamonds recovered from sampling programs must specify the number and total weight (in carats) of diamonds recovered. Details of the type and size of samples which produced the diamonds must also be specified including the lower cut-off sieve size used in the recovery.

- The weight of diamonds recovered may only be omitted from the report when the diamonds are less than 0.4 mm in size (ie. when the diamonds recovered are microdiamonds).

REPORTING OF MINERAL RESOURCES

20. **A 'Mineral Resource' is a concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such form and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred[D21], Indicated[D22] and Measured[D23] categories.**

Portions of a deposit that do not have reasonable prospects for eventual economic extraction must not be included in a Mineral Resource.

The term 'Mineral Resource' covers mineralisation which has been identified and estimated through exploration and sampling and within which Ore Reserves may be defined by the consideration and application of technical, economic, legal, environmental, social and governmental factors.

The term 'reasonable prospects for eventual economic extraction' implies a judgement (albeit preliminary) by the Competent Person in respect of the technical and economic factors likely to influence the prospect of economic extraction, including the approximate mining parameters. In other words, a Mineral Resource is not an inventory of all mineralisation drilled or sampled, regardless of cut-off grade, likely mining dimensions, location or continuity. It is a realistic inventory of mineralisation which, under assumed and justifiable technical and economic conditions, might, in whole or in part, become economically extractable.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for many coal, iron ore, bauxite and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However for the majority of gold deposits, application of the concept would normally be restricted to perhaps 20 to 30 years, and frequently to much shorter periods of time.

Certain reports (eg: coal inventory reports, exploration reports to government and other similar reports not intended primarily for providing information for investment purposes) may require full disclosure of all mineralisation, including some material that does not have reasonable prospects for eventual economic extraction. Such estimates of mineralisation would not qualify as Mineral Resources or Ore Reserves in terms of the JORC Code (refer also to the guidelines to Clauses 5 and 37).

Where considered appropriate by the Competent Person, Mineral Resource estimates may include material below the selected cut-off grade to ensure that the Mineral Resources comprise bodies of mineralisation of adequate size and continuity to properly consider the most appropriate approach to mining. Documentation of Mineral Resource estimates should clearly identify any such inclusions, and Public Reports should include commentary on the matter if considered material.

21. **An 'Inferred Mineral Resource' is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.**

Note: Code is in normal typeface, guidelines are in indented italics, definitions are in bold.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource.

The category is intended to cover situations where a mineral concentration or occurrence has been identified and limited measurements and sampling completed, but where the data are insufficient to allow the geological and/or grade continuity to be confidently interpreted. Due to the uncertainty which may attach to some Inferred Mineral Resources, it cannot be assumed that all or part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is usually not sufficient to allow the appropriate application of technical and economic parameters or to enable an evaluation of economic viability. Caution should be exercised if this category is considered in economic studies.

22. **An 'Indicated Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.**

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource, but has a higher level of confidence than that applying to an Inferred Mineral Resource.

Mineralisation may be classified as an Indicated Mineral Resource when the nature, quality, amount and distribution of data are such as to allow confident interpretation of the geological framework and to assume continuity of mineralisation. Confidence in the estimate is sufficient to allow the appropriate application of technical and economic parameters and to enable an evaluation of economic viability.

23. **A 'Measured Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and/or grade continuity.**

Mineralisation may be classified as a Measured Mineral Resource when the nature, quality, amount and distribution of data are such as to leave no reasonable doubt, in the opinion of the Competent Person determining the Mineral Resource, that the tonnage and grade of the mineralisation can be estimated to within close limits and that any variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit. Confidence in the estimate is sufficient to allow the appropriate application of technical and economic parameters and to enable an evaluation of economic viability.

24. The choice of the appropriate category of Mineral Resource depends upon the quantity, distribution and quality of data available and the level of confidence that attaches to those data. The appropriate Mineral Resource category must be determined by a Competent Person or Persons.

Mineral Resource classification is a matter for skilled judgement and Competent Persons should take into account those items in Table 1 which relate to confidence in Mineral Resource estimation.

In deciding between Measured Mineral Resources and Indicated Mineral Resources, Competent Persons may find it useful to consider, in addition to the phrases in the two definitions relating to geological and grade continuity in Clauses 22 and 23, the phrase in the guideline to the definition for Measured Mineral Resources: '.... any variation from the estimate would not significantly affect potential economic viability'.

In deciding between Indicated Mineral Resources and Inferred Mineral Resources, Competent Persons may wish to take into account, in addition to the phrases in the two definitions in Clauses 21 and 22 relating to geological and grade continuity, the guideline to the definition for Indicated Mineral Resources: 'Confidence in the estimate is sufficient to allow the appropriate application of technical and economic parameters and to enable an evaluation of economic viability', which contrasts with the guideline to the definition for Inferred Mineral Resources: 'Confidence in the estimate is usually not sufficient to allow the appropriate application of technical and economic parameters or to enable an evaluation of economic viability. Caution should be exercised if this category is considered in economic studies'.

25. Mineral Resource estimates are not precise calculations, being dependent on the interpretation

of limited information on the location, shape and continuity of the occurrence and on the available sampling results. Reporting of tonnage and grade figures should reflect the order of accuracy of the estimate by rounding off to appropriately significant figures and, in the case of Inferred Mineral Resources, by qualification with terms such as 'approximately'.

In most situations, rounding to the second significant figure should be sufficient. For example 10,863,000 tonnes at 8.23 per cent should be stated as 11 million tonnes at 8.2 per cent. There will be occasions, however, where rounding to the first significant figure may be necessary in order to convey properly the uncertainties in estimation. This would usually be the case with Inferred Mineral Resources.

To emphasise the imprecise nature of a Mineral Resource or Ore Reserve estimate, the final result should always be referred to as an estimate not a calculation.

26. Public Reports of Mineral Resources must specify one or more of the categories of 'Inferred', 'Indicated' and 'Measured'. Categories must not be reported in a combined form unless details for the individual categories are also provided. Mineral Resources must not be reported in terms of contained metal or mineral content unless corresponding tonnages and grades are also presented. Mineral Resources must not be aggregated with Ore Reserves.

27. Table 1 provides, in a summary form, a list of the main criteria which should be considered when preparing reports on exploration results, Mineral Resources and Ore Reserves. These criteria need not be discussed in a Public Report unless they materially affect estimation or classification of the Mineral Resources.

    Where diamond Mineral Resource grades are based on the correlation of macrodiamond grade with the grade of microdiamonds, this must be stated and its reliability explained.

    It is not necessary, when publicly reporting, to comment on each item in Table 1, but it is essential to discuss any matters which might materially affect the reader's understanding or interpretation of the results or estimates being reported. This is particularly important where inadequate or uncertain data affect the reliability of, or confidence in, a statement of exploration results or an estimate of Mineral Resources and/or Ore Reserves; for example, poor sample recovery, poor repeatability of assay or laboratory results, limited information on tonnage factors etc.

    If there is doubt about what should be reported, it is better to err on the side of providing too much information rather than too little.

    Mineral Resource or Ore Reserve estimates are sometimes reported after adjustment by cutting of high grades, or after the application of modifying factors arising from reconciliation with mill data. If any of the data are materially adjusted or modified for the purpose of making the estimate, or if the estimate is subsequently adjusted, this should be clearly stated in a Public Report of Mineral Resources or Ore Reserves and the nature of the adjustment or modification described.

28. The words 'ore' and 'reserves' must not be used in stating Mineral Resource estimates as the terms imply technical feasibility and economic viability and are only appropriate when all relevant technical, economic, marketing, legal, environmental, social and governmental factors have been considered. Reports and statements should continue to refer to the appropriate category or categories of Mineral Resources until technical feasibility and economic viability have been established. If re-evaluation indicates that the Ore Reserves are no longer viable, the Ore Reserves must be reclassified as Mineral Resources or removed from Mineral Resource/Ore Reserve statements.

    It is not intended that re-classification from Ore Reserves to Mineral Resources should be applied as a result of changes expected to be of a short term or temporary nature, or where company management has made a deliberate decision to operate on a non-economic basis. Examples of such situations might be a commodity price drop expected to be of short duration, mine emergency of a non-permanent nature, transport strike etc.

REPORTING OF ORE RESERVES

29. **An 'Ore Reserve' is the economically mineable part of a Measured or Indicated Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments**

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demonstrate at the time of reporting that extraction could reasonably be justified. Ore Reserves are sub-divided in order of increasing confidence into Probable Ore Reserves[D30] and Proved Ore Reserves[D31].

Ore Reserves are those portions of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Competent Person or Persons making the estimates, can be the basis of a viable project after taking account of all relevant metallurgical, economic, marketing, legal, environmental, social and governmental factors. Ore Reserves are inclusive of diluting material which will be mined in conjunction with the Ore Reserves and delivered to the treatment plant or equivalent.

The term 'economic' implies that extraction of the Ore Reserve has been established or analytically demonstrated to be viable and justifiable under reasonable investment assumptions. The term 'Ore Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

It should be noted that the Code does not imply that an economic operation must have Proved Ore Reserves. Situations arise where Probable Ore Reserves alone may be sufficient to justify extraction, as for example with some alluvial tin or gold deposits.

Some countries use the term 'Mineral Reserve' instead of 'Ore Reserve'. The Joint Ore Reserves Committee has retained the term 'Ore Reserve' because it assists in maintaining a clear distinction between a 'Mineral Resource' and an 'Ore Reserve', a distinction which might be less clear if 'Mineral Reserve' was substituted. However, if preferred by the reporting company, 'Ore Reserve' and 'Mineral Resource' estimates for coal may be reported as 'Coal Reserve' and 'Coal Resource' estimates.

30. **A 'Probable Ore Reserve' is the economically mineable part of an Indicated, and in some circumstances Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified.**

A Probable Ore Reserve has a lower level of confidence than a Proved Ore Reserve.

31. **A 'Proved Ore Reserve' is the economically mineable part of a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified.**

32. The choice of the appropriate category of Ore Reserve is determined primarily by the classification of the corresponding Mineral Resource and must be made by the Competent Person or Persons.

The Code provides for a direct relationship between Indicated Mineral Resources and Probable Ore Reserves and between Measured Mineral Resources and Proved Ore Reserves. In other words, the level of geoscientific confidence for Probable Ore Reserves is the same as that required for the in situ determination of Indicated Mineral Resources and for Proved Ore Reserves is the same as that required for the in situ determination of Measured Mineral Resources.

The 1999 edition of the Code also provides, for the first time, for a two-way relationship between Measured Mineral Resources and Probable Ore Reserves. This is to cover the situation where uncertainties associated with any of the modifying factors considered when converting Mineral Resources to Ore Reserves may result in there being a significantly lower degree of confidence in the Ore Reserves than in the corresponding Measured Mineral Resources. Such a conversion would not imply a reduction in the level of geological knowledge or confidence.

If the uncertainties in the modifying factors preventing the Measured Mineral Resource being converted to a Proved Ore Reserve are removed, the Measured Mineral Resource may be converted to a Proved Ore Reserve. However modification is only acceptable to an equivalent or lower level of confidence. No amount of confidence in the modifying factors for conversion of a Mineral Resource to an Ore Reserve can override the upper level of confidence which exists in the Mineral Resource. Under no circumstances can an Indicated Mineral Resource be converted directly to a Proved Ore Reserve.

Application of the category of a Proved Ore Reserve implies the highest degree of confidence in the estimate with consequent expectations in the minds of readers of the report. These expectations should be borne in mind when categorising a Mineral Resource as Measured.

Refer also to the guidelines to Clause 24 regarding classification of Mineral Resources.

33. Ore Reserve estimates are not precise calculations and tonnage and grade figures in Public Reports should be expressed so as to convey the order of accuracy of the estimates by rounding off to appropriately significant figures.

Refer to the guidelines to Clause 25, regarding rounding of Mineral Resource estimates.

34. Except for the special provisions relating to coal (see Clause 39) Public Reports of Ore Reserves must specify one or both of the categories of 'Proved' and 'Probable'. Categories must not be reported in a combined form unless details for the individual categories are also provided. Ore Reserves must not be reported in terms of contained metal or mineral content unless corresponding tonnages and grades are also presented.

In reporting Ore Reserves, information on assumed metallurgical recovery factors is very important, and should always be included in Public Reports.

35. In situations where figures for both Mineral Resources and Ore Reserves are reported, a clarifying statement must be included in the report which clearly indicates whether the Mineral Resources are inclusive of, or additional to the Ore Reserves.

The committee recognises that there are legitimate reasons, in some situations, for reporting Mineral Resources inclusive of Ore Reserves and, in other situations, for reporting Mineral Resources additional to Ore Reserves. The committee does not express a preference but it does require that reporting companies make it clear which form of reporting has been adopted. Appropriate forms of clarifying statements may be:

'The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Ore Reserves.' or:

'The Measured and Indicated Mineral Resources are additional to the Ore Reserves.'

Inferred Mineral Resources are, by definition, always additional to Ore Reserves.

Where there is a substantial difference between the statement of Mineral Resources and the statement of Ore Reserves in a Public Report, an explanation of the reasons for the difference should be included in the report. This will assist the reader of the report in making a judgement of the likelihood of the remaining Mineral Resources eventually being converted to Ore Reserves.

Ore Reserves may incorporate material (dilution) which is not part of the original Mineral Resource. It is essential that this fundamental difference between Mineral Resources and Ore Reserves is borne in mind and caution exercised if attempting to draw conclusions from a comparison of the two.

For the same reason, Ore Reserves should not be added to Mineral Resources. The resulting total can be very misleading in economic terms and is capable of being misunderstood or, more seriously, of being misused to give a false impression of a company's mineral prospects.

Public Reporting of tonnage and grade estimates using terms other than Mineral Resources and Ore Reserves is not permitted under the Code.

In preparing the Ore Reserve statement, the relevant Mineral Resource statement on which it is based should first be developed. This can be reconciled with the Mineral Resource statement estimated for the previous comparable period and differences (due, for example, to mine production, exploration etc) identified. The application of cut-off and other criteria to the Mineral Resource can then be made to develop the Ore Reserve statement which can also be reconciled with the previous comparable statement. Companies are encouraged whenever possible to reconcile estimates in their reports. A detailed account of differences between estimates is not essential, but sufficient comment should be made to enable significant variances to be understood by the reader.

36. Table 1 provides, in a summary form, a list of the main criteria which should be considered when preparing reports on exploration results, Mineral Resources and Ore Reserves. These criteria need not be discussed in a Public Report unless they materially affect estimation or classification of the Ore Reserves. Changes in economic or political factors alone may be the basis for significant changes in Ore Reserves and should be reported accordingly.

Where diamond Ore Reserve grades are based on the correlation of macrodiamond grade with the grade of microdiamonds, this must be stated and its reliability explained. If a valuation of a parcel of diamonds is reported, the weight in carats and size range of the contained diamonds must be stated and the value of the diamonds must be given in US dollars per carat.

Refer also to Clause 19 and to the guidelines to Clause 27.

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REPORTING OF COAL RESOURCES AND RESERVES

37. Clauses 38 to 40 of the Code address matters which relate specifically to the Public Reporting of Coal Resources and Reserves. Unless otherwise stated, clauses 1 to 36 of this Code (including Figure 1) apply. Table 1, as part of the guidelines, should also be considered persuasive when reporting on Coal Resources and Reserves.

    For guidance on the estimation of black Coal Resources and Reserves and on statutory reporting not primarily intended for providing information to the investing public, readers are referred to the 1999 edition of the "Guidelines for the Estimation and Reporting of Australian Black Coal Resources and Reserves", *a document drawn up by a committee of coal industry and government representatives and consultants from New South Wales and Queensland.*

    Coal is of particular interest to State and Federal Governments because of its impact on government planning and land use implications. Reports to governments may require estimates of coal resources which are not constrained by short to medium term economic considerations. Such reports and estimates of strategic resources are not covered by the JORC Code. Refer also to the guidelines to Clauses 5 and 20.

38. The terms 'Mineral Resource(s)' and 'Ore Reserve(s)', and the subdivisions of these as defined above, apply also to coal reporting, but if preferred by the reporting company, the terms 'Coal Resource(s)' and 'Coal Reserve(s)' and appropriate subdivisions may be substituted.

39. For coal reporting only, Probable and Proved Ore (Coal) Reserves may be combined and reported as Recoverable Reserves.

40. Reports of 'Marketable Coal Reserves', representing beneficiated or otherwise enhanced coal product, may be used in Public Reports in conjunction with, but not instead of, reports of Ore (Coal) Reserves. The basis of the predicted yield to achieve Marketable Coal Reserves should be stated.

REPORTING OF MINERALISED STOPE FILL, STOCKPILES, REMNANTS, PILLARS, LOW GRADE MINERALISATION AND TAILINGS

41. The Code applies to the reporting of all potentially economic mineralised material including mineralised stope fill, stockpiles, remnants, pillars, low grade mineralisation and tailings.

    For the purposes of the Code, mineralised stope fill and stockpiles of mineralised material can be considered to be similar to in situ mineralisation when reporting Mineral Resources and Ore Reserves. Consequently the Competent Person assessing the fill or stockpiles must use the bases of classification outlined in the Code. In most cases, the opinion of a relevant professional should be sought when making judgements about the mineability of fill, remnants and pillars.

    If there are not reasonable prospects for the eventual economic extraction of a particular portion of the fill or stockpile, this material cannot be classified as either Mineral Resources or Ore Reserves. If some portion is currently sub-economic but there is a reasonable expectation that it will become economic, then this material may be classified as a Mineral Resource. Such stockpile material may include old dumps and tailings dam material. If technical and economic studies have demonstrated that economic extraction could reasonably be justified under realistically assumed conditions, the material may be classified as an Ore Reserve.

    The above guidelines apply equally to low grade in situ mineralisation, sometimes referred to as 'mineralised waste' or 'marginal grade material', and often intended for stockpiling and treatment towards the end of mine life. For clarity of understanding, it is recommended that tonnage and grade estimates of such material be itemised separately in Public Reports, although they may be aggregated with total Mineral Resource and Ore Reserve figures.

    Stockpiles are defined to include both surface and underground stockpiles, including broken ore in stopes, and can include ore currently in the ore storage system. Mineralised material being processed (including leaching), if reported, should be reported separately.

    Mineralised remnants, shaft pillars and mining pillars which are potentially mineable are in situ mineralisation and consequently are included in the Code definitions of Mineral Resources and Ore Reserves.

    Mineralised remnants, shaft pillars and mining pillars which are not potentially mineable must not be included in Mineral Resource and Ore Reserve statements.

TABLE 1
CHECK LIST OF ASSESSMENT AND REPORTING CRITERIA

Table 1 is a check list and guideline which those preparing reports on exploration results, Mineral Resources and Ore Reserves should use as a reference. The check list is not prescriptive and, as always, relevance and materiality are overriding principles that determine what information should be publicly reported. It is, however, important to report any matters that might materially affect a reader's understanding or interpretation of the results or estimates being reported. This is particularly important where inadequate or uncertain data affect the reliability of, or confidence in, a statement of exploration results or an estimate of Mineral Resources and/or Ore Reserves.

The order and grouping of criteria in Table 1 reflects the normal systematic approach to exploration and evaluation. Criteria in the first group 'Sampling Techniques and Data' apply to all succeeding groups. In the remainder of the table, criteria listed in preceding groups would often apply to succeeding groups and should be considered when estimating and reporting.

CRITERIA	**EXPLANATION**
	SAMPLING TECHNIQUES AND DATA ***(criteria in this group apply to all succeeding groups)***
Drilling techniques	*Drill type (eg. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka etc.) and details (eg. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, etc.). Measures taken to maximise sample recovery and ensure representative nature of the samples.*
Logging	*Whether core and chip samples have been logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. Whether logging is qualitative or quantitative in nature. Core (or costean, channel etc.) photography.*
Drill sample recovery	*Whether core and chip sample recoveries have been properly recorded and results assessed. In particular whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*
Other sampling techniques	*Nature and quality of sampling (eg. cut channels, random chips etc.) and measures taken to ensure sample representivity.*
Sub-sampling techniques and sample preparation	*If core, whether cut or sawn and whether quarter, half or all core taken. If non-core, whether riffled, tube sampled, rotary split etc. and whether sampled wet or dry. For all sample types, the nature, quality and appropriateness of the sample preparation technique. Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. Measures taken to ensure that the sampling is representative of the in situ material collected. Whether sample sizes are appropriate to the grainsize of the material being sampled.*
Quality of assay data and laboratory tests	*The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. Nature of quality control procedures adopted (eg. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie. lack of bias) and precision have been established.*
Verification of sampling and assaying	*The verification of significant intersections by either independent or alternative company personnel. The use of twinned holes.*
Location of data points	*Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. Quality and adequacy of topographic control.*
Data density and distribution	*Data density for reporting of exploration results. Whether the data density and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. Whether sample compositing has been applied.*
Audits or reviews	*The results of any audits or reviews of sampling techniques and data.*

Note: Code is in normal typeface, guidelines are in indented italics, definitions are in bold.

REPORTING OF EXPLORATION RESULTS ***(criteria listed in the preceding group apply also to this group)***	
Mineral tenement and land tenure status	*Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. In particular the security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*
Exploration done by other parties	*Acknowledgement and appraisal of exploration by other parties.*
Geology	*Deposit type, geological setting and style of mineralisation.*
Data aggregation methods	*In reporting exploration results, weighting averaging techniques, maximum and/or minimum grade truncations (eg. cutting of high grades) and cut-off grades are usually material and should be stated. Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. The assumptions used for any reporting of metal equivalent values should be clearly stated.*
Relationship between mineralisation widths and intercept lengths	*These relationships are particularly important in the reporting of exploration results. If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. If it is not known and only the down-hole lengths are reported, there should be a clear statement to this effect (eg. 'downhole length, true width not known').*
Diagrams	*Where possible, maps and sections (with scales) and tabulations of intercepts should be included for any material discovery being reported if such diagrams significantly clarify the report.*
Balanced reporting	*Where comprehensive reporting of all exploration results is not practicable, representative reporting of both low and high grades and/or widths should be practised to avoid misleading reporting of exploration results.*
Other substantive exploration data	*Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples - size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*
Further work	*The nature and scale of planned further work (eg. tests for lateral extensions or depth extensions or large-scale step-out drilling).*
ESTIMATION AND REPORTING OF MINERAL RESOURCES ***(criteria listed in the first group, and where relevant in the second group, apply also to this group)***	
Database integrity	*Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. Data validation procedures used.*
Geological interpretation	*Nature of the data used and of any assumptions made. The effect, if any, of alternative interpretations on Mineral Resource estimation. The use of geology in guiding and controlling Mineral Resource estimation. The factors affecting continuity both of grade and geology.*
Estimation and modelling techniques	*The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters, maximum distance of extrapolation from data points. The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. The assumptions made regarding recovery of by-products. In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. Any assumptions behind modelling of selective mining units (eg. non-linear kriging). The process of validation, the checking process used, the comparison of model data to drillhole data, and use of reconciliation data if available.*

Cut-off grades or parameters	*The basis of the cut-off grade(s) or quality parameters applied, including the basis, if appropriate, of equivalent metal formulae.*
Mining factors or assumptions	*Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It may not always be possible to make assumptions regarding mining methods and parameters when estimating Mineral Resources. Where no assumptions have been made, this should be reported.*
Metallurgical factors or assumptions	*The basis for assumptions or predictions regarding metallurgical amenability. It may not always be possible to make assumptions regarding metallurgical treatment processes and parameters when reporting Mineral Resources. Where no assumptions have been made, this should be reported.*
Tonnage factors (in situ bulk densities)	*Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, the frequency of the measurements, the nature, size and representativeness of the samples.*
Classification	*The basis for the classification of the Mineral Resources into varying confidence categories. Whether appropriate account has been taken of all relevant factors. ie. relative confidence in tonnage/ grade computations, confidence in continuity of geology and metal values, quality, quantity and distribution of the data. Whether the result appropriately reflects the Competent Person(s)' view of the deposit.*
Audits or reviews	*The results of any audits or reviews of Mineral Resource estimates.*
ESTIMATION AND REPORTING OF ORE RESERVES ***(criteria listed in the first group, and where relevant in other preceding groups, apply also to this group)***	
Mineral Resource estimate for conversion to Ore Reserves	*Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve. Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*
Cut-off grades or parameters	*The basis of the cut-off grade(s) or quality parameters applied, including the basis, if appropriate, of equivalent metal formulae. The cut-off grade parameter may be economic value per block rather than metal grade.*
Mining factors or assumptions	*The method and assumptions used to convert the Mineral Resource to an Ore Reserve (ie either by application of appropriate factors by optimisation or by preliminary or detailed design). The choice of, the nature and the appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc. The assumptions made regarding geotechnical parameters (eg. pit slopes, stope sizes, etc.), grade control and pre-production drilling. The major assumptions made and Mineral Resource model used for pit optimisation (if appropriate). The mining dilution factors, mining recovery factors, and minimum mining widths used and the infrastructure requirements of the selected mining methods.*
Metallurgical factors or assumptions	*The metallurgical process proposed and the appropriateness of that process to the style of mineralisation. Whether the metallurgical process is well-tested technology or novel in nature. The nature, amount and representativeness of metallurgical testwork undertaken and the metallurgical recovery factors applied. Any assumptions or allowances made for deleterious elements. The existence of any bulk sample or pilot scale testwork and the degree to which such samples are representative of the orebody as a whole.*
Cost and revenue factors	*The derivation of, or assumptions made, regarding projected capital and operating costs. The assumptions made regarding revenue including head grade, metal or commodity price(s), exchange rates, transportation and treatment charges, penalties, etc. The allowances made for royalties payable, both Government and private.*
Market assessment	*The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future. A customer and competitor analysis along with the identification of likely market windows for the product. Price and volume forecasts and the basis for these forecasts. For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*

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Others	*The effect, if any, of natural risk, infrastructure, environmental, legal, marketing, social or governmental factors on the likely viability of a project and/or on the estimation and classification of the Ore Reserves. The status of titles and approvals critical to the viability of the project, such as mining leases, discharge permits, government and statutory approvals.*
Classification	*The basis for the classification of the Ore Reserves into varying confidence categories. Whether the result appropriately reflects the Competent Person(s)' view of the deposit. The proportion of Probable Ore Reserves which have been derived from Measured Mineral Resources (if any).*
Audits or reviews	*The results of any audits or reviews of Ore Reserve estimates.*
ESTIMATION AND REPORTING OF DIAMOND MINERALISATION ***(criteria listed in the first group, and where relevant in other preceding groups, apply also to this group)***	
Primary rock source	*Primary sources of diamonds in nature are variable and complex. Accordingly, information relating to primary sources should contain details of the nature of the rock type together with its form, shape and size.*
Diamond value	*Diamond valuation is a highly specialised process and is only possible on parcels containing appropriate numbers of macrodiamonds. It is not possible to evaluate diamond quality from microdiamonds. Classification of diamonds as, for example, gem, near gem and industrial should be made by recognised experts who should be identified in the valuation report and their independence stated. The number of stones, the total carat weight and size range for the parcel valued should be stated.*
Secondary rock source	*Secondary sources of diamonds in nature, including alluvial deposits, are variable and complex. Accordingly, information relating to secondary sources should contain details of the nature of the geological environment together with its form, age and size.*
Microdiamonds	*Current practice in Australia defines microdiamonds as diamonds which will pass through a screen with 0.4 mm aperture, ie. diamonds weighing less than 0.001 carats. Reports of microdiamond recoveries should specify both the number of stones recovered and the top and bottom screen or crushing sizes used in the recovery process.*
Macrodiamonds	*Macrodiamonds are defined as diamonds larger than 0.4 mm in size. Reports of macrodiamond recoveries should specify both the number of stones and the total carat weight recovered above a specified screen size.*
Indicator minerals	*Conventional indicator minerals include garnet, ilmenite, chrome spinel and chrome diopside having the requisite chemical and physical attributes that distinguish them from otherwise similar minerals found in non-diamond associated rock types. Reports of indicator minerals should be prepared by a suitably qualified laboratory.*
Sampling parameters	*Reported discoveries of diamonds or indicator minerals from all samples must be accompanied by details of the sampling parameters and sampling equipment used. Type of sample (stream sediment, soil, bulk, rock etc.) as well as sample size, sample density and screening or jigging parameters are required.*
Cut-off grades	*Assumptions regarding cut-off grades should specify minimum screen size.*
Carat	*One fifth (0.2) of a gram (often defined as a metric carat or MC).*
Grades	*Internationally, diamond grades for primary deposits are stated both in carats per tonne and carats per 100 tonnes. The Joint Ore Reserves Committee recommends the use of carats per tonne. In the case of alluvial deposits, industry practice is to quote grades in carats per tonne or carats per cubic metre. Volumes: state bank or loose cubic metres and basis of volume to tonnage conversions.*

APPENDIX 1
THE JORC CODE AND AUSTRALASIAN STOCK EXCHANGES

The Australian and New Zealand Stock Exchanges ('ASX' and 'NZSX') have, since 1989 and 1992 respectively, incorporated the Code into their listing rules. Under these listing rules, a Public Report must be prepared in accordance with the Code if it includes a statement on exploration results, Mineral Resources or Ore Reserves. The incorporation of the Code imposes certain specific requirements on mining or exploration companies reporting to the ASX and NZSX. The guidelines in this section of the Code which paraphrase these requirements should not be used as a replacement for the relevant listing rules, and it is strongly recommended that users of the Code familiarise themselves with those listing rules which relate to Public Reporting of exploration results, Mineral Resources and Ore Reserves.

ASX listing rules require the Competent Person(s), on whose work the Public Report of Mineral Resources or Ore Reserves is based, to be named in the report. The report or attached statement must say that the person consents to the inclusion in the report of the matters based on their information in the form and context in which it appears, and must include the name of the person's firm or employer. Refer also to Clause 8 of the Code.

Appropriate forms of compliance statements may be as follows (delete bullet points which do not apply):

- *If the required information is in the report:*

  "The information in this report that relates to Mineral Resources or Ore Reserves is based on information compiled by (insert name of Competent Person), who is a Fellow or Member of The Australasian Institute of Mining and Metallurgy or the Australian Institute of Geoscientists (select as appropriate)": or

- *If the required information is included in an attached statement:*

  "The Information in the report to which this statement is attached that relates to Mineral Resources or Ore Reserves is based on information compiled by (insert name of Competent Person), who is a Fellow or Member of The Australasian Institute of Mining and Metallurgy or the Australian Institute of Geoscientists (select as appropriate)".

- *If the Competent Person is a full-time employee of the company:*

  "(Insert name of Competent Person) is a full-time employee of the company".

- *If the Competent Person is not a full-time employee of the company:*

  "(Insert name of Competent Person) is employed by (insert name of Competent Person's employer)".

- *For all reports:*

  "(Insert name of Competent Person) has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he (or she) is undertaking to qualify as a Competent Person as defined in the 1999 Edition of the "Australasian Code for Reporting of Mineral Resources and Ore Reserves". (Insert name of Competent Person) consents to the inclusion in the report of the matters based on their information in the form and context in which it appears".

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Appendix 6

Nickel assay data used for the resource estimates

bhid	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	each hole length	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm
bhid	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	each hole length	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm
A003	223.7	224.7	1.0	1.02	0.96	10	200	15	95	34								
A003	224.7	225.7	1.0	0.56	0.55	10	100	X	140	26								
A003	225.7	227.0	1.3	1.04	1.09	10	200	X	85	34								
A003	227.0	228.2	1.2	0.50	0.50	5	100	X	85	30								
A003	228.2	229.2	1.0	0.82	0.84	15	150	10	95	32								
A003	229.2	230.2	1.0	0.62	0.61	10	100	X	50	22								
A003	230.2	231.2	1.0	0.72	0.73	5	150	X	80	28								
A003	231.2	232.2	1.0	1.12	1.13	15	200	35	130	28								
A003	232.2	233.2	1.0	0.61	0.67	20	100	X	145	18								
A003	233.2	234.2	1.0	0.82	0.86	20	150	X	60	22								
A003	234.2	235.2	1.0	0.59	0.67	15	100	X	85	24								
A003	235.2	236.2	1.0	0.53	0.61	20	100	X	115	20								
A003	236.2	237.2	1.0	0.93	1.06	35	150	X	70	14								
A003	237.2	238.2	1.0	1.64	2.07	95	250	X	105	30								
A003	238.2	239.2	1.0	0.82	1.02	55	150	X	80	22								
A003	239.2	240.2	1.0	1.80	2.29	110	300	X	85	36								
A003	240.2	241.2	1.0	0.99	1.23	70	150	X	75	28								
A003	241.2	242.2	1.0	0.87	1.03	50	150	X	60	20								
A003	242.2	243.2	1.0	1.84	2.21	105	350	X	80	22								
A003	243.2	244.2	1.0	0.72	0.82	35	150	X	105	18								
A003	244.2	244.9	0.7	1.22	1.24	70	250	X	85	20								
A003	244.9	245.9	1.0	2.08	2.34	115	500	X	70	22								
A003	245.9	246.9	1.0	1.83	2.30	55	350	15	95	24								
A003	246.9	247.9	1.0	0.36	0.75	25	50	80	90	16								
A003	247.9	248.7	0.8	0.19	0.33	10	X	50	95	40								
A003	248.7	249.7	1.0	0.18	0.48	15	X	145	700	22								
A003	249.7	250.7	1.0	0.42	0.69	20	100	40	90	63								
A003	250.7	251.7	1.0	0.58	1.31	30	100	30	65	200								
A003	251.7	252.7	1.0	0.25	0.76	20	X	120	85	190								
A003	252.7	253.7	1.0	0.38	1.09	25	50	40	135	22								
A003	253.7	254.7	1.0	0.32	0.29	X	50	X	35	42								
A003	254.7	255.7	1.0	0.40	0.28	30	50	X	65	110								
A003	255.7	256.7	1.0	0.17	0.15	5	X	X	100	18								
A003	256.7	257.7	1.0	0.12	0.07	X	X	X	35	26								
A003	257.7	258.7	1.0	0.24	0.19	5	50	X	35	24								
A003	258.7	259.7	1.0	0.15	0.09	X	X	X	40	26								
A003	259.7	260.7	1.0	0.20	0.14	X	X	X	40	48								
A003	260.7	262.0	1.3	0.20	0.13	X	50	X	40	30								
A003	262.0	263.3	1.3	0.74	0.67	30	200	X	45	30								
A003	263.3	264.3	1.0	1.47	1.40	45	300	X	70	83								
A003	264.3	265.3	1.0	0.19	0.14	X	50	X	30	32								
A003	265.3	266.3	1.0	0.42	0.37	X	100	X	20	26								
A003	266.3	267.3	1.0	0.61	0.52	5	100	X	35	65								
A003	267.3	268.3	1.0	1.02	0.93	15	150	X	30	85								
A003	268.3	269.3	1.0	0.67	0.61	5	100	X	25	61								
A003	269.3	270.1	0.8	0.92	0.78	5	150	X	30	74								
A003	270.1	271.1	1.0	0.89	0.76	35	100	10	70	135								
A003	271.1	272.1	1.0	2.98	2.68	70	350	15	90	210								
A003	272.1	273.1	1.0	1.06	0.94	20	150	X	25	105								

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A003	273.1	274.1	1.0	2.83	2.35	40	350	15	80	260								
A003	274.1	275.1	1.0	0.34	0.24	10	50	X	35	46								
A003	275.1	276.1	1.0	0.26	0.13	X	100	X	40	20								
A003	276.1	277.1	1.0	0.35	0.24	X	100	X	35	28								
A003	277.1	278.1	1.0	0.90	0.73	5	200	X	45	67								
A003	278.1	279.1	1.0	1.31	1.31	40	250	X	185	500								
A003	**279.1**	**280.0**	**0.9**	**1.63**	**0.77**	**92**	**640**	**X**	**250**	**13600**	**56.3**	**0.82**	**0.87**	**27**	**148**	**11**	**86**	**276**
A004	131.1	132.1	1.0	0.55	0.52	X	120	X	150	50								
A004	132.1	133.9	1.8	0.62	0.56	X	120	X	100	200								
A004	133.9	134.9	1.0	0.50	0.47	X	110	X	100	50								
A004	134.9	136.1	1.2	0.52	0.51	X	110	X	50	X								
A004	136.1	137.3	1.2	0.35	0.38	X	80	X	50	X								
A004	137.3	138.3	1.0	1.19	1.44	50	280	X	50	100								
A004	138.3	139.3	1.0	1.34	1.47	50	270	100	200	100								
A004	139.3	140.3	1.0	1.57	1.85	100	330	150	450	100								
A004	140.3	141.3	1.0	1.04	1.17	X	240	X	150	100								
A004	141.3	142.3	1.0	0.36	0.42	X	100	X	100	50								
A004	142.3	143.3	1.0	1.44	1.68	100	340	X	100	350								
A004	143.3	144.3	1.0	0.71	0.81	X	150	X	100	50								
A004	144.3	145.3	1.0	0.39	0.49	X	90	X	50	X								
A004	145.3	146.3	1.0	1.11	1.46	100	260	X	100	50								
A004	146.3	147.3	1.0	0.28	0.37	X	70	X	50	X								
A004	147.3	148.3	1.0	0.43	0.57	X	110	X	100	50								
A004	148.3	149.3	1.0	0.28	0.42	X	70	X	100	X								
A004	149.3	150.1	0.8	0.19	0.14	X	70	X	1500	100								
A004	150.1	151.1	1.0	0.53	0.79	X	120	X	200	100								
A004	151.1	152.1	1.0	1.38	1.80	100	320	X	150	350								
A004	152.1	153.1	1.0	1.62	1.97	100	370	X	100	1150								
A004	153.1	154.1	1.0	0.61	0.78	X	150	X	100	600								
A004	154.1	155.1	1.0	0.62	0.77	X	150	X	100	750								
A004	155.1	156.1	1.0	1.92	2.31	100	400	X	100	2750								
A004	156.1	157.1	1.0	1.60	1.04	X	750	X	100	11400								
A004	157.1	158.1	1.0	0.19	0.17	X	90	X	50	900								
A004	158.1	159.1	1.0	0.35	0.18	X	230	X	100	2950								
A004	159.1	159.9	0.8	1.00	0.69	X	500	X	150	7300								
A004	159.9	160.9	1.0	0.48	0.32	X	280	X	50	2950								
A004	160.9	161.9	1.0	0.42	0.47	X	120	X	100	500								
A004	161.9	162.9	1.0	0.36	0.41	X	90	X	100	50								
A004	162.9	163.9	1.0	0.63	0.86	50	150	X	100	150								
A004	163.9	164.9	1.0	1.88	2.71	150	400	X	100	250								
A004	164.9	165.9	1.0	0.55	0.61	X	300	X	50	2000								
A004	165.9	166.9	1.0	0.40	0.30	X	330	X	100	2250								
A004	166.9	167.9	1.0	0.25	0.33	X	100	X	100	100								
A004	167.9	168.9	1.0	0.41	0.57	50	110	X	100	600								
A004	168.9	169.9	1.0	0.71	0.25	X	330	X	100	6750								
A004	169.9	170.9	1.0	1.06	1.09	50	240	X	100	5750								
A004	170.9	172.0	1.1	0.84	0.47	X	300	X	100	7300								
A004	172.0	173.1	1.1	0.49	0.24	X	290	X	150	5050								
A004	173.1	174.9	1.8	0.22	1.50	150	80	X	50	100								
A004	174.9	175.9	1.0	0.23	2.52	300	110	X	50	X								
A004	175.9	176.9	1.0	0.81	4.88	700	320	X	X	X								
A004	176.9	177.9	1.0	0.41	2.74	400	190	X	X	X								
A004	177.9	178.9	1.0	0.38	2.55	550	220	X	50	300								
A004	178.9	179.9	1.0	0.61	4.11	800	360	X	100	200								

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A004	179.9	180.9	1.0	0.24	2.60	350	150	X	250	50								
A004	180.9	182.2	1.3	0.15	2.09	1350	190	X	100	X								
A004	182.2	183.2	1.0	0.15	1.10	1150	170	X	200	50								
A004	183.2	184.2	1.0	0.22	1.62	700	220	X	100	500	53.1	0.67	1.17	150	212	5	126	1215
A006	436.0	437.5	1.5	0.47	0.87	145	170	X	130	X	1.5	0.47	0.87	145	170	X	130	X
A007	88.4	89.4	1.0	0.19	0.14	X	28	X	265	1090								
A007	89.4	90.7	1.3	0.61	0.20	X	64	X	250	2920	2.3	0.37	0.17	0	48	0	257	2124
A008	357.3	358.3	1.0	1.04	1.26	36	230	X	1940	X								
A008	358.3	359.3	1.0	0.67	0.68	38	145	X	265	X								
A008	359.3	360.6	1.3	1.15	1.20	30	225	X	50	X								
A008	360.6	361.6	1.0	1.32	1.83	44	325	X	30	X								
A008	361.6	362.6	1.0	0.72	1.03	X	200	60	520	X								
A008	362.6	363.6	1.0	0.34	0.55	X	115	105	455	X								
A008	363.6	364.6	1.0	0.36	0.43	26	120	X	200	X								
A008	364.6	365.6	1.0	0.35	0.39	X	120	X	120	X								
A008	365.6	366.6	1.0	0.32	0.35	X	110	X	155	X								
A008	366.6	367.6	1.0	0.32	0.37	X	105	X	48	X								
A008	367.6	368.6	1.0	0.85	1.09	X	210	X	62	X								
A008	368.6	369.6	1.0	1.64	2.00	34	390	X	52	X								
A008	369.6	370.4	0.8	0.97	1.20	X	240	X	78	X	13.1	0.78	0.95	17	195	13	303	0
A009	129.3	130.3	1.0	0.19	0.23	115	115	50	145	145								
A009	130.3	131.3	1.0	0.38	0.08	X	84	X	34	2590								
A009	131.3	132.3	1.0	0.80	0.09	X	100	X	X	7390	3.0	0.46	0.13	38	100	17	60	3375
A011	283.9	285.1	1.2	9.79	15.40	1770	2540	X	X	X								
A011	285.1	286.1	1.0	3.20	6.55	615	840	X	44	X								
A011	286.1	287.2	1.1	2.24	4.00	310	870	X	X	2450								
A011	287.2	287.6	0.4	5.86	9.40	820	1780	X	X	1260								
A011	287.6	288.6	1.0	2.04	3.80	260	770	X	X	300								
A011	288.6	289.6	1.0	1.54	2.85	165	615	X	X	X								
A011	289.6	290.7	1.1	1.90	3.35	185	645	X	X	110								
A011	290.7	291.7	1.0	1.08	2.30	125	565	X	70	610								
A011	291.7	292.7	1.0	2.32	4.15	325	795	X	X	1020								
A011	292.7	293.7	1.0	1.79	3.45	280	515	X	30	650								
A011	293.7	294.7	1.0	1.46	3.35	225	445	X	52	320								
A011	294.7	295.7	1.0	2.04	4.20	280	605	X	X	230								
A011	295.7	296.7	1.0	1.53	3.70	275	460	X	X	130								
A011	296.7	297.7	1.0	2.08	4.45	425	580	X	28	X								
A011	297.7	298.7	1.0	1.29	2.90	315	395	X	X	X								
A011	298.7	299.7	1.0	1.55	3.95	375	490	X	26	X								
A011	299.7	300.7	1.0	3.09	7.00	655	890	X	70	X								
A011	300.7	301.9	1.2	3.38	8.60	675	1110	X	X	X	18.0	2.65	5.18	451	818	0	18	366
A012	391.9	393.3	1.4	4.25	7.75	415	1140	X	53	24								
A012	393.3	394.6	1.3	2.45	4.25	230	705	X	66	X								
A012	394.6	395.6	1.0	0.50	0.80	59	140	X	31	X								
A012	395.6	396.9	1.3	1.07	1.67	94	290	X	59	X	5.0	2.21	3.87	212	606	0	54	7
A013	391.9	392.9	1.0	3.35	4.53	218	770	X	258	140								
A013	392.9	393.9	1.0	2.18	3.13	165	773	X	205	X	2.0	2.77	3.83	192	772	0	232	70
A018	214.4	215.4	1.0	2.05	2.35	78	415	X	245	170								
A018	215.4	216.4	1.0	1.47	1.90	56	290	X	250	36								
A018	216.4	217.4	1.0	1.56	1.84	44	275	X	255	360								
A018	217.4	218.2	0.8	1.11	1.15	42	200	X	290	565								
A018	218.2	219.2	1.0	0.92	0.94	42	165	X	165	140								
A018	219.2	220.2	1.0	1.61	1.35	34	515	X	170	4750								
A018	220.2	221.2	1.0	1.27	1.05	18	355	X	275	2860								

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A018	221.2	222.2	1.0	0.84	0.82	18	230	X	170	995								
A018	222.2	223.4	1.2	0.80	0.83	X	160	X	135	X								
A018	223.4	224.4	1.0	0.33	0.29	X	84	X	145	X								
A018	224.4	225.9	1.5	0.84	0.87	24	185	X	230	X								
A018	225.9	227.0	1.1	0.87	0.81	X	200	X	215	56								
A018	227.0	228.0	1.0	1.42	1.47	X	310	X	180	X								
A018	228.0	229.0	1.0	1.97	2.10	X	425	X	215	48								
A018	229.0	230.0	1.0	0.22	0.19	X	80	X	50	X								
A018	230.0	231.0	1.0	0.31	0.32	X	100	X	48	X								
A018	231.0	232.0	1.0	0.67	0.74	X	180	X	74	X								
A018	232.0	233.0	1.0	1.01	1.14	16	245	X	110	X								
A018	233.0	234.0	1.0	0.49	0.58	X	140	X	42	X								
A018	234.0	235.0	1.0	0.35	0.49	X	110	X	82	X								
A018	**235.0**	**236.0**	**1.0**	**0.94**	**1.07**	**X**	**285**	**X**	**210**	**X**	**21.6**	**1.00**	**1.04**	**17**	**234**	**0**	**170**	**457**
A019	241.8	243.0	1.2	1.62	2.75	135	435	65	285	1330								
A019	243.0	244.0	1.0	2.20	3.00	140	550	60	360	360								
A019	244.0	245.0	1.0	1.85	1.98	62	405	65	270	150								
A019	245.0	246.0	1.0	3.70	3.95	175	805	70	370	270								
A019	246.0	247.0	1.0	3.95	4.55	220	895	55	355	280								
A019	247.0	248.0	1.0	2.90	3.50	170	685	65	305	220								
A019	248.0	249.0	1.0	2.40	2.85	98	585	50	270	250								
A019	249.0	250.0	1.0	1.82	2.40	115	455	65	265	230								
A019	250.0	251.0	1.0	1.57	1.91	82	375	70	215	180								
A019	251.0	252.0	1.0	0.98	1.22	40	255	55	185	160								
A019	252.0	253.0	1.0	0.61	0.70	X	170	X	145	160								
A019	253.0	254.0	1.0	0.76	0.79	X	295	70	180	820								
A019	254.0	255.0	1.0	0.99	0.98	X	265	X	140	190								
A019	255.0	256.6	1.6	0.66	0.68	X	190	50	120	170								
A019	256.6	257.7	1.1	0.38	0.37	X	110	90	210	140								
A019	257.7	258.7	1.0	0.40	0.40	X	120	55	170	110								
A019	258.7	259.7	1.0	0.64	0.57	X	195	X	240	200								
A019	259.7	260.7	1.0	0.46	0.41	X	150	55	160	150								
A019	260.7	261.7	1.0	1.07	0.98	X	275	65	235	130								
A019	261.7	262.6	0.9	0.89	0.94	40	225	90	185	160								
A019	262.6	263.6	1.0	0.31	0.32	30	105	X	195	X								
A019	263.6	265.0	1.4	0.72	0.68	X	155	60	220	150								
A019	**265.0**	**266.4**	**1.4**	**1.32**	**1.38**	**28**	**275**	**X**	**195**	**160**	**24.6**	**1.37**	**1.59**	**56**	**339**	**50**	**227**	**263**
A020	**225.8**	**227.0**	**1.2**	**0.40**	**0.46**	**X**	**68**	**X**	**240**	**120**	**1.2**	**0.40**	**0.46**	**X**	**68**	**X**	**240**	**120**
A021	376.0	377.0	1.0	1.18	1.55	30	325	X	150	X								
A021	**377.0**	**378.0**	**1.0**	**0.47**	**0.56**	**X**	**165**	**X**	**125**	**X**	**2.0**	**0.83**	**1.05**	**15**	**245**	**0**	**138**	**0**
A023	356.7	357.7	1.0	0.02	0.19	640	120	X	190	X								
A023	357.7	358.7	1.0	0.07	0.10	175	110	X	110	X								
A023	358.7	359.7	1.0	0.23	0.20	X	135	X	175	X								
A023	359.7	360.7	1.0	0.57	1.03	100	230	X	185	X								
A023	360.7	361.4	0.7	1.26	2.55	140	360	X	165	X								
A023	361.4	362.5	1.1	0.43	0.85	50	175	X	150	X								
A023	**362.5**	**363.6**	**1.1**	**1.08**	**1.54**	**145**	**325**	**X**	**155**	**X**	**6.9**	**0.50**	**0.86**	**178**	**202**	**0**	**161**	**0**
averages			**arithmetic**	**1.05**	**1.50**	**106**	**275**	**11**	**137**	**582**	**210.6**	**1.04**	**1.50**	**107**	**273**	**11**	**135**	**562**
correlation coefficients with % Ni				**1.00**	**0.90**	**0.47**	**0.94**	**-0.19**	**0.07**	**0.05**								

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bhid	from	to	length	Ni	S	Cu	Co	Pb	Zn	As	each hole	Ni	S	Cu	Co	Pb	Zn	As
	m	m	m	%	%	ppm	ppm	ppm	ppm	ppm	length	%	%	ppm	ppm	ppm	ppm	ppm
A014	254.7	255.7	1.0	6.10	6.65	285	1020	X	715	1340								
A014	255.7	256.7	1.0	0.88	0.99	46	175	X	490	X								
A014	256.7	257.7	1.0	2.50	3.70	195	445	X	355	X								
A014	257.7	258.7	1.0	0.63	0.92	X	110	X	195	X								
A014	258.7	259.7	1.0	1.14	2.10	52	235	X	180	X								
A014	259.7	260.7	1.0	1.37	1.79	52	255	X	245	X								
A014	260.7	261.7	1.0	1.66	2.65	115	330	X	315	150								
A014	261.7	262.7	1.0	1.44	3.40	235	310	X	255	210								
A014	262.7	264.0	1.3	0.71	1.86	115	165	X	185	310								
A014	264.0	265.3	1.3	0.90	2.35	160	220	X	230	1020								
A014	265.3	266.3	1.0	1.01	2.95	210	290	X	365	740								
A014	266.3	267.3	1.0	0.88	2.90	275	265	X	305	390								
A014	267.3	268.3	1.0	1.71	4.75	390	470	X	330	940								
A014	268.3	269.3	1.0	0.94	2.85	205	260	X	260	X								
A014	269.3	270.3	1.0	0.95	3.40	200	260	X	225	300								
A014	270.3	271.3	1.0	0.85	2.70	165	230	X	205	270								
A014	271.3	272.3	1.0	0.80	2.55	200	195	X	220	700								
A014	272.3	273.3	1.0	0.88	2.45	170	205	X	260	950								
A014	273.3	274.3	1.0	0.85	2.90	175	220	X	245	X								
A014	274.3	275.3	1.0	0.89	3.15	130	235	X	185	110								
A014	275.3	277.0	1.7	0.53	1.84	96	140	X	135	230								
A014	**277.0**	**278.0**	**1.0**	**0.61**	**1.64**	**78**	**115**	**X**	**135**	**290**	**23.3**	**1.24**	**2.71**	**159**	**273**	**0**	**268**	**365**
A026	224.5	225.5	1.0	1.98	1.99	48	385	X	155	230								
A026	225.5	226.6	1.1	0.14	0.11	X	62	X	155	X								
A026	226.6	228.0	1.4	0.65	0.65	X	185	X	205	650								
A026	228.0	229.0	1.0	0.10	0.11	X	76	X	250	X								
A026	229.0	230.0	1.0	2.75	4.95	430	650	X	190	X								
A026	230.0	231.0	1.0	1.19	2.35	84	240	X	160	140								
A026	231.0	232.0	1.0	1.31	2.40	72	265	X	150	X								
A026	232.0	233.0	1.0	1.78	3.50	100	315	X	180	310								
A026	233.0	234.0	1.0	1.33	2.80	80	215	X	120	X								
A026	234.0	235.0	1.0	1.07	2.35	62	190	X	165	X								
A026	235.0	236.0	1.0	1.17	1.37	X	170	X	125	470								
A026	236.0	237.0	1.0	1.07	1.30	26	170	X	160	190								
A026	237.0	238.0	1.0	1.08	2.00	84	210	X	145	120								
A026	238.0	239.0	1.0	1.39	3.20	185	310	X	145	970								
A026	239.0	240.0	1.0	0.91	2.05	130	215	X	165	1330								
A026	240.0	241.0	1.0	1.25	3.10	255	345	X	235	1760								
A026	241.0	242.0	1.0	0.89	2.35	165	260	X	195	1570								
A026	242.0	243.0	1.0	0.76	2.35	170	240	X	180	680								
A026	243.0	244.2	1.2	0.57	1.75	140	175	X	180	760								
A026	244.2	245.4	1.2	0.48	1.48	120	160	X	145	290								
A026	245.4	246.4	1.0	0.23	0.42	32	64	X	125	X								
A026	246.4	247.6	1.2	0.34	0.70	48	105	X	145	X								
A026	247.6	248.8	1.2	1.65	2.85	130	335	X	125	1640								
A026	248.8	250.0	1.2	0.27	0.37	X	58	X	120	X								
A026	250.0	251.0	1.0	0.21	0.37	X	56	X	115	X								
A026	251.0	252.0	1.0	0.28	0.46	X	64	X	120	X								
A026	252.0	253.0	1.0	0.66	0.99	32	140	X	170	X								
A026	253.0	254.0	1.0	0.22	0.41	X	58	X	120	130								
A026	254.0	255.2	1.2	0.39	0.55	26	96	X	145	220								
A026	255.2	256.2	1.0	0.55	0.85	30	135	X	230	160								

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A026	256.2	257.0	0.8	0.34	0.38	X	74	X	110	X								
A026	257.0	258.0	1.0	0.52	0.78	30	130	X	145	X								
A026	258.0	259.0	1.0	0.25	0.33	X	68	X	115	X								
A026	259.0	260.0	1.0	0.19	0.20	X	48	X	96	X								
A026	260.0	261.0	1.0	0.22	0.23	X	60	X	135	X								
A026	261.0	262.0	1.0	0.33	0.32	X	72	X	115	X								
A026	262.0	263.0	1.0	1.79	2.00	76	350	X	320	X								
A026	263.0	264.0	1.0	1.00	1.04	44	200	55	130	X								
A026	264.0	265.0	1.0	1.79	1.87	74	340	X	175	110								
A026	**265.0**	**266.0**	**1.0**	**2.40**	**2.55**	**98**	**450**	**50**	**140**	**180**	**41.5**	**0.88**	**1.48**	**69**	**193**	**3**	**158**	**307**
A026	293.8	295.0	1.2	1.10	1.64	56	265	X	115	X								
A026	295.0	296.0	1.0	0.38	0.54	X	120	X	525	X								
A026	296.0	297.4	1.4	0.67	0.97	36	180	X	325	350								
A026	297.4	298.2	0.8	1.34	1.64	68	265	X	74	X								
A026	298.2	299.0	0.8	0.85	1.14	52	175	X	125	210								
A026	299.0	300.0	1.0	1.96	2.65	155	400	X	200	360								
A026	300.0	301.0	1.0	1.94	2.45	115	375	X	150	420								
A026	301.0	302.0	1.0	1.86	2.25	80	390	X	105	2480								
A026	302.0	303.0	1.0	2.70	3.35	155	465	X	52	250								
A026	303.0	304.0	1.0	0.59	0.93	48	140	X	200	150								
A026	304.0	305.0	1.0	0.89	1.46	42	185	X	250	160								
A026	305.0	306.0	1.0	0.35	0.53	30	80	X	155	X								
A026	306.0	307.0	1.0	0.94	1.37	135	200	X	235	X								
A026	307.0	308.0	1.0	1.49	1.82	92	320	X	220	X								
A026	308.0	309.0	1.0	3.20	3.60	200	680	X	260	140								
A026	309.0	310.0	1.0	1.88	2.05	56	470	X	390	460								
A026	310.0	311.0	1.0	1.45	2.05	175	315	X	270	X								
A026	311.0	312.0	1.0	2.65	3.85	175	665	X	135	290								
A026	312.0	313.0	1.0	0.68	1.44	60	170	X	115	X								
A026	313.0	314.0	1.0	1.09	1.52	94	265	X	120	X								
A026	314.0	315.0	1.0	2.80	4.00	175	665	X	105	110								
A026	315.0	316.0	1.0	2.35	2.90	135	525	X	120	120								
A026	316.0	316.9	0.9	4.05	4.50	195	830	55	130	140								
A026	316.9	318.0	1.1	0.91	1.11	60	175	X	80	X								
A026	318.0	319.0	1.0	0.95	1.36	80	215	X	105	X								
A026	319.0	320.0	1.0	1.25	1.60	70	300	X	88	X								
A026	320.0	321.0	1.0	0.17	0.09	X	56	X	155	X								
A026	321.0	322.0	1.0	0.88	0.74	X	140	X	520	630								
A026	322.0	323.0	1.0	2.25	2.65	50	430	X	165	450								
A026	323.0	324.0	1.0	2.25	3.05	62	425	X	150	X								
A026	324.0	325.0	1.0	0.48	0.70	X	110	X	145	640								
A026	325.0	326.2	1.2	0.28	0.43	X	90	X	180	1090								
A026	326.2	327.2	1.0	0.20	0.18	X	46	X	145	X								
A026	327.2	328.0	0.8	0.19	0.18	X	32	X	175	X								
A026	328.0	329.0	1.0	2.35	1.92	X	245	X	325	680								
A026	329.0	330.0	1.0	1.08	0.85	X	165	X	185	800								
A026	330.0	331.0	1.0	1.55	1.34	32	225	X	225	800								
A026	**331.0**	**331.9**	**0.9**	**1.24**	**1.11**	**X**	**145**	**X**	**170**	**130**	**38.1**	**1.39**	**1.72**	**70**	**286**	**1**	**191**	**293**
averages			**arithmetic**	**1.17**	**1.86**	**90**	**246**	**2**	**195**	**307**	**102.9**	**1.15**	**1.85**	**90**	**246**	**2**	**195**	**315**
correlation coefficients with % Ni				**1.00**	**0.82**	**0.42**	**0.96**	**0.05**	**0.31**	**0.05**								

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Appendix 7

Manual resource estimates

TO File
FROM Mick McKeown
DATE 28 November 2000

SUBJECT AVEBURY PROJECT - MANUAL RESOURCE ESTIMATES

Two manual resource estimates have been made for the North Avebury nickel mineralisation. Both estimates are of the mineralised zone defined for the recent Mineral Resource report (McKeown, 2000).

On a longitudinal projection showing the mineralisation outline, the centres of the drill hole intersections, the horizontal north-south widths of the intersections, and the grades of the intersections, a traditional **polygonal estimate** was made by drawing polygons around the drill hole centres (Figure 1). For each drill hole, the grade and thickness of the intersection were attributed to the area of the surrounding polygon. A tonnage was estimated for each polygon using a bulk density of 3.0 tonnes per cubic metre, and a tonnage weighted grade was estimated for the mineralisation:

bhid	area	horizontal thickness m	tonnes	grade %Ni	grade* tonnes
A007	4600	1.0	13,800	0.37	5,106
A009	4600	3.0	41,400	0.46	19,044
A020	6300	1.0	18,900	0.40	7,560
A004	4700	32.0	451,200	0.67	302,304
A018	5300	20.0	318,000	1.00	318,000
A003	2900	20.0	174,000	0.82	142,680
A011	2000	12.0	72,000	2.65	190,800
A008	5000	12.0	180,000	0.78	140,400
A019	4800	20.0	288,000	1.37	394,560
A023	4200	8.0	100,800	0.50	50,400
A021	5800	1.0	17,400	0.83	14,442
A013	6200	1.5	27,900	2.77	77,283
A006	3900	0.8	9,360	0.47	4,399
A012	5700	3.5	59,850	2.21	132,269
totals			**1,772,610**	**1.02**	1,799,247

Using the polygonal method, the North Avebury mineralisation contains 1.77 million tonnes at 1.02% Ni.

The resource was also estimated using an **isoline method**. On a longitudinal projection showing the mineralisation outline, the centres of the drill hole intersections,

the horizontal north-south widths of the intersections, and the nickel grades of the intersections, grade and thickness contours were drawn by hand (Figures 2 and 3). The area of mineralisation as shown on the longitudinal projection was divided into 50 metre by 50 metre squares, and for each square an average nickel grade and thickness was estimated by eye. From this information, a tonnage was estimated using a bulk density of 3.0 tonnes per cubic metre, and a tonnage weighted grade was estimated for the mineralisation; note that the pattern of 50 metre blocks in the table replicates the pattern on the longitudinal projection:

horizontal north south widths					**totals**
1	1	11	14	8	
1	1	30	22	17	
9	5	16	18	20	
11	9	12	13	17	
8	7	4	4	8	
3	3	3	2	1	
nickel grades					
2.2	1.2	0.4	0.4	0.5	
2.1	1.1	0.7	0.8	0.8	
2.1	0.7	0.7	1.2	1.1	
2.5	0.8	0.8	1.4	1.4	
2.0	1.4	0.9	2.0	1.1	
2.3	2.0	1.2	2.3	1.5	**0.97**
areas					
500	2500	2500	2500	2500	
500	2500	2500	2500	2500	
500	2500	2500	2500	2500	
500	2500	2500	2500	2500	
500	2500	2500	2500	2500	
500	2500	2500	2500	2500	
tonnes of mineralisation					
1500	7500	82500	105000	60000	
1500	7500	225000	165000	127500	
13500	37500	120000	135000	150000	
16500	67500	90000	97500	127500	
12000	52500	30000	30000	60000	
4500	22500	22500	15000	7500	**1,894,500**
tonnes of nickel					
33	90	330	420	300	
32	83	1575	1320	1020	
284	263	840	1620	1650	
413	540	720	1365	1785	
240	735	270	600	660	
104	450	270	345	113	**18,467**

Using the isoline method, the North Avebury mineralisation contains 1.90 million tonnes at 0.97% Ni.

For the South Avebury mineralisation, a **cross-sectional estimate** of the tonnage was made. For each cross-section, the area of each lens was measured and a volume of mineralisation was estimated by assuming a distance of influence half way to each adjacent section, except for the end sections which were terminating sections:

section	north lens ore width	south lens ore width	north lens area square m	south lens area square m	north lens volume cubic m	south lens volume cubic m
354410	30	-	4200	-	94,500	0
354455	30	-	4200	-	126,000	0
354470	34	-	4760	-	107,100	0
354500	46	44	6440	6160	225,400	246,400
354540	44	42	6160	5880	184,800	176,400
354560	42	34	5880	4760	58,800	47,600
totals					**796,600**	**470,400**

Using the cross-sectional method the volume of the South Avebury mineralisation is 1.27 million cubic metres.

661278

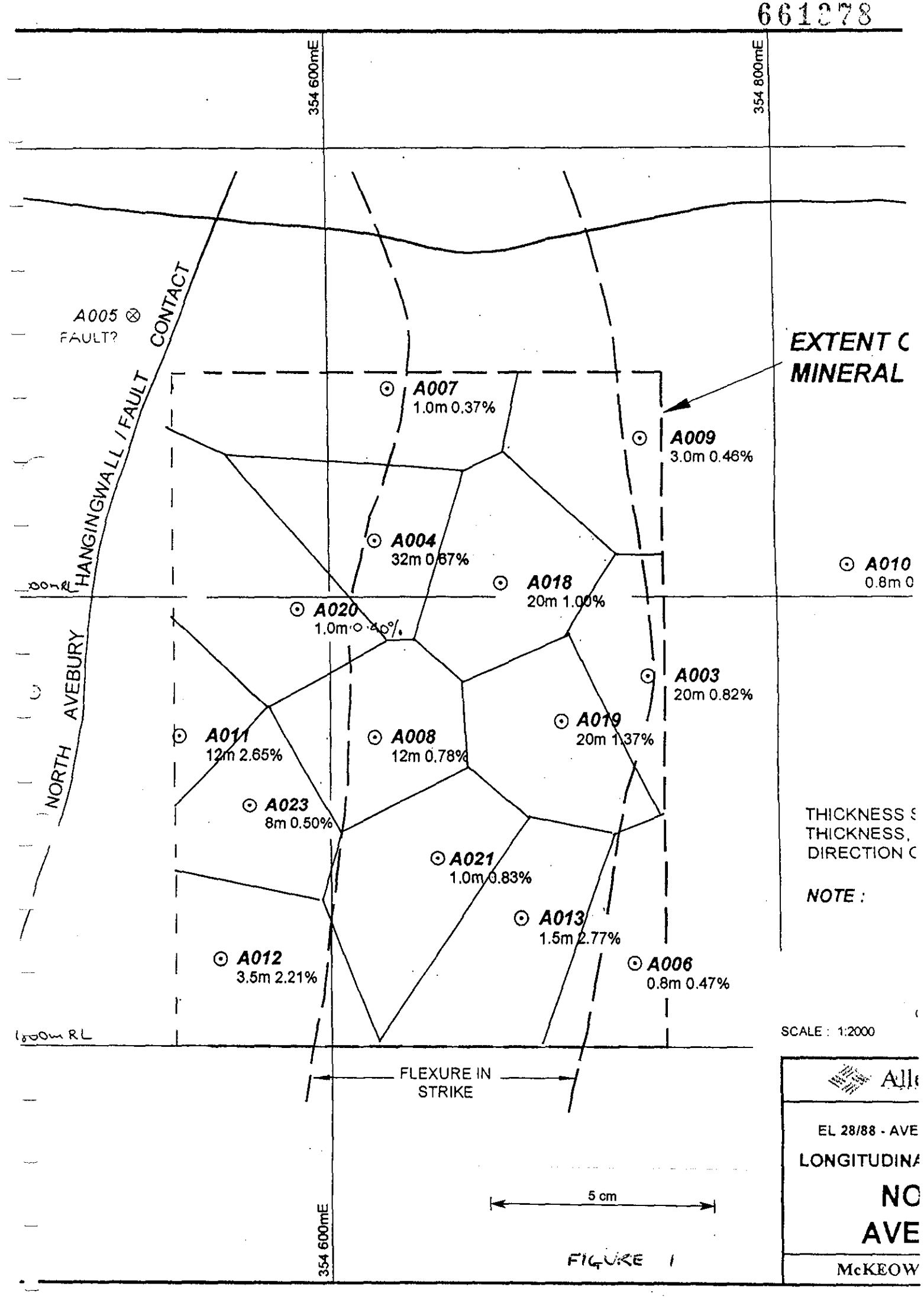

FIGURE 1

661379

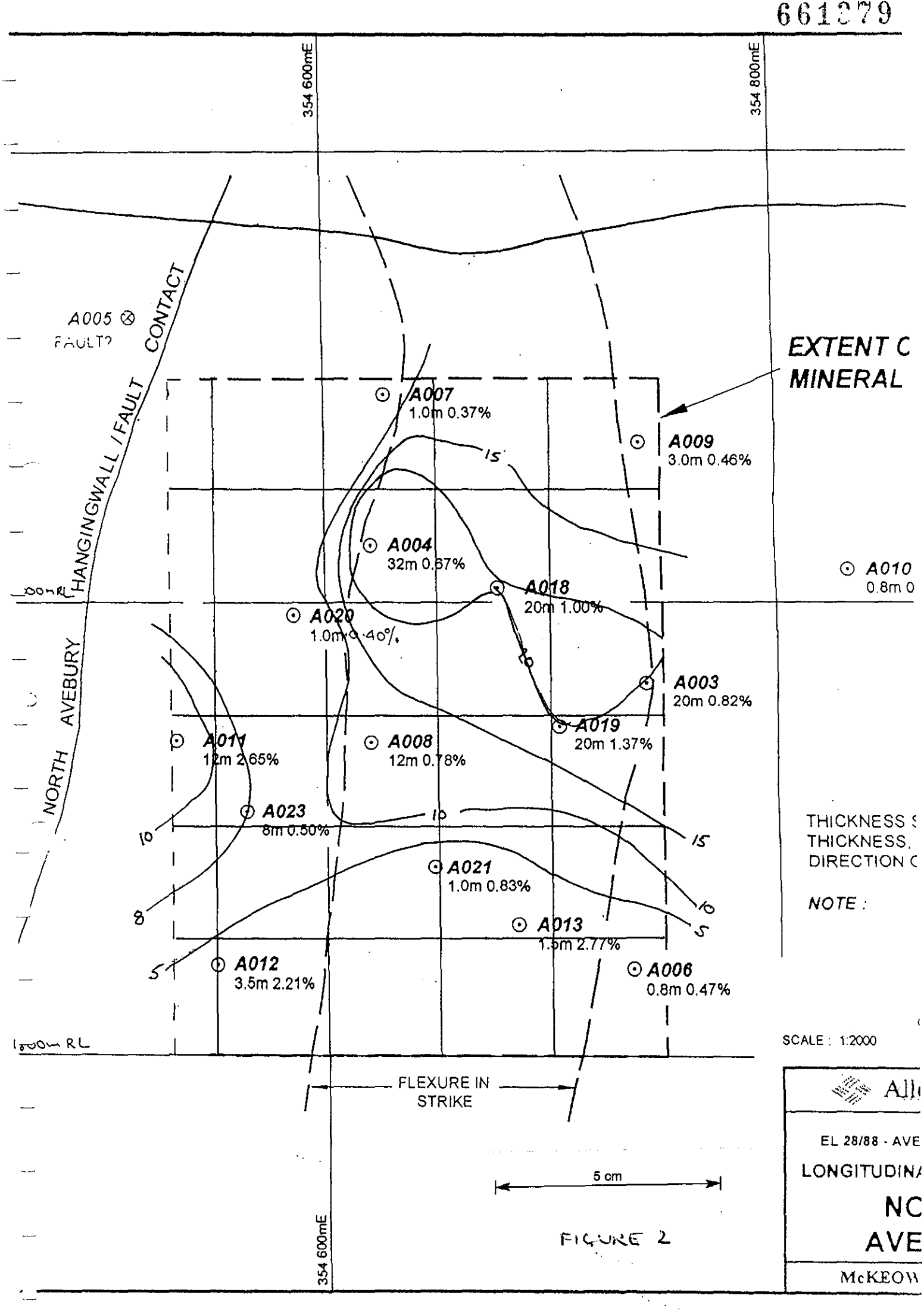

FIGURE 2

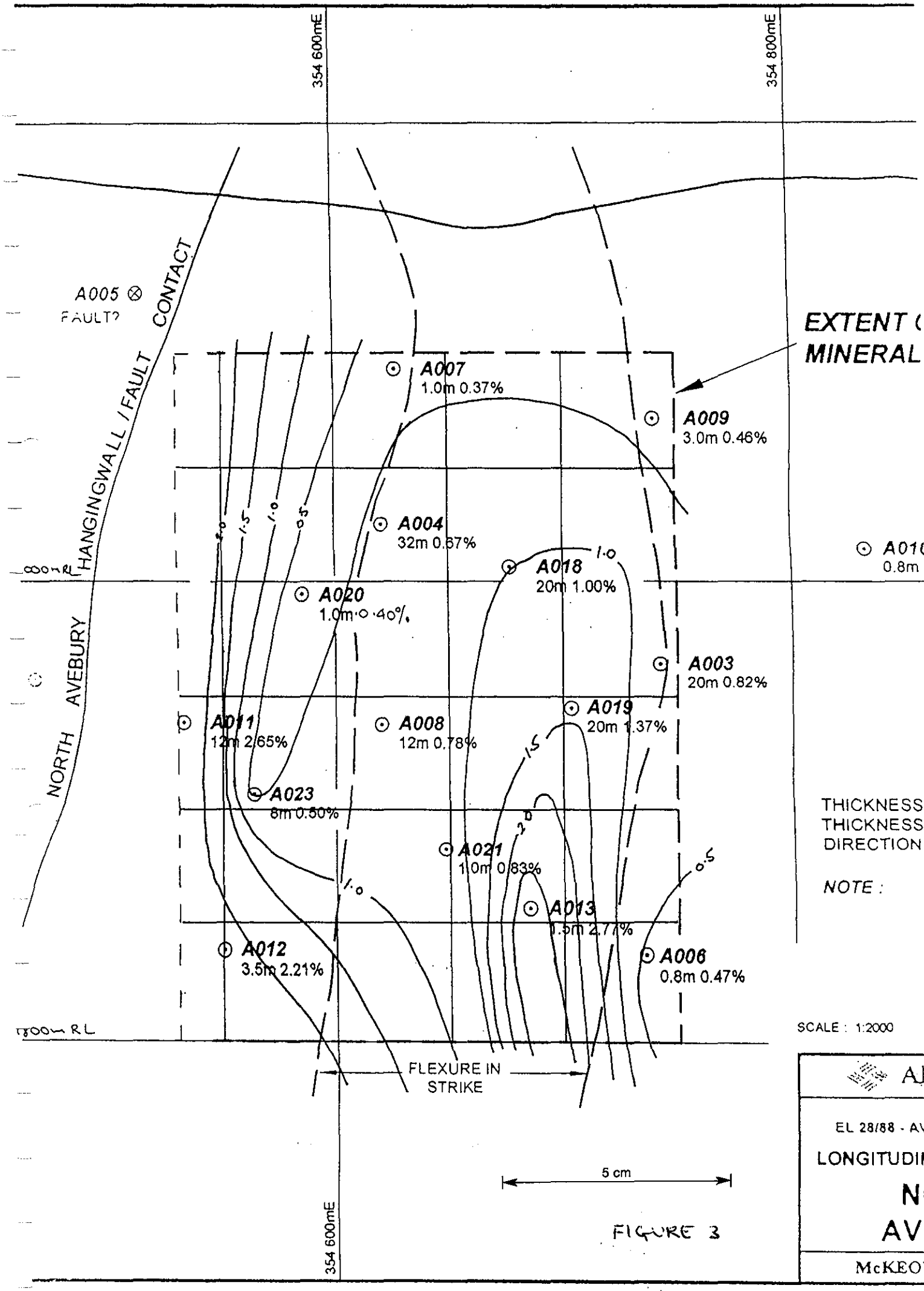
354 600mE
354 800mE
A005 ⊗
FAULT?
NORTH AVEBURY HANGINGWALL / FAULT CONTACT
EXTENT (
MINERAL
A007
1.0m 0.37%
A009
3.0m 0.46%
A004
32m 0.67%
A018
20m 1.00%
A020
1.0m 0.40%
A003
20m 0.82%
A011
12m 2.65%
A008
12m 0.78%
A019
20m 1.37%
A023
8m 0.50%
A021
1.0m 0.83%
A013
1.5m 2.77%
A012
3.5m 2.21%
A006
0.8m 0.47%
1.0
1.5
1.0
0.5
1.5
2.0
1.0
0.5
FLEXURE IN STRIKE
THICKNESS
THICKNESS.
DIRECTION
NOTE :
SCALE : 1:2000
EL 28/88 - AV
LONGITUDIN
NC
AVE
McKEOV
5 cm
354 600mE

FIGURE 3

Appendix 8

Cut-off grade estimate for Renison Bell operation

TO File
FROM Mick McKeown
DATE 15 December 2000

SUBJECT RENISON BELL MINE - ESTIMATED CUT-OFF GRADE

The cut-off grade calculated here is the grade which will pay for all variable costs.

The table outlines the calculation of a cut-off grade for the Renison Bell tin mine, based on recently published data, the sources of which are indicated. The total cash cost includes the following charges: mining, milling, administration, environmental, site specific corporate charges, inventory charges, refining costs and transportation costs and royalties, that is, all the variable costs associated with the operation.

inputs			
ore mined	174,294	tonnes	Murchison Ltd Quarterly Report, 30 Sep 2000
ore milled	173,204	tonnes	Murchison Ltd Quarterly Report, 30 Sep 2000
tin produced	1,978	tonnes	Murchison Ltd Quarterly Report, 30 Sep 2000
cash cost	7,856	$A per tonne	Murchison Ltd Quarterly Report, 30 Sep 2000
cash cost	4,511	$US per tonne	Murchison Ltd Quarterly Report, 30 Sep 2000
tin price	10,702	$A per tonne	Murchison Ltd Quarterly Report, 30 Sep 2000
mill recovery	70.1	%	Murchison Ltd Quarterly Report, 30 Sep 2000
calculations			
ore treated	173,749	tonnes	(ore mined + ore milled)/2
total production cost	15,539,168	$A	tin produced*production cost
unit production cost	89	$A per tonne treated	total production cost/ore treated
tin content for cost recovery	0.84	%Sn	(unit production cost/tin price)*100
calculated cut-off grade	1.19	%Sn	tin content for cost recovery*100/mill recovery

661283

Quarterly Report
30 September 2000

Renison Bell Tin Mine (MUR 100%)

Operations	Sept Qtr
Ore Mined (tonnes)	174,294
Ore Treated (tonnes)	173,204
Ore Grade (% tin)	1.63
Recovery (%)	70.1
Tin in Concentrate Produced (tonnes)	1,978
Tin Sales (tonnes)	1,914
Total Cash Costs (A$/tonne) [1]	7,856
Total Production Costs (A$/tonne) [2]	8,355
Total Cash Costs (US$/tonne)	4,511
Total Production Cost (US$/tonne)	4,797
Tin Price Received ($A/tonne) (including benefit of currency hedging)	10,702

(1) Total Cash Costs include the following charges:

Mining, milling, administration, environmental, site specific corporate charges, inventory changes, refining and transportation costs and royalties.

(2) Total Production Costs include:

Total Cash Costs plus depreciation, amortisation and costs of estimated rehabilitation expenditure, decommissioning and closure costs.

Appendix 9

Drill logs - cover pages

HOLE NO	A001	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,648.1	AMG
NORTHING	5,357,224.9	AMG
RL	2,152.1	m
FINAL DEPTH	532.6	m
HOLE SIZES	0.0m to 3.0m	HW
	3.0m to 53.8m	HQ
	53.8m to 529.7m	NQ
COMMENCED	15-Jan-98	
COMPLETED	22-Jan-98	
LOGGED BY	Mick McKeown	
DRILLED BY	Diamond Drilling Tasmania Pty Ltd	
ANALYSES BY	Amdel Pty Ltd and Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	365	365	-47	Collar bearing and dip are as provided by surveyor.
50	377	390	365	-51	All downhole bearings and dips were surveyed using a downhole camera.
100	346	359	365	-51.5	Down-hole bearings as read were subject to large magnetic errors because this
151	353	366	365	-51	hole was drilled parallel to the hangingwall of the South Avebury serpentinite.
202	361	374	365	-50.5	(see cross section 354 650mE)
250	384	397	365	-50.5	It has been assumed that the collar bearing was maintained to the end of the hole
300	362	375	365	-50.5	(analagous to A006).
367	358	371	365	-51.5	For more information regarding the protocol used to correct the bearings, refer to the
423	355	368	365	-52.5	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
480	366	379	365	-53	
522	366	379	365	-53	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	chert and hornfels	0	243.9												
Cum	Central Avebury mineralisation	243.9	257.6	13.7	1.46	1.33	57	223	30	143	19				
Cum	serpentinite	257.6	529.7												

HOLE NO	A002	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	355,225.2	AMG
NORTHING	5,357,585.6	AMG
RL	2,175.0	m
FINAL DEPTH	307.0	m
HOLE SIZES	0.0m to 26.7m	HW
	26.7m to 54.4m	HQ
	54.4m to 307.0m	NQ
COMMENCED	26-Jan-98	
COMPLETED	03-Feb-98	
LOGGED BY	Mick McKeown	
DRILLED BY	Diamond Drilling Tasmania	
ANALYSES BY	Amdel Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	169	182	182	-52	Collar bearing and dip are as provided by surveyor.
50	169	182	182	-52	All downhole bearings and dips were surveyed using a downhole camera.
100	7	20	182	-52	The bearing as read at 100m was ignored and a sensible bearing was interpolated.
150	170	183	183	-52	The bearing as read at 300m was unreadable and a bearing was extrapolated.
200	166	179	179	-52	
250	166	179	179	-52.5	
300	?	?	179	-53	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Ag ppm	Pt ppb	Pd ppb
CCF	chert and hornfels	0	117.3											
Cum	assay of first metre after contact	117.3	118.3	1.0	0.06	0.07	49	120	25	170	82	<1		
Cum	serpentinite	118.3	157.8											
Cum	assay of last metre before contact	157.8	158.8	1.0	0.09	<0.05	65	5	10	160	26	<1		
CCF	chert and hornfels	158.8	187.6											
Cum	East Avebury mineralisation	187.6	188.0	0.4	0.90	0.72	260	32	60	600	99	<1		
Cum	serpentinite	188.0	307.0											

HOLE NO	A003	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,730.2	AMG
NORTHING	5,357,281.6	AMG
RL	2,163.4	m
FINAL DEPTH	339.0	m
HOLE SIZES	0.0m to 3.0m	HW
	3.0m to 57.0m	HQ
	57.0m to 309.0m	NQ
COMMENCED	15-Jan-99	
COMPLETED	04-Feb-99	
LOGGED BY	Mick McKeown	
DRILLED BY	Diamond Drilling Tasmania	
ANALYSES BY	Anatabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	365	365	*-48*	*Collar bearing and dip are as provided by surveyor.*
52	204	217	365	-51	All downhole bearings and dips were surveyed using a downhole camera.
100	387	400	364	-51	The bearing as read at 52m, 100m and 201m were ignored and sensible
151	350	363	363	-51.5	bearings were interpolated.
201	357	370	363	-52	For more information regarding the protocol used to correct the bearings, refer to the
252	350	363	363	-53	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
301	351	364	364	-53	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	serpentinite	0.0	223.7												
Cum	*North Avebury mineralisation*	223.7	*280.0*	*56.3*	*0.82*	*0.87*	27	*148*	*11*	*86*	*276*				
CCF	hornfels and chert	280.0	339.0												

HOLE NO	A004	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,625.3	AMG
NORTHING	5,357,371.9	AMG
RL	2,146.0	m
FINAL DEPTH	226.3	m
HOLE SIZES	0.0m to 3.0m	HW
	3.0m to 56.0m	HQ
	57.0m to 226.3m	NQ
COMMENCED	05-Feb-99	
COMPLETED	08-Feb-99	
LOGGED BY	Mick McKeown	
DRILLED BY	Diamond Drilling Tasmania	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	353	353	-49.5	Collar bearing and dip are as provided by surveyor.
61	351	364	357	-50	All downhole bearings and dips were surveyed using a downhole camera.
112	360	373	359	-50	Down-hole bearings as read have been corrected for local magnetic error by subtracting
160	357	370	363	-51	7 degrees (determined by comparing the true collar bearing as determined by theodolite
211	360	373	366	-50.5	survey with the collar bearing as determined by compass).
					Obviously erratic down-hole bearings have been replaced by interpolation or
					extrapolation to fit the general trend of bearing changes in the hole.
					For more information regarding the protocol used to correct the bearings, refer to the
					memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	44.7												
CCF	serpentinite	44.7	131.1												
Cum	North Avebury mineralisation	131.1	184.2	53.1	0.67	1.17	150	212	5	126	1215				
Cum	serpentinite	184.2	226.3												

HOLE NO	A005	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,520.9	AMG
NORTHING	5,357,467.4	AMG
RL	2,171.3	m
FINAL DEPTH	202.5	m
HOLE SIZES	0.0m to 3.0m	HW
	3.0m to 75.8m	HQ
	75.8m to 202.5m	NQ
COMMENCED	22-Feb-99	
COMPLETED	05-Mar-99	
LOGGED BY	Mick McKeown	
DRILLED BY	Diamond Drilling Tasmania	
ANALYSES BY		

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	354	354	-47.5	Collar bearing and dip are as provided by surveyor.
52	355	368	354	-49	All downhole bearings and dips were surveyed using a downhole camera.
100	356	369	355	-48.5	Down-hole bearings as read have been corrected for local magnetic error by subtracting
151	361	374	355	-47.5	14 degrees (determined by comparing the true collar bearing as determined by theodolite
202	355	368	354	-47.5	survey with the first down-hole camera survey).
					Obviously erratic down-hole bearings have been replaced by interpolation or
					extrapolation to fit the general trend of bearing changes in the hole.
					For more information regarding the protocol used to correct the bearings, refer to the
					memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

The fault intersected between 59.8m and 61.5m occupies the position where both Central Avebury and North Avebury might have been expected, see cross-section 354 500mE.

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Co ppm	Cu ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	59.8												
Fault	see comments	59.8	61.5												
CCF	hornfels and chert	61.5	202.5												

69693

HOLE NO	A006		
COMPANY	ALLEGIANCE MINING NL		
PROJECT	AVEBURY		
EASTING	354,735.8	AMG	
NORTHING	5,357,189.7	AMG	
RL	2,150.0	m	
FINAL DEPTH	481.5	m	
HOLE SIZES	0.0m to 3.0m	HW	
	3.0m to 30.0m	HQ	
	30.0m to 481.5m	NQ	
COMMENCED	26-Mar-99		
COMPLETED	30-Apr-99		
LOGGED BY	Mick McKeown		
DRILLED BY	Diamond Drilling Tasmania		
ANALYSES BY	Analabs Pty Ltd		

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	360	360	-43.5	Collar bearing and dip are as provided by surveyor.
50	349	362	360	-44.5	All downhole bearings and dips were surveyed using a downhole camera.
100	343	356	360	-44.5	Hole apparently maintained collar bearing to at least 150m, probably to 253m;
150	349	362	360	-45	it has been assumed that the collar bearing has been maintained to the end of the
200	366	379	360	-46	hole.
253	354	367	360	-46	For more information regarding the protocol used to correct the bearings, refer to the
300	366	379	360	-46.5	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
370	409	422	360	-48	
420	402	415	360	-48.5	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	98.9												
Cum	Central Avebury mineralisation	98.9	99.9	1.0	0.17	0.16	42	32	385	225	X				
Cum	serpentinite	99.9	436.0												
Cum	North Avebury mineralisation	436.0	437.5	1.5	0.47	0.87	145	170	X	130	X				
CCF	hornfels and chert	437.5	481.5												

0063699

HOLE NO	A007	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,622.7	AMG
NORTHING	5,357,521.4	AMG
RL	2,157.0	m
FINAL DEPTH	251.0	m
HOLE SIZES	0.0m to 3.0m	HW
	3.0m to 251.0m	HQ
		NQ
COMMENCED	19-Nov-99	
COMPLETED	06-Dec-99	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	175	175	-45	Collar bearing and dip are as provided by surveyor.
50	171	184	175	-46	All downhole bearings and dips were surveyed using a downhole camera.
101	184	197	179	-45.5	Down-hole bearings as read have been corrected for local magnetic error by subtracting
150	171	184	184	-46	9 degrees (determined by comparing the true collar bearing as determined by theodolite
200	141	154	186	-47	survey with the first down-hole camera survey).
251	184	197	188	-46	Obviously erratic down-hole bearings have been replaced by interpolation or extrapolation to fit the general trend of bearing changes in the hole.
					For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S ppm	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	31.0												
CCF	hornfels and chert	31.0	88.4												
Cum	North Avebury mineralisation	88.4	90.7	2.3	0.37	0.17	X	48	X	257	2124				
Cum	serpentinite	90.7	215.2												
Cum	Central Avebury mineralisation	215.2	216.2	1.0	0.38	0.47	X	68	X	88	X				
CCF	chert and hornfels	216.2	251.0												

HOLE NO	A008	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,629.3	AMG
NORTHING	5,357,744.1	AMG
RL	2,178.8	m
FINAL DEPTH	575.0	m
HOLE SIZES	0.0m to 32.0m	HQ
	32.0m to 575.0m	NQ
COMMENCED	04-Jan-00	
COMPLETED	15-Jan-00	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	camera bearing as read magnetic	camera bearing as read AMG	Maxibor bearing as applied AMG	camera dip	comments
0	-	178	178	-46	Collar bearing and dip are as provided by surveyor.
50	169	182	178	-45	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	173	186	179	-43	survey instrument.
150	174	187	181	-41.5	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
200	179	192	183	-40	camera dips.
250	178	191	184	-39	For more information regarding the protocol used to correct the bearings, refer to the
300	184	197	186	-40	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
350	188	201	187	-40	
400	119	132	188	-40	
454	177	190	190	-41	
498	186	199	190	-40.5	
560	163	176	190	-39	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	289.1												
CCF	hornfels and chert	289.1	357.3												
Cum	North Avebury mineralisation	357.3	370.4	13.1	0.78	0.95	17	195	13	303	X				
Cum	serpentinite	370.4	444.3												
Cum	Central Avebury mineralisation	444.3	445.3	1.0	0.38	0.46	22	140	X	89	X				
CCF	chert	445.3	464.4												
Cum	South Avebury mineralisation	464.4	478.4	14.0	0.75	0.82	25	173	X	93	<100				
Cum	serpentinite	478.4	575.0												

2363969

HOLE NO	A009		
COMPANY	ALLEGIANCE MINING NL		
PROJECT	AVEBURY		
EASTING	354,736.1	AMG	
NORTHING	5,357,522.6	AMG	
RL	2,164.2	m	
FINAL DEPTH	525.0	m	
HOLE SIZES	0.0m to 5.5m	HW	
	5.5m to 35.0m	HQ	
	35.0m to 525.0m	NQ	
COMMENCED	07-Dec-99		
COMPLETED	21-Dec-99		
LOGGED BY	Mick McKeown		
DRILLED BY	Almac Pty Ltd		
ANALYSES BY	Analabs Pty Ltd		

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	177	177	-45	Collar bearing and dip are as provided by surveyor.
50	170	183	177	-46	All downhole bearings and dips were surveyed using a downhole camera.
100	169	182	176	-45.5	Down-hole bearings as read have been corrected for local magnetic error by subtracting
150	163	176	179	-45.5	6 degrees (determined by comparing the true collar bearing as determined by theodolite
200	161	174	181	-45.5	survey with the first down-hole camera survey).
250	157	170	184	-46	Obviously erratic down-hole bearings have been replaced by interpolation or
300	157	170	186	-46	extrapolation to fit the general trend of bearing changes in the hole.
353	135	148	188	-46.5	For more information regarding the protocol used to correct the bearings, refer to the
404	169	182	190	-46.5	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
450	186	199	193	-46	
500	189	202	196	-46	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	40.9												
CCF	hornfels and chert	40.9	129.3												
Cum	North Avebury mineralisation	129.3	132.3	3.0	0.46	0.13	38	100	17	60	3375				
Cum	serpentinite	132.3	505.4												
Cum	South Avebury mineralisation	505.0	506.4	1.4	0.69	3.45	355	855	X	435	7290				
CCF?	chert, conglomerate, very altered rocks	506.4	525.0												

A009

99 03 69

HOLE NO	A010							
			DOWN-HOLE SURVEYS					
COMPANY	ALLEGIANCE MINING NL			camera	camera	Maxibor		
			at	bearing	bearing	bearing	dip	comments
PROJECT	AVEBURY			as read	as read	as applied		
				magnetic	AMG	AMG		
EASTING	354,835.6	AMG	0	-	360	360	-46	Collar bearing and dip are as provided by surveyor.
NORTHING	5,357,291.0	AMG	50	-	13	358	-47.5	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
RL	2,169.3	m	100	335	348	358	-49	survey instrument.
FINAL DEPTH	373.0	m	150	330	343	358	-49	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
			200	344	357	358	-49	camera dips.
HOLE SIZES	0.0m to 35.8m	triconed	250	347	360	358	-48.5	For more information regarding the protocol used to correct the bearings, refer to the
	35.8m to 69.2m	HQ	298	349	362	360	-47.5	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
	69.2m to 373.0m	NQ	352	351	364	360	-47	
COMMENCED	17-Jan-00							
COMPLETED	28-Jan-00							
LOGGED BY	Mick McKeown							
DRILLED BY	Almac Pty Ltd							
ANALYSES BY	Analabs Pty Ltd							

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	79.2												
Cum	Central Avebury mineralisation	79.2	80.2	1.0	not assayed										
Cum	serpentinite	80.2	206.6												
Cum	North Avebury mineralisation	206.6	208.4	1.8	0.90	2.70	140	175	50	725	270				
CCF	hornfels and chert	208.4	373.0												

69134

HOLE NO	A011	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,545.1	AMG
NORTHING	5,357,311.7	AMG
RL	2,158.8	m
FINAL DEPTH	452.0	m
HOLE SIZES	0.0m to 6.0m	HW
	6.0m to 46.3m	HQ
	46.3m to 452.0m	NQ
COMMENCED	01-Nov-99	
COMPLETED	18-Nov-99	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	351	351	-50.5	Collar bearing and dip are as provided by surveyor.
15	346	359	351	-51	All downhole bearings and dips were surveyed using a downhole camera.
52	349	362	354	-51	Down-hole bearings as read have been corrected for local magnetic error by subtracting
101	350	363	355	-50	8 degrees (determined by comparing the true collar bearing as determined by theodolite
150	356	369	355	-49	survey with the first down-hole camera survey).
200	360	373	359	-48	Down-hole bearings as read were subject to large magnetic errors because this
250	360	373	358	-47.5	hole was drilled parallel to the hangingwall of the South Avebury serpentinite.
302	357	370	359	-48	(see cross section 354 540mE); it has been assumed that bearing changes for most
350	364	377	359	-48	of the hole were similar to those in A006.
400	366	379	363	-48	For more information regarding the protocol used to correct the bearings, refer to the
450	367	380	364	-48.5	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	283.9												
Cum	Central Avebury mineralisation	283.9	301.9	18.0	2.65	5.18	451	818	X	18	366				
CCF	hornfels and chert	301.9	452.0												

HOLE NO	A012	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,559.6	AMG
NORTHING	5,357,742.7	AMG
RL	2,178.5	m
FINAL DEPTH	544.0	m
HOLE SIZES	0.0m to 27.3m	HQ
	27.3m to 544.0m	NQ
COMMENCED	29-Jan-00	
COMPLETED	16-Feb-00	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	camera bearing as read magnetic	camera bearing as read AMG	Maxibor bearing as applied AMG	camera dip	comments
0	-	177	177	-59.5	Collar bearing and dip are as provided by surveyor.
50	169	182	178	-59.5	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	181	194	179	-60	survey instrument.
152	177	190	180	-60	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
203	178	191	183	-60	camera dips.
257	173	186	185	-59	For more information regarding the protocol used to correct the bearings, refer to the
305	184	197	186	-59	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
350	172	185	188	-59	
400	207	220	190	-59	
452	175	188	191	-59	
500	174	187	190	-58.5	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	338.0												
CCF	hornfels and chert	338.0	391.9												
Cum	North Avebury mineralisation	391.9	396.9	5.0	2.21	3.87	212	606	0	54	7				
Cum	serpentinite	396.9	458.0												
Cum	Central Avebury mineralisation	458.0	459.0	1.0	0.33	0.36	28	95	55	57	52				
CCF	altered rock	459.0	504.3												
Cum	South Avebury mineralisation	504.3	506.8	2.5	0.92	1.41	50	257	X	97	X				
Cum	serpentinite	506.8	544.0												

A012

HOLE NO	A013	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,674.0	AMG
NORTHING	5,357,744.5	AMG
RL	2,176.7	m
FINAL DEPTH	545.6	m
HOLE SIZES	0.0m to 44.8m	HQ
	44.8m to 545.6m	NQ
COMMENCED	17-Feb-00	
COMPLETED	02-Mar-00	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	173	173	-58.5	Collar bearing and dip are as provided by surveyor.
59	169	182	173	-60	All downhole bearings and dips were surveyed using a downhole camera.
101	171	184	175	-59	Down-hole bearings as read have been corrected for local magnetic error by subtracting
152	172	185	176	-57.5	9 degrees (determined by comparing the true collar bearing as determined by theodolite
200	166	179	177	-55	survey with the first down-hole camera survey).
250	175	188	179	-53	Obviously erratic down-hole bearings have been replaced by interpolation or
300	176	189	180	-51	extrapolation to fit the general trend of bearing changes in the hole.
350	174	187	178	-48	For more information regarding the protocol used to correct the bearings, refer to the
400	113	126	178	-46	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
450	113	126	179	-46	
500	171	184	180	-44.5	
542	35	48	181	-43.5	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	284.6												
CCF	chert and hornfels	284.6	391.9												
Cum	North Avebury mineralisation	391.9	393.9	2.0	2.77	3.83	192	772	X	232	70				
Cum	serpentinite	393.9	545.6												

HOLE NO	A014		
COMPANY	ALLEGIANCE MINING NL		
PROJECT	AVEBURY		
EASTING	354,427.7	AMG	
NORTHING	5,357,459.0	AMG	
RL	2,180.0	m	
FINAL DEPTH	379.5	m	
HOLE SIZES	0.0m to 2.0m	HW	
	2.0m to 27.2m	HQ	
	27.2m to 379.5m	NQ	
COMMENCED	04-Mar-00		
COMPLETED	16-Mar-00		
LOGGED BY	Mick McKeown		
DRILLED BY	Almac Pty Ltd		
ANALYSES BY	Analabs Pty Ltd		

DOWN-HOLE SURVEYS

at	camera bearing as read magnetic	camera bearing as read AMG	Maxibor bearing as applied AMG	camera dip	comments
0	-	172	172	-45	Collar bearing and dip are as provided by surveyor.
50	307	320	172	-45	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	169	182	171	-44	survey instrument.
154	168	181	172	-42.5	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
200	167	180	171	-41	camera dips.
250	157	170	171	-42.5	For more information regarding the protocol used to correct the bearings, refer to the
300	171	184	170	-43.5	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
350	165	178	171	-44	
379	164	177	171	-43.5	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	158												
CCF	chert and hornfels	158.0	254.7												
Cum	South Avebury mineralisation	254.7	278.0	23.3	1.24	2.71	159	273	0	268	365				
Cum	serpentinite	278.0	303.7												
CCF	chert	303.7	310.8												
Cum	serpentinite	310.8	333.8												
CCF?	chert, conglomerate, very altered rocks	333.8	379.5												

A014

69143

HOLE NO	A014A	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	*354,453.4*	AMG
NORTHING	*5,357,286.9*	AMG
RL	*2,016.0*	m
FINAL DEPTH	280.1	m
HOLE SIZES	239.2m to 280.1m	NQ

DOWN-HOLE SURVEYS

at	camera bearing as read magnetic	camera bearing as read AMG	Maxibor bearing as applied AMG	camera dip	comments
0	-	172	172	-45	Collar bearing and dip are as provided by surveyor.
50	307	320	172	-45	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	169	182	171	-44	survey instrument.
154	168	181	172	-42.5	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
200	167	180	171	-41	camera dips.
280	160	173	171	-41.5	For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENCED	17-Mar-00
COMPLETED	20-Mar-00
LOGGED BY	Mick McKeown
DRILLED BY	Almac Pty Ltd
ANALYSES BY	Analabs Pty Ltd

COMMENTS

Note that 2000m has been added to the actual RL.

This hole was wedged off A014 at 239.2m.
The coordinates given above in italics are the coordinates of the point at 239.2m depth in A014; downhole depths are relative to the collar of A014.

This hole was drilled to acquire drill core for metallurgical testwork and the core has not been split and assayed.

The lithological headings in this log are decribed more fully in the log for A014.

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	chert and hornfels	239.2	254.7												
Cum	South Avebury mineralisation	254.7	267.3	12.6	1.96	3.34	195	395							
Cum	serpentinite	267.3	280.1												

69 Feb 00

HOLE NO	A015	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,907.5	AMG
NORTHING	5,357,752.5	AMG
RL	2,168.0	m
FINAL DEPTH	467.0	m
HOLE SIZES	0.0m to 2.0m	HW
	2.0m to 39.7m	HQ
	39.7m to 467.0m	NQ
COMMENCED	20-Mar-00	
COMPLETED	02-Apr-00	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	174	174	-51.5	Collar bearing and dip are as provided by surveyor.
50	167	180	174	-49	All downhole bearings and dips were surveyed using a downhole camera.
100	167	180	174	-48	Down-hole bearings as read have been corrected for local magnetic error by subtracting
150	167	180	174	-47	6 degrees (determined by comparing the true collar bearing as determined by theodolite
203	171	184	178	-45	survey with the first down-hole camera survey).
250	173	186	180	-43	Obviously erratic down-hole bearings have been replaced by interpolation or
300	176	189	183	-40	extrapolation to fit the general trend of bearing changes in the hole.
350	177	190	184	-39	For more information regarding the protocol used to correct the bearings, refer to the
400	181	194	188	-37	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
450	176	189	192	-36.5	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	414.5												
CCF	hornfels and chert	0.0	414.5												
Cum	North Avebury mineralisation	414.5	415.5	1.0	0.83	1.05	28	170	X	205	X				
Cum	serpentinite	415.5	467.0												

661400

HOLE NO	A016	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,316.4	AMG
NORTHING	5,357,457.4	AMG
RL	2,181.2	m
FINAL DEPTH	355.5	m
HOLE SIZES	0.0m to 2.0m	HW
	2.0m to 27.5m	HQ
	27.5m to 355.0m	NQ
COMMENCED	04-Mar-00	
COMPLETED	14-Mar-00	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	175	175	-45	Collar bearing and dip are as provided by surveyor.
50	169	182	175	-45	All downhole bearings and dips were surveyed using a downhole camera.
100	173	186	179	-44	Down-hole bearings as read have been corrected for local magnetic error by subtracting
150	177	190	183	-42	7 degrees (determined by comparing the true collar bearing as determined by theodolite
200	171	184	184	-40	survey with the first down-hole camera survey).
250	160	173	185	-38	Obviously erratic down-hole bearings have been replaced by interpolation or
297	180	193	186	-36	extrapolation to fit the general trend of bearing changes in the hole.
352	346	359	187	-36.5	For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

The South Avebury serpentinite and mineralisation appears to have been faulted out, probably by a more or less north-south striking fault.

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	227												
CCF	chert and hornfels	227.0	264.2												
Fault	see comments	264.2	265.4												
CCF?	chert	265.4	337.0												
CCF?	conglomeratic chert	337.0	355.5												

HOLE NO	A017	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,431.5	AMG
NORTHING	5,357,362.2	AMG
RL	2,179.4	m
FINAL DEPTH	187.0	m
HOLE SIZES	0.0m to 6.0m	HW
	6.0m to 187.0m	HQ
COMMENCED	15-Apr-00	
COMPLETED	29-Apr-00	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	174	174	-45	Collar bearing and dip are as provided by surveyor.
50	168	181	174	-46	All downhole bearings and dips were surveyed using a downhole camera.
100	174	187	174	-46	Down-hole bearings as read have been corrected for local magnetic error by subtracting
150	167	180	173	-47.5	7 degrees (determined by comparing the true collar bearing as determined by theodolite survey with the first down-hole camera survey).
					Obviously erratic down-hole bearings have been replaced by interpolation or extrapolation to fit the general trend of bearing changes in the hole.
					For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

This hole was stopped just short of the South Avebury serpentinite as projected up from A014.

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Co ppm	Cu ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	chert and hornfels	0.0	187.0												

661102

HOLE NO	A018	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,718.7	AMG
NORTHING	5,357,599.0	AMG
RL	2,166.4	m
FINAL DEPTH	396.9	m
HOLE SIZES	0.0m to 5.5m	HW
	5.5m to 49.8m	HQ
	49.8m to 396.9m	NQ
COMMENCED	01-May-00	
COMPLETED	10-May-00	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac	
ANALYSES BY	Analabs	

DOWN-HOLE SURVEYS

at	*camera* bearing as read magnetic	*camera* bearing as read AMG	*Maxibor* bearing as applied AMG	*camera* dip	comments
0	-	193	193	-47	Collar bearing and dip are as provided by surveyor.
50	181	194	193	-47	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	184	197	194	-46.5	survey instrument.
151	185	198	196	-45	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
202	186	199	197	-43	camera dips.
250	211	224	198	-42	For more information regarding the protocol used to correct the bearings, refer to the
300	71	84	200	-41.5	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
350	188	201	201	-40	
396	185	198	202	-39.5	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from *m*	to *m*	length *m*	Ni %	S %	Cu *ppm*	Co *ppm*	Pb *ppm*	Zn *ppm*	As *ppm*	Au *ppb*	Ag *ppb*	Pt ppb	Pd ppb
CCF	hornfels	0.0	120.6												
CCF	chert and hornfels	120.6	214.4												
Cum	North Avebury mineralisation	214.4	236.0	21.6	1.00	1.04	17	234	X	170	457				
Cum	serpentinite	236.0	298.0												
Cum	Central Avebury mineralisation	298.0	326.0	28.0	0.95	1.05	17	166	X	217	140				
CCF	chert	326.0	349.5												
Cum	South Avebury mineralisation	349.5	358.0	8.5	0.59	0.99	216	143	X	268	1050				
Cum	serpentinite	358.0	396.9												

661A03

HOLE NO	A018A		
COMPANY	ALLEGIANCE MINING NL		
PROJECT	AVEBURY		
EASTING	*354,686.7*	AMG	
NORTHING	*5,357,474.0*	AMG	
RL	*2,032.3*	m	
FINAL DEPTH	241.0	m	
HOLE SIZES	186.2m to 241.0m	NQ	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	193	193	-47	Collar bearing and dip are as provided by surveyor.
50	181	194	193	-47	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	184	197	194	-46.5	survey instrument.
151	185	198	196	-45	To 151m, the downhole bearings used for this hole are Maxibor bearings;
199	184	197	197	-41	bearings after 151m are based on the camera surveys; all dips are downhole camera dips.
240	175	188	198	-41	For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENCED	17-Mar-00
COMPLETED	20-Mar-00
LOGGED BY	Mick McKeown
DRILLED BY	Almac Pty Ltd
ANALYSES BY	analyses from metallurgical testwork at Metcon Laboratories, supervised by John Butler

COMMENTS

Note that 2000m has been added to the actual RL.

This hole was wedged off A018 at 186.2m

The coordinates given above in italics are the coordinates of the point at 186.2m depth in A018; downhole depths are relative to the collar of A018.

This hole was drilled to acquire drill core for metallurgical testwork and the core has not been split and assayed.

The lithological headings in this log are decribed more fully in the log for A018.

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Fe %	Co ppm	Cu ppm
CCF	chert and hornfels	186.2	213.9						
Cum	North Avebury mineralisation	213.9	217.0	3.1	1.43	1.38	17.0	270	48
Cum	North Avebury mineralisation	217.0	221.5	4.5	1.29	0.97	15.7	365	32
Cum	North Avebury mineralisation	221.5	230.5						
Cum	serpentinite	230.5	241.0						

6641041

HOLE NO	A019		
COMPANY	ALLEGIANCE MINING NL		
PROJECT	AVEBURY		
EASTING	354,699.1	AMG	
NORTHING	5,357,598.1	AMG	
RL	2,165.2	m	
FINAL DEPTH	281.7	m	
HOLE SIZES	0.0m to 5.4m	HW	
	5.4m to 33.6m	HQ	
	33.6m to 281.7m	NQ	
COMMENCED	15-May-00		
COMPLETED	21-May-00		
LOGGED BY	Mick McKeown		
DRILLED BY	Almac		
ANALYSES BY	Analabs		

DOWN-HOLE SURVEYS

at	camera bearing as read magnetic	camera bearing as read AMG	Maxibor bearing as applied AMG	camera dip	comments
0	-	175	175	-60.5	Collar bearing and dip are as provided by surveyor.
50	170	183	177	-62	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	171	184	178	-61	survey instrument.
150	173	186	178	-60	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
200	172	185	180	-59	camera dips.
250	304	317	182	-56.5	For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

This hole was stopped in Central Avebury mineralisation.

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	139.4												
CCF	chert and hornfels	139.4	241.8												
Cum	North Avebury mineralisation	241.8	266.4	24.6	1.37	1.59	56	339	50	227	263				
Cum	serpentinite	266.4	272.8												
Cum	nickel mineralisation	272.8	281.7	8.9	1.03	1.15	45	212	17	190	209				

66140 CR

HOLE NO	A019A	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	*354,703.1*	AMG
NORTHING	*5,357,496.7*	AMG
RL	*1,985.0*	m
FINAL DEPTH	279.2	m
HOLE SIZES	206.9m to 279.2m	NQ

DOWN-HOLE SURVEYS

at	camera bearing as read magnetic	camera bearing as read AMG	Maxibor bearing as applied AMG	camera dip	comments
0	-	175	175	-60.5	Collar bearing and dip are as provided by surveyor.
50	170	183	177	-62	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	171	184	178	-61	survey instrument.
150	173	186	178	-60	To 200m, the downhole bearings used for this hole are Maxibor bearings;
200	172	185	180	-59	the bearing at 265m has been extrapolated; all dips are downhole camera dips.
265	192	205	182	-55	For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENCED
COMPLETED

LOGGED BY Mick McKeown

DRILLED BY Almac Pty Ltd

ANALYSES BY analyses from metallurgical testwork at Metcon Laboratories, supervised by John Butler

COMMENTS

Note that 2000m has been added to the actual RL.

This hole was wedged off A019 at 206.9m.
The coordinates given above in italics are the coordinates of the point at 206.9m depth in A019; downhole depths are relative to the collar of A019.

This hole was drilled to acquire drill core for metallurgical testwork and the core has not been split and assayed.

The lithological headings in this log are decribed more fully in the log for A019.

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Fe %	Cu ppm	Co ppm
CCF	chert and hornfels	206.9	240.5						
Cum	North Avebury mineralisation	240.5	244.4	3.9	3.10	4.34	13.8	340	805
Cum	North Avebury mineralisation	244.4	247.4	3.0	3.50	4.24	15.8	200	845
Cum	North Avebury mineralisation	247.4	250.4	3.0	3.53	4.06	15.2	220	825
Cum	serpentinite	250.4	279.2						

661406

HOLE NO	A020	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,607.7	AMG
NORTHING	5,357,623.3	AMG
RL	2,167.0	m
FINAL DEPTH	300.1	m
HOLE SIZES	0.0m to 3.0m	triconed
	3.0m to 30.4m	HQ
	30.4m to 300.1m	NQ
COMMENCED	24-May-00	
COMPLETED	07-Jun-00	
LOGGED BY	Mick McKeown	
DRILLED BY	*Almac Pty Ltd*	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	camera bearing as read magnetic	camera bearing as read AMG	Maxibor bearing as applied AMG	camera dip	comments
0	-	189	189	-52	Collar bearing and dip are as provided by surveyor.
50	160	173	189	-51.5	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
100	183	196	189	-50	*survey instrument.*
150	189	202	189	-48	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
200	190	203	190	-46	camera dips.
250	26	39	190	-45	For more information regarding the protocol used to correct the bearings, refer to the
295	183	196	190	-45	*memo "Avebury project - down-hole surveys"* (McKeown, 12 October 2000).
400	185	198	192	-43.5	
458	135	148	193	-42	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	chert and hornfels	0.0	225.8												
Cum	North Avebury mineralisation	225.8	227.0	1.2	0.40	0.46	X	68	X	240	120				
Cum	serpentinite	227.0	250.2												
Cum	Central Avebury mineralisation	250.2	257.7	7.5	0.79	1.93	23	155	77	441	176				
Cum	chert	257.7	422.3												
Cum	South Avebury mineralisation	422.3	435.5	13.2	0.49	0.57	23	85	4	95	14				
Cum	serpentinite	435.5	465.9												

66740-1

HOLE NO	A021	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,657.2	AMG
NORTHING	5,357,750.6	AMG
RL	2,177.5	m
FINAL DEPTH	507.6	m
HOLE SIZES	0.0m to 36.8m	HQ
	36.8m to 507.6m	NQ
COMMENCED		
COMPLETED		
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	camera bearing as read magnetic	camera bearing as read AMG	Maxibor bearing as applied AMG	camera dip	comments
0	-	179	179	-54	Collar bearing and dip are as provided by surveyor.
54	172	185	179	-54	All downhole bearings and dips were surveyed using a downhole camera and a Maxibor
104	174	187	179	-53	survey instrument.
155	174	187	181	-52	The downhole bearings used for this hole are Maxibor bearings; dips are downhole
203	180	193	183	-51.5	camera dips.
251	180	193	184	-51	For more information regarding the protocol used to correct the bearings, refer to the
305	182	195	186	-50	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
353	179	192	187	-49	
400	196	209	189	-47.5	
450	162	175	190	-48	
500	43	56	191	-48	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	299.4												
CCF	hornfels and chert	299.4	376.0												
Cum	North Avebury mineralisation	376.0	378.0	2.0	0.83	1.05	15	245	X	138	X				
Cum	serpentinite	378.0	507.6												

3696

HOLE NO	A022	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,629.0	AMG
NORTHING	5,357,743.0	AMG
RL	2,179.0	m
FINAL DEPTH	172.9	m
HOLE SIZES	0.0m to 33.2m	HQ
	33.2m to 172.9m	NQ
COMMENCED		
COMPLETED		
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	178	178	-52	Collar bearing and dip are as provided by surveyor.
65	168	181	178	-52	All downhole bearings and dips were surveyed using a downhole camera.
100	170	183	180	-52	Down-hole bearings as read have been corrected for local magnetic error by subtracting
150	170	183	180	-52	3 degrees (determined by comparing the true collar bearing as determined by theodolite
158	171	184	181	-52	survey with the first down-hole camera survey).
172.9	170	183	180	-51	Obviously erratic down-hole bearings have been replaced by interpolation or extrapolation to fit the general trend of bearing changes in the hole. For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

This hole was stopped at 172.9m due to deviation problems.

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S ppm	Co ppm	Cu ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	172.9												

6614709

HOLE NO	A023	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,569.4	AMG
NORTHING	5,357,742.6	AMG
RL	2,179.5	m
FINAL DEPTH	524.6	m
HOLE SIZES	0.0m to 6.6m	HQ
	6.6m to 524.6m	NQ
COMMENCED		
COMPLETED		
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	174	174	-52	Collar bearing and dip are as provided by surveyor.
25	148	161	174	-51	All downhole bearings and dips were surveyed using a downhole camera.
51	170	183	174	-50.5	Down-hole bearings as read have been corrected for local magnetic error by subtracting
75	171	184	175	-50.5	9 degrees (determined by comparing the true collar bearing as determined by theodolite
102	172	185	176	-50	survey with the first down-hole camera survey).
125	175	188	179	-50	Obviously erratic down-hole bearings have been replaced by interpolation or
150	160	173	180	-50	extrapolation to fit the general trend of bearing changes in the hole.
174	177	190	181	-49.5	For more information regarding the protocol used to correct the bearings, refer to the
198	178	191	182	-49	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
225	1	14	184	-49	
250	183	196	187	-48.5	
300	184	197	188	-47.5	
350	192	205	189	-45	
400	186	199	190	-44	
450	186	199	190	-43.5	
500	182	195	186	-41.5	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

formation	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	302.4												
CCF	hornfels and chert	302.4	356.7												
Cum	North Avebury mineralisation	356.7	363.6	6.9	0.50	0.86	178	202	X	161	X				
Cum	serpentinite	363.6	402.2												
Cum	Central Avebury mineralisation	402.2	403.8	1.6	1.43	1.36	40	341	X	103	69				
CCF	chert	403.8	482.4												
Cum	South Avebury mineralisation	482.4	483.4	1.0	0.10	0.25	68	36	50	1300	X				
Cum	serpentinite	483.4	524.6												

A023

HOLE NO	A024	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,465.7	AMG
NORTHING	5,357,753.7	AMG
RL	2,174.5	m
FINAL DEPTH	613.0	m
HOLE SIZES	0.0m to 21.4m	HQ
	21.4m to 613.0m	NQ
COMMENCED		
COMPLETED		
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

COMMENTS

Note that 2000m has been added to the actual RL.

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	180	180	-63	Collar bearing and dip are as provided by surveyor.
11	164	177	180	-62.5	All downhole bearings and dips were surveyed using a downhole camera.
25	172	185	180	-62.5	Down-hole bearings as read have been corrected for local magnetic error by subtracting
50	173	186	181	-62	5 degrees (determined by comparing the true collar bearing as determined by theodolite
75	172	185	180	-62	survey with the first down-hole camera survey).
101	172	185	180	-61.5	Obviously erratic down-hole bearings have been replaced by interpolation or
125	173	186	181	-61	extrapolation to fit the general trend of bearing changes in the hole.
150	187	200	182	-61	For more information regarding the protocol used to correct the bearings, refer to the
175	152	165	183	-61	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
200	177	190	185	-61	
250	181	194	189	-61	
300	182	195	190	-60	
350	184	197	192	-59	
400	185	198	193	-59	
450	187	200	195	-58	
500	186	199	194	-58	
548	187	200	195	-58	
600	180	193	195	-58	

The fault intersected between 395.8m and 396.3m occupies the position where both Central Avebury and North Avebury might have been expected, see cross-section 354 445mE.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

zone	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	0.0	358.0												
CCF	hornfels and chert	358.0	395.8												
Fault	see comments	395.8	396.3												
CCF	hornfels and chert	396.3	548.1												
Cum	South Avebury mineralisation	548.1	550.4	2.3	1.01	1.13	81	217	73	455	457				
Cum	serpentinite	550.4	613.0												

HOLE NO	A024A	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,452.0	AMG
NORTHING	5,357,586.2	AMG
RL	1,869.5	m
FINAL DEPTH	580.7	m
HOLE SIZES	348.5m to 580.7m	NQ
COMMENCED		
COMPLETED		
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

COMMENTS

Note that 2000m has been added to the actual RL.

This hole was wedged off A024 at 348.5m.

The coordinates given above in italics are the coordinates of the point at 348.5m depth in A024; downhole depths are relative to the collar of A024.

The lithological headings in this log are decribed more fully in the log for A024.

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	*180*	*180*	*-63*	*Collar bearing and dip are as provided by surveyor.*
11	164	177	180	-62.5	All downhole bearings and dips were surveyed using a downhole camera.
25	172	185	180	-62.5	Down-hole bearings as read have been corrected for local magnetic error by subtracting
50	173	186	181	-62	5 degrees (determined by comparing the true collar bearing as determined by theodolite
75	172	185	180	-62	survey with the first down-hole camera survey).
101	172	185	180	-61.5	Obviously erratic down-hole bearings have been replaced by interpolation or
125	*173*	*186*	*181*	*-61*	*extrapolation to fit the general trend of bearing changes in the hole.*
150	187	200	182	-61	For more information regarding the protocol used to correct the bearings, refer to the
175	152	165	183	-61	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
200	177	190	185	-61	
250	181	194	189	-61	
300	182	195	190	-60	
358	*185*	*198*	*193*	*-57*	
402	183	196	191	-57	
450	187	200	195	-56	
500	188	201	196	-55	
550	210	223	196	-55	

SIGNIFICANT INTERSECTIONS

zone	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels	348.5	358.0												
CCF	hornfels and chert	358.0	393.0												
Fault	breccia	393.0	393.7												
CCF	hornfels and chert	393.7	548.2												
Cum	South Avebury mineralisation	548.2	550.0	1.8	2.69	3.93	131	483	20	135	663				
Cum	serpentinite	549.4	580.7												

HOLE NO	A025	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	*354,494.6*	AMG
NORTHING	5,357,735.7	AMG
RL	2,177.1	m
FINAL DEPTH	559.4	m
HOLE SIZES	0.0m to 35.0m	HQ
	35.0m to 559.4m	NQ
COMMENCED		
COMPLETED		
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	179	179	-50	Collar bearing and dip are as provided by surveyor.
15	168	181	179	-50	All downhole bearings and dips were surveyed using a downhole camera.
34	170	183	181	-50	Down-hole bearings as read have been corrected for local magnetic error by subtracting
50	171	184	182	-49	2 degrees (determined by comparing the true collar bearing as determined by theodolite
100	174	187	185	-48	survey with the first down-hole camera survey).
152	178	191	189	-47.5	Obviously erratic down-hole bearings have been replaced by interpolation or
200	178	191	189	-47	extrapolation to fit the general trend of bearing changes in the hole.
250	183	196	194	-46.5	For more information regarding the protocol used to correct the bearings, refer to the
299	186	199	197	-45	memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).
350	189	202	200	-44	
400	189	202	200	-43	
450	186	199	203	-41.5	
500	196	209	207	-40	
556	*229*	*242*	*210*	-38	

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

zone	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	293.4												
CCF	North Avebury: serpentinitic zone	293.4	308.5												
CCF	hornfels and chert	308.5	534.0												
Cum	South Avebury mineralisation	534.0	543.0	9.0	0.39	0.70	22	129	7	173	X				
Cum	serpentinite	543.0	559.4												

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HOLE NO	A026	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,509.6	AMG
NORTHING	5,357,453.0	AMG
RL	2,171.0	m
FINAL DEPTH	351.9	m
HOLE SIZES		HW
		HQ
		NQ
COMMENCED	September 2000	
COMPLETED	September 2000	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	181	181	-45	Collar bearing and dip are as provided by surveyor.
57	179	192	181	-46	All downhole bearings and dips were surveyed using a downhole camera.
105	180	193	182	-45.5	Down-hole bearings as read have been corrected for local magnetic error by subtracting
152	179	192	181	-45	11 degrees (determined by comparing the true collar bearing as determined by theodolite
200	181	194	183	-44.5	survey with the first down-hole camera survey).
251	182	195	184	-43.5	Obviously erratic down-hole bearings have been replaced by interpolation or
305	180	193	182	-43	extrapolation to fit the general trend of bearing changes in the hole.
350	179	192	181	-44	For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

The outer contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at the contact between serpentinite and country rock.
The inner contact of each lode intersection (North Avebury, Central Avebury, South Avebury) has been taken at a point known as the 0.5% Ni drop-off; for more information regarding the method used, refer to "Avebury nickel project - resource report" (McKeown, 30 November 2000).

SIGNIFICANT INTERSECTIONS

zone	comment	from m	to m	length m	Ni %	S %	Cu ppm	Co ppm	Pb ppm	Zn ppm	As ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	hornfels and chert	0.0	189.2												
CCF	chert and hornfels	189.2	224.5												
Cum	South Avebury - north lens	224.5	266.0	41.5	0.88	1.48	69	193	3	158	307				
Cum	serpentinite	266.0	293.8												
Cum	South Avebury - south lens	293.8	331.9	38.1	1.39	1.72	70	286	1	191	293				
CCF?	very altered conglomeratic rocks	331.9	351.9												

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HOLE NO	A026A	
COMPANY	ALLEGIANCE MINING NL	
PROJECT	AVEBURY	
EASTING	354,505.6	AMG
NORTHING	5,357,306.2	AMG
RL	2,022.6	m
FINAL DEPTH	333.7m	m
HOLE SIZES	208.8m to 333.7m.	NQ
COMMENCED	September 2000	
COMPLETED	September 2000	
LOGGED BY	Mick McKeown	
DRILLED BY	Almac Pty Ltd	
ANALYSES BY	Analabs Pty Ltd	

DOWN-HOLE SURVEYS

at	bearing as read magnetic	bearing as read AMG	bearing as applied AMG	dip	comments
0	-	181	181	-45	Collar bearing and dip are as provided by surveyor.
57	179	192	181	-46	All downhole bearings and dips were surveyed using a downhole camera.
105	180	193	182	-45.5	Down-hole bearings as read have been corrected for local magnetic error by subtracting
152	179	192	181	-45	11 degrees (determined by comparing the true collar bearing as determined by theodolite
200	181	194	183	-44.5	survey with the first down-hole camera survey).
224	186	199	188	-42.5	Obviously erratic down-hole bearings have been replaced by interpolation or
250	198	211	188	-42.5	extrapolation to fit the general trend of bearing changes in the hole.
300	150	163	188	-41.5	For more information regarding the protocol used to correct the bearings, refer to the memo "Avebury project - down-hole surveys" (McKeown, 12 October 2000).

COMMENTS

Note that 2000m has been added to the actual RL.

This hole was wedged off A026 at 208.8m; the collar coordinates given above are of the point at 208.8m in A026; downhole depths are relative to the collar of A026.

This hole was drilled to acquire drill core for metallurgical testwork and the core has not been split and assayed.

The lithological headings in this log are decribed more fully in the log for A026.

SIGNIFICANT INTERSECTIONS

zone	comment	from m	to m	Ni %	S %	As ppm	Cu ppm	Co ppm	Pb ppm	Zn ppm	Au ppb	Ag ppb	Pt ppb	Pd ppb
CCF	chert and hornfels	208.8	226.3											
Cum	South Avebury - north lens	226.3	269.0											
Cum	serpentinite	269.0	296.1											
Cum	South Avebury - south lens	296.1	327.2											
CCF?	very altered conglomeratic rocks	327.2	333.7											

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DECEMBER 2000

MINERAL RESOURCES
14 FEB 2001
TASMANIA

AVEBURY NICKEL PROJECT
METALLURGICAL SCOPING STUDY

FOR

MINERAL RESOURCES
FILE REF: EL28/88PT5
16 FEB 2001
DOC. REF:
OFFICER | FOR ACTION | FOR INFO
See folio 61F
RESUBMIT TO | DATE

ALLEGIANCE MINING N.L.

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01_4526(7)
Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL*; Newnham Exploration and Mini Newnham, L.A.
EL28/1988
VOL 5 of 10

1. INTRODUCTION

Following the receipt of proposals from Metcon Laboratories, Sydney and AMMTEC, Perth for the completion of a preliminary metallurgical testwork programme, John E. Butler (JEB) obtained approval from Allegiance Mining NL on 28th July 2000 to commission Metcon Laboratories for the duty. Both organisations quoted for similar programmes and almost identical costs, and the final decision was largely based upon JEB's past experience with Metcon Laboratories and their more convenient Sydney location.

The basic testwork schedule included the areas of sample preparation, ore specific gravity, Bond ball mill work index, head assays including multi-element scans, Davis tube magnetic separation of ore and concentrates, roughing/ scavenging flotation tests, cleaner flotation tests, locked cycle flotation evaluation, concentrate and tailings assaying and sizing, and multi-element/penalty element scans on final concentrate and tailings. The programme demanded frequent telephone and facsimile communications between Metcon Laboratories and JEB, and JEB visited the Sydney laboratories on several occasions for testwork participation/discussions and final report editing. On the latter subject, it must be pointed out that JEB did not modify any part of the attached Metcon Laboratories report, but restricted his contribution to confirmation of technical accuracy.

It was originally envisaged that two wedged drillcores would be examined, but when an additional geologically acceptable core became available midway through the testwork period it was submitted for the production/expansion of metallurgical data. This action was largely responsible for an increased volume of flotation tests, a situation which proved beneficial in giving further confidence on the amenability of Avebury nickel sulphide mineralisation to acceptable nickel recovery and concentrate properties.

Several significant results of the metallurgical testwork exercise included the development of a preliminary beneficiation circuit/flowsheet quantification, and estimated capital/operating costs, the latter being of a very approximate nature with a probable ±30% accuracy.

An additional metallurgical requirement of the commission was to make an evaluation of the input and estimated expenditure against a definitive feasibility programme, and comment on the anticipated involvement of external consulting organisations.

2. SIGNIFICANT ESTIMATED DATA

As a result of the findings from the metallurgical testwork programme the following estimates were determined. They are currently of a preliminary nature and subject to possible significant future amendments.

Treatment rate	300,000 dry tonnes ore p.a.
Ore head grade	1.6% Ni
Concentrate grade	22% Ni
Tailings grade	0.19% Ni
Nickel recovery	88.9%
Nickel in ore	4,800 tonnes p.a.
Nickel in concentrate	4,270 tonnes p.a.
Concentrate production	19,400 dry tonnes p.a.
Concentrate moisture	8%
Saleable concentrate	21,100 wet tonnes p.a.
Tailings production	280,600 dry tonnes p.a.
Tailings to storage	Approx. 60% solids
Concentrate size distribution	100% minus 150 microns 98% minus 75 microns 88.7% minus 45 microns
Tailings size distribution	100% minus 150 microns 83.8% minus 75 microns 63.1% minus 45 microns
Capital cost	$18,250,000
Operating cost	$19 per dry tonne ore processed

3. METALLURGICAL TESTWORK/FLOWSHEET DEVELOPMENT

The attached Metcon Laboratories report 00900, dated December 2000, contains a detailed account of the testwork programme evolution, quantitative results/data, technical comments on the findings, and important conclusions. It is not intended to cover the report's contents in depth, but the following tabulation of the conclusions is included as a quick reference guide to the current interpretation of the metallurgical response of Avebury nickel sulphide mineralisation:

1. Of the three drillcores examined AO18A was found to have a significantly higher grinding hardness than AO14A and AO19A. A Bond ball mill work index of 18.4 kWh per tonne of ore was determined with a composite derived from AO18A and AO19A.

2. Only sample AO19A was tested for heavy media separation response, and it proved to be non-beneficial with respect to weight rejection and preconcentration.

3. The three samples were very amenable to flotation treatment, giving extremely acceptable and encouraging nickel recoveries and concentrate quality.

4. A straightforward flotation reagent regime, including sodium isobutyl xanthate collector, copper sulphate as a pentlandite activator, and methyl isobutyl carbinol frother appears adequate, with probably the addition of carboxy methyl cellulose being of value for gangue depression.

5. Flotation circuit optimisation would almost certainly enhance flotation response even further.

6. The high natural pH of flotation feed slurry, at approximately 10.0, was an additional positive property as most nickel sulphide ores require an alkali, such as burnt lime, to adjust the pulp to the required pH level.

7. Although the primary grind size distribution was not optimised it was shown that the 80% minus 75 microns level used in flowsheet development gave improved nickel recoveries as compared to those at an 80% minus 106 microns grind.

8. A simple roughing/scavenging/two stage cleaning flotation circuit appears to be adequate.

9. Regrinding of rougher/scavenger concentrates ahead of cleaning may be beneficial, but has not been included in the preliminary flowsheet.

10. The abovementioned possible flotation reagent addition of carboxy methyl cellulose may be invaluable in minimising the magnesium oxide content of saleable concentrate, this being a compound which at high levels is less than popular in nickel smelter feedstock, and it has been allowed for in the adopted flowsheet.

11. Cobalt recoveries are virtually identical with those for nickel in the flotation system developed.

12. In the locked cycle test, including six cycles and the recycling of cleaner tailings within the system, very satisfactory response was achieved. It was sufficient to give confidence that at a 1.6% Ni ore head grade, a 22% Ni concentrate with a 0.19% tailings is readily achievable.

13. Due to the limited weight availability of suitable concentrate product no settling or filtering tests were undertaken, but from past experience it is obvious that it will not give any operational difficulties.

To assist in the derivation of estimated capital and operating costs, a preliminary quantitative flowsheet was developed after analysis of the flotation testwork data, and this is illustrated in the following three figures:

Figure 1	Simplified process flowsheet
Figure 2	Schematic flowsheet/solids and water balance - comminution circuit
Figure 3	Schematic flowsheet/solids and water balance - flotation and concentrate handling circuit.

A further processing stage which may prove to be of benefit to concentrate quality is the magnetic separation of reground cleaner concentrate; however it has not been considered in the current exercise.

FIGURE 1

SIMPLIFIED PROCESS FLOWSHEET

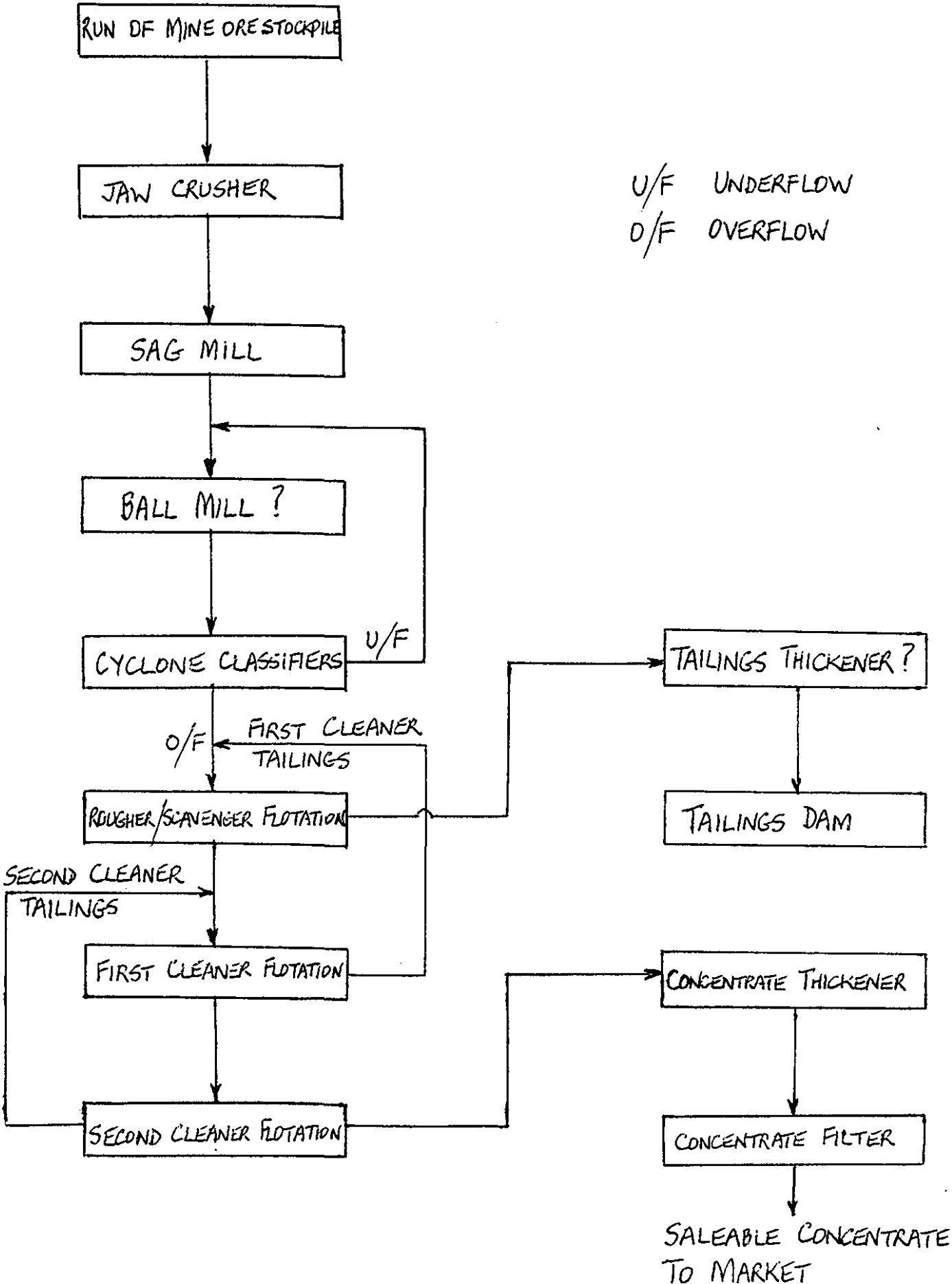


FIGURE 2

SCHEMATIC FLOWSHEET/SOLIDS AND WATER BALANCE

COMMINUTION CIRCUIT

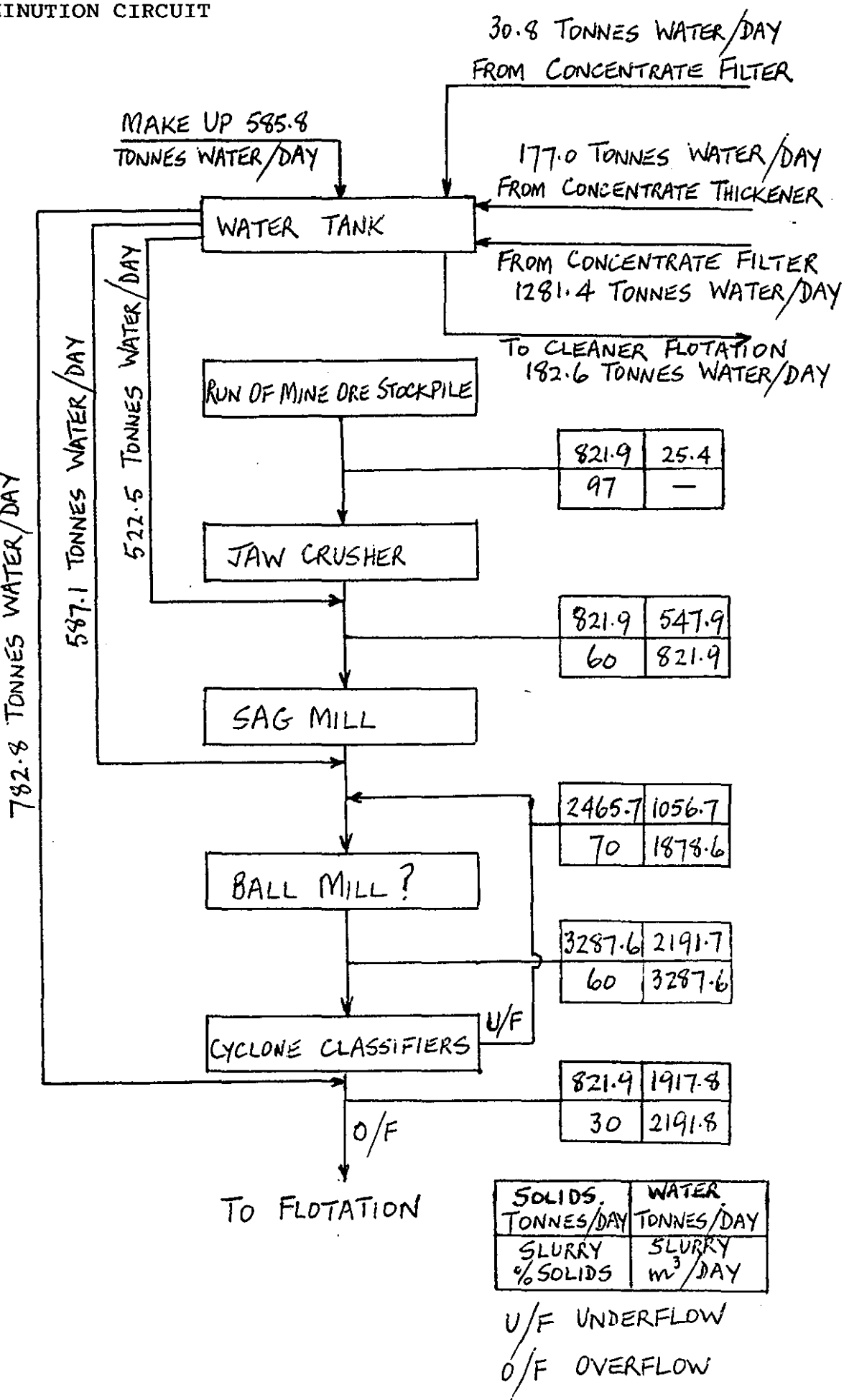
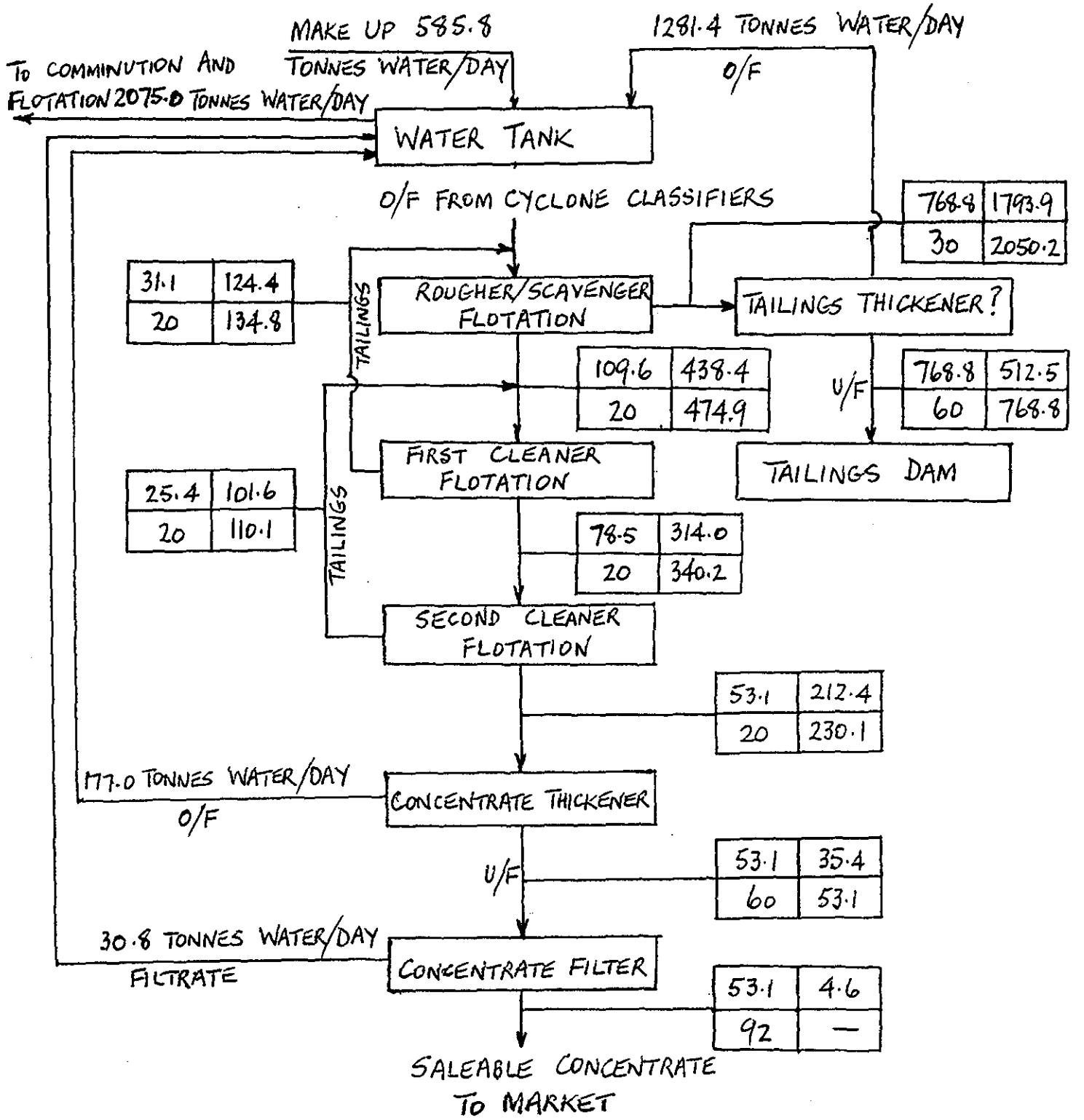


FIGURE 3

SCHEMATIC FLOWSHEET/SOLIDS AND WATER BALANCE

FLOTATION AND CONCENTRATE HANDLING CIRCUIT



SOLIDS TONNES/DAY	WATER TONNES/DAY
SLURRY % SOLIDS	SLURRY m^3/DAY

U/F UNDERFLOW

O/F OVERFLOW

4. ESTIMATED CAPITAL AND OPERATING COSTS

In view of the relatively embryonic nature of the current scoping study, and to minimise expenditure, it was decided to use JEB's available cost input and adapt it to the factored method outlined in the AusIMM "Cost Estimation Handbook for the Australian Mining Industry", published in 1993. This approach was expanded upon in the form of discussions and input from nickel project authorities having up to date experience of the industry.

(a) Capital cost estimate

		$
Major equipment items (Approx. basic cost)		
Jaw crusher		500,000
SAG mill		1,500,000
Ball mill		900,000
Roughing/scavenging flotation machines		300,000
Cleaning flotation machines		100,000
Conveyors/motors/screens/piping		1,000,000
Miscellaneous tanks (conditioners, surge tanks, etc.)		250,000
Concentrate thickener		150,000
Concentrate filter		150,000
Tailings thickener		600,000
Sub-total (A)		5,450,000
Factored items		say
Equipment installation	(A) x 0.21	1,150,000
Piping installation	(A) x 0.14	750,000

Electrical installation	(A) x 0.19	1,050,000
Instrumentation	(A) x 0.08	450,000
Process buildings	(A) x 0.25	1,350,000
Auxiliary buildings	(A) x 0.11	600,000
Plant services (compressors, water pumps, etc.)	(A) x 0.11	600,000
Site preparation	(A) x 0.12	650,000
Sub-total (B)		12,050,000
EPCM	(B) x 0.15	1,800,000
Total installed cost sub-total (C)		13,850,000
Contingency	(C) x 0.10	1,400,000
Tailings dam		3,000,000
Grand total		**18,250,000**

Installed power approximately 1.5MW

(b) Operating cost estimate

Labour		$/annum
Mill Superintendent	1 off	120,000
General Foreman	1 off	90,000
Senior Operators	4 off	320,000
Operators	4 off	280,000
Daycrew/relief/crushing	4 off	280,000
Maintenance Foreman	1 off	90,000
Tradesmen (2 electricians/ instruments, 2 fitters, 1 boilermaker)	5 off	375,000
Junior Metallurgist	**1 off**	**75,000**
Total	21 off	1,630,000

Unit cost $5.43 per dry tonne ore

Power	$/annum
40 kWh per dry tonne ore at 8 cents per kWh	960,000
Unit cost $3.20 per dry tonne ore	
Liners and grinding media	
Liners (jaw crusher, SAG mill and ball mill) 50 tonnes at $7,000/tonne	350,000
Grinding media (SAG mill and ball mill) 450 tonnes at $1,250/tonne	562,500
Total	912,500
Unit cost $3.04 per dry tonne ore	
Reagents	
Copper sulphate 120 tpa @ $1,600/tonne	192,000
Sodium isobutyl xanthate 24 tpa @ $3,050/tonne	73,200
Methyl isobutyl carbinol 18 tpa @ $5,050/tonne	90,900
Carboxyl methyl cellulose 33 tpa @ $4,700/tonne	155,100
Flocculant 30 tpa @ $6,000/tonne	180,000
Total	691,200
Unit cost $2.30 per dry tonne ore	
Maintenance materials	
4% per annum of capital cost sub-total (B)	say 500,000
Unit cost $1.67 per dry tonne ore	

Tailings dam operations	$/annum
Reticulation (piping, cyclones, wall construction, etc.)	450,000
Unit cost $1.50 per dry tonne ore	

Miscellaneous

Includes loader operation (ROM ore), assays, office supplies, environmental monitoring, communications, computing, training, site vehicles, safety/ first aid/firefighting, concentrate bagging, etc.	550,000
Unit cost $1.83 per dry tonne ore	

Estimated overall operating cost of $18.97,

say **$19 per dry tonne ore**

5. CONCLUSIONS AND RECOMMENDATIONS

With the samples examined to date it has been confirmed that they gave an excellent response with respect to nickel recovery and saleable concentrate quality, the high nickel product grades, in excess of 20%, being particularly encouraging. At the present time even the most cautious and critical mining industry professionals would be positively optimistic that the essential confirmatory evaluation with a wide spectrum of additional samples will continue the successful processing trend.

With respect to the further metallurgical requirements, should a definitive feasibility study be undertaken, the following is submitted for future deliberations and consideration:

Stage 1 - Mineralogical evaluation of samples with particular emphasis upon nickel mineralisation, liberation properties and grain size.

Stage 2 - Develop/optimise the flowsheet (comminution and flotation)which will require inputs on grinding characteristics, classification , possible magnetic separation, flotation reagents, regrinding of rougher/scavenger concentrates, etc.

Stage 3 - Evaluate the optimised flowsheet on a wide spectrum of samples representing nickel head grade variations, mineralisation types, mining schedules, etc.

Stage 4 - Treat a relatively large ore/drillcore sample to produce sufficient cleaner concentrate for settling (thickening) and filtration tests, and to provide marketing samples for prospective treatment/refining companies. The large scale tests would also provide a large quantity of tailings for any settling, acid mine drainage and mine backfill evaluations.

JEB's preliminary assessment is that at least 250 kg of ore/drillcore would be required at an estimated laboratory cost of $150,000. An additional expenditure would be incurred against consulting, probably of the order of $50,000. Of course, larger costs become expendable when the engineering/construction fraternity are engaged, and although there is a proliferation of such helpful organisations no attempt has been made to quantify a likely cost in the current exercise.

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Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL*; Newnham Exploration and Mini
Newnham, L.A. EL28/1988 Vol 6 of 10

Avebury nickel project

Marketing scoping study

Report for Allegiance Mining NL
November 2000

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01_4526 (E)
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FILE REF: EL28/98PT5
16 FEB 2001
DOC. REF:
OFFICER | FOR ACTION | FOR INFO
See folio 61E
RESUBMIT TO | DATE

Avebury nickel project

Marketing scoping study

Report for Allegiance Mining NL
November 2000

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Avebury nickel project – marketing scoping study

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1. Introduction

In early 1999, Allegiance Mining NL discovered extensive nickel-sulphide mineralisation near Zeehan, in western Tasmania. By August 2000, Allegiance Mining had identified a substantial high-grade nickel-sulphide resource at Avebury, 10 kilometres west of Zeehan. Allegiance Mining estimates that there is potential to significantly expand Avebury's initial resource base. Allegiance Mining recently commenced a scoping study of an Avebury project that would produce 4,000–4,500 tonnes per year of nickel in concentrate for at least ten years.

Allegiance Mining's goal is to develop a medium-sized underground mining operation at Avebury, processing ore on-site to produce nickel-sulphide flotation concentrate for sale. Because the sale of flotation concentrate would be Avebury's only source of income, Allegiance Mining needs to ensure that it optimises the returns from selling concentrate, including taking advantage of opportunities to obtain the benefits that might be associated with a concentrate sale and purchase agreement.

As part of the scoping study, Allegiance Mining commissioned John O'Shea & Associates to produce this marketing scoping study for Avebury flotation concentrate. The report begins by reviewing the global nickel market, and the past and present arms-length trade in intermediate products – including nickel-laterite ore, nickel-sulphide concentrate and nickel matte. Next, it reviews possible and likely buyers of Avebury concentrate, analyses the key commercial issues in any concentrate sale and purchase agreement, and suggests appropriate marketing parameters to use in the cashflow model for the Avebury scoping study. Finally, it reviews likely opportunities for attracting partnership for Allegiance Mining in investment in Avebury by potential purchasers of concentrate. Concentrate buyers have, on occasion:

- purchased equity in projects
- partly financed project construction
- provided working capital by making early payment for product
- provided technical assistance to projects.

In parallel with the production of this report, John O'Shea & Associates has contacted several likely buyers of Avebury concentrate, asking if they would be interested in purchasing Avebury concentrate. Interested parties have been invited to submit indicative concentrate-purchase terms. Results of these enquiries will be presented in a supplementary report.

2. Executive summary

Allegiance Mining NL is conducting a scoping study of its Avebury nickel-sulphide project to assess the economic attractiveness of an underground mining operation and associated concentrator, producing nickel-sulphide concentrate for sale. Avebury is expected to produce 4,000–4,500 tonnes per year of nickel in high-grade nickel-sulphide concentrate.

Global consumption of refined nickel is expected to reach 1,136,000 tonnes in 2000; global consumption of refined nickel is growing at around 4–5% per year. The additional production from Avebury will have negligible impact on global nickel supply.

Global arms-length trade in nickel-sulphide concentrates has been quite small – less than 50,000 tonnes per year of contained nickel – primarily because there have been very few dedicated custom smelters or refineries seeking to purchase concentrate. In addition, the large, fully integrated nickel producers – including Inco, Falconbridge and WMC – used not be inclined to purchase concentrate that would occupy smelting capacity better used for concentrate produced in-house. The large nickel producers used also to be disinclined to assist smaller companies to bring into being additional nickel concentrate production capacity.

However, in the last few years this position has changed. Although the tonnage of nickel in traded nickel-sulphide concentrate is still less than 50,000 tonnes per year, the development of several small nickel mining operations in Western Australia – including Titan Resources' Radio Hill mine, Jubilee Mines' Cosmos mine and Tectonic Resources' RAV 8 mine, has been characterised by the sale of nickel-sulphide concentrate to either Inco or WMC.

Allegiance Mining should be able to attract competitive offers to purchase Avebury concentrate. The concentrate is high-grade; its chemical composition appears well suited to either smelting or to direct leaching; and arsenic appears to be the only deleterious impurity likely to be of concern to potential buyers. The challenge facing Allegiance Mining is to maximise the benefits it receives from selling concentrate. This report names 17 potential buyers of concentrate, and identifies seven companies considered highly likely to be interested in purchasing Avebury's concentrate.

This report recommends that Allegiance Mining uses, for the Avebury scoping study, the following key marketing parameters: payment, at LME prices, for 97% of the contained nickel in concentrate; treatment and refining charges of 35% of the payable nickel in concentrate; receipt of 90% of invoice value two months following the arrival of concentrate at buyer's works; and receipt of the outstanding amount one month later.

In addition to optimising the financial returns from selling concentrate and to minimising associated working capital requirements, Allegiance Mining can explore opportunities for potential purchasers of concentrate to contribute to the development of Avebury. A concentrate buyer could purchase from Allegiance Mining a joint-venture interest in the Avebury project; it could invest in Allegiance Mining by purchasing already issued shares or by subscribing for new shares; and it could reduce the working capital required for the Avebury project by reducing the time taken to pay for concentrate.

3. Global nickel market

3.1 Overview

The International Nickel Study Group (INSG) recently estimated that global consumption of primary nickel would reach 1,136,000 tonnes in 2000, up from 1,080,000 tonnes in 1999. Global nickel consumption is growing at around 4–5% per year, principally because of growing demand for nickel-bearing stainless steel. Stainless-steel production accounts for around 65–70% of global nickel consumption.

Mined production of nickel (nickel content of sulphide concentrates and laterite ores) is estimated to reach 1,190,000 tonnes in 2000; sulphide concentrates will supply around 720,000 tonnes (60%) and laterite ores around 470,000 tonnes (40%). Russia, Canada, Australia, southern Africa, China and Brazil will produce all but a few thousand tonnes of the 720,000 tonnes of nickel in nickel-sulphide concentrates. Three countries – Russia (225,00 tonnes), Canada (205,000 tonnes) and Australia (140,000 tonnes) – will between them produce 570,000 tonnes (80%). Tropical countries dominate mined production of nickel in laterite ores. In 2000, Asia–Pacific countries – New Caledonia, Indonesia, Philippines and Australia – will produce around 265,000 tonnes, and Latin-American countries – Cuba, Colombia, Dominican Republic and Brazil – will produce 185,000 tonnes of nickel in laterite ores.

The INSG is currently forecasting global production of refined nickel (including nickel in ferronickel and nickel oxide final products) of 1,127,000 tonnes in 2000, and 1,209,000 tonnes in 2001. In 1998, global nickel supply exceeded global nickel consumption by 25,000 tonnes; in 1999, the global nickel market moved into deficit; consumption exceeded supply by around 58,000 tonnes. The INSG is currently forecasting a small (9,000 tonnes) global market deficit for this year, followed by a substantial surplus (35,000 tonnes) in 2001. The largest uncertainty in the INSG's current estimate of supply in 2000 and 2001 is production from the three Western Australian pressure acid leach (PAL) nickel–cobalt laterite plants – Bulong, Cawse and Murrin Murrin.

3.2 Trade in intermediate products

Arms-length trade in nickel intermediate products is relatively small, because most nickel producers are vertically integrated from mine production to final product output. Global trade in nickel intermediates falls almost entirely into four categories:

- nickel-laterite ore
- nickel-laterite matte
- nickel-sulphide concentrate
- nickel-sulphide matte

Nickel-laterite ore (1.5–2.5% Ni) is sold by miners in New Caledonia, Indonesia and the Philippines to Japanese ferronickel smelters and to QNI Yabulu. In 2000, this trade will account for production of around 95,000–100,000 tonnes of refined nickel, of which the three Japanese ferronickel smelters will produce around 70,000 tonnes. Traded laterite ore is sold under long-term contracts; in any one year, the quantity bought and sold depends solely upon the requirements of the buyer, and the price for nickel contained in ore depends almost entirely on the average LME nickel price.

The only traded **nickel-laterite matte** (70% Ni) is that produced by PT Inco in Indonesia. Some 80% of PT Inco's output is processed by Tokyo Nickel (a Japan-based refiner, majority owned by Inco Limited); the remaining 20% is processed by Sumitomo Metal Mining at its Niihama refinery, also in Japan).

Nickel-sulphide concentrate (15–20% Ni) trade is currently restricted to five main contracts, although at times some concentrate producers may sell some smaller quantities of concentrate on a spot basis. There are also two recently announced agreements covering projects not yet in operation.

- WMC sells 14,000 tonnes per year of nickel in nickel-sulphide concentrate from Mt Keith to Outokumpu (smelted by Outokumpu and refined by OM Group in Finland).
- Titan Resources sells 4,000–4,500 tonnes per year of nickel in concentrate to WMC (smelted at Kalgoorlie).
- Jubilee Mines sells around 10,000 tonnes per year of nickel in concentrate to Inco (smelted at Thompson, Manitoba).
- Tectonic Resources sells around 2,000 tonnes per year of nickel in concentrate to WMC (smelted at Kalgoorlie).
- Tati Nickel sells around 5,000 tonnes of nickel in concentrate and in direct-smelting ore to BCL Limited (smelted by BCL at Selebi Phikwe, Botswana).
- LionOre Australia has an agreement in place with Inco covering the sale of around 6,500 tonnes per year of nickel in concentrate from LionOre's Emily Ann project (to be smelted at Thompson, Manitoba or Sudbury, Ontario).
- Mincor Resources/Clough Mining/Donegal Resources joint venture recently purchased mining tenements at Kambalda from WMC. The purchase agreement provides for WMC to toll treat joint-venture ore at WMC's Kambalda concentrator, and for WMC to purchase around 7,000 tonnes per year of nickel in concentrate produced from joint-venture ore (to be smelted at Kalgoorlie).

Nickel in **nickel-sulphide matte** (40–75% Ni) is probably the most easily traded nickel-intermediate, partly because so many refineries have been designed – or modified – to process nickel-sulphide or nickel-copper-sulphide matte. Inco transports considerable quantities of matte from Canada to Wales, Taiwan and Korea to be refined; Falconbridge transports matte from Canada to its refinery in Norway. There are currently four significant arms-length nickel-sulphide matte contracts, all of them long-term, and all of them very important to seller and buyer.

- WMC sells around 25,000 tonnes per year of nickel in matte to Sumitomo (refined by Sumitomo Metal Mining at Niihama, Japan).
- WMC sells around 6,000 tonnes per year of nickel in matte to Outokumpu/OM Group (refined by OM Group at Harjavalta, Finland).
- BCL Limited sells about 20,000–25,000 tonnes per year of nickel in matte. Most (around 15,000 tonnes per year) is sold to Falconbridge (refined at Kristiansand, Norway); around 7,000 tonnes is toll refined for a Switzerland-based third party by Rio Tinto Zimbabwe.
- Rio Tinto Brazil sells around 10,000 tonnes per year of nickel in matte to Outokumpu/OM Group (refined by OM Group at Harjavalta, Finland).

3.3 Trade in nickel-sulphide flotation concentrate

Until recently, arms-length trade in nickel-sulphide flotation concentrate has been sparse. This was attributable to the paucity of custom smelters and of custom-smelting capacity, and to the unwillingness of major nickel producers to purchase nickel-sulphide concentrate. For many years, there were only two custom smelters capable of processing significant quantities of nickel-sulphide flotation concentrate – Sherritt's refinery at Fort Saskatchewan, Alberta and Outokumpu's smelter at Harjavalta, Finland. Sherritt's Fort Saskatchewan refinery – like WMC's Kwinana refinery – was designed to process concentrate; both plants found, after years of operation, that nickel-sulphide matte was a better feedstock than nickel-sulphide concentrate. Other smelters, including BCL's smelter in Botswana, were prepared to consider custom smelting nickel-sulphide concentrate, but few agreements were executed.

In 1993, Sherritt's Fort Saskatchewan refinery became part of the fully integrated Sherritt/CubaNiquel joint venture, processing mixed nickel–cobalt sulphide intermediate from Moa Bay. As Outokumpu's in-house production of nickel-sulphide concentrate diminished, Outokumpu's Harjavalta smelter came to rely more and more on purchased concentrate for feedstock. Despite protracted and valiant efforts to find feedstock for its 'hungry bear' smelter/refinery complex at Harjavalta, Outokumpu eventually sold the refinery to OM Group earlier this year. Outokumpu will continue to operate the smelter, toll smelting nickel-sulphide concentrate for OM Group. This toll-smelting arrangement may cease, and the smelter may close, around 2004, when Outokumpu's Black Swan nickel-sulphide mine in Western Australia exhausts its currently known reserves, and when Outokumpu's contract to buy 14,000 tonnes per year of nickel in concentrate from WMC's MT Keith mine expires.

Two recent developments have changed, quite dramatically, global arms-length trade in nickel-sulphide concentrate. First, the supply of suitable concentrates increased. Junior mining companies discovered and wished to develop several medium-sized, relatively high-grade nickel-sulphide orebodies in Western Australia. Five of the seven trades listed above (see section 3.2) involve small, recently developed or yet to be developed, nickel-sulphide mining operations in Western Australia. Secondly, major fully integrated nickel producers decided to purchase nickel-sulphide concentrate from smaller would-be competitors, whereas previously, they had been reluctant to assist the development of competing new mine production capacity by providing smelting capacity for nickel-sulphide concentrate from those new mines. If the new development had been sufficiently attractive, major nickel producers used to be more interested in purchasing the orebody than in purchasing concentrate produced from that orebody.

Inco and WMC have been at the forefront of this change of heart by major nickel producers; WMC accounts for three, and Inco for two, of the seven concentrate purchasing agreements listed in section 3.2. However, the long-term commitment of Inco and WMC to custom smelting is yet to be demonstrated. Both companies stress that their over-riding economic objective is to be more profitable, and to enhance returns to shareholders. In WMC's case especially, this has been evidenced as a focus on high-margin production (margin being sales revenue less cost of sales); it would seem consistent for WMC to agree to custom smelt concentrate only if that activity would be high-margin business. Although custom smelting may be high-margin for WMC when WMC's in-house production of concentrate is insufficient to fully feed its Kalgoorlie smelter, WMC would be unlikely to commit long-term to allocating smelting capacity to custom smelting if it constrained WMC's opportunity to smelt WMC's own concentrate. Exploration success by WMC would probably end WMC's interest in custom smelting. The same goes for Inco, whose recent enthusiasm for custom smelting is likely to evaporate should Inco reach agreement, with the government of Newfoundland and Labrador, for Voisey's Bay nickel-sulphide concentrate to be smelted at Inco's existing smelters at Thompson and Sudbury.

3.4 Potential developments

Two recent developments may provide opportunities for a potential producer and seller of nickel-sulphide concentrate.

- QNI has developed, to pilot-plant scale, its BioNic nickel-sulphide concentrate leaching process. This process is designed to produce final nickel product (probably electrowon nickel cathode) directly from nickel-sulphide concentrate, using bacterial leaching in stirred atmospheric-pressure reactors. QNI is currently seeking suitable a development opportunity to establish the first commercial-scale BioNic operation.

- Bulong, Cawse and Murrin Murrin – the three Western Australia PAL operations commissioned in the last two years – should be capable of producing final nickel product (electrowon cathode or hydrogen-reduced powder/briquette) by directly leaching nickel-sulphide concentrate purchased from external sources. The scope of the necessary capital works is probably not yet apparent; however processing of high-grade nickel-sulphide concentrate would help to compensate the three PAL operators for the decline in nickel grade of autoclave feed that is scheduled for each of them.

4. Nickel-sulphide concentrate market

4.1 Potential buyers

Potential buyers of Avebury nickel-sulphide concentrate include established nickel smelters and nickel refineries capable of directly leaching concentrate; recently commissioned nickel smelters and nickel refineries capable of directly leaching concentrate; nickel smelters and refineries not yet built; and trading companies that would themselves negotiate necessary toll-smelting and toll-refining arrangements.

Established smelters and refineries include:

- Inco, with its smelters at Thompson and Sudbury;
- Falconbridge, with its smelter at Sudbury, and possibility of arranging toll smelting at BCL Botswana;
- OM Group, which might arrange toll smelting by Outokumpu at Harjavalta, before processing the resultant matte at OM Group's adjacent refinery;
- BCL Limited, with its smelter in Botswana – although this would almost certainly need to be negotiated and managed by an experienced global concentrates trader, or perhaps by Falconbridge which has close links to BCL Limited;
- Impala Platinum, with its smelter at Rustenburg and refinery at Springs in South Africa;
- Amplats, with its smelters and refinery at Rustenburg in South Africa;
- Bindura Nickel, an Anglo American subsidiary, with its smelter and refinery in Zimbabwe;
- Jinchuan Nonferrous Metals Company, with its smelter and refinery at Jinchang, Gansu province, China – although this would almost certainly need to be negotiated and managed by an experienced global concentrates trader;
- WMC, with its smelter at Kalgoorlie.

Recently commissioned smelters and refineries include:

- Rio Tinto Brazil, with its smelter at Fortaleza de Minas;
- Murrin Murrin joint venture, or one of the Murrin Murrin joint venturers – Anaconda Nickel or Glencore International – with its PAL plant at Murrin Murrin;
- Preston Resources, with its PAL plant at Bulong;
- Centaur Mining & Exploration, with its PAL plant at Cawse.

Smelters and refineries yet to be built include:

- QNI, which might be interested in building a demonstration BioNic plant at Yabulu.

Trading companies include:

- Marc Rich AG;
- Glencore International Limited;
- Sumitomo Corporation, which might arrange toll smelting before processing the resultant matte at Sumitomo Metal Mining's Niihama refinery:

The companies considered most likely to be interested in purchasing Avebury nickel-sulphide concentrate are Inco, Falconbridge, OM Group, WMC, QNI, Marc Rich and Glencore. It is recommended that Allegiance Mining defer contacting the two trading companies – Marc Rich and Glencore – until after Allegiance Mining has established direct contact with the five likely concentrate processors – Inco, Falconbridge, OM Group, WMC and QNI.

4.2 Pricing and payment terms

Based on Allegiance Mining's indicative Avebury concentrate specification (see section 6), nickel is likely to be the only contained metal for which payment is received; cobalt content at 0.5% is likely to be too low for any meaningful payment for cobalt to be received. However, cobalt recovery does vary widely from smelter to smelter, and some potential buyers of Avebury nickel-sulphide concentrate may be willing to pay for some of the contained cobalt. Arsenic appears to be the only deleterious impurity present in sufficient quantity to cause concern to some buyers. Avebury's indicative concentrate assay of 4,640 parts per million arsenic means that Avebury's indicative annual production of 19,400 tonnes of concentrate will contain 90 tonnes of arsenic – sufficient to concern many smelters. Smelter are finding it increasingly difficult – and costly – to manage the long-term disposal and storage of arsenic-bearing residues in an environmentally acceptable way. Indicative pricing and payment terms follow.

4.2.1 Quotational period

The quotational period (abbreviated to QP) is the days, weeks or months that will be specified in the Avebury concentrate sale and purchase agreement, relative to the date of some fixed and easily determined event in the production, transport or processing of any lot of Avebury concentrate, over which the reference price (see 4.2.2) will be averaged to obtain the base price for the lot. For example, the QP might be 'the second calendar month following the month of arrival at buyer's works', or 'the month following the month of shipment from Australia'. Allegiance Mining should study carefully any proposal to give the buyer the option to choose the QP to be applied to any lot from several alternatives, because this flexibility is usually managed to the advantage of the buyer and the disadvantage of the seller.

A likely quotational period is 'the second calendar month following the month of arrival at buyer's works'.

4.2.2 Reference price

The reference price will undoubtedly be the London Metal Exchange (LME) price for nickel, averaged over the quotational period. There are many LME prices established each LME trading day – the most often used prices in concentrate sale and purchase agreements are the official cash seller's and official cash buyer's prices, and the official three-month buyer's and seller's prices. One or more of these prices will be selected in the Avebury concentrate sale and purchase agreement as the reference price. For example, the reference price might be 'the mean of the official cash buyer's and seller's prices, averaged over the quotational period' or 'the mean of the official cash buyer's, cash seller's, three-month buyer's and three-month seller's prices averaged over the quotational period'. The latter is sometimes referred to as the 'four corners' average. The pricing and payment consequences of selecting any one reference price need to carefully considered. Allegiance Mining should study carefully any proposal to give the buyer the option to choose, from several alternatives, the reference price to be applied to any lot; this flexibility is usually managed to the advantage of the buyer and the disadvantage of the seller.

A likely reference price is 'the mean of the official London Metal Exchange cash buyer's, cash seller's, three-month buyer's and three-month seller's prices averaged over the quotational period'.

4.2.3 Payable metal

The buyer will seek, and expect to obtain, acknowledgment in the concentrate sale and purchase agreement that the buyer does not recover all the metal contained in the concentrate into the buyer's final product. Some of the nickel and possibly much of the cobalt in Avebury's nickel-sulphide concentrate will be lost to slag and other waste products. The concentrate sale and purchase agreement will therefore specify the percentage of contained metal for which the buyer will pay; the percentage usually varies from metal to metal, and can be more or less complex. For example, a simple payable metal clause would be 'buyer shall pay seller for 97% of the nickel contained in concentrate'. A more complex clause would be 'buyer shall pay seller for 97.5% of the nickel contained in concentrate, subject to a minimum deduction of 0.5% from the nickel assay of the concentrate'. The 'minimum deduction' provision operates, in this example, if the concentrate assays less than 20%Ni, thereby protecting the buyer against expected increasing loss of nickel to slag as the concentrate nickel grade falls, because the quantity of slag produced per tonne of nickel increases. Concentrate assaying 15% Ni would be assessed as containing 14.5% payable nickel, and the buyer would pay for 14.5/15.0, or 96.67%, of nickel contained in concentrate.

A likely reference price is 'buyer shall pay seller for 97% of nickel contained in concentrate'.

4.2.4 Treatment and refining charges

Treatment and refining charges are intended to provide the buyer with sufficient income to pay its expected operating costs associated with processing Avebury nickel-sulphide concentrate, and to provide it with adequate return on its capital invested. Treatment and refining charges may be expressed in several ways. One extreme is to express them as percentages of payable metal; the other extreme is to express them as monetary charges – in a specified currency – per tonne of concentrate and per pound of payable nickel. An example of the former would be 'seller shall pay buyer a treatment and refining charge of 35% of the payable nickel in concentrate'. An example of the latter would be 'seller shall pay buyer a treatment charge of US$200 per dry tonne of concentrate, and a refining charge of US$0.50 per pound of payable nickel in concentrate'. If the concentrate sale and purchase agreement incorporates a monetary charge per tonne of concentrate or per pound of nickel, the buyer will always seek (but not always receive) ongoing adjustment of these charges in line with one or more cost indices; for example, in line with the cost of living index in the buyer's country, or the inflation-adjusted price of fuel or power in the buyer's country. Allegiance Mining should study very carefully any proposal to adjust monetary charges in line with general inflation in the buyer's country.

Should a potential buyer consider that its ability to process Avebury concentrate would be adversely affected by the quantity of a deleterious element – for example, arsenic – then the buyer will insist on receiving an appropriate additional monetary charge linked to the assay content of the deleterious element. As well, a potential buyer is likely to insist on the inclusion of a permitted maximum assay content of the deleterious element; the buyer will not be obliged to purchase concentrate that exceeds the maximum permissible assay content.

A likely – but perhaps over-simplified – treatment and refining charge would be 'seller shall pay buyer a treatment and refining charge of 35% of the payable nickel in concentrate'.

4.2.5 Payment timing

Negotiation of the timing of payment for deliveries of concentrate involves buyer and seller reaching a mutually satisfactory compromise between the seller's understandable desire to minimise its working capital by being paid as soon as each lot of concentrate is ready to be despatched from the seller's works, and the buyer's understandable desire to minimise its working capital by not paying for the concentrate until it has produced – if not sold – the finished nickel produced from each lot. In most cases, timing of payment is advanced to suit the seller's needs, but the resulting financial cost incurred by the buyer is recovered by it, most commonly through increased treatment and refining charges.

In special circumstances – often where the seller would have difficulty in financing working capital – the buyer may be prepared to make advance payments for concentrate as soon as securing title to the concentrate. Again, the resultant financing charge incurred by the buyer will be recovered from the seller.

The seller in particular needs to study carefully the clause or clauses that specify how payment is made in circumstances where the final weights and assays have not been determined – usually because the difference between buyer's and seller's assays is outside the agreed 'splitting limit' (see section 4.3.3). In this situation, the seller should insist on receiving payment, at the agreed time, for the estimated value of the concentrate. Delaying payment until final weights and assays are known can create unexpected cashflow difficulties for the seller, In addition, the process of determining final weights and assays may be outside the control of the seller.

A likely payment timing clause is 'Buyer shall pay seller 90% of the estimated value of each lot – calculated using latest information about the quantity and quality of the lot, and latest known reference price – by the end of the first month following the month of arrival of that lot at buyer's works. Full payment for each lot – calculated using latest information about the quantity and quality of the lot, and latest known reference price – shall be received by seller no later than the end of the second month following the month of arrival of that lot at buyer's works'.

4.3 Other important contract terms

4.3.1 Delivery terms

Delivery terms are commonly negotiated to suit the particular requirements of the seller, and to take advantage of any possible logistical synergy that the buyer may be able to develop. Avebury, located close to Tasmania's west coast, may have to transport concentrate to Burnie or to Hobart for loading onto ocean-going vessels. Because the frequency of shipping from Burnie or Hobart may be less than Allegiance Mining or any potential buyer might wish, transhipment to a mainland-Australia port may be cost effective.

Allegiance Mining has indicated production of around 21,100 wet tonnes per year of concentrate from Avebury. As there appears to be no factor preventing this concentrate being shipped evenly throughout the year, monthly shipments could average around 1,750 wet tonnes. Each monthly shipment would be worth around US$1.6 million (A$3 million) to Allegiance Mining (calculated using an LME nickel price of US$3.25 per pound; a US$/A$ exchange rate of 0.55; payable metal of 97% of contained metal; and a treatment and refining charge of 35% of payable metal value). The as-shipped concentrate would contain 445 pounds of nickel per wet tonne, giving it an intrinsic value of around US$1,450 (A$2,600) per wet tonne.

The cheapest way to transport this quantity of nickel-sulphide concentrate would be in bulk, rather than in plastic-lined shipping containers holding, say, 23 tonnes of concentrate, or in intermediate bulk containers (IBCs, or 'bulk bags') holding, say, two tonnes each. However, savings in packaging and transport costs would be offset by the risk of loss of concentrate at each stage of handling. The optimum trade-off will need to be determined by Allegiance Mining and the potential concentrate buyer.

The delivery terms clause, and some associated clauses – including those dealing with insurance, and with the passing of risk and title – in the concentrate sale and purchase agreement will almost certainly refer to one of the Incoterms developed and refined by the International Chamber of Commerce. The latest version – Incoterms 2000 – entered into force on 1 January 2000.

Likely delivery terms might be 'seller shall deliver goods in bulk to buyer Free on Board Burnie Tasmania (Incoterms 2000)'.

4.3.2 Lot size

Lot size specifies the quantity of concentrate to be weighed, sampled and assayed as one parcel. The more valuable the commodity being bought and sold, the smaller the lot size tends to be. Avebury nickel-sulphide concentrate will be worth around US$1,450 (A$2,600) per wet tonne at an LME nickel price of US$3.25 per pound, so it would seem sensible to begin with a lot size of around 200 tonnes. This would mean Avebury would sell around 100 lots per year, each lot containing around 42 tonnes of nickel worth around US$300,000 (A$550,000). The lot size could be increased by mutual agreement of buyer and seller after sufficient experience has been accumulated in administering the Avebury concentrate sale and purchase agreement.

If Avebury concentrate is shipped in bulk, lots will be determined at the final discharge port, or upon receipt at the buyer's works. The critical requirement is that there be appropriately designed and managed means to determine weight and moisture content of each lot, and to obtain a satisfactory representative sample of each lot.

Likely lot size might be 'around 200 wet tonnes'.

4.3.3 Weighing, sampling and assaying

If concentrate is shipped in bulk, weighing and sampling must be done at the point where lots are determined; if concentrate is shipped in shipping containers or IBCs, weighing and sampling can be done when the containers or IBCs are being packed – probably at the seller's works – or when they are being emptied – at the buyer's works. The buyer may be willing to accept weighing and sampling at the seller's works, or at loading port, if weighing and sampling procedures are sufficiently rigorous, and demonstrably free of bias. Weighing and sampling at the seller's works maximises the time available for the seller and the buyer to reach agreement on assays, facilitates early payment to the seller, and gives the buyer precise information about concentrate deliveries long before the deliveries arrive at its works.

Sampling, in addition to providing a definitive sample for determination of the moisture content of a lot, usually provides at least four identical samples to be used to determine the chemical analysis of the lot. One sample is given to the buyer; one to the seller; one is held in reserve for use if an umpire assay is required; and the fourth sample is held as an archived reserve sample to be used as needed.

The usual practice is for buyer and seller to assay the sample they have been given, and to exchange assays according to a predetermined procedure within a certain time. Seller and buyer will have agreed on a 'splitting limit'. If the difference between seller's and the buyer's assay is less than or equal to the splitting limit, seller and the buyer accept the mean of their assays as final and binding. Should the difference between seller's and the buyer's assays exceed the splitting limit, the final assay is determined by umpire analysis (see section 4.3.4).

A likely weighing, sampling and analysis clause might be 'Weighing and sampling shall be carried out at buyer's works in accordance with the procedures described in Attachment 1. Moisture determination shall be carried out in accordance with the procedure specified in Attachment 2. Buyer and seller shall assay their respective samples, and exchange assays within 30 days of receipt of samples. If the assays differ by no more than 0.5% nickel, the mean of buyer's and seller's assays shall be the final assay used to prepare buyer's final invoice to seller for the lot. If the assays differ by more than 0.5% nickel, the final assay of the lot shall be determined in accordance with the procedure specified in Attachment 3'.

4.3.4 Resolution of assay differences

Resolution of assay differences that fall outside the agreed splitting limits is usually done by submitting a reserve sample to an umpire assayer. The concentrate sale and purchase agreement may specify only one umpire, or it may list a panel of umpire assayers, to be used in rotation. The final assay may be that of the umpire, or that of seller or buyer falling closest to that of the umpire; the cost of the umpire assay is usually borne by the party (seller or buyer) whose assay falls furthest from the umpire assay.

A likely assay resolution clause might be 'If the difference between seller's and buyer's exchanged assays falls outside the agreed splitting limit of 0.5% Ni, the umpire reserve sample shall be sent to an umpire, selected in rotation from the agreed panel of umpires. The umpire's assay shall be binding on the parties, and the cost of the umpire assay shall be borne by the party – buyer or seller – whose assay was furthest from the umpire assay'.

4.4 Suggested cashflow modelling parameters

For the purposes of the cashflow model being prepared for Allegiance Mining's scoping study of its Avebury project, the following simplified parameters are suggested as being appropriate. The suggested parameters are simpler than the likely clauses described in sections 4.2 and 4.3, mainly because in a cashflow model there is no uncertainty about weight, moisture content and assay of concentrate shipments.

Quotational period: the second month following the month of arrival at buyer's works.

Reference price: the mean of the official London Metal Exchange cash buyer's, cash seller's, three-month buyer's and three-month seller's prices averaged over the quotational period.

Payable metal: 97% of nickel contained in concentrate.

Treatment and refining charges: 35% of the payable nickel in concentrate.

Payment timing: 90% of the estimated value of each lot by the end of the first month following the month of arrival of that lot at buyer's works, and the remaining 10% by the end of the second month following the month of arrival of that lot at buyer's works.

Delivery terms: seller to deliver concentrate in bulk, Free on Board Burnie Tasmania (Incoterms 2000).

Shipping schedule: evenly spread monthly shipments.

5. Marketing leverage

5.1 Competition for concentrate

The major fully integrated nickel producers are keen to operate their smelters and refineries at full capacity provided that the marginal cost of producing concentrate to feed their smelters allows them to make an adequate return on capital. In general the major nickel producers are facing declining resources as exploration success in recent years has failed to match depletion of resources. The one exception to this generality is Inco; provided that Inco is successful in its efforts to obtain government permission to process Voisey's Bay concentrate through its existing smelters and refineries outside the province of Newfoundland and Labrador, Inco should have no difficulty in operating its smelters at maximum capacity.

WMC has demonstrated its willingness to purchase concentrate in recent years. However this is a marked turn-around from WMC's previous practice. Significant exploration success by WMC may lead to it placing higher priority on smelting concentrate produced in-house than on smelting concentrate purchased from others.

Outokumpu's smelter at Harjavalta used to be the most assertive custom nickel smelter in the world, as Outokumpu struggled to find sufficient concentrate to ensure it ran at capacity. However, Outokumpu sold its adjacent refinery to OM Group earlier this year, and Outokumpu's interest in maximising Harjavalta's smelting rate may have diminished considerably. On the other hand, OM Group now owns a refinery with a capacity of at least 50,000 tonnes of nickel per year, but has no assured feedstock for that refinery post 2003.

Allegiance Mining has indicated that its Avebury project will produce around 4,000–4,500 tonnes per year of nickel in concentrate. This quantity represents probably no more than 1 or 2 years incremental continuous-improvement growth in capacity for any of the world's large nickel smelters, including Inco's Sudbury smelter and WMC's Kalgoorlie smelter.

In addition, the construction and commissioning of Bulong, Cawse and Murrin Murrin may well lead to the opening up of new homes for nickel-sulphide concentrates. This may not be fully apparent for at least another 12 months – but by the end of 2001 all three plants should have ramped up to close to nameplate throughput, had time to consider the benefits they could obtain from processing purchased nickel-sulphide concentrate, and had time to estimate the capital cost required for the necessary plant modifications.

QNI's development of BioNic appears to have reached the stage where QNI is extremely keen to build a commercial-scale plant to demonstrate in practice the capital and operating costs so far estimated by pilot-plant test work. The annual quantity, and the high quality, of Avebury concentrate would seem to suit QNI's requirements very closely.

5.2 Potential partnership and investment opportunities

Inco, as well as contracting to purchase concentrate from Jubilee Mines' Cosmos project, also purchased a significant – around 3.5% – equity stake in Jubilee Mines. In addition to contracting to purchase concentrate from LionOre Nickel's Emily Ann project, Inco has also undertaken to finance the construction of the Emily Ann concentrator. Inco may well be interested in taking an equity stake in Allegiance Mining, and may be prepared to assist Allegiance Mining with financing of Avebury.

QNI has demonstrated its keenness to take an equity stake in mining projects that will send intermediate product to QNI's Yabulu refinery for processing. The history of QNI's involvement with Comet Resources Limited's Ravensthorpe PAL project shows that QNI, in addition to purchasing all nickel–cobalt from Ravensthorpe, has so far purchased a 20% equity stake in Comet Resources Limited. If QNI assesses that BioNic processing of Avebury nickel concentrate is economically attractive – at Yabulu, in Tasmania or elsewhere – it is highly likely to insist upon taking an equity share in the Avebury project, and may well be interested in taking an equity stake in Allegiance Mining.

Other partnership opportunities will undoubtedly surface over the next year or so, especially as Allegiance Mining progresses product-offtake discussions with potential concentrate buyers.

6. Appendix: Indicative concentrate quantity and quality

6.1 General description

Material: Nickel-sulphide flotation concentrate produced at Avebury, near Zeehan, western Tasmania.

Quantity: 19,400 dry tonnes per year.

Moisture: Approximately 8%.

Quality: Nickel concentrate containing 22% nickel, 0.5% cobalt and 0.15% copper.

Detailed assays as per part (B).

Sizing: 100% passing 150 microns.

92% passing 53 microns.

30% passing 8 microns.

Detailed sizing as per part (C).

6.2 Indicative concentrate assay (by weight, dry basis)

Nickel	22.0%	
Cobalt	0.5%	
Copper	0.15%	
Au	1,821	parts per billion
Ag	less than 1000	parts per billion
Pt	less than 2	parts per billion
Pd	20	parts per billion
Iron	28.9%	
Sulphur	27.8%	
SiO_2	9.6%	
Al_2O_3	0.3%	
CaO	1.1%	
MgO	4.3%	
TiO_2	0.01%	
Na_2O	0.01%	
K_2O	0.05%	
P_2O_5	0.03%	
MnO	0.11%	
As	4,640	parts per million
Ba	13	parts per million
Bi	616	parts per million
Cd	less than 5	parts per million
Cr	1,310	parts per million
Mo	75	parts per million
Pb	397	parts per million
Sb	160	parts per million
Sr	less than 10	parts per million
V	16	parts per million
Zn	537	parts per million
Zr	22	parts per million
Nb	0.9	parts per million
Sc	0.9	parts per million
Se	17.8	parts per million
Sn	1.2	parts per million
Ta	0.7	parts per million
Te	1.8	parts per million
W	9.1	parts per million
Hg	0.2	parts per million
Cl	300	parts per million
F	220	parts per million

6.3 Indicative concentrate size distribution

Microns	% retained	% passing
150	0.0	100.0
106	0.4	99.6
90	0.6	99.0
75	1.1	97.9
63	1.5	96.4
53	4.6	91.8
45	3.2	88.6
32	16.9	71.7
22	11.6	60.1
16	12.7	47.4
11	10.6	36.8
8	7.7	29.1
-8	29.1	0.0

ICR 01-4526F (F)

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DECEMBER 2000

MINERAL RESOURCES
14 FEB 2001
TASMANIA

AVEBURY NICKEL PROJECT
MARKETING SCOPING STUDY

FOR

MINERAL RESOURCES
FILE REF: EL28/88 PT5
16 FEB 2001
DOC. REF:
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ALLEGIANCE MINING N.L.

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01_4526 (F)

Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL*; Newnham Exploration and Mini EL28/1988
Newnham, L.A.

VOL 1 of 10

1. INTRODUCTION

In July 2000, John E. Butler (JEB) contacted John O'Shea, of John O'Shea and Associates Pty. Ltd., with the request for a proposal to carry out an Avebury nickel project marketing scoping study. On 20th July 2000 a submission was received with a scope of work including (a) the global nickel market, (b) nickel sulphide concentrate marketing, and (c) marketing leverage, and the commission was approved on 20th October 2000. O'Shea was selected for the following reasons:

1. His areas of specialisation include marketing and commodity studies, and project and business development.

2. His career has embraced extensive senior management with several major international resource companies, such as Rio Tinto, QNI, RGC and MIM.

3. Since establishing John O'Shea and Associates Pty. Ltd. in 1997 he has carried out assignments for Rio Tinto, Macquarie Bank, Highlands Pacific, Warrior International, North Limited, etc. and was appointed an External Associate of Rio Tinto Technical Services in 1997.

4. From mid-1998 he has published "Nickel Australasia", a twice monthly subscription newsletter reporting and analysing developments in nickel and cobalt in the Asia-Pacific region.

Regular O'Shea/JEB dialogue has been undertaken, with JEB supplying a wide range of information and data to enable O'Shea to evaluate the estimated Avebury nickel concentrate production. Apart from drawing upon his considerable in-house data base, O'Shea has also contacted the major nickel concentrate smelting and refining companies with a request for intentions of processing interest and indicative financial and technical input. It must be pointed out that

the latter schedule was not included in the original proposal and JEB advised O'Shea that the obtaining of specific potential processing details should not be allowed to delay the submission of his final report.

The attached report was received in early December 2000 and does not include the responses from "interested processing parties" which will be detailed in a supplementary report. It is unlikely that the additional input will be available before early 2001, thus it will not form part of the current *scoping study documentation*.

2. SIGNIFICANT MARKETING DATA

The following tabulations were derived from the interpretation of metallurgical testwork results, the attached John O'Shea marketing scoping study, and JEB's quantification of the aforementioned documentation:

Concentrate production	19,400 dry tonnes p.a.
Concentrate grade	22% Ni
Nickel in concentrate	4,270 tonnes p.a.
Assumed nickel price	US$7,500 per tonne

Payable metal

O'Shea recommendation 97% of nickel in concentrate, thus 4,141.9 tonnes payable nickel per annum, with value of **US$31,064,250 annually.**

Alternative 97.5% of nickel in concentrate, requiring a minimum concentrate grade of 20% Ni, thus 4,163.25 tonnes payable nickel per annum, with value of **US$31,224,375 annually.**

Treatment and refining charges

O'Shea recommendation 35% of payable nickel, thus **US$10,872,488 annual charge.**

Alternative treatment charge of US$200 per dry tonne of concentrate and US$0.50 per pound of payable nickel in concentrate, with concentrate grade 20% Ni and above, thus **US$8,469,150 annual charge.**

Alternative payable metal and treatment and refining formulae illustrate the potential for an annual revenue improvement of **US$2,563,463** when compared with the O'Shea recommendations for the two components.

3. SIGNIFICANT MARKETING FEATURES

In O'Shea's opinion Allegiance Mining should not experience difficulty in negotiating a contract for the processing (treatment and refining) of an Avebury nickel sulphide concentrate. One of the more positive aspects of the anticipated concentrate properties is the relatively high 22% nickel grade, a value which appears to put it into a highly desirable category. The only adverse comment is on page 13 of the attached report, with the statement that if the saleable concentrate contains a deleterious element such as arsenic "the buyer will insist on receiving an appropriate additional monetary charge linked to the assay content of the deleterious element". This observation resulted from the 4,640 ppm (0.464%) arsenic value contained in the indicative assay tabulation (item 6.2), but it must be emphasised that JEB is aware of this treatment/refining reaction and would evaluate the depression of arsenic mineralisation in future flotation testwork.

The following significant treatment and refining contract terms are recommended by O'Shea for adoption in the current Allegiance Mining evaluation:

1. Payable metal

   Buyer shall pay seller for 97% of nickel contained in concentrate.

2. Treatment and refining charges

   Seller shall pay buyer a treatment and refining charge of 35% of the payable nickel in concentrate.

Reference was made to possible alternatives as illustrated below:

1. Payable metal

Buyer shall pay seller for 97.5% of the nickel contained in concentrate , subject to a minimum deduction of 0.5% from the nickel assay of the concentrate.

2. Treatment and refining charges

Seller shall pay buyer a treatment charge of US$200 per dry tonne of concentrate, and a refining charge of US$0.50 per pound of payable nickel in concentrate.

O'Shea did not include any production and financial calculations in his report, but JEB has included comparative data for the recommended and alternative terms in the item 4 conclusions and basic production/financial comparisons section.

A wide range of possible processing organisations are detailed in the attached report, including INCO, Falconbridge, OM Group, Outokumpu, BCL Limited, Impala Platinum, Amplats, Bindura Nickel, Jinchuan Nonferrous Metal Company, WMC, Rio Tinto Brazil, Murrin Murrin Joint Venture, Preston Resources, Centaur Mining and Exploration and QNI, but JEB considers that the emphasis should be on negotiating terms with the better known, well established, organisations in the short term.

In the section on pricing and payment terms (4.2) O'Shea covers quotational period, reference price, payable metal, treatment and refining charges, and payment timing, and they will have to be carefully considered and evaluated when deriving the overall financial estimates. It is of interest that O'Shea makes the comment "cobalt content at 0.5% is

likely to be too low for any meaningful payment for cobalt to be received", but in JEB's opinion this is an area demanding careful consideration. Section 4.3 outlines other important contract terms, such as delivery terms, lot size, weighing/sampling/assaying, and resolution of assay differences.

Suggested cashflow modelling parameters are tabled in section 4.4, whilst section 5 covers marketing leverage involving competition for concentrate and potential partnership and investment opportunities.

4. CONCLUSIONS AND BASIC PRODUCTION/FINANCIAL COMPARISONS

Although Allegiance Mining will wish to make their own production and financial evaluations, JEB has carried out the following preliminary quantitative comparisons of the alternative payable metal and treatment/refining charges included in the attached report:

Assumptions

1. Price of nickel US$7,500 per tonne.
2. Concentrate grade 22% Ni.
3. Concentrate production 19,400 dry tonnes per annum.
4. Concentrate contained nickel 4,270 tonnes per annum.
5. O'Shea recommended payable metal 97% of nickel in concentrate.
6. Alternative payable metal 97.5% of nickel in concentrate. This requires a minimum concentrate grade of 20% Ni.
7. Treatment and refining charges 35% of payable nickel in concentrate.
8. Alternative treatment and refining charges US$200 per dry tonne of concentrate (treatment) and US$0.50 per pound of payable nickel in concentrate (refining).

Payable metal estimates

O'Shea recommendation

4,270 x 0.97 = 4,141.9 tonnes nickel per annum
Value = US$7,500 x 4,141.9 = US$31,064,250 per annum

Alternative (concentrate 20% Ni and above)

4,270 x 0.975 = 4,163.25 tonnes nickel per annum
Value = US$7,500 x 4,163.25 = US$31,224,375 per annum
Additional value = US$160,125 per annum

Treatment and refining charge estimates

O'Shea recommendation

US$31,064,250 x 0.35 = US$10,872,488 per annum
Therefore Allegiance Mining income US$20,191,762 p.a.

Alternative (concentrate 20% Ni and above)

Treatment US$200 x 19,400 = US$3,880,000 per annum
Refining US$0.50 x 4,163.25 x 2,204.6 = US$4,589,150 per annum
Total charges US$8,469.150 per annum
Allegiance Mining income US$22,755,225 per annum
Reduced charges US$2,403,338 per annum
Increased Allegiance Mining income US$2,563,463 per annum, comprising US$160,125 increase from payable metal and $2,403,338 reduction in treatment and refining charges.

A definitive feasibility study will require detailed attention to a possible financial return against contained cobalt in concentrate, and laboratory testwork will include evaluation of arsenic minimisation into saleable concentrate. The latter should be readily achievable, but care will have to be taken to ensure that arsenic depression in flotation does not have an adverse effect on overall nickel recovery.

In the event that the Avebury nickel project is further advanced, it is recommended that O'Shea should have a continuing participation, but with an Allegiance Mining appointee having closer involvement with concentrate processing negotiations.

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Allegiance Mining NL

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Avebury Nickel Project

Scoping Study - Environment

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December 2000
CR 938_1_v2

01_4526 (G)
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Allegiance Mining NL*; Newnham Exploration and Mini Newnham, L.A.
EL28/1988
VOL 8 of 10

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Allegiance Mining NL

Avebury Nickel Project

Scoping Study - Environment

December 2000
CR 938_1_v2

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9 Exploration track on the boundary of rainforest and mixed forest communities (Tea-tree mixed forest shown).

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12 Little Henty River east of Zeehan.

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1 Sewage treatment plant option

2 Letter to Newnham Exploration and Mining Services

1. Introduction

1.1 Background

Allegiance Mining NL (Allegiance) is undertaking a Scoping Study to determine the technical feasibility and preliminary costs for the mining and processing of a nickel sulfide deposit at Avebury, near Zeehan, Tasmania (Figure 1). The project concept involves mining 300,000 tpa of ore and extraction of 4,270 tpa nickel in sulfide concentrates for transport to a beneficiation plant elsewhere.

This report presents the results of a preliminary environmental assessment of the project. We understand that this report (along with other reports, e.g., mine engineering and metallurgy) will form part of the Scoping Study to be assessed by Allegiance before a decision is made as to whether to proceed further with the project.

1.2 Scope of Work

The following scope of work was required by Allegiance (correspondence from Lindsay Newnham (Newnham Exploration and Mining Services), 17 May 2000):

1. Identify environmental issues.
2. Determine the project approval processes and environmental requirements.
3. Quantify necessary environmental studies and propose an appropriate time frame for completion of this work.
4. Provide indicative cost estimates to firstly, undertake these further studies and, secondly, to prepare a full EMP.

During the course of the program NSR was asked to provide additional information on possible sewage treatment options and environment operating costs.

1.3 Basis for this Report

This report has been prepared on the basis of the following:

- NSR's proposal to Lindsay Newnham of 31 May 2000.
- Allegiances's acceptance of NSR's proposal on 14 July 2000.
- Site visit to the Avebury Nickel Project area by Michael Jones and Luci David (NSR), 3 August 2000.
- On site (and subsequent) discussions between Michael Jones, Luci David and Lindsay Newnham.

- Phone conversations with Leigh Clark (Forestry Tasmania), Kevin Bird and David Gatehouse (Mineral Resources Tasmania), Gary Walker and Trevor Jones (West Coast Council), Robin Gilmour, Vortec Grun, Jim Lockley and Lynne Powell (Department of Primary Industries, Water and Environment), Ian Newman (Burnie City Council).
- NSR's attendance at the project team meetings in Launceston on 14 November and 18 December 2000.
- Review of relevant documents provided by Lindsay Newnham and other reference material (see Section 8 for details).
- NSR's experience with mining projects, including specific experience of such operations in Tasmania.

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NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Location map	Figure No: 1
	File No: 938_F1_HB	Allegiance Mining NL		

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2. Project Concept

The current project concept provides for the mining and beneficiation of pentlandite-magnetite-serpentinite (Fe,Ni)S ore to produce nickel concentrates for shipment to a smelter. For this preliminary assessment, it has been assumed that 19,400 dry tonnes of nickel concentrate (NiS) (22% nickel, 7-10% moisture) will be produced annually from 300,000 t of mined ore.

A preliminary project outline is normally required to determine the appropriate environmental investigations program. Notwithstanding that the current stage of project development does not permit such a project outline to be described with any confidence, it is likely that the key components of the project will involve the following:

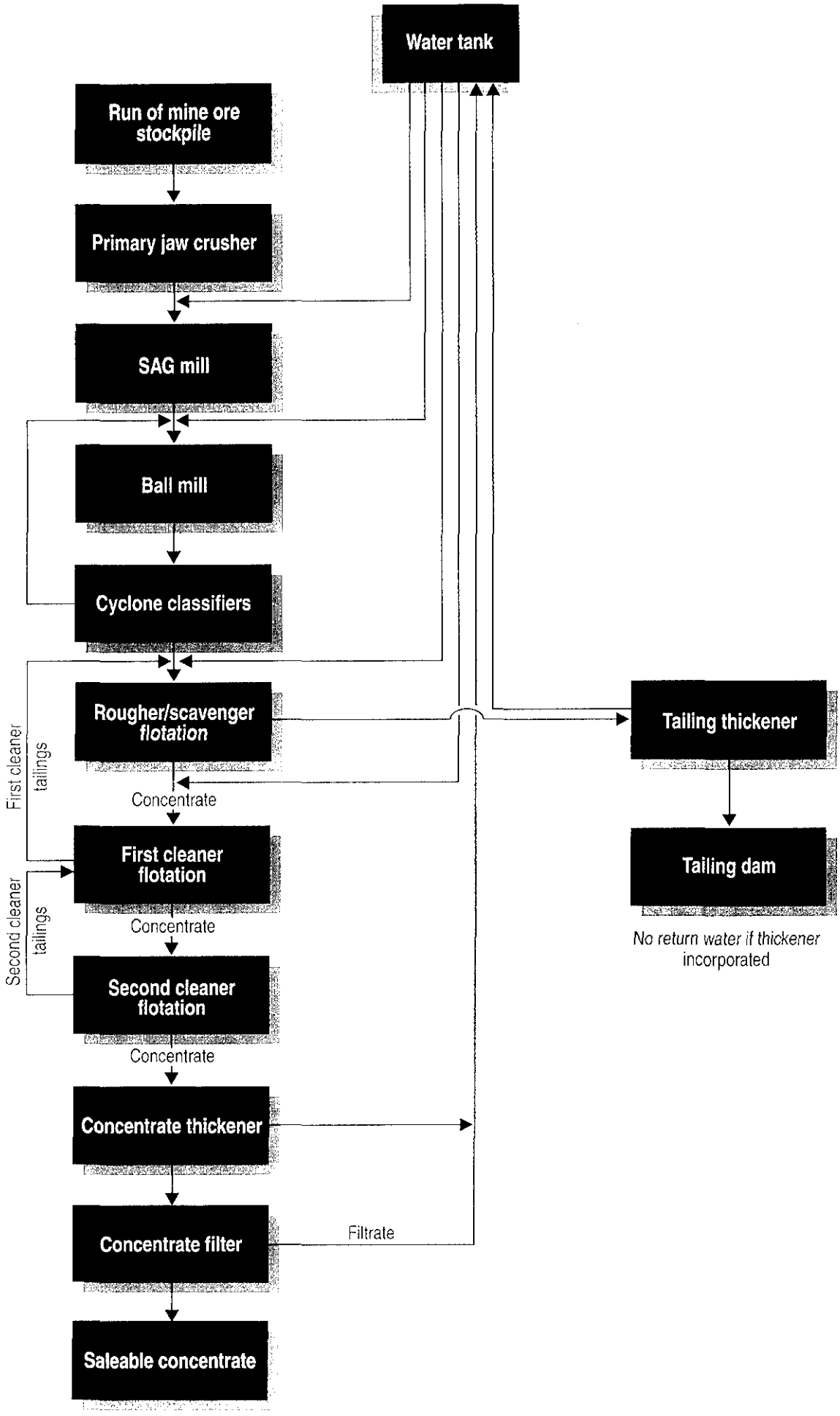
Mining

- Mining is envisioned at 300,000 tpa over a mine life of at least 11 years.
- The mine will be an underground operation accessed by a decline. It will be a mechanised small-medium scale operation, with the two proposed mining methods being benching and open-stoping.
- The footprint of the operation would cover about 45 ha and would encompass the portal, stockpiles, beneficiation plant, tailing dam and offices.
- Where possible waste rock will be used as underground backfill. There will be no caving, therefore no subsidence issues. Any deficit in fill material will be sourced from a quarry (see waste rock below).

Beneficiation and concentrate transport

The process plant would be located on the scrub and buttongrass and heath plains adjacent to degraded rainforest (Plate 1) and west of Comstock Creek in an area concealed from the Zeehan to Trial Harbour Road. The footprint of the development would not impinge on adjacent rainforest or high quality wilderness areas (Section 3.2.3 and 3.2.6).

- Pentlandite will be recovered by conventional comminution and flotation methods (Figure 2).
- Preliminary testwork has indicated that acceptable nickel recovery can be obtained.



NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Nickel concentrate production flowsheet	Figure No: 2
	File No: 938_F2_HB	Allegiance Mining NL		

- Transport of the nickel concentrate (three trucks per day, five days per week) between the beneficiation facility at the mine site to the smelter could involve trucking (using existing public roads) the concentrate from the mine to either:
  - Burnie for shipment to the smelter.
  - Melba Flats for rail transport (using the existing Emu Bay Railway line) to Burnie for shipment to the smelter.
- Waste solids (in slurry form) from the flotation circuit will report to the tailing dam. Filtrate will be recycled through the water tank.
- Reagents (flotation agents and depressors) will be recycled. However, a small proportion will not be recovered and will report to the tailing dam.
- There will be no gaseous emissions.

Waste rock

Waste rock material will be consumed by surface civil works (providing the material is geochemically inert) such as the construction of pads for the mill and beneficiation plant and possibly in the construction of the tailing storage facility), or as underground backfill. Any excess will be stockpiled on the surface in waste rock dumps. This approach includes:

- Waste rock storage requirements will be greatest at the start when the decline is under construction. There will be approximately 1 km of decline before ore is accessed and waste rock that is not consumed by civil works will require surface storage.
- Initially, with no production in the first year during mine development, all waste rock produced (198,000 t) will require surface stockpiling.
- During the second year, when production commences, only 58,000 t will require surface storage (in addition to that from the first year) with the remainder used as backfill underground.
- For each subsequent year development waste will be consumed underground with any short fall sourced from the surface stockpile, until Year 5 when approximately 200,000 t will need to be sourced elsewhere from the surface (i.e., quarry).
- The temporary surface storage area could be coincident with the site of a quarry which may need to be developed to provide sufficient volumes of fill.
- Waste rock material will contain some sulfides (intersects indicate sulfur assays up to about 3%). The management implications are discussed in Section 3.3.

Tailing

- The sulfur concentration in the tailing will be 0.4% since most sulfides will be recovered in the beneficiation process.
- Tailing will be stored in tailing storage facilities (TFSs) located in a small catchment northwest of the beneficiation plant (Figure 3).
- The flat and open terrain offers opportunities for stable and permanently safe tailing storage.
- Annual tailing discharge rate will be 280,600 dry tonnes at about 60% solids.

Infrastructure

Ancillary infrastructure includes a port and storage facilities, road access, electrical power, water supply, sewage treatment system, quarries for fill and construction materials, and workforce requirements. A preliminary infrastructure layout (for the mine area only) is shown in Figure 3.

- It has been assumed that the existing port facilities at Burnie will be used for all project-related shipping activities. If the concentrate is transported by rail to Burnie, it is also assumed that the existing facilities at Melba Flats will be used.
- The projected electrical power demand is approximately 4 MW and preliminary investigations indicate that this would be obtained from the existing Tasmanian Grid via the Zeehan substation.
- Water supplies for mining and the beneficiation process are available from the Kynance Creek/Comstock Creek system.
- Access to the mine will be via a new access road running south from Trial Harbour Road west of Kynance Creek, approximately 2 km to the mine site. The West Coast Council have upgraded the Trial Harbour Road to the Comstock Zinc Mine. A further 3.5 km of road will require upgrading (widening, straightening and drainage) to the mine access road.
- The workforce will comprise approximately 60-70 people and will operate in two 10 hr shifts. The workforce will be sourced locally from the West Coast, if not Zeehan. Local people will continue to live within their existing communities and be bussed to site. Non-local people will be accommodated by existing housing in Zeehan. The construction workforce will be approximately equivalent to the size of the operations workforce.
- A sewage treatment system has not been included in the current stage of planning, although a possible option is described in Appendix 1.

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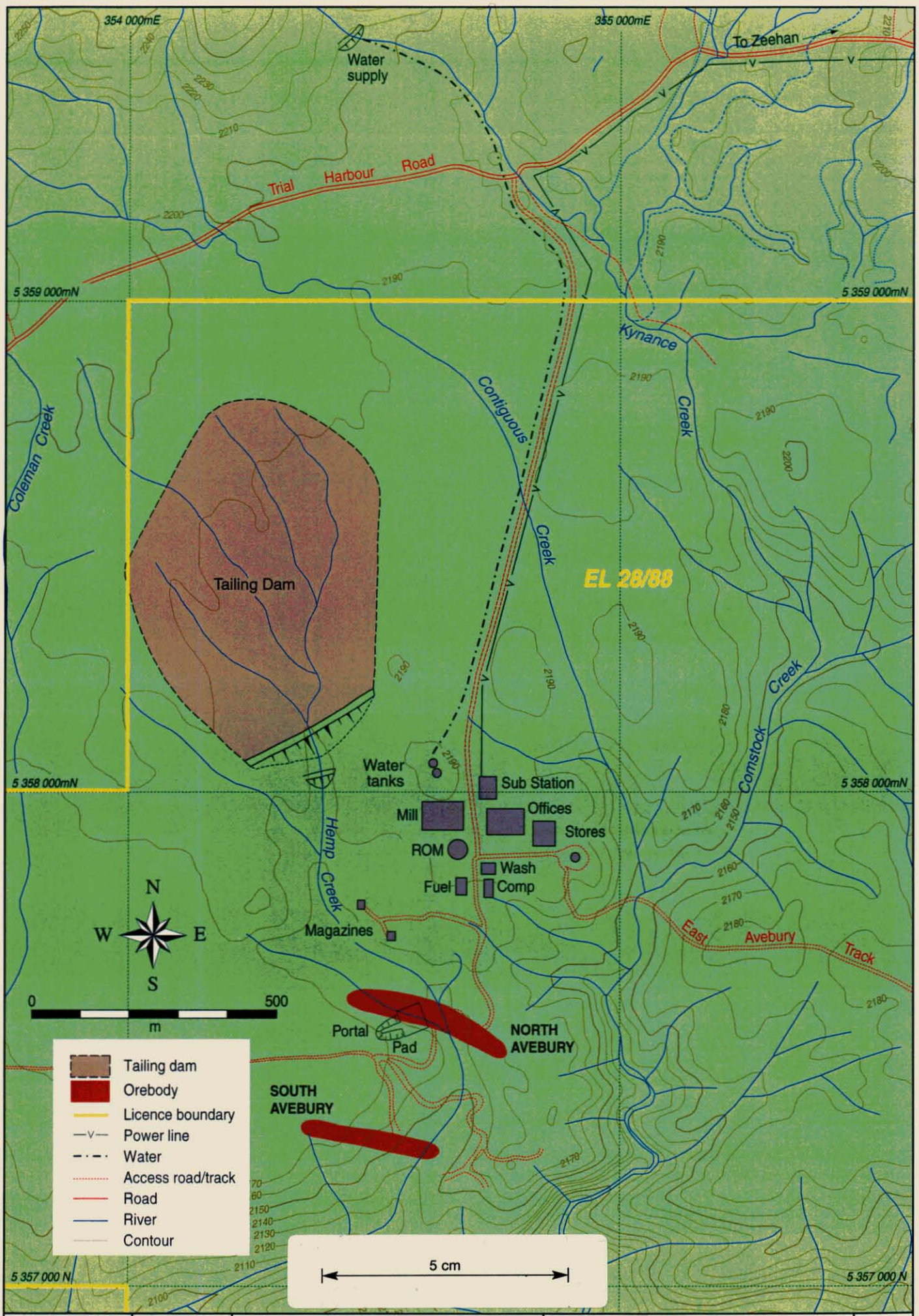

NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Preliminary infrastructure layout	Figure No: 3
	File No: 938_F3_HB	Allegiance Mining NL		

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3. Environmental Setting and Issues

3.1 Background

This section outlines the environmental setting and sensitivities of the Avebury Nickel Project and allows the environmental issues to be placed in context.

3.2 Environmental Setting

3.2.1 General

The Avebury Nickel Project area is situated about 8 km southwest of Zeehan (see Figure 1). The nearest localities are Trial Harbour (permanent population 12; West Coast Council, Planning Department, pers. comm.) to the west and Zeehan (population 1116, ABS (1996)). The nearest major towns by road are Roseberry, 29 km northeast, and Queenstown, 38 km southeast.

The general area has a history of exploration and prospecting since the 1800s, although this was impeded by the rugged terrain and difficult vegetation. Queenstown and Roseberry are historically mining towns and today their economy is still centred around this industry. More recently (1991) the Henty Gold Mine near Mt Read and the Anthony Power Development commenced operations.

The township of Zeehan started as a mining town in the 1880s and in 1903 was known as the Silver City. With a population of 10,000 it was the third largest town in Tasmania. Although the nearby Renison Bell remains one of the world's largest underground tin mines, Zeehan is now one of the smaller towns in Tasmania.

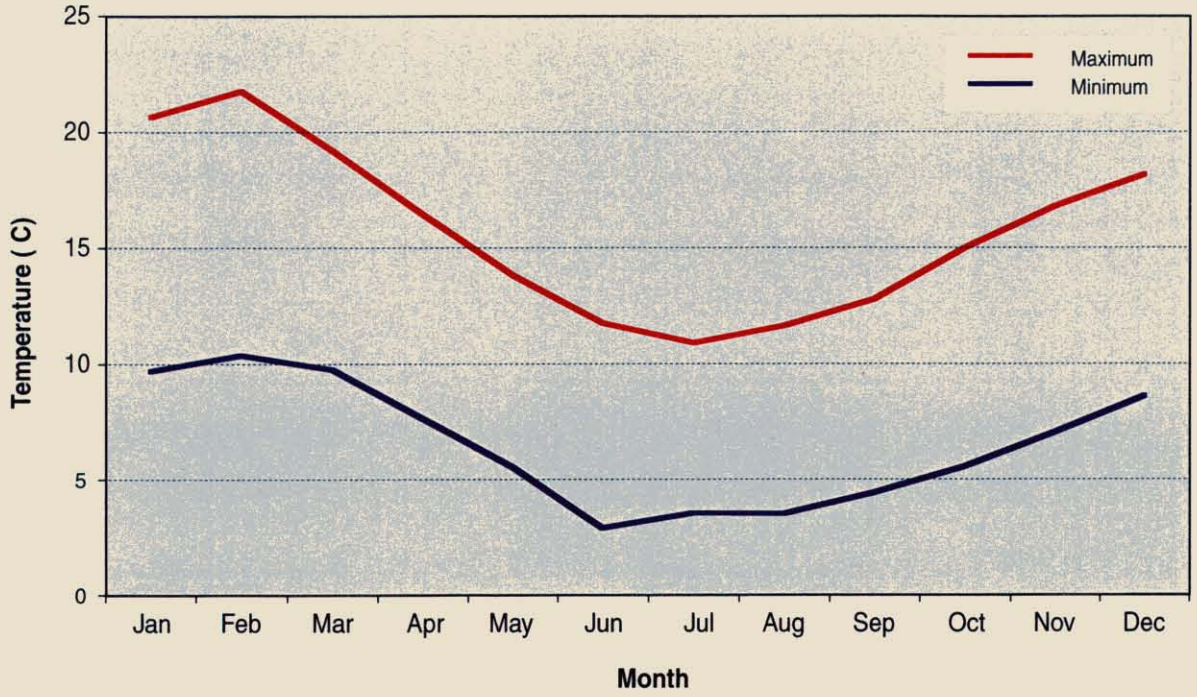
3.2.2 Climate

The Bureau of Meteorology established a meteorological station at the Zeehan Memorial Museum (latitude 41°53'01" S, longitude 145°20'06" E, elevation 172 m) in February 1968. Temperature and wind speed and direction were recorded from 1969 to 1978, while rainfall records commenced in 1968 and rainfall continues to be recorded.

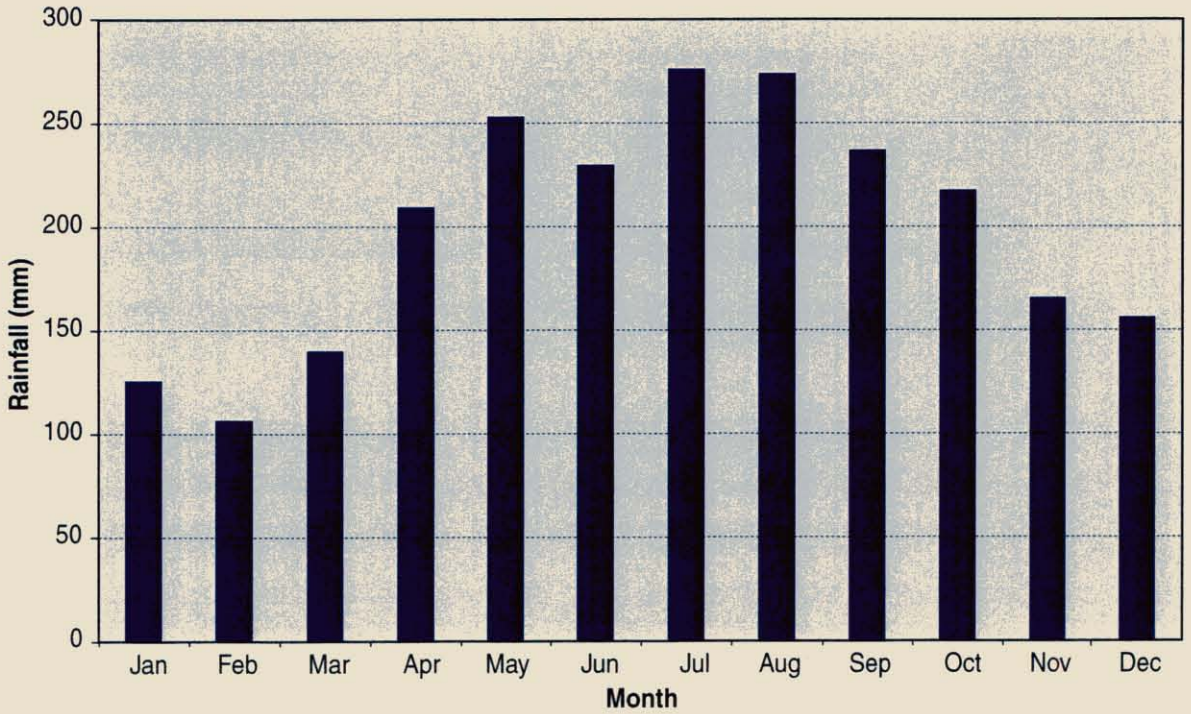
The climate of the project area is characterised by cool temperatures, and high and consistent annual rainfall. Rainfall distribution in western Tasmania is generally high throughout the year, with June to September being the wettest months and December to March the driest. Drought conditions are rare. The data shown in Figure 4 at Zeehan confirms this distribution, with an average annual rainfall at the Zeehan Memorial Museum site of almost 2,290 mm.

Mean monthly minimum temperatures at Zeehan range from 2.9°C in June to 10.3°C in February and maximum temperatures range from 10.9 to 21.8°C (July to February) (Figure 4). Although not exposed to extreme and persistent winter conditions, the area is subject to an average of 13.8 rain days per month.

Average daily maximum and minimum temperatures



Average monthly rainfall



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NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Rainfall and temperature at Zeehan	Figure No: 4
	File No: 938_F4_HB	Allegiance Mining NL		

The proposed mine is located in the pathway of the strong westerly winter winds known as the 'Roaring Forties'. However, the local occurrence of strong winds is highly site-specific and depends on topographic exposure (Bureau of Meteorology, 1996, pers. comm.). Westerlies prevail throughout the entire year: northwesterly during winter, southwesterly during summer.

3.2.3 Land Tenure and Land Use

The mine area is located within the Municipality of West Coast and the general area has been zoned 'rural' by the Zeehan Section 46 Planning Scheme No.1 1992. This is a zone in which extractive industries are a permitted land use. Allegiance holds several exploration licences in the Zeehan area as shown in Figure 5. In particular, exploration has focused on the Avebury and Melba Flats prospects, however, it is the Avebury prospect which is the subject of this report. The closest adjoining lease is the Comstock Zinc Mine located approximately 2 km northeast of the Avebury Nickel Project.

The Zeehan area and surrounds have been under commercial and recreational land use for many years. Current land uses comprise mineral exploration and mining (along the Zeehan to Trial Harbour Road), general nature conservation, walking, fishing, photography and off-road vehicle driving, particularly on the dunes along the coast.

The project area is within multiple use State Forest designated under the *Forestry Act 1920.* Part of this area is classified as Informal Reserve (stream side corridor) (Figure 6) through Forestry Tasmania's administrative and management process as part of the Comprehensive Adequate and Representative (CAR) Reserve System[1]. These land classifications are recognised and accredited under the Regional Forest Agreement (RFA), which was signed between the Australian and Tasmanian governments in November 1997. The area of State Forest east of Comstock Creek has been classed as high quality wilderness (i.e., wilderness value greater than or equal to 12) (Plate 2) area through the RFA process (Figure 6). (P. Wells, pers. comm.)

Under the RFA, these classifications are subject to periodic review. The RFA permits mineral exploration and mining within State Forest, Informal Reserves and wilderness areas (K. Bird and P. Wells, pers. comm.).

3.2.4 Physiography and Topography

The Avebury Nickel Project is located in a relatively open area of undulating buttongrass plains mixed with eucalypt and tea-tree forest and rainforest. The area is bounded to the northwest by steep, forested, mountainous terrain (e.g. Mt

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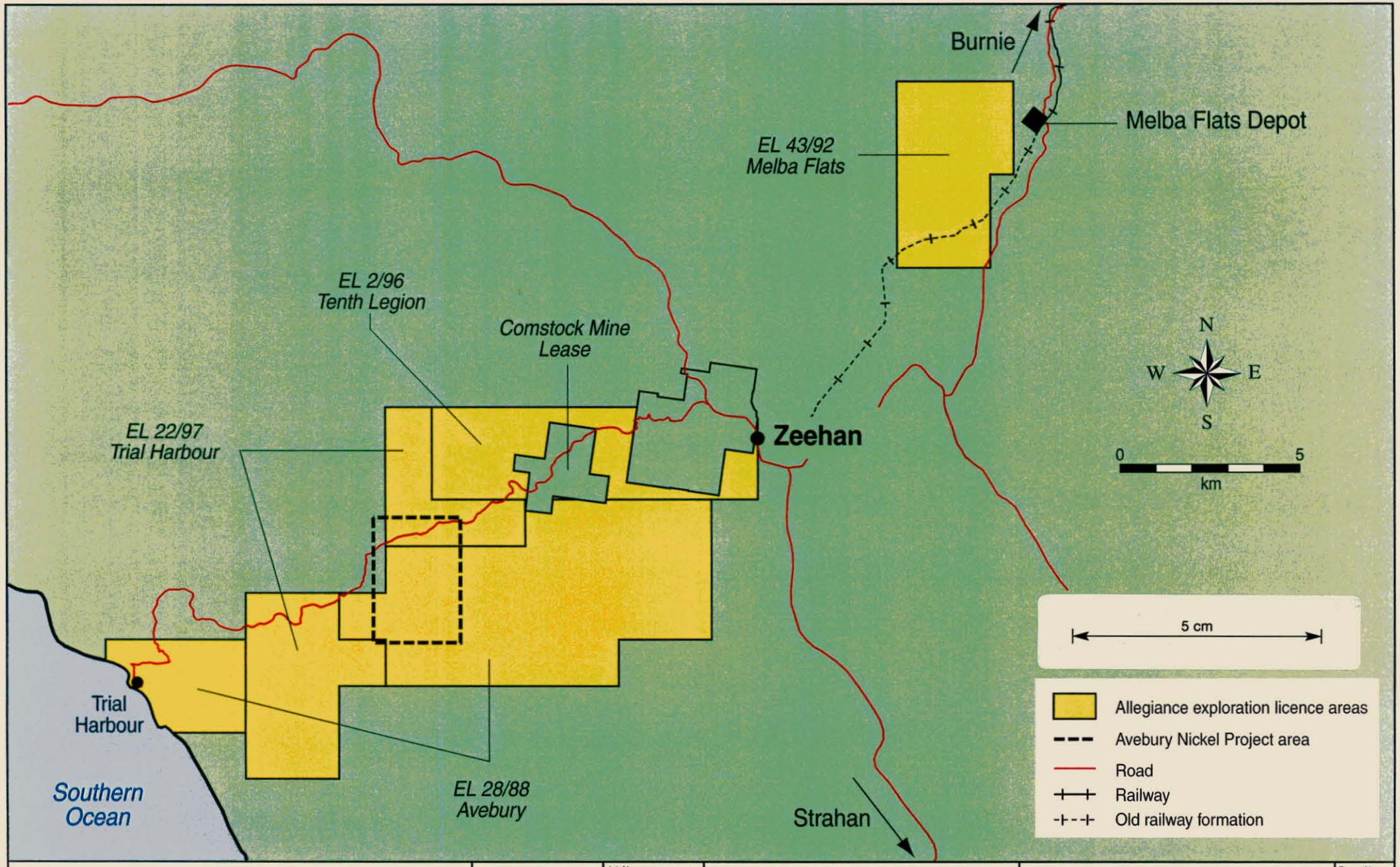
[1] The CAR Reserve System was developed under the National Forest Policy Statement to protect biodiversity, old growth forest and wilderness values (Mineral Resources Tasmania (1999)). It is based on JANIS, the national reserve criteria. (Regional Forest Agreement (1996)).



	NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Project area	Figure No: 5
		File No: 938_F5_HB	Allegiance Mining NL		

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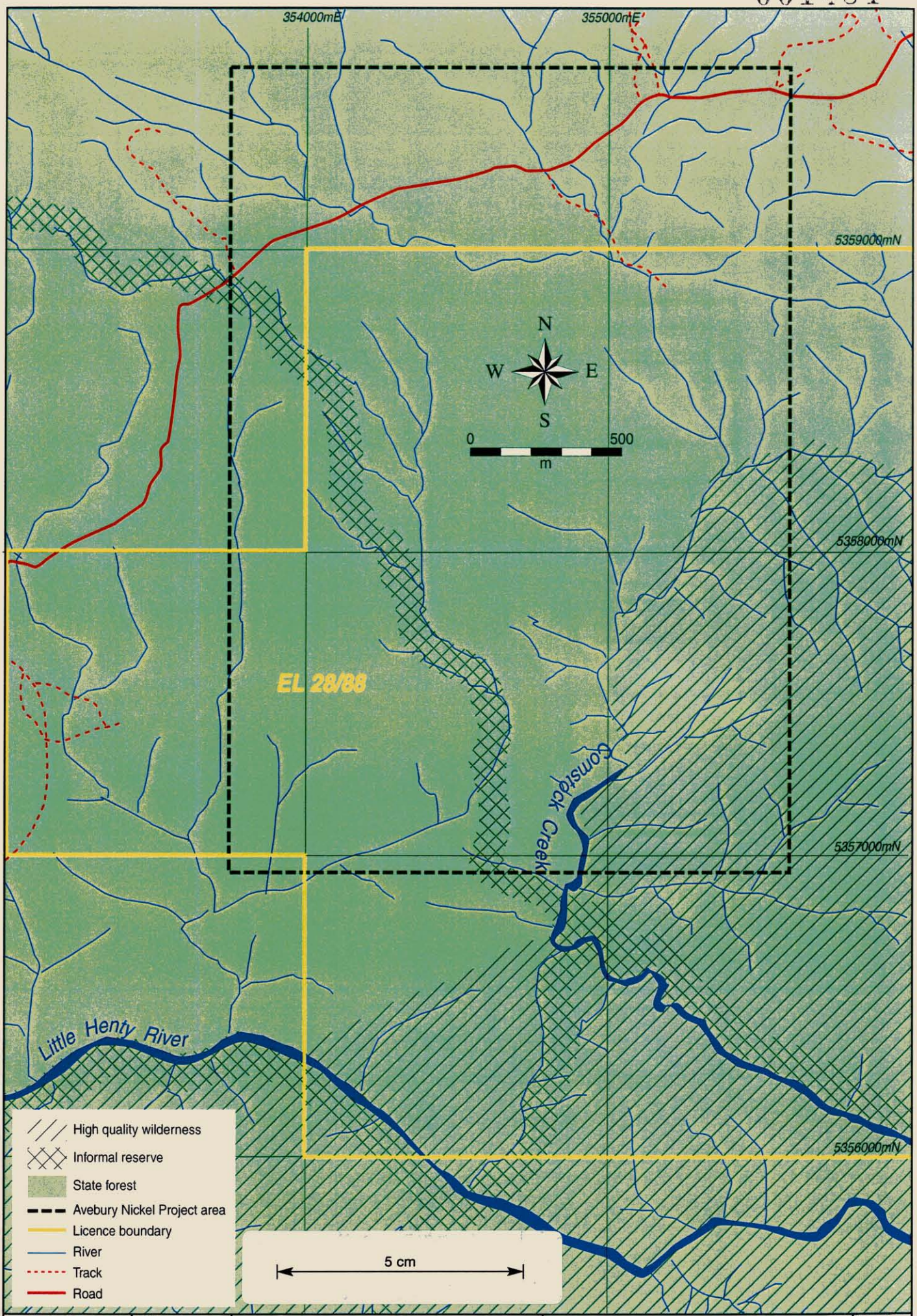


NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Land use	Figure No: 6
	File No: 938_F6_HB	Allegiance Mining NL		

Agnew at 846 m), to the north-northwest by a gentle ridgeline marking the catchment divide with the Pieman River, to the east by Mount Zeehan (702 m) (Plate 3), to the west by coastal dunes and to the south by the incised valley of the westerly-flowing Little Henty River.

The area is incised by a series of sub-parallel, south-trending tributaries of the Little Henty River, which include Comstock Creek.

3.2.5 Geology and Geochemistry

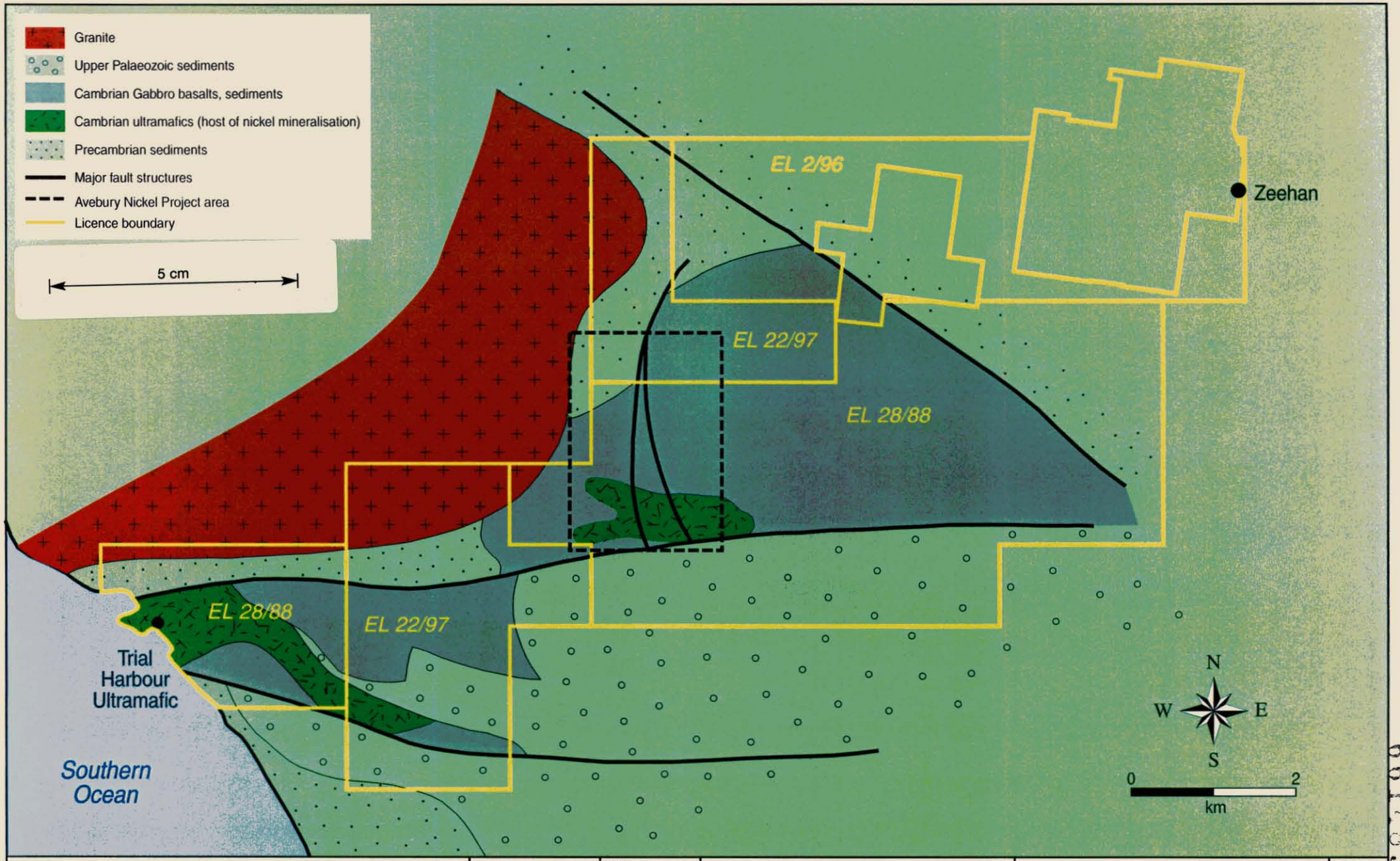
The Avebury nickel sulfide mineralisation occurs within moderately to steeply dipping Cambrian serpentinite bodies lying within a sequence of Cambrian metamorphic rocks dominated by hornfels and chert. The serpentinites were derived from rocks of the McIvor Hill ultramafic complex (Figure 7); the other rocks were originally siltstones and volcaniclastics of the Crimson Creek Formation. The serpentinites in the Avebury area appear to be concordant with the enclosing Crimson Creek Formation. The whole sequence was metamorphosed and metasomatised during the intrusion of the Heemskirk granite at the time of the Devonian Tabberabberan Orogeny.

During the metamorphic and metasomatic events, nickel sulfide mineralisation was mobilised and concentrated in places within the serpentinite bodies, particularly within and immediately adjacent to their contacts (Figure 8). There are sporadic occurrences of nickel sulfide outside the serpentinites but the outer boundary of the nickel mineralisation generally coincides with the sharp serpentinite contacts. The internal boundary of the nickel sulfide mineralisation is defined by the transition from high grade to background grade nickel mineralisation.

The rocks carrying the nickel mineralisation are serpentinite and altered serpentinite (Plate 4). The nickel mineralisation carries significant magnetite, ranging from sparse to massive, and this will report to the tailings. Sulfides, arsenides and sulph-arsenides are present, dominated by pentlandite and pyrrhotite, however, these should mostly be recovered to concentrate (see Section 2 – Tailing).

Waste rock requiring temporary surface storage will predominantly originate from decline development through the enclosing Crimson Creek Formation, dominated by hornfels and chert with lesser siltstone and shale and much smaller amounts of dolomite and conglomerate. These rocks are extensively altered by metamorphic processes. Narrow sporadic zones of pyrrhotite/pyrite mineralisation have been intersected in drill holes with sulfur assays of about 3%. A key feature of pyritic material is its potential to oxidise and form acid (see Section 3.3). Waste rock will also originate from parts of the serpentinite which, although mineralised, do not reach ore grade.

Some fibrous material (chrysotilic serpentine), similar to that seen at Savage River, has been identified. However, a negative relationship exists between the zones of nickel mineralisation and this material.



	NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Avebury - district geology	Figure No: 7
		File No: 938_F7_HB	Allegiance Mining NL		

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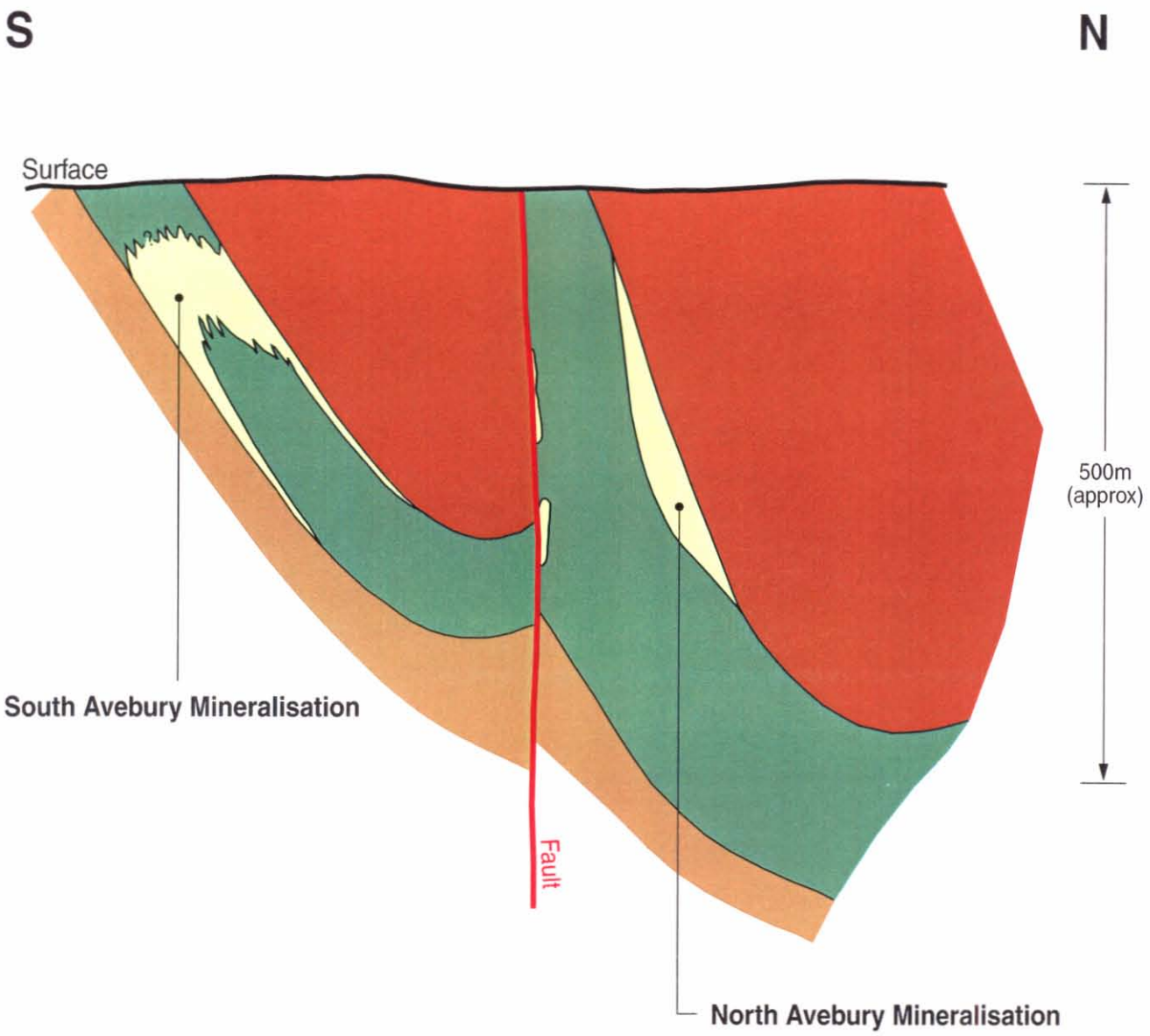

Legend

- Nickel mineralisation
- Serpentinite
- Hornfels and chert
- Chert, conglomerate, very altered rocks
- Fault

5 cm

NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Avebury - local geology	Figure No: 8
	File No: 938_F8_HB	Allegiance Mining NL		

3.2.6 Terrestrial Flora

Like much of the cold temperate rainforest climax vegetation of western Tasmania, the vegetation of the project area is characteristically altered into a complex mosaic of differing floral associations in various stages of successional development after fire (Plates 5 and 6). Apart from regular control burns through the buttongrass plains (every 5-6 years) the last bushfire through the area is thought to have occurred in the early 1980s (L. Clark , pers. comm.).

No flora reports are available concerning the vegetation conditions in the project area. There is, however, a Forestry Tasmania map showing forest type. This has been simplified and is shown in Figure 9.

The vegetation can be described as mainly comprising rainforest communities (Plate 7) and mixed forest (as the rainforest degrades to wet sclerophyll communities, Plate 8) or a combination of both (Duncan and Packham, 1994). The rainforest is a poorer quality, small-crowned, shorter, thamnic rainforest dominated by myrtle with leatherwood. The understorey comprises leatherwood, myrtle, sassafras, and wet fern species. The Eucalypt forest is dominated by a mixture of *Eucalyptus delegatensis* and *Eucalyptus obliqua*, with secondary species of *E. delegatensis* and *obliqua* with *E. nitida* around the poorer sites near the edge of the buttongrass plains (L. Clark, pers. comm.). Tea-tree is also present in the understorey (Plate 9) (Duncan and Packham, 1994).

A search was conducted on the Tasmania Parks and Wildlife Service database for rare and threatened species, in accordance with Schedule 5 of the *Threatened Species Protection Act 1995,* within the project area. The closest plant of any significance (352500 E, 5357500 N, approximately 1.5 km west of the project area), is the rare *Orthoceras strictum* (Horned orchid). However, it was last recorded in 1973 and its current status at this location is unknown.

Few weeds were evident during the site inspection. While broom and gorse are not present large clumps of cutting grass are evident along the tracks (Plate 10). This is thought to have been introduced when the area was selectively logged for celery-top pine and blackwood, using horses in the 1950s. Signs of *Phytophthora cinnamomi,* (Dieback or Rootrot)[1] or myrtle root, while not noted during the site visit, have been recorded elsewhere on the West Coast and this issue will require further investigation.

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[1] This is an introduced plant fungus which causes dieback and death in many native plant species. The fungus is water-borne and lives in soil. One of the main means of spread of the fungus is by carriage of soil particles from infected to uninfected areas via walkers, vehicles and earth-moving equipment.

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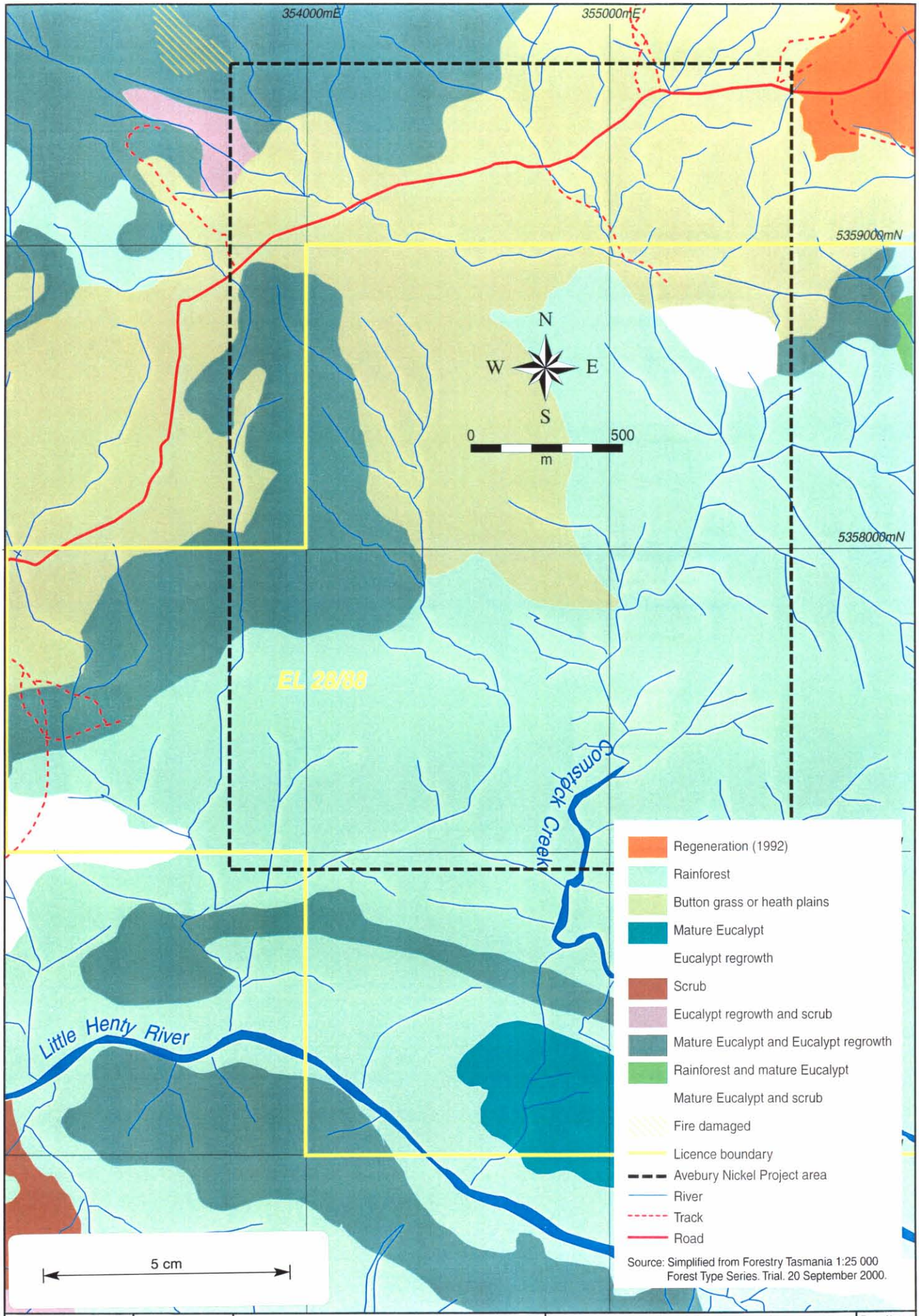


NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Vegetation	Figure No: 9
	File No: 938_F9_HB	Allegiance Mining NL		

3.2.7 Terrestrial Fauna

Compared to other parts of mainland Australia, the range of Tasmanian mammalian species is depauperate and is associated with reduced competition between species and a broadening of the range of habitats occupied (Hocking and Guiler, 1983).

No fauna studies are available concerning the terrestrial fauna in the project area. It may be reasonable to assume that faunal species found in the project area are common along the West Coast in similar habitats of temperate rainforest and wet sclerophyll forest. A low level of mammal activity is a feature of the dense wet forests of western Tasmania and hence is also expected to be a feature of the project area, with the dominant mammal fauna being small ground foragers. Rainforest avifauna is expected to be somewhat depauperate in the project area (as elsewhere on the West Coast) compared with the drier eastern parts of Tasmania. This can be attributed to the closed canopy reducing the availability of soft food and restricting access to invertebrate food.

A search was conducted on the Tasmania Parks and Wildlife Service database for rare and threatened species, in accordance with Schedule 5 of the *Threatened Species Protection Act 1995,* within the project area. Three vulnerable species were recorded (354877 E, 5357979 N) within the project area on a tributary of Comstock Creek (Table 1). However, as with the significant flora species, they are not recent recordings and will require verification.

Table 1 Vulnerable fauna species

Species name	Common name	Date of record
Aquila audux fleayi	Wedge-tailed eagle	28 February 1981
Pachyptila turtur subantarctica	Fairy Prion southern sub-sp	13 December 1977
Halobaena caerula	Blue petrel	3 December 1978

3.2.8 Archaeology and Heritage

No archaeological or heritage surveys have been conducted in the project area. However, work undertaken in the general Savage River area, located approximately 45 km north of the project area, shows a paucity of archaeological sites, which has previously been attributed to the densely vegetated and rugged nature of the region (Miedecke, 1996). In work undertaken for Savage Resources Ltd, du Cros and Associates (1993, 1995) and CHC (1995) surveyed a number of areas in the Savage River area and reported finding no Aboriginal or historic European archaeological sites, nor were areas of archaeological sensitivity noted. Similar findings will be expected for Allegiance's mine area footprint, although this will need to be confirmed by survey.

A search of the Australian Heritage Commission's Register of the National Estate Database indicated that there are no places of natural, historic or indigenous significance located within the project area that are currently registered. The

Tarkine Wilderness Area is located approximately 18 km to the north of the Avebury Nickel Project.

3.2.9 Hydrology

The Avebury Nickel Project area is located both east and west of Comstock Creek which is a tributary of the Little Henty River; its confluence is approximately 2 km downstream of the proposed mine area. The Little Henty River flows west into the Southern Ocean, south of Trial Harbour (see Figures 1 and 6). The Comstock Creek catchment has an area of some 20 km^2, in which the Comstock Zinc Mine is located approximately 2 km upstream of the project area.

Comstock Creek flows through a relatively steeply incised valley which is characterised by rapids and small waterfalls (Plate 11).

There are no gauging stations on either the Little Henty River or Comstock Creek, therefore there is no discharge data. For the reasons documented in Section 3.2.10 it is recommended that gaugings are taken, so that a stream discharge rating curve can be established and river flow characterised. This information will provide input data to the water quality, suspended sediment concentrations and sediment transport and aquatic biology investigations.

3.2.10 Surface Water Quality

Water quality objectives to determine the required level of protection are based on the environmental values (also known as beneficial uses) of the particular waterbodies. ANZECC (1992) defines five environmental values:

- Ecosystem protection.
- Recreation and aesthetics.
- Raw water for drinking water supplies.
- Agricultural water (i.e. for irrigation and livestock watering).
- Industrial water.

For water quality indicators such as pH, conductivity and toxicants (e.g. trace metals), the highest quality water is generally that required for the protection of aquatic ecosystems. For Comstock Creek, ecosystem protection and recreation are considered to be the applicable values.

Streams in western Tasmania are characterised by:

- High humic acid concentrations reflected in elevated dissolved organic carbon (DOC) levels and neutral to slightly acidic pH values.
- Low alkalinity, conductivity and total suspended solids (TSS) values.
- Low concentrations of trace metals.
- Significant ability of the dissolved humic material to complex trace metals such as copper (Cu).

These features are clearly shown by Koehnken (1992) in the 15-month Pieman River Environmental Monitoring Program. The waters of Comstock Creek and Little Henty River (Plate 12) are expected to exhibit these characteristics, however, the quality of Comstock Creek may differ to expectation due to input from the Comstock Zinc Mine upstream of the project area.

The Department of Primary Industries, Water and Environment (DPIWE) has taken a series of grab samples from Comstock Creek at the Comstock Zinc Mine upstream of the proposed project over the last 4 years and the results are presented in Table 2. There may have been recent changes to water quality in the river and these results are indicative only (R. Gilmour, pers. comm.). The results indicate that above the Comstock Zinc Mine adit, concentrations of aluminium, copper, iron, lead and zinc all exceed both the ANZECC (1992) and ANZECC/ARMCANZ (2000) guidelines for the protection of freshwater aquatic ecosystems. Below the adit, in addition to these metals, cadmium and manganese are also outside these guidelines. Although within the ANZECC/ARMCANZ (2000) guidelines above the adit, cobalt and nickel exceed them below the adit.

With little information available concerning water quality data for Comstock Creek, and nothing available for the Little Henty River, it is important that baseline conditions are established so that the environment can be managed in accordance with the objectives of the State Policy on Water Quality Management 1997. The responsibilities for this may be complicated by inputs to the system from the Comstock Zinc Mine.

It is recommended that Allegiance establishes a water quality monitoring program as soon as possible. Various approaches to this were detailed in a letter to Newnham Exploration and Mining Services on 30 November 2000 (Appendix 2).

3.2.11 Groundwater

No information is available concerning the hydrogeological conditions in the project area. However, the evidence from Allegiance's exploration program at Avebury suggests that the groundwater is below a depth of 500 m (drill core depth). A good water return on the drill holes was noted and this reflects a geological environment of tightly folded and fractured bedrock and a high rainfall environment. This is consistent with the experience of other mines in the region.

3.3 Environmental Issues

Given the notional project outline described in Section 2 and the environmental setting described in Section 3.2, the key environmental issues raised by the future development of a mine, beneficiation plant and associated infrastructure are:

- The extent of direct land alienation, in particular:

- Loss of existing land uses where project components will be located. This includes both the mine and beneficiation plant areas. This is not expected to

Table 2 Water quality in Comstock Creek (µg/L)

Site	Comstock Creek above adit	Comstock Creek below adit	Comstock Creek below adit	Comstock creek below adit	ANZECC (1992) **	ANZECC/ ARMCANZ (2000)†
Date	10/08/99	22/10/97	10/06/99	10/08/99	-	
Al (D)	287		1780	1430		
Al (T)	370	2560	2190	2110	Note 1	Note 2
Cd (D)	<1		19	19		
Cd (T)	1	39	19	20	0.2-2*	2*
Co (D)	<1		23	25		
Co (T)	<1	66	23	26	-	5
Cr (D)	1		<1	1		
Cr (T)	<1	7	<1	<1	10	10
Cu (D)	11		10	14		
Cu (T)	9	<5	12	13	2-5*	1.4*
Fe (D)	681		351	678		
Fe (T)	1300	59100	25100	30700	1000	[B]
Mn (D)	129		4290	4220		
Mn (T)	138	10800	4270	4230	1000^	330
Ni (D)	3		47	51		
Ni (T)	4	147	42	51	15-150*	11*
Pb (D)	52		81	80		
Pb (T)	85	380	147	155	1-5*	3.4*
Zn (D)	284		14700	14400		
Zn (T)	298	33000	14900	14900	5-50*	8*[C]

Notes:

** ANZECC (1992); water quality guidelines for the protection of freshwater aquatic ecosystems.

† ANZECC/ARMCANZ (2000); draft water quality guidelines for the protection of freshwater aquatic ecosystems.

D – dissolved, T – total

Note 1: Al<5 µg/L if pH≤6.5, Al<100 µg/L if pH>6.5

Note 2: 40 if pH>6.5, if pH<6.5 insufficient data to give a reliable value.

* Dependent on hardness.

^ Mn; USEPA (1988), State Water Quality standards Summaries. (Final Report). Battelle, Arlington, Va. September. 408 p. PB89-141634.

[B] Insufficient data to derive a reliable trigger value.

[C] Figure may not protect key test species from chronic toxicity.

Data source:

Department of Primary Industries, Water and Environment

be a significant issue given that the mine will be an underground operation and the main infrastructure for the project will be located on the plains off the Trial Harbour Road and not within areas of rainforest.

- Effects on biodiversity and conservation values. From a regional context, it is likely that the species and vegetation communities and the associated fauna in areas which will be disturbed by the mine are adequately represented elsewhere in Tasmania, particularly along the West Coast.

- Water quality. Water management and the effects of runoff from the project area and/or project facilities on beneficial uses of the downstream waters, with particular emphasis on:

  - Impacts associated with possibly elevated suspended sediment concentrations or other contaminants (e.g. trace metals such as Pb or Zn).

  - The potential for ore, overburden and tailing to generate acid due to the oxidation of contained sulfide minerals.

    Given that a small amount of waste rock material associated with the project contains up to 3% sulfur, and is likely to be either acid forming or potentially acid forming, testwork to define the nature and proportion of acid generating material and whether any buffering capacity exists in the surrounding material will be required. Potentially acid forming material should be preferentially used as backfill (ahead of non-acid forming material) and encapsulated while on the surface. If testwork indicates that acid mine drainage is not expected to be a problem then selective handling can be minimised, runoff monitored and contingency methods for managing runoff (eg., identifying source, dosing runoff with lime) identified.

- The potential for Allegiance to be held responsible for degradation of the aquatic environment downstream of the mine area. As noted previously, the waters of Comstock Creek are already impacted by mining activities at the Comstock Zinc Mine upstream of the proposed mine. Mining at Comstock Zinc Mine could result in ongoing contamination of Comstock Creek and ultimately Little Henty River. Therefore, the following needs to be addressed:

  - The existing status of Comstock Creek needs to be ascertained so that baseline conditions prior to Allegiance's activities can be established.

  - If the water quality monitoring program indicates that Allegiance will be operating in a existing degraded environment then consideration should be given to establishing an appropriate framework (legislative or otherwise) whereby Allegiance is not held responsible for pollution caused by others.

- The 'outrage' factor i.e., the extent to which general antagonism to the project might be generated throughout the community (and not necessarily

related to the facts). A review of articles which have been published in the Mercury (Hobart) newspaper since December 1997 and which concern nickel projects shows that nothing of a negative nature has been reported. The articles simply report the encouraging results from the exploration program.

However, part of a proposed exploration track will be located in a high quality wilderness area (see Section 3.2.3) and, although there may be minimal impact, could be perceived to threaten wilderness values. Therefore, consultation will be required with groups such as the Tasmanian Greens, The Wilderness Society, Tasmanian Conservation Trust, Friends of the Tarkine, Tarkine National Coalition and the North West Walking Club, if only to allay fears and misconceptions about the project and to ensure that the general public understands the nature of the development and its interaction with the environment.

- Permanent storage of waste solids. This will be an issue if it is not possible to use all waste rock as backfill. However, use as backfill is not an option for tailing. As described in Section 2, and shown in Figure 3, tailing will be stored in a dam northwest of the mine and processing areas. The area of the dam will occupy almost the entire catchment of Hemp Creek. Coleman and Contiguous creeks, to the west and east of Hemp Creek respectively, drain most of the catchment north of the tailing dam. A natural north-south divide between Hemp Creek and the infrastructure area, excluding the portal and pad, is located downstream of the dam within the Hemp Creek catchment. Due to the possibility (albeit remote) of failure or uncontrolled overflow, which could threaten the entire operation, additional management measures may be required. There are no statutory requirements which dictate that infrastructure can not be located downstream of tailing dams, however, such a situation will increase the hazard rating of the dam (V. Grun, pers. comm.). DPIWE would recommend that the situation is assessed by geotechnical engineers.

- Socio-economic effects on towns and communities associated with or affected by the development. While both positive and negative effects are possible, it is likely that the positive effects such as annual royalties, direct contributions to the local economy through the purchase of goods and services, and increased employment opportunities (both direct and indirect) will outweigh the negative effects.

- The scale and nature of rehabilitation of the mine site and beneficiation plant, and the long-term integrity, end use and rehabilitation of tailing impoundment(s).

- Die back due to *Phytophthora cinnamomi.* This is recognised as a problem throughout Tasmania and the potential impacts of the operation with regard to this issue will require investigation.

- Potential air quality impacts. These include dust in the mine area and along haul roads. With appropriate management, these will be minimal.

- Fibrous materials. Although there is thought to be no fibrous material in the mineralised zone it does occur in some waste rock material and laboratory analysis should be undertaken to fully assess occupational health and safety risks.

- Mining operation noise and ground vibration arising from drills, shovels, blasting, and dumping/haulage of ore and overburden. The absence of nearby human habitation ensures that no significant noise or vibration levels are likely to be experienced by non-mining personnel.

- Potential effects on archaeological or cultural sites. Given the findings described in Section 3.2, this is unlikely to be a major issue.

Other minor issues which will need to be addressed include aesthetics, chemicals management, hydrocarbons management, fire control, weed and pest (e.g. feral cats/bees) control, general waste management, public safety risks due to increased road traffic, and hazards or risks associated with shipping consumables/product through the Port of Burnie (e.g. ship discharges, risk of collision, etc).

4. Approvals and Permitting Process

4.1 General Regulatory Environment

Some of the more relevant acts, regulations and statutes which are likely to be applicable to the project are shown in Table 3.

4.1.1 Environmental Management and Pollution Control Act 1994 and the Land Use Planning and Approvals Act 1993

Since the *Environment Protection Act* (EPA) *1973* and associated regulations were enacted in the early 1970s, Tasmania has undergone a complete review of all planning and environmental legislation. This has led to the formulation of the *Environmental Management and Pollution Control Act* (EMPCA) *1994*, which differs significantly to the EPA by integrating environmental approvals with the approvals process under the *Land Use Planning and Approvals Act* (LUPAA) *1993* and adopting sustainable development philosophies.

4.1.2 Environment Protection and Biodiversity Conservation Act 1999

The Commonwealth *Environment Protection and Biodiversity Conservation Act 1999* (EPBC Act), in force since 16 July 2000, enables the Commonwealth to join with the States and Territories to provide a national scheme of environment protection and biodiversity conservation.

The EPBC Act replaces five Commonwealth statutes: *Environment Protection (Impact of Proposals) Act 1974; Endangered Species Protection Act 1992; National Parks and Wildlife Conservation Act 1975; World Heritage Properties Conservation Act 1983*; and *Whale Protection Act 1980.*

Under the Act, actions that are likely to have a significant impact on matters of national environmental significance (i.e., world heritage areas, Ramsar wetlands of international significance, nationally threatened species and ecological communities, migratory species, Commonwealth marine areas and nuclear actions) will trigger the Commonwealth's involvement in the state assessment and approval process.

DPIWE does not anticipate that the Avebury Nickel Project will trigger the EPBC Act. This requires confirmation by discussion with appropriate federal agencies.

Table 3 Relevant Tasmanian Acts, Regulations and Statutes and Australian Standards

Primary Acts and Regulations • Environmental Management and Pollution Control Act (1994) • Land Use Planning and Approvals Act (1993) • Environment Protection Act (Water Pollution) Regulation 1974 • Environment Protection Act (Water Pollution by Oil and Noxious Substances) Regulations 1989 • Environment Protection (Waste Disposal) Regulation 1974 • Environment Protection (Noise) Regulation 1977 • Environment Protection (General) Regulations 1974 • Environment Protection (Atmospheric Pollution) Regulations 1974 • Dangerous Goods Act 1976 and Regulations • Chloro Fluorocarbons and Other Ozone Depleting Substances Control Act 1988 • Forest Practices Act 1985 • Forestry Legislation (Transitional Provisions) Act 1994 • Forestry Rights Registration Act 1990 • Mineral Resources Act 1951 • Mineral Resources Development Act 1995 • Mines Inspection Act 1968 • Mining Act 1958 • Native Forestry Agreement Act 1980 • Native Title (Tasmania) Act 1994 • Pollution of Waters by Oil and Noxious Substances Act 1987 • Threatened Species Protection Act 1995 • Mines Inspection Regulations 1991 • Mines Inspection (Medical Examinations) Regulations 1991 • National Parks and Reserves (General) Regulations 1971 • Regional Forest Agreement (Land Classification) Act 1998 • Water Management Act 1999 • Workplace Health and Safety Act 1995
Ancillary Statutes • State Policies and Projects Act (1993) • State Policy on Water Quality Management 1997 • Aboriginal Relics Acts (1975) • Mines Inspection Act (1968) and (1972) • Fire Services Act (1979) • Groundwater Act (1985)
Australian Standards • AS1216 Classification Hazard Identification and Information Systems for Dangerous Goods • AS1940-1992 Storage and Handling of Flammable and Combustible Liquids • AS2243-10 Chemical Storage • AS2508 Safe Storage and Handling Information Cards for Hazardous Materials • AS/NZS ISO 14001 (Int):1995 Environmental Management Systems - Specification with Guidance for Use • ASNZS ISO 14004 (Int): 1995 Environmental Management Systems - General Guidelines on Principles, Systems and Supporting Techniques

4.1.3 State Policy on Water Quality Management 1997

This policy was prepared pursuant to the *State Policies and Projects Act 1993* and gazetted in September 1997. The objectives of this policy are to:

> (a) focus water quality management on the achievement of water quality objectives which will maintain or enhance water quality and further the objectives of Tasmania's Resource Management and Planning System;
>
> (b) ensure that diffuse source and point source pollution does not prejudice the achievement of water quality objectives and that pollutants discharged to waterways are reduced as far as is reasonable and practical by the use of best practice environmental management;
>
> (c) ensure that efficient and effective water quality monitoring programs are carried out and that the responsibility for monitoring is shared by those who use and benefit from the resource, including polluters, who should bear an appropriate share of costs arising from their activities, water resource managers and the community;
>
> (d) facilitate and promote integrated catchment management through the achievement of objectives (a) to (c) above; .....

In a process managed by DPIWE and based on public consultation, the Environmental Management Pollution Control Board and the West Coast Council will define protected environmental values and the area to which they will apply. Based on these, DPIWE will set water quality objectives, and consequently emission guidelines, which the operation must meet.

Given that the Comstock Zinc Mine is located upstream of the Avebury Nickel Project on Comstock Creek, it is important that Allegiance establishes the current quality of receiving waters.

4.1.4 Other policies

DPIWE was consulted regarding the current status of relevant policies and legislation. The following is a summary of this discussion with Lynne Powell (December 2000) concerning relevant legislation.

DPIWE is in the process of preparing Draft Environmental Protection Policies for air quality and noise. These must be gazetted by 1 January 2002 when the *Environment Protection (Noise) Regulation 1977* and *Environment Protection (Atmospheric Pollution) Regulations 1974* will be rescinded.

Also in process is a review of EMPCA (1994). The three main areas this will focus on are:

- Activities listed in Schedule 2 (status of mining has not been included in this review).
- The fee structure.
- The assessment timeframe.

No timeframe to complete this review has been imposed.

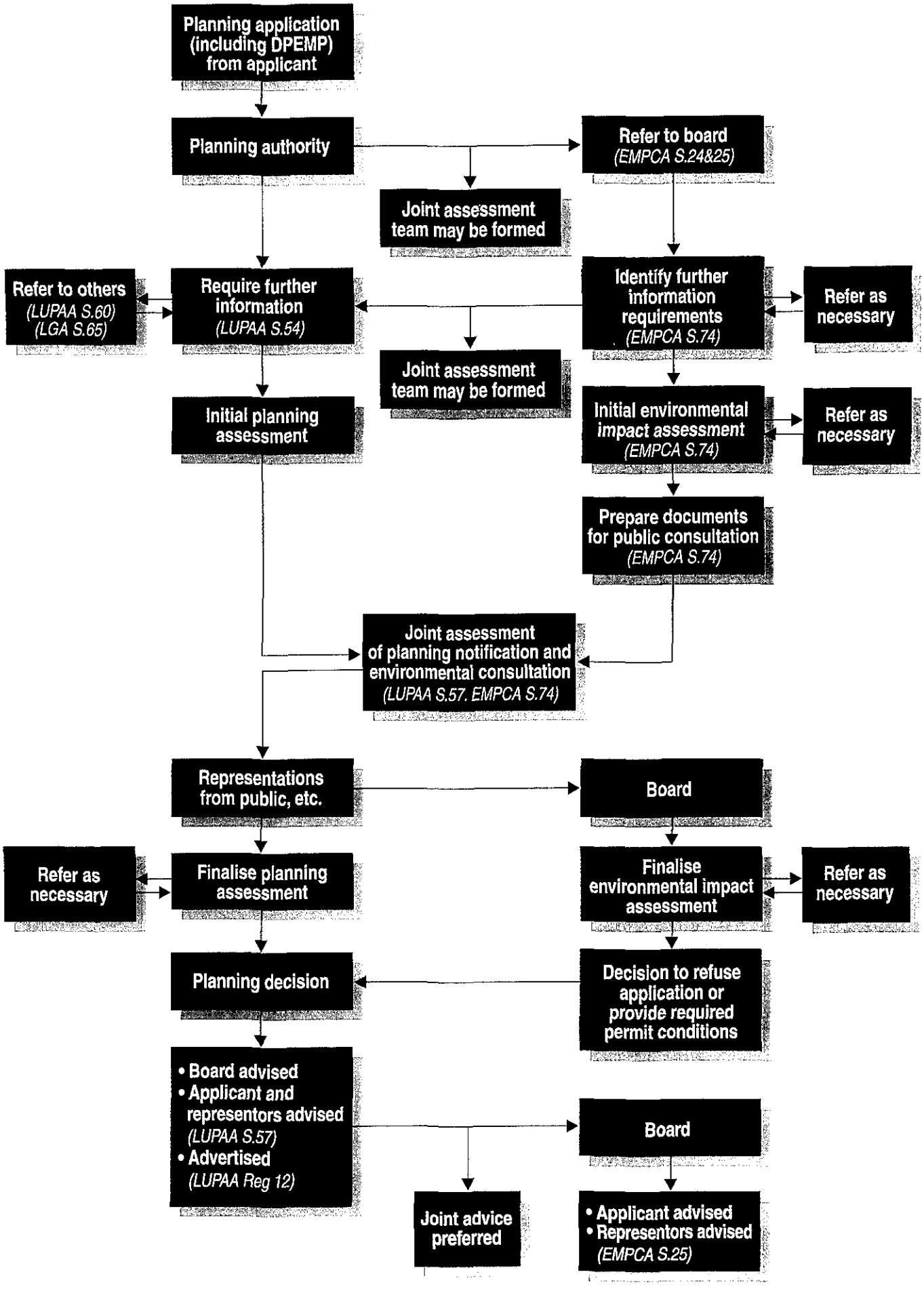
4.2 Environmental Approval Process

According to Schedule 2 of the EMPCA (1994), both the mine and beneficiation plant can be classified as Level 2 activities. The application for environmental approval for these activities is part of an application made to the local Planning Authorities under the LUPAA (1993), as shown in Figure 10. The environmental assessment focuses on the process described in Section 25 of the EMPCA (1994), which in turn makes reference to the environmental impact assessment (EIA) provisions described in Sections 73 and 74 of the EMPCA (1994).

In practice, the Planning Permit application(s) submitted to the West Coast Council will include the Development Proposal and Environmental Management Plan (DPEMP) as supporting documentation, where a single DPEMP will be prepared which addresses all aspects of the project[1]. Guidelines for the DPEMP will probably be issued by DPIWE and are likely to require a description of the proposed operation, an environmental management plan, environmental management strategies and a summary of environmental commitments. The application will be referred by the West Coast Council to DPIWE (in its role as an advisory body to the Board of Environmental Management and Pollution Control) and, providing that adequate information is contained within the document, the DPEMP will be placed on public display. DPIWE will assess the resulting comments and, in conjunction with its own assessment of the document, will prepare a report to be considered by the Board. After public consultation, the Board will either notify the Planning Authorities of conditions which need to be incorporated in the planning permit or direct the West Coast Counil to refuse to grant a permit. The permit may contain additional conditions inserted by the West Coast Council.

It is likely that an Environmental Management Plan (EMP) review will need to be conducted 12 months after commencement of operations. Subsequent updates of the EMP are then likely to be required at 3-yearly intervals.

[1] The operation will use existing facilities at the Port of Burnie for the storage and loading of concentrate. Therefore a planning permit application to the Burnie City Council is not required (I. Newman, pers. comm.).



NSR Environmental Consultants Pty Ltd	Job No: 938	Avebury Nickel Project	Environmental approval process	Figure No: 10
	File No: 938_F10_HB	Allegiance Mining NL		

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5. Future Work Program

5.1 Approach and Timing

Although some information is available concerning the project and the project area, additional work is required before the feasibility of developing a mine and process plant can be demonstrated. While we understand that Allegiance is encouraged by initial exploration results and preliminary economic evaluations, there is no certainty that a mine will be developed in the future. However, systematic environmental investigations at this early stage will ensure that environmental issues are thoroughly researched and resolved in a timely fashion prior to development. The proposed environmental work program is based on the principle that it proceeds in parallel with the bankable feasibility study (BFS), where interdependencies will exist between the engineering and environmental studies. It is recommended that the environmental scheduling is determined with the engineers selected for the BFS.

The proposed environmental work program involves both long- and short-elapsed time investigations, where:

- *Long-elapsed time investigations* focus on those variables where the range in natural fluctuations is unlikely to be captured on a seasonal basis and longer-term measurements are required.

- *Short-elapsed time investigations* include those studies that can be adequately completed within a short period of time, some of which will may need to be completed at an early stage.

We recommend that the long-elapsed time environmental investigations (some of which are proposed in Appendix 2) of the project planning phase should start as soon as possible. The short-elapsed time environmental investigations can be undertaken later as project planning progresses. Once these investigations commence, the preparation of the DPEMP is likely to be completed within six months.

Environmental baseline investigations normally follow the decision to proceed with the project and are not initiated until the relevant government approvals have been obtained. The purpose of the environmental baseline is to characterise the predevelopment conditions and thus provide a statistically- and scientifically-valid basis for comparison with future conditions. This ensures that any future allegation of unforeseen environmental damage caused by the project can be addressed by reference to the baseline.

This section outlines the scope of the proposed environmental investigations for the DPEMP that are envisaged.

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5.2 Program

Once approval to proceed with the feasibility study, and related environmental program, has been received from the Allegiance Board, the following program will be initiated:

- Inception Report (IR): Statement of intent to undertake preparation of a DPEMP, outlining the project, potential environmental concerns and the proposed scope of investigations and schedule to address all issues.
- Consultation Program: Although earlier than the timing requirement in the regulatory framework, identify and engage stakeholders. Effective community consultation is a key component of the investigation and approvals process and the project will benefit greatly if the program can commence as soon as possible after preparation of the IR.
- Detailed investigations (project planning phase): See Sections 5.1 and 5.3.
- DPEMP (environmental impact assessment and management strategy phase): Preparation of the documentation for submission to DPIWE.

5.3 Long-elapsed and Short-elapsed Time Investigations

5.3.1 Hydrological/Suspended Sediment Investigations

Objectives

- To provide a suitable hydrological database to design the operation's water supply, water diversion and drainage structures and to enable the confident prediction of sediment transport and deposition in the downstream drainage.
- To establish the range of background (pre-mining) suspended sediment concentrations in the downstream drainage[1].
- To establish the background riverine sediment load passing through the operation area.

Long-elapsed time investigations

The initial site visit determined that there are no stream gauging stations in the immediate project area at Avebury. We propose that Allegiance:

- Install, monitor and provide routine maintenance of a rising-stage sampler and gauging station in the Comstock Creek and Little Henty River to establish the range of background total suspended sediment (TSS) concentrations.

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[1] In most rivers the suspended sediment concentration is related to river discharge and increases as the discharge increases. The quantification of this relationship is essential for the prediction of downriver environmental impact and will allow utilisation of case study information on the effects of mining on downriver aquatic biota.

The installation of the stream gauging station was proposed in a letter to Mr Lindsay Newnham dated 21 November 2000 (see Appendix 2).

Determining existing background TSS concentrations and loads is crucial given that the Comstock Zinc Mine site is located upstream of the Avebury prospect within the same catchment.

Short-elapsed time investigations

The following supplemental investigations are proposed:

- Calibration of the rising-stage sampler results by undertaking a series of intensive campaigns using depth-integrated suspended sediment sampling.

5.3.2 Meteorological Investigations

Objective

- To provide a suitable meteorological database to design those parts of the mine and process plant requiring knowledge of meteorological variables.
- To provide rainfall data which can be used in conjunction with stream discharge data for impact assessment.

Long-elapsed time investigations

- Install a meteorological station at the Avebury prospect (this was proposed in a letter to Mr Lindsay Newnham dated 21 November 2000 (see Appendix 2)).

5.3.3 Characterisation of Waste Material/Impacts on Receiving Waters

Objectives

- To characterise the ore, waste rock and tailing and define any potential for acid generation and dissolution of any contained metals.
- To assess the suitability of the waste material as a substrate for revegetation.
- To make an assessment of the potential for acid rock drainage (ARD) and/or drainage with elevated metal contents for input into waste disposal design and management.
- If necessary, to evaluate pre-emption or mitigation and control measures.
- To determine the impacts on receiving water quality relative to Tasmanian and Australian water quality standards and guidelines.

Long-elapsed time investigations

The following investigations are proposed:

- Routine analysis of drill core (both ore and waste rock) for total sulfur and total carbon as an initial assessment of the potential for acid generation.

- Conduct a laboratory program (kinetic testwork) on representative rock types. The various rock types will be watered and the leachate routinely sampled and analysed for evidence of sulfide oxidation and metal dissolution.

Short-elapsed time investigations

The following supplemental investigations are proposed:

- Undertake a detailed program to fully assess the variability and nature of waste rock and tailing and its potential to release dissolved metals (static testwork for different rock types, etc).
- Once representative samples have been taken, the testwork has been completed and the results are available, the potential for acid formation and dissolution of any contained metals in the pit, the waste dumps and the tailing will be assessed and implications for mine design, management and control will be addressed.
- If necessary, undertake mixing tests to assess the geochemical behaviour and fate of key contaminants after discharge into the receiving waters. The latter will involve mixing representative samples of material which will potentially report to the receiving waters with local stream water at various dilution ratios/stirring periods and determining both filtered and unfiltered concentrations of key water quality variables. Dissolved oxygen, pH, temperature, conductivity will also be determined.

5.3.4 Environmental Characterisation

Objectives

- To characterise key environmental quality indicators in the water, soil, sediment and biota prior to project development.

Long-elapsed time investigations

The following investigations are proposed:

- Obtain and review existing water quality data from DPIWE.
- Depending on the results of this review, undertake a water quality monitoring program involving an intensive, initial survey at selected sites on Comstock Creek and Little Henty River, taking into account variations in flow, i.e. flood sampling[1], followed by periodic sampling at the same sites.

[1] Water quality in river systems often deteriorates with increasing flow. Sampling of high-flow events is therefore crucial in terms of characterising existing conditions and determining the total loads of suspended solids and associated trace metals moving downstream.

- Collect samples of soil, sediment and aquatic biota. The data will be used to characterise these environmental compartments in terms of relevant variables (e.g. trace metals) and will form the basis of a baseline dataset.

Short-elapsed time investigations

The following supplemental investigations are proposed:

- Additional collection of samples of water, soil, sediment and aquatic biota to expand the baseline dataset to give a statistically- and scientifically-valid dataset.

5.4 Short-elapsed Time Investigations Only

5.4.1 Tailing Toxicity Testwork Program

Objectives

- To assess the toxicity of the tailing.
- If necessary, to identify and evaluate toxicity to determine which components of the tailing are toxic, allowing mitigative action to be undertaken.

Short-elapsed time investigations

The tailing testwork program requires a tailing sample (solids and waste liquor) representative of the tailing to be discharged to the tailing dam. The program is divided into four assessment levels:

- Assessment Level 1 is the minimum required to compare the tailing to Tasmanian standards, and assumes that the tailing is treated in a manner that renders it non-toxic (metallurgical testwork indicates that the tailing will have a pH of approximately 10 at which metals that might be toxic to biota, if they were to remain in solution, will precipitate out). The requirements are described in Section 5.3.3 and will take into account possible dilution of tailing liquor within the TSF prior to any discharge that might occur.
- Assessment Level 2 extends assessment Level 1 to a point where a defensible case can be developed that the tailing is of minimum risk to the freshwater environment. Assessment Level 2 includes screening-level toxicity testing to show that the tailing is non-toxic to sensitive freshwater organisms.
- Assessment Level 3 would be required if the treated tailing was shown to be toxic in Assessment Level 2. It extends toxicity testwork, and examines potentially bioavailable metal and other potential contaminant concentrations. It would provide the basis for a defensible case that the (and potentially toxic) tailing is of minimum risk to the freshwater environment.
- Assessment Level 4 would only be required if (i) the tailing contains high toxicant concentrations and (ii) these concentrations and significant toxicity are likely to be associated with discharge from the TSF. It provides a method

for identifying problematic toxicants, with the aim of allowing targeted modification of the tailing treatment process.

5.4.2 Sediment Transport Investigations

Objective

- To predict mine-derived bed sedimentation in the downstream drainage and the nature and extent of its impact with respect to associated beneficial uses.

Short-elapsed time investigations

The following preliminary investigations are proposed:

- Determination of the channel characteristics and variation in longitudinal slope downriver.
- Sediment sizing assessment by analysis of existing river bed sediment samples and estimation of mine-derived sediment sizing and abrasion.
- Identification of sources and pathways of mine-derived sediment entering the receiving waters.
- Estimation of quantities and timing of mine-derived sediment entering the receiving waters.
- Estimation of sediment transport capacity in the receiving waters.
- Prediction of the likely increment in suspended sediment concentrations during both construction and operations (by numerical modelling and also comparison to actual data for other river systems).
- Prediction of areas of likely riverbed sedimentation and estimate rates of aggradation (by numerical modelling and also comparison to actual data for other river systems).
- Estimation of the long-term behaviour and fate of project-derived sedimentation in the river system.

5.4.3 Aquatic Biota Investigations

Objectives

- To identify species of conservation and/or recreational importance in the receiving waters.
- To characterise the aquatic ecosystems in the receiving waters.

Short-elapsed time investigations

The following investigations are proposed:

- Identify and review local information sources, to determine the known occurrence and abundance of aquatic fauna.

- If appropriate, undertake netting and/or electro-fishing to determine the range of species in the receiving waters.
- If appropriate, employ the AusRivAS[1] stream assessment procedures.

5.4.4 Terrestrial Conservation Values

Objectives

- To determine whether the project area contains any flora and fauna species or other features of special conservation significance.

Short-elapsed time investigations

The following investigations are proposed (using local biologists/conservationists if appropriate):

- Review existing information, including discussions with DPIWE, Forestry Tasmania and other relevant bodies, to ascertain the existing flora and fauna in the project area and other features of potential significance, e.g., high quality wilderness area.
- Depending on the results of this review, undertake surveys to examine and record plant species and other features that occur in areas to be disturbed.
- If necessary, evaluate pre-emption or mitigation and control measures.

5.4.5 *Phytophthora cinnamomi* Investigations

Objectives

- To determine the extent to which *P. cinnamomi* already exists within the project area.
- To identify within the project area where there is a significant risk of introducing or spreading *P. cinnamomi* in native vegetation.
- To advise on appropriate management procedures during construction and operations to minimise the risk of introducing or spreading the fungus to areas at high risk of infection.

Short-elapsed time investigations

We propose that the following be undertaken:

- Review existing soils and vegetation data.
- Identify areas at high risk to infection by *P. cinnamomi* due to project development.

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[1] AusRivAS is the Australian Rivers Assessment System which has only recently been developed to assess river health and involves the examination of aquatic macroinvertebrate fauna.

- Draft safeguards to be implemented during construction and operations to minimise the risk of introducing or spreading the fungus to areas at high risk to infection.

5.4.6 Archaeological and Cultural Investigations

Objectives

- To determine whether the project area contains any sites of archaeological significance or of cultural significance.

Short-elapsed time investigations

We propose that the following be undertaken:

- Consult with relevant government authorities, and appropriate aboriginal groups, and examine the relevant literature to identify the location of any known archaeological sites in the project area.
- If necessary, carry out archaeological surveys of the project area and identify any archaeological sites and describe their significance and implications for siting.

5.4.7 Atmospheric/Noise Investigations

Objectives

- To determine the likely impacts due to air emissions and noise associated with the project.

Short-elapsed time investigations

We propose that the following be undertaken:

- Prepare an air dispersion model using an appropriate gaussian plume model and meteorological data file.
- Assess the potential for environmental nuisance or harm to be caused, particularly in relation to dust and odour.
- If necessary, evaluate pre-emption or mitigation and control measures.

5.4.8 Socio-economic Investigations

Objectives

- To determine ways to maximise the likely positive and minimise the negative socio-economic effects resulting from the development of the project.

Short-elapsed time investigations

We propose that the following be undertaken with particular reference to:

- Employment
- Income
- Cost of living

- Housing
- Community facilities

- Identify the existing education and skill levels, employment and income levels of potentially affected populations and relate these to likely Allegiance recruitment, labour training requirements and payroll.
- Identify local economies, cost structures and standards of living of potentially affected populations.
- Identify existing government services such as health, education, etc. in potentially affected populations.
- Determine potentially affected peoples' expectations for employment, training, business development, infrastructure and services both with and without the project.
- Recommend practical and achievable measures to enhance the positive and minimise the negative socio-economic effects of the project.

5.5 Reporting

We envisage that a series of reports will be prepared for the individual investigations. These individual investigation reports will ultimately become supporting documents (appendices) to the DPEMP. Should Allegiance decide at any point not to proceed with the project, a report will be prepared summarising all investigations and results up to that point in time.

5.6 Environmental Impact Assessment and Management Plan

Experience of mining projects in similar environments shows that the environmental effects of the construction phase are important and, in some cases, are more severe than during the operations phase. Therefore, we propose to structure the environmental impact assessment and management plan formulation so that appropriate emphasis is given to both the construction and operations effects.

5.6.1 Land Impacts and Management

In the mine area, any siting constraints identified during the project planning phase will be highlighted together with implications for the project's general arrangement. Proposed mitigation and management measures to reduce the project's effects on land and particularly relevant conservation or archaeological values will be identified. The extent of the project's unavoidable land alienation and the resulting implications will be identified. The suitability of rehabilitated surfaces for revegetation and other possible end uses will also be described.

5.6.2 Stream Impacts and Management

Changes to stream flows and the local drainage pattern, such as diversion channels, will be described. Implications of these hydrological changes with respect to existing downstream aquatic biota will be described.

Localised bed sedimentation and high concentrations of suspended sediment in the downstream drainage are generally associated with mining developments. Therefore, we suggest that appropriate emphasis be placed on integrating mitigation and management measures and controls into the project design and operation to minimise these effects. These measures and their likely degree of success will be described.

Notwithstanding controls to minimise sediment generation, the transport of mine-derived sediment into and along the stream system will be simulated and predictions made on the incremental sediment loading to the river and its likely behaviour and fate in terms of bed sedimentation and timetable for its ultimate transport downriver.

The potential for acid rock drainage will determine chemical effects on downstream water quality and any toxicity to aquatic biota; however, elevated concentrations of suspended sediment in the downstream drainage are expected to be the principal cause of effects on aquatic biota. The combined physical and chemical effects on identified downstream environmental values will be estimated.

5.6.3 Air Quality Impacts and Management

The air quality impact associated with mining is expected to be small and not significant either locally or regionally. There may be some impacts associated with the process plant, hence management measures may be required and these will be discussed.

5.6.4 Socio-economic Impacts

The socio-economic effects of the project will be described in terms of the positive and negative effects. Components to be covered will include:

- Population, settlement and migration.
- Existing transportation, education and health.
- Opportunities for employment and business development.
- Workforce composition and training.
- Royalty and compensation payments.
- Local, regional and national issues.

5.6.5 Rehabilitation

The DPEMP will address the integration of a possible rehabilitation concept with the mine plan.

5.7 Government Liaison and Approval Process

Approval of the project is likely to be facilitated by the establishment of an effective working relationship with relevant government agencies and local community groups.

We therefore recommend that Allegiance establish and maintain open communications with government agencies and relevant non-government organisations (NGOs) and community groups on the environmental aspects of the project. In practice, this means, as soon as possible, identifying and maintaining contact with key personnel, progressively reviewing results of ongoing investigations, holding regular liaison meetings and agreeing to changes in the scope of work should the project's environmental concepts change.

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6. Schedule

The DPEMP for the mine and process plant will be developed as a single document which would address all project components. The following provides an indication of the overall Environmental Program Schedule culminating in issuing the Land Use Permit(s):

• Commencement of program	Day 1, Month 1, Year 1
• Briefs prepared for DPEMP studies	Day 15, Month 1, Year 1
• Preparation and submission of Inception Report	Day 20, Month 1, Year 1
• Government/NGO/comm. liaison commences	Day 30, Month 1, Year 1
• DPEMP studies commence	Day 30, Month 2, Year 1
• Short-term DPEMP studies finish	Day 30, Month 6, Year 1
• Submission of first draft DPEMP to Allegiance	Day 1, Month 9, Year 1
• Allegiance review of draft DPEMP complete	Day 21, Month 9, Year 1
• Finalisation of DPEMP complete	Day 14, Month 10, Year 1
• DPEMP submitted	Day 17, Month 10, Year 1
• DPIWE's report finalised	Day 21, Month 2, Year 2
• Land Use Permit(s) issues	Day 14, Month 3, Year 2

This schedule is based on our best estimate of the likely time required to complete the relevant studies and obtain approval of the DPEMP.

A key component of the process is the design and implementation of an effective consultation process, as described previously. This is an ongoing and continual process. However, the intensity of the process is likely to vary, depending on the particular stage of the project and initial responses. Extensive consultation is likely to be required at the following stages:

- Initiation of the environmental program.
- Throughout the preparation of the DPEMP, but particularly as results from the studies become available.
- Finalisation of the DPEMP.

It should be noted that provision of ongoing advice and support (which includes ensuring that all commitments contained in the DPEMP are implemented) will occur as required during construction and operation.

7. Cost Estimate

This section provides indicative cost estimates for preparation of the DPEMP and the public consultation process. Costs are itemised for the various studies described in Section 5.

All costs are given in Australian dollars.

7.1 Pre-development costs

Inception Report (total = $5,000-10,000[1])

DPEMP - Technical Studies (total = $276,000)

- Hydrological/Suspended Sediment Investigations ($13,500)
- Meteorological Investigations ($7,500)
- Characterisation of Waste Material/Impacts on Receiving Waters ($40,000)
- Environmental Characterisation ($50,000)
- Tailing Toxicity Testwork Program ($30,000)
- Sediment Transport Investigations ($20,000)
- Aquatic Biota Investigations ($25,000)
- Terrestrial Conservation Values ($20,000)
- *Phytophthora cinnamomi* Investigations ($10,000)
- Archaeological and Cultural Investigations ($15,000)
- Atmospheric/Noise Investigations ($15,000)
- Socio-economic Investigations ($10,000)
- Technical Study Reports ($20,000)

DPEMP - Preparation and Community Consultation (total = $75,000)

- Report Preparation ($55,000)
- Consultation ($20,000)

[1] Depends on formatting

Project Management (total = $40,000)

Total Pre-development cost $401,000

7.2 Operating costs

Based on the following assumptions, the operating cost for the environment would be approximately $0.58/tonne:

- One full-time Environment/Occupational Health and Safety person employed ($80,000/yr).
- Environmental monitoring costs ($20,000/yr).
- Reporting and consultation consumables ($15,000/yr).
- Trouble shooting – consultant fees ($40,000/yr)[1].

[1] These may not be required.

8. References

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Personal communications

Kevin Bird	Mineral Resources Tasmania
Leigh Clark	Forestry Tasmania
Robin Gilmour	Department of Primary Industries, Water and Environment
Vortec Grun	Department of Primary Industries, Water and Environment

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Lynne Powell	Department of Primary Industries, Water and Environment
Ian Newman	Burnie City Council
Penny Wells	Forestry Tasmania
Bureau of Meteorology	
West Coast Council	Planning Department

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Plate 1 View north along track to possible process plant location.

Plate 2 High quality wilderness area.

Plate 3 View across rainforest (deposit) to buttongrass plains (Mt Zeehan left background).

Plate 4 Core tray.

Plate 5 Burnt out tree.

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Plate 6 Evidence of bushfires in the overstorey canopy.

Plate 7 Small crowned shorter Thamnic rainforest.

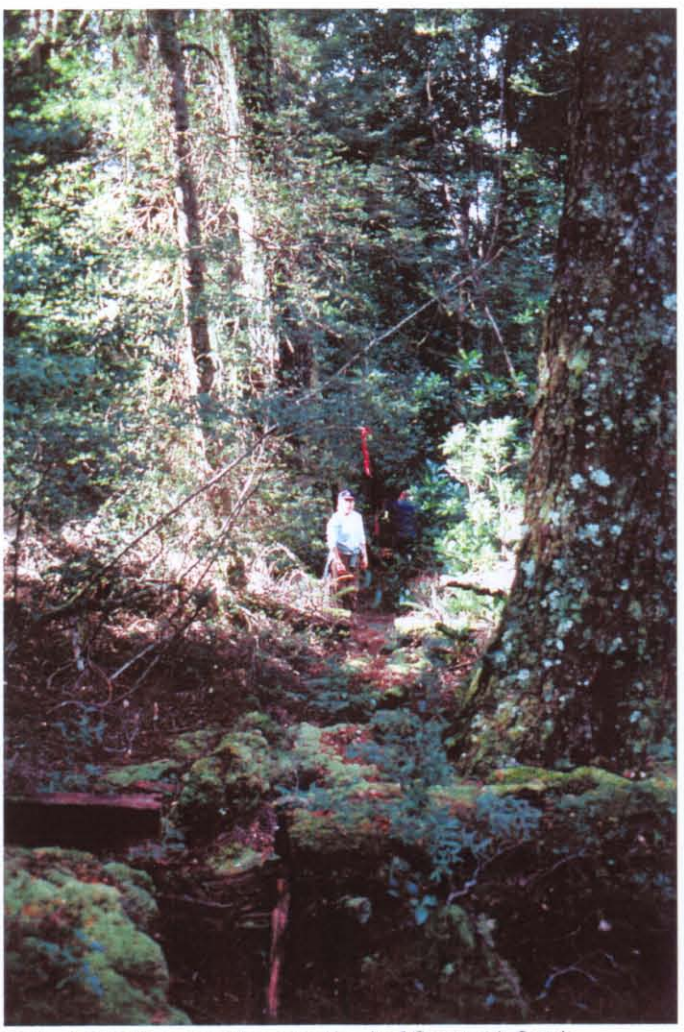

Plate 8 Riparian vegetation, west bank of Comstock Creek.

Plate 9 Exploration track on the boundary of rainforest and mixed forest communities (Tea-tree mixed forest shown).

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Plate 10 Weed invasion.

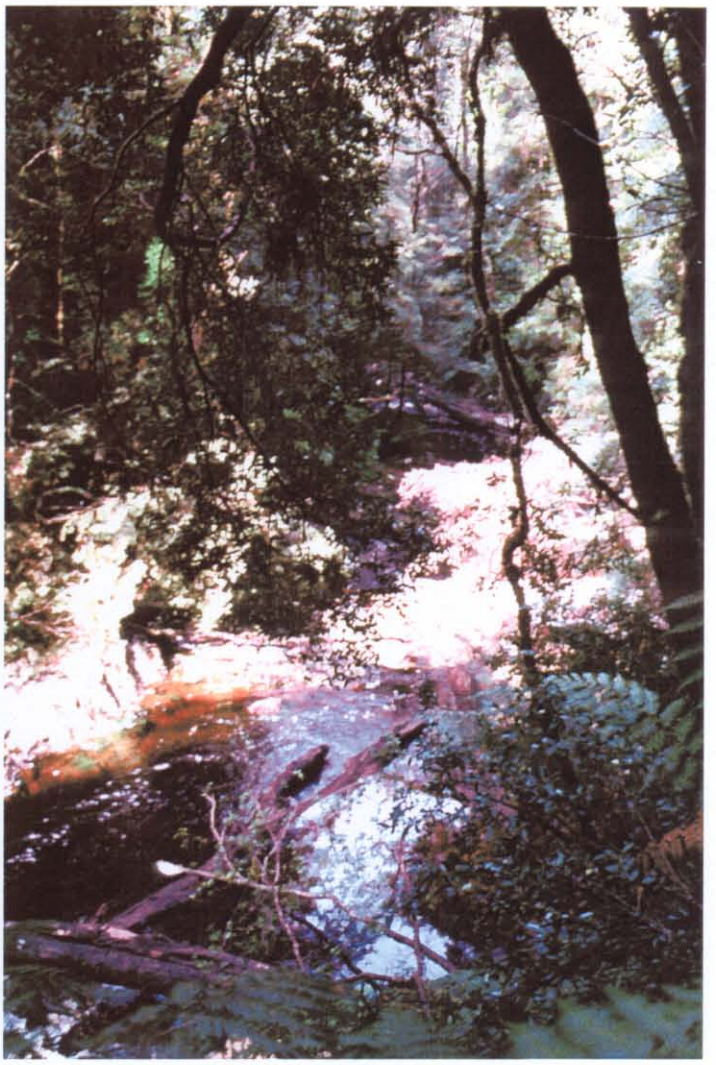

Plate 11 Comstock Creek looking south.

Plate 12 Little Henty River east of Zeehan.

Appendix 1

Sewage treatment plant option

A description of the Biocycle system is provided below to provide Allegiance with an indication of the operating principles for this system and the associated costs. Other accredited systems such as Envirocycle and Turbo Jet 3000 could be investigated.

A possible sewage treatment option, based on a maximum workforce of 40 at any one time, is the installation of a package treatment plant to treat sewage and grey water. Biocycle[1] is an aerated wastewater treatment system (AWTS) which processes effluent to a standard suitable for irrigation. This system is based upon a four stage mini sewage system using two 9 KL reinforced concrete tanks and a series of filters to clarify the effluent, separate dead bacteria and suspended solids and finally, with the use of chlorine tablets, discharge the treated water via drip irrigation. (Disposal to trenches is an option but chlorine would not be used.)

The maximum discharge flow is calculated at 1,800 L/day and seeks to achieve BOD_5 < 20 ppm, suspended solids < 30 ppm and E. coli < 10 ppm. This would meet the *Environment Protection Act (Water Pollution) Regulation 1974* for inland waters and the Guidelines for Reuse of Waste Water in Tasmania, for drip irrigation in areas of human access.

This system would require 240 V power and 250 W for the air blowers which run constantly. The pumps operate as required and use little power. The cost for the design, installation and commissioning of this system would be approximately $28,700 (inlcuding GST). This does not include connection of waste pipes, electricity, filling the tanks with water, fencing, landscaping or the consumption of chlorine tablets (calculated to be 100 tablets/year at $2.20 each). In addition it does not include the cost of spares, which would include blowers and pumps ($1,770 and $540 respectively, including GST).

[1] Biocycle is an accredited plumbing and drainage system which meets the performance requirements of Part C of the *Tasmanian Plumbing Code 1994*. This system was accredited 26 February 1996 and is currently used by Pipers Brook Vineyard, Hoolybank Forestry, HEC and Eidelweiss Restaurant (to name a few of the commercial users).

Appendix 2

Letter to Newnham Exploration and Mining Services

30 November 2000

Mr L Newnham
Newnham Exploration and Mining Services
PO Box 183
EXETER TAS 7275

Dear Lindsay

Re: Avebury Nickel Project – Water quality monitoring

In response to your request, please find following NSR's proposed water quality monitoring program to commence the characterisation of Comstock Creek water quality.

We have proposed two alternatives, depending on the objectives of the program. If the intent is to demonstrate that the water quality has been degraded by upstream activities then NSR proposes that three locations be sampled. On the other hand, if the intent is simply to characterise Comstock Creek then the program could be simplified to one location immediately upstream of the proposed Avebury operation.

1. Sampling locations

Three locations are proposed. These would depend on accessibility (lack of which may result in elimination) but would include a site:

- Downstream of the Comstock Mine, preferably upstream of any tributaries.
- Immediately upstream of the proposed Avebury operation footprint.
- Immediately downstream of the proposed Avebury operation footprint.

2. Frequency of sampling

Summer

During low flow conditions, two times at each location.

Winter

At higher flows, sample twice at each location.

During a flood event sample the rising and falling limb (at a range of flows) at the site immediately upstream of the Avebury operation only.

This winter sampling regime can be simplified if the objective is to provide a very simple characterisation of water quality in Comstock Creek.

This suggested sampling program will provide a preliminary characterisation of Comstock Creek only and would build on any requirements for a DPEMP. It is assumed that only when the Feasibility Study is approved would a more rigorous sampling program be implemented to provide the appropriate level of data for the DPEMP.

3. Analyses

Samples should be analysed for the following:

- General water quality: major ions (Ca, K, Mg, Na, Cl, SO_4), alkalinity, nutrients (total P, total N [unfiltered], nitrite, nitrate, ammonia, SRP [filtered], turbidity, pH, conductivity.
- TSS
- Metals: Ag, Al, As, Cd, Cu, Cr, Hg, Ni, Pb, Se, Zn (unfiltered and filtered). Alternatively multi-element scans could be considered.

4. Estimation of costs

For the greatest ease NSR would suggest that the DPIWE laboratory, Analytical Services Tasmania, is used for the analysis (they will be able to supply all sample bottles).

A cost estimate for analysis is approximately $400 per sample (see Table 1). This includes the supply of sampling bottles.

Table 1 Approximate analysis costs

Parameter	Cost unfiltered ($)	Cost filtered ($)
Metals	90	90
Nutrients	25	80
Major ions	45	-
Turbidity	10	-
Alkalinity	15	-
TSS	15	-
Conductivity	10	-
Filtering	-	~ 20
Multi-element scan	120	120

If Allegiance were prepared to purchase (approximately $2,000) or hire a pH metre, which could also measure DO and temperature, then these parameters could be measured in situ, and on an ongoing basis, which would be beneficial to the program.

This suggested program does not take into consideration QA/QC sampling and analyses. There would be additional costs associated with this, however, given the limited sampling such a program would not be warranted at this stage.

Please find following some indicative costs for the installation of a stream gauging station and meteorological station. They have not been derived from formal quotes.

Stream gauging station

The following lists details the various components of the gauging station.

- Unidata Starlogger Model 6004B
- Unidata Field Termination Strip and support frame Model 6004B
- Unidata Weatherproof Enclosure Model 6701A
- Unidata Water Depth Probe Model 6508E (20 m range)
- Hydrological Services Staff Gauge

This would cost approximately $3,500 plus consumables such as concrete, tubing (steel or PVC) for the cable, steel pipe for the staff gauges, hose clamps and the cost for power supply.

There would be labour costs on top of this which would vary depending on who is commissioned to install the station. In total, it would cost HEC approximately $5,000 to install a gauging station that could form part of their network.

Meteorology station

The approximate cost for installing an automatic meteorology station varies depending on the parameters to be measured and whether the station would be part of a telemetry network.

The Bureau of Meteorology has stated that with their current program commitments they would not have the resources available to install a meteorology station for at least 12 months.

HEC recently installed an automatic telemetry station, measuring most parameters, at Port Latta for ABM, for approximately $24,000. They advised that a simpler station measuring wind speed and direction, rainfall, temperature and humidity, which would require the data to be downloaded periodically rather than telemetry, would cost approximately $15,000.

NSR could install a similar system, including telemetry (the components of which are listed below) for approximately $5,500 plus consumables, labour and airfare.

- 6504-FS wind speed/direction
- 6501-DU temperature/humidity/solar radiation sensor
- 6700A telescopic tower
- 6103-1 field termination strip and frame
- 6603B/10 10m SQL half extension cable for 6504-FS
- 6004-11 12 channel data logger and battery pack
- 6506A 0.5mm tipping bucket rain gauge
- 6301B/AUE software package
- 6807-101 GSM data, steel enclosure, antenna
- 6904 C solar pannel (7W)
- 6907C sealed lead acid battery 12Ah
- Sim card

Please ring if you wish to discuss any of these matters .

Yours faithfully
NSR Environmental Consultants Pty Ltd

Luci David
Senior Consultant

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A J BRUCE CONSULTING PTY LTD
ABN 11 085 961 977

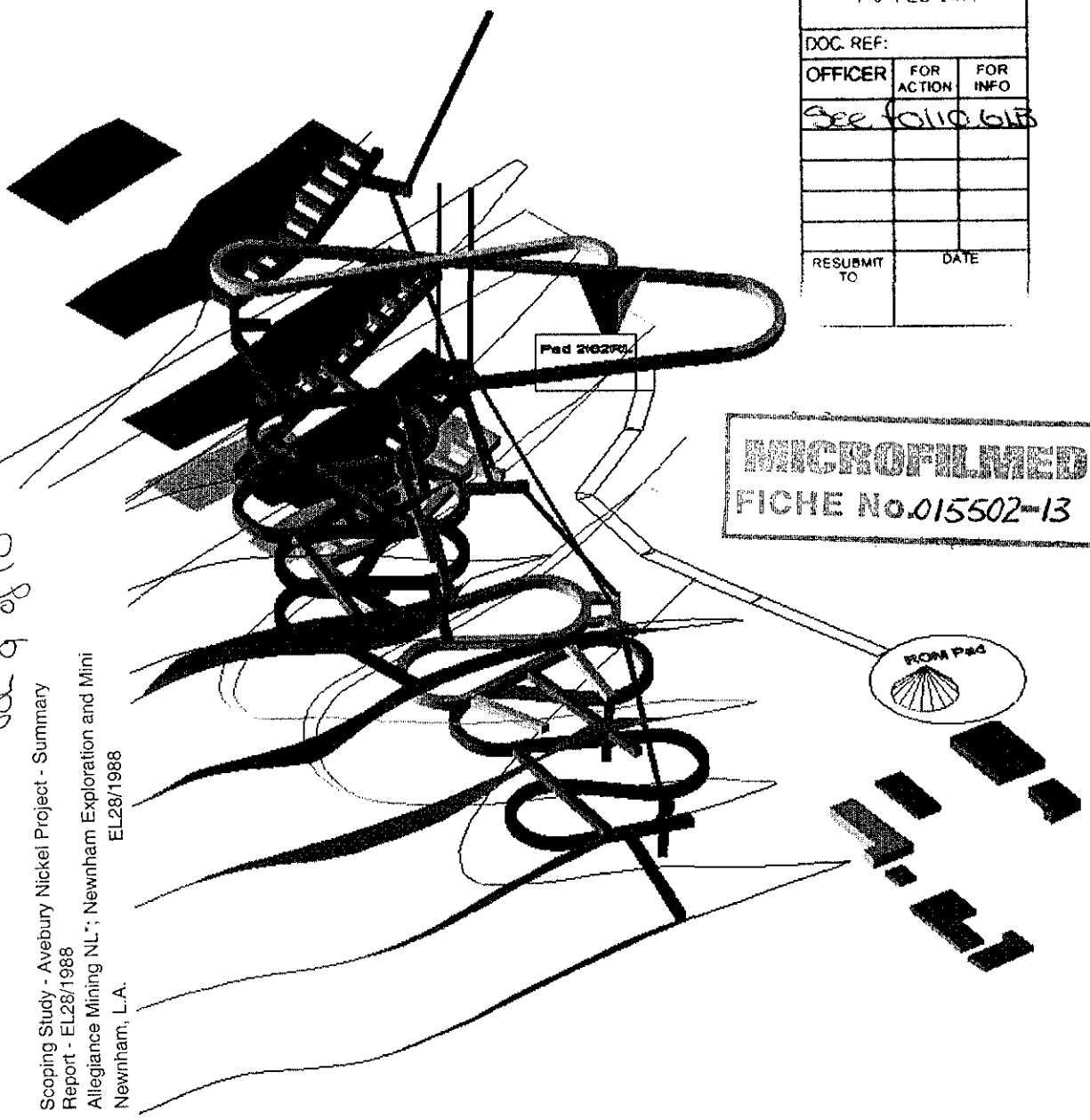
MINERAL RESOURCES
14 FEB 2001
TASMANIA

AVEBURY PROSPECT

Mine Scoping Study

December 2000

MINERAL RESOURCES
FILE REF: EL28/88 PT5
16 FEB 2001
DOC. REF:
OFFICER | FOR ACTION | FOR INFO
See folio 613
RESUBMIT TO | DATE



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Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL"; Newnham Exploration and Mini EL28/1988
Newnham, L.A.

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EXECUTIVE SUMMARY

INTRODUCTION

In summary the broad aims of this Report were:

- Conduct a Scoping study in regard to underground mining of the Avebury nickel prospect.
- Briefly review the implications arising from surface (open cut) mining of the deposit.
- Identify the necessary follow-ups to advance the project from a Scoping to Feasibility study stage.

KEY ASSUMPTIONS:

- The mining study was limited to delivery of ore or waste rock to the so called run-of-mine (ROM) pad.
- All services were assumed to be in place and available to the underground operation at no cost, via connection points near the mine entrance.
- The target underground ore production rate would be 300,000 tonnes per annum delivered to the concentrator.
- That mining will not be adversely affected by unusual environmental conditions such as gaseous or radiation emissions, excess ground water or toxic minerals.

It is assumed that the underground operations will be largely contracted out but the Owner would maintain a small mine management and maintenance team. The Owner's mining team would number eighteen.

GEOLOGY AND GEOTECHNICAL

Most of the mine access development can be done in medium-strength waste rock. However, development and stoping within the orebodies is expected to be challenging due to the weak nature of the serpentinite rocks. Substantial ground support costs will be incurred and have been included in the costing models.

PROPOSED MINING METHOD

For the thinner and steeper dipping North Avebury, the preferred mining method is Benching. A modified Open Stoping method with cemented fill is proposed for the larger South Avebury lenses.

Some waste rock-fill will have to be quarried from a surface location and this has been allowed for in the costings. Some sand fill will be sourced from the concentrator.

Total waste development required to fully extract the known resources is 7,669 metres.

Open Cut mining of the South Avebury lenses is possible, but at the current resource size a full scale open cut is not competitive with underground mining.

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TIMESCALE

- Planned production is 300,000 tonne per annum delivered to the concentrator.
- First ore production will be achieved in the second year of mine operations, with the full production rate being achieved in the third year.
- Total mine life will be just over 11 years, including the development only first year.
- Total extraction of ore will be 3,030,000 tonnes at an approximate grade of 1.6% nickel.

OPERATING COSTS

Mining costs are summarised in the Table below. The level of accuracy is consistent with a "Scoping Study". All calculations are based on October 2000 Avebury resource estimates.

SCENARIO:	CONTRACT MINING	OWNER OPERATOR
INFRASTRUCTURE, EQUIPMENT & START-UP	$4.6 million	$10.6 million
WASTE DEVELOPMENT	$20.0 million	$18.3 million
BENCH STOPING	$23.10 per Bench tonne	$19.37 per Bench tonne
OPEN STOPING	$18.03 per OS tonne	$15.12 per OS tonne
SERVICES	$3.62 per tonne	$3.62 per tonne
SUPERVISION	$3.81 per tonne	$3.81 per tonne
TOTAL EXPENDITURE OVER MINE LIFE:	$107 million	$101 million
OVER-ALL COST PER TONNE:	$35.29	$33.51

START-UP COSTS

The Table below summarises the main capital and operating cash flows over the mine life at a 300,000 tpa production rate. Expenditure in Year zero is related to pre-development costs; mainly contractor mobilisation and the mine portal construction.

ANNUAL EXPENDITURE FOR THE CONTRACT MINING SCENARIO ($MILLIONS)

YEAR:	0	1	2	3	4	5	6	7	8	9	10	11	12	LOM
CAPITAL DEVELOPMENT		$6.09	$1.15	$1.46										$8.7
OTHER CAPITAL	$0.92	$2.11	$1.33	$0.24										$4.6
OPERATING EXPENDITURE		$1.57	$9.40	$9.51	$11.6	$10.9	$9.60	$8.23	$8.42	$7.64	$7.64	$7.64	$1.99	$94.1
ANNUAL TOTAL COST	$0.92	$9.77	$11.9	$11.2	$11.6	$10.9	$9.60	$8.23	$8.42	$7.64	$7.64	$7.64	$1.99	$107

OPPORTUNITIES

Open cut mining of South Avebury is marginal based on the current Resources, but even low grade extensions to this orebody may change the economics to favour open cut mining.

The South Avebury orebody also looks to be suited to sub-level cave, which is a low cost underground mining method that involves no filling. Consequently $4 per tonne would be saved. A surface subsidence zone would however be created. This mining method, and open cut mining, were excluded from detailed review in this scoping study due to their perceived environmental impacts.

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It is likely that some thin zones of high-grade mineralisation identified in some diamond drill holes would be mined, adding to the value of the prospect.

A higher production rate will marginally lower unit mining costs (the reduction is more substantial for an owner operator). A higher production rate than 300,000 tonnes per annum would be easily achieved for Resources of the Avebury size.

The Avebury East exploration target may provide additional resources and it is likely that extraction of the East prospect could be accomplished off the same underground workings as North & South Avebury. It is important that this is considered prior to committing to a decline design.

THREATS

Based on the current geology the orebodies look relatively simple to mine.

The most significant mining threats are likely to result from poor ground conditions or excessive ground water. A detailed geotechnical program is required to establish accurate estimates for the rock mass quality and identify any hydrological hazards.

FUTURE FEASIBILITY STUDIES

To advance the Avebury Prospect to the Feasibility stage the following steps need to be completed:

- Considerably more high quality recording of geotechnical data is necessary.
- Hydrological and Geotechnical data needs to be evaluated by external Geotechnical specialist.
- A mine feasibility study must be conducted fully integrating the hydrological and geotechnical studies.

To gather the diamond drill data, triple tube drilling to give coverage at 40m by 40m throughout the orebodies is recommended. Logging and hydrological fieldwork is likely to take 40 days and the Geotechnical/Hydrological analysis and reporting a further 20 days. At least fifty core strength tests are needed. The total estimated cost is $84,000 for a feasibility level Geotechnical Study.

The mine feasibility study is estimated to take about 60 man-days, it may involve significant travel costs and some research costs. The estimated total cost for a Mine Feasibility Study is $82,000.

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1. INTRODUCTION

Mr. Andrew Bruce B.E. (Mining), MAusIMM of A J Bruce Consulting Pty Ltd was responsible for compilation of this report on mining of the Avebury Nickel Prospect. He has more than twenty years of mining experience.

Assistance was provided by Mr. Michael McKeown M.Eng.Sc., B.Sc., FAusIMM in preparation and analysis of the geotechnical data.

Mr. Lindsay Newnham, acting as Allegiance's project manager for the Zeehan Nickel Project, was the principal point of contact with respect to all primary data on the Avebury Prospect. Mr. Michael McKeown was Allegiance's resource geologist for the project.

In preparing this report we have relied on information provided to us by Newnham and have no reason to believe that the information is misleading or incomplete or contains any material errors. Newnham was provided with a draft of this report for correction of any material errors or noting of any material omissions.

Unless otherwise stated any reference to Allegiance can be taken to mean Allegiance Mining N.L. or it's agents.

1.1 Project Aims

In summary the broad aims of the mining study were to:

- Review potential mining methods and provide an evaluation of the advantages and disadvantages of the most likely methods, including a brief review of open cut mining.
- Recommend a suitable mining method, and provide capital and operating costs to a level of accuracy consistent with the resource estimate.
- Produce life-of-mine production and development schedules for the Avebury deposit based on preliminary mine designs, parametric studies of the deposit and the potential reserve contained in the North Avebury lens.
- Complete a conceptual underground mine design. Supply schematic mining layouts and designs to convey the concepts of the proposed mine design and extraction methodology.
- Identify the likely surface infrastructure and services required for an underground mine of 300,000 tonnes of ore production per annum.
- Recommend a portal location with due regard to environmental considerations.

- Quantify the likely waste rock requirements for the mine, including such issues as storage, placement, sources and cost.
- Provide capital and operating mine budgets for the first 10 years of operation.
- Examine the alternative options for "owner vs. contractor" mining.
- Undertake a preliminary geotechnical study of the Avebury deposit as per the attached program of work in Table 1.2b.
- Identify any shortcomings (from a mining and geotechnical perspective) of the completed resource drilling to aid in scoping the next diamond drilling program.
- Provide realistic mining parameters as inputs to assist in producing a Probable Ore Reserve (subject to the appropriate metallurgical, economic, legal, social, environmental and government factors) once an Indicated Resource estimate is available.

1.2 Terms of Reference

The scope of work included in the original Proposal by A J Bruce Consulting Pty Ltd for the Mine Design Study included the following:

Table 1.2a: Components of the Mining Study

- Review existing data and reports including Client liaison to ensure the outcomes of the study are in line with the Client expectation. Attend a number of project briefings in Zeehan.
- Conduct a review of the available plans, sections and core photographs.
- Conduct Main Creek site inspection(s) and review diamond drill core at Zeehan core shed.
- Examine possible mining methods; determine advantages and disadvantages then short-list suitable mining techniques.
- Complete a basic conceptual mine design and parametric study for the "North Avebury".
- Review of mine filling and stockpile issues and alternatives.
- Select mining equipment and obtain budget quotations from typical equipment suppliers and contract mining companies.
- Review and layout of surface facilities, mine portal, stockpiles etc. Including a liaison with the Client's environmental consultants.
- Final examination of geotechnical parameters and constraints with subsequent modification of mine plan (if necessary).
- Finalise mining method and layout and proceed to short and long-term production and development schedules.
- Capital and operating budget estimation.
- Develop a work program of additional mining investigations required for the next project stage.
- Final report compilation.

Table 1.2b: Proposed Geotechnical Study

- Review existing data including core logs.
- Conduct a small number of point load strength tests on drill core from the hangingwall, orezone and footwall rock units.
- Estimate rock mass classification parameters for hangingwall, orezone and footwall rock units.
- Estimate stable stope sizes and basic rock properties.
- Review geotechnical constraints with regard to mine layout.
- Propose ground support patterns for access development and stopes.
- Suggest a work program of additional mining geotechnical investigations.

1.3 Disclaimer

This report summarises the results of a mine design study conducted on the Avebury prospect to a level of accuracy consistent with a Scoping Study[a]. As such the level of accuracy is expected to be ±30%. Use of this report will be applicable when "Developing long-term corporate strategies. Deciding on level of ongoing exploration. Assessing project advancement from exploration to pre-development".

Figures 1, 2 and 3 were supplied by Allegiance and are reproduced in this report for information only (minor editorial changes have been made for clarity of presentation).

It is important to note that this mining study is based on the geology as supplied on 13 October 2000 and a subsequent update on 25 October 2000. As a consequence, the Resources used for this report may not be exactly the same as the latest estimate. Discussions with the Avebury's resource geologist on this matter indicate that any discrepancies will not have a substantive effect on the outcome of the mine scoping study.

The opinions expressed below are the opinion of A J Bruce Consulting Pty Ltd and may not reflect the opinions of Allegiance Mining N.L. or it's Agents.

1.4 Definitions

References in this report to ore or orebody are generic references to that part of the mineralisation that the mining method, as described, will extract. The uses of the word ore or orebody are not intended to indicate that the mineralisation in question can or will be extracted economically.

[a] *M. E White, "Feasibility Studies – Scope & Accuracy", see Reference 8.*

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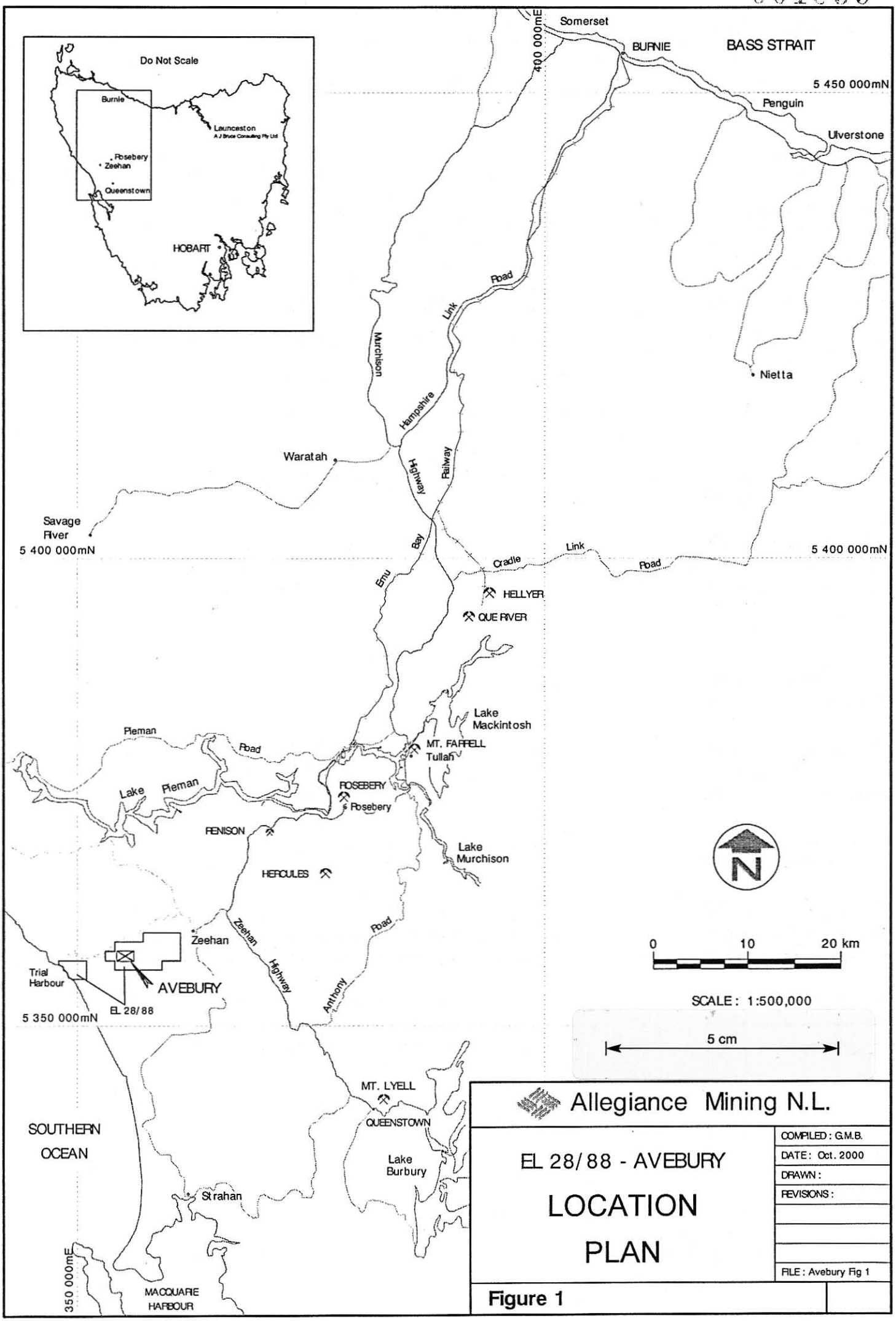

Allegiance Mining N.L.

EL 28/88 - AVEBURY

LOCATION

PLAN

COMPILED : G.M.B.
DATE : Oct. 2000
DRAWN :
REVISIONS :
FILE : Avebury Fig 1

Figure 1

2. BACKGROUND

No previous mining studies have been conducted on the Avebury nickel mineralisation. Other than *Zeehan Nickel Project – Summary Profile;* L. Newnham, May 2000, no other Allegiance reports were available during the preparation of this mine design study.

2.1 General Description of the Deposit[b]

"The Avebury deposit is located within Exploration Licence 28/88 and is now owned 100% by Allegiance. Rio Tinto retains a 2% net smelter royalty on future production.

The deposit lies approximately 7 km due west of Zeehan in an area of rain forest." See Figure 1.

"Allegiance is keen to ensure the deposit is developed not only in a technically efficient, low cost manner, but also in a way which minimises impacts on the environment.

The Avebury area is underlain by a sequence of Cambrian sediments and ultramafic bodies, which have been intruded and altered by the Heemskirk Granite. It is thought that the intrusion of this granite remobilised low levels of nickel and iron in the ultramafic and redeposited them in a more concentrated form near the tops of steeply dipping fingers of ultramafics.

To date, several such deposits have been recognised and tentatively named North Avebury, Central Avebury and South Avebury.

Most attention to date has been focused on North Avebury. A drilling program designed to delineate an Identified Mineral Resource is currently in progress on North Avebury and is due for completion by end of August 2000.

Mineralisation consists principally of Pentlandite and magnetite and is concentrated along plunging zones close to the ultramafic-sediment contacts.

Whilst these higher grade (1-3% Ni) zones may be the focus of early development plans, it is important to note that there are very substantial bodies of lower grade (0.5-1.0% Ni) sulfide mineralisation typically developed in the footwall of the higher grade zones.

At this early stage, our belief is that North Avebury may contain a resource of 1.5-2 Mt of 1.0-2.0% Ni. Our vision is to develop a modest-sized underground mine of, say, 200,000 tpa, producing approximately 3,000 tpa nickel contained in nickel sulfide concentrates.

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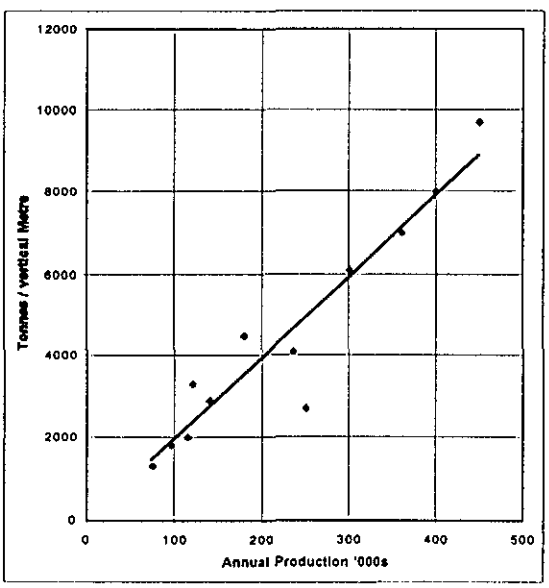
[b] *As provided by Newnham, 10 July 2000.*

Extensions of North Avebury or additional resources subsequently identified at Central, South or East Avebury would provide a basis for either extending mine life or increasing output (or both)."

2.2 Mine Production Rate

Prior to mine (or production plant) design for the Avebury deposit or any other deposit, a target annual production rate must be set. Newnham's comments in Section 2.1 propose an annual production rate of about 200,000 tonnes. While this rate is similar to that calculated using Taylor's Rule[c] for a deposit around 1.5M tonnes, inclusion of Resources from the South Avebury implies that the 200,000 tpa target is too low.



In the Australian context a production rate of 200,000 tpa is very low (other than for mostly small high grade gold mines). The graph to the left shows typical Australian mine annual production rates compared to the tonnage of reserve per vertical metre. The implication from the graph is that for a resource of 1.5M tonnes over 250 vertical metres, Avebury mine should have a production rate not less than 300,000 tonne per annum.

It was subsequently agreed that the Avebury Scoping Study would be based on an annual production rate of 300,000 tonnes of ore delivered to the concentrator.

2.3 Resources Included in the Mining Study

The original scope of work for this mine design study covered only the North Avebury resource. Delays in finalising the North Avebury geology and contemporary successful diamond drill holes into the South lens allowed time for inclusion of South Avebury in the mine design study. It should be noted that the South Avebury Resource has an Inferred status and North Avebury has an Indicated Resource status.

McKeown indicated that the mining study should be restricted to 1800RL to 2050RL and 1920RL to 2050RL for North and South Avebury respectively. (To maintain consistent sub-level intervals, very minor resource extensions were included to allow for the top sill in each lens.) For an in-depth description of the Avebury resources see *Avebury Nickel Project Mineral Resource Report*, M McKeown, November 2000. Refer to Figure 2 and 3 for generalized representations of the Avebury zones of mineralisation in plan and section views.

[c] *See Reference 2*

All mine design work was based on geological interpretations of the deposit supplied by Allegiance as CAD or data files in the following formats:

Avebury Ore RLs-R13.dxf
Ore & serpentinite outlines at 50m levels (plan views), North & South lenses.
Source: G Bennett
Received: 13/10/2000

ajbmodel.zip
Nickel grades throughout the North lens as point data.
Source: M McKeown
Received: 23/10/2000

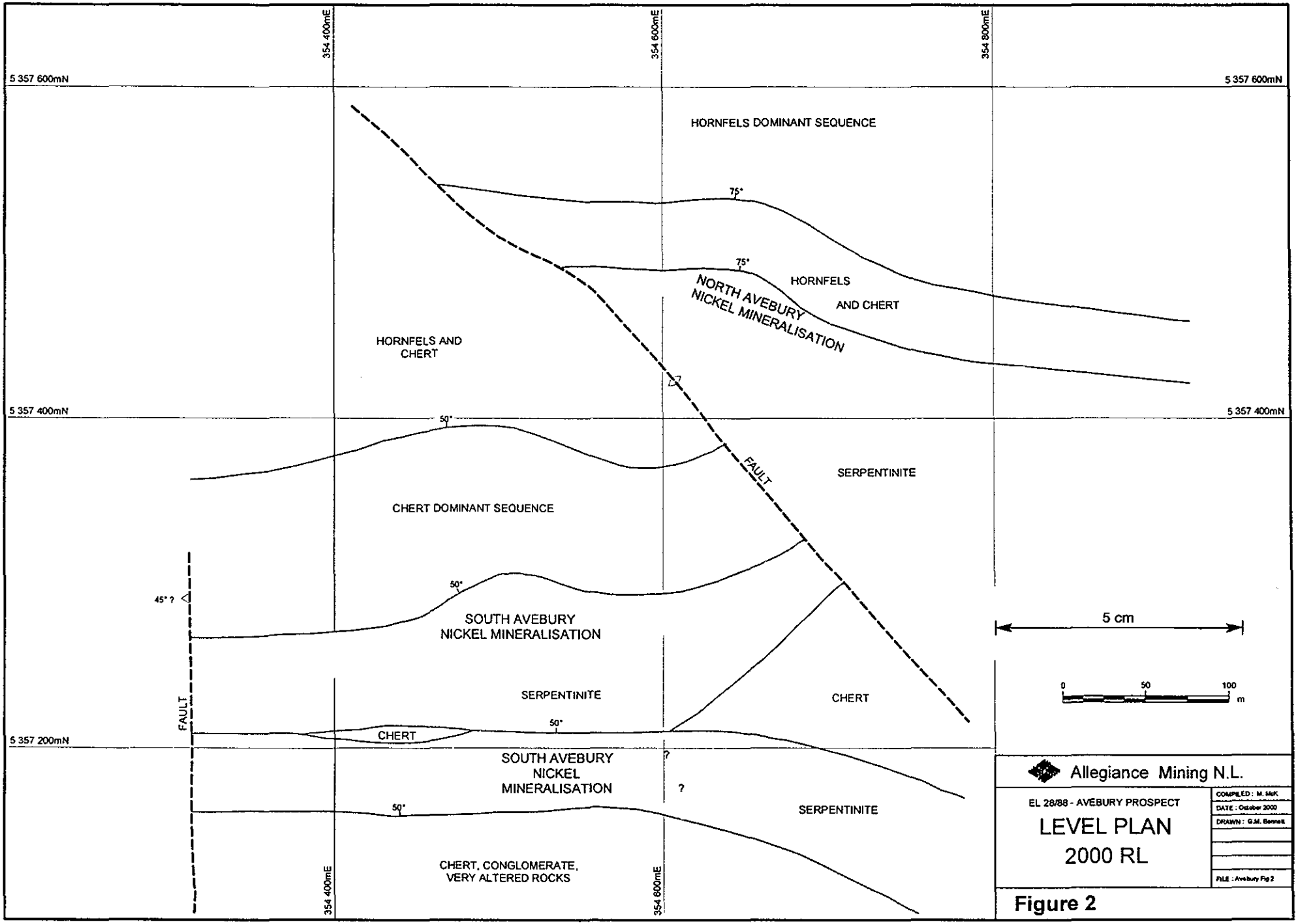
E-mailed coordinates
New outline of the South lens at 0.0% Ni cut-off as 3D coordinates.
(A copy of the coordinates is attached in Appendix A).
Source: M McKeown
Received: 25/10/2000; superseding the 13/10/2000 outlines of the south lens.

Surface profile, existing tracks and river paths were taken from recently completed photogrammetric work completed for Allegiance by the Survey & Geographic Information section of the Hydro-Electric Corporation, and in particular the file **G3049_10.dwg**.

2.4 Basic Parameters for the Deposit

The following basic parameters were used in the preparation of this report on the Avebury deposit:

- In-situ ore density 3.0 t/m^3
- In-situ waste density 2.9 t/m^3 (or 2.8 t/m^3 for quarried waste fill).
- In-situ to broken rock swell factor is taken as 40%.
- All coordinates are relative to the Australian Mine Grid.
- Reduced Levels (RL) are based on mean sea level plus 2000 metres.
- Unless otherwise stated all dimensions are in metres.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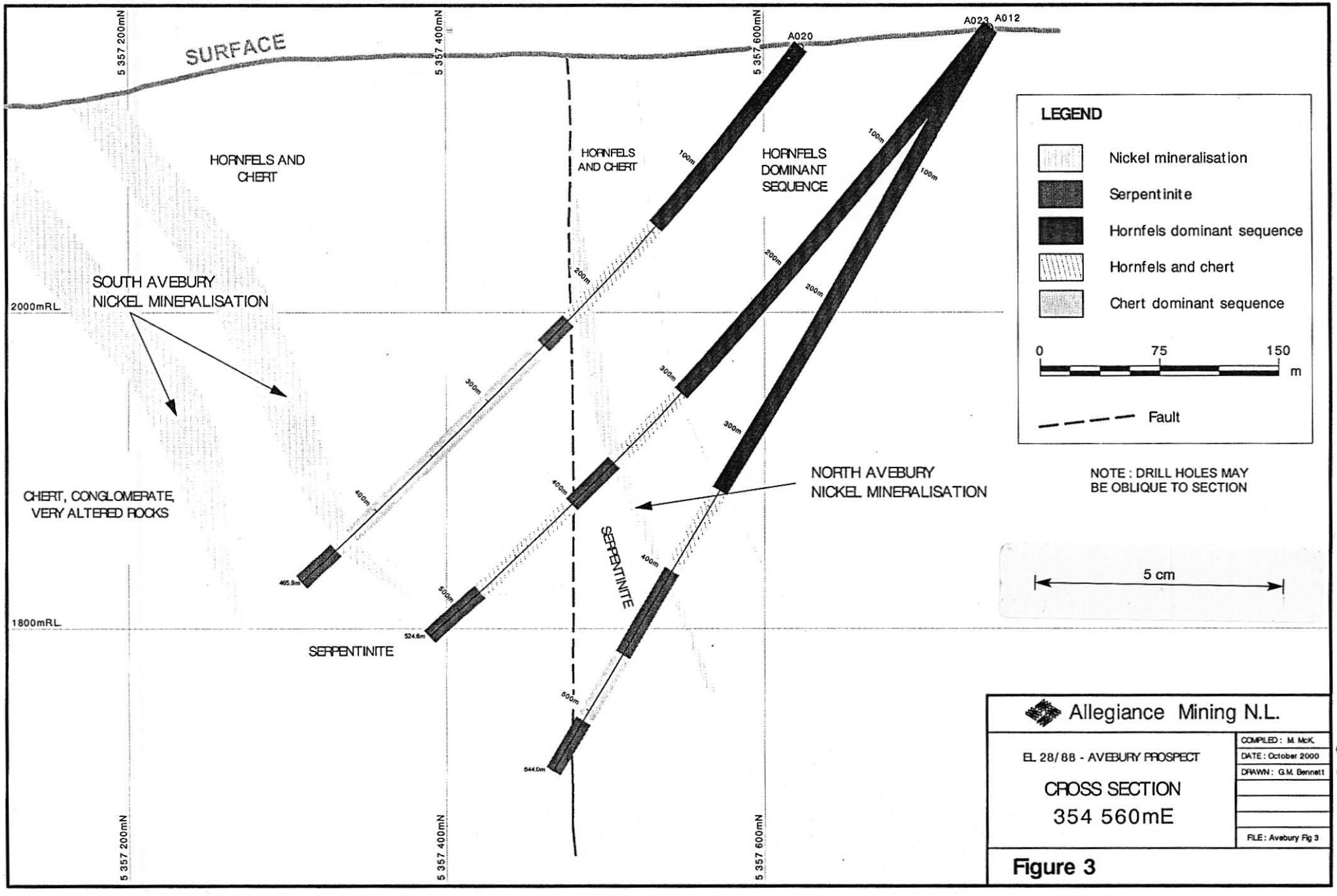
SERPENTINITE
CHERT, CONGLOMERATE,
VERY ALTERED ROCKS
Allegiance Mining N.L.
EL 28/88 - AVEBURY PROSPECT
LEVEL PLAN
2000 RL
COMPILED : M. McK
DATE : October 2000
DRAWN : G.M. Bennett
FILE : Avebury Fig 2
Figure 2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5 357 600mN
A020
A023
A012
HORNFELS AND CHERT
HORNFELS AND CHERT
HORNFELS DOMINANT SEQUENCE
SOUTH AVEBURY NICKEL MINERALISATION
2000mRL
1800mRL
CHERT, CONGLOMERATE, VERY ALTERED ROCKS
SERPENTINITE
SERPENTINITE
NORTH AVEBURY NICKEL MINERALISATION
100m
200m
300m
400m
500m
465.9m
524.6m
544.0m
LEGEND
Nickel mineralisation
Serpentinite
Hornfels dominant sequence
Hornfels and chert
Chert dominant sequence
0
75
150
m
Fault
NOTE : DRILL HOLES MAY BE OBLIQUE TO SECTION
5 cm
Allegiance Mining N.L.
EL 28/88 - AVEBURY PROSPECT
CROSS SECTION
354 560mE
COMPILED : M. McK.
DATE : October 2000
DRAWN : G.M. Bennett
FILE : Avebury Fig 3
Figure 3

3. GEOTECHNICAL

In combination with nickel grade distribution, geotechnical variables of the Avebury deposit will have a major influence on the selection of mining method, stope or access sizes, ground support requirements etc.

With relatively little diamond drilling into the deposit, no specific geotechnical holes and no bulk surface or underground exposures, only rough estimates of the geotechnical behaviour of the deposit can be made. Despite this comment, the relatively conservative approach adopted below should provide estimates of the rock mass properties sufficient for a mining study to a 'scoping' level of accuracy.

The geotechnical assumptions are partially based on discussions with personnel from the Renison Bell tin mine. The Renison Bell mine is the closest operational mine to the Avebury site.

3.1 Geotechnical Overview

In Situ Stresses

In the situation where site specific in-situ stress measurements have not been conducted, it is generally accepted that reasonable estimates can be based on the following formulae:

$\sigma_v = 0.027 \bullet H$ and,
$\sigma_1 = 2\text{-}3 \bullet \sigma_v$.
Where σ_v is the vertical stress in MPa, H metres below surface and,
σ_1 is the major principal stress in MPa at that point (acting in near horizontal direction).

Estimates of the stress concentration factors[d] that typically occur around stope excavations are tabulated below. These stress concentration factors are applied to σ_1.

Table 3.1a Stope Stress Concentration Factors

Vertical Lift Height	Hangingwalls & Footwall	Stope Backs 5 wide	7 wide	10 wide	10+ wide
18 metres	0.55	1.6	1.4	1.25	1.1
30 Metres	0.45	2.6	2.1	1.8	1.45

Rock Strengths

As no rock strength testing had been completed on any of the Avebury diamond drill core, a point load testing program was conducted at the Zeehan core shed. Sufficient tests were performed to determine indicative unconfined compressive strengths (UCS) for the

[d] *J Slade, Engineering Geologist; Renison Bell Ltd.*

Avebury North and South lenses, associated serpentinite rock and nearby metasediments. The results are tabulated in Appendix B and summarised in Table 3.1b.

The results obtained from point load testing are sufficiently accurate for a scoping study. The relationship between PLI and UCS needs to be confirmed by comparative laboratory testing prior to any final design.

Table 3.1b Intact Rock Strength Summary

Rock Type	PLI (MPa)	UCS (MPa)	Description
Mineralisation/serpentinites	3.0	67	Moderate strength
Immediate Hangingwall	2.0	44	Moderate strength
Remote Hangingwall	3.9	86	Medium high strength

Core Logging

Basic geotechnical logging is now being conducted for all Avebury diamond drill holes, although a number of the earlier holes were missed. McKeown supplied copies of the logs for holes numbered A008, A010, A012, A013, A014, A015, A016, A017, A018, A019, A020, A021, A023, A024, A025, and A026. A copy of a page for a typical log is attached in Appendix C.

The system being used to describe the geotechnical condition of the core is the proprietary McKeown Mining – Basic Geotechnical Log (BGL) and not one of the more widely recognised Rock Mass Rating (RMR) or Tunnel Quality Index (Q) systems. Despite this, sufficient information was available to calculate an approximate RMR or Q rating for the core.

3.2 Rock Mass Classification

Although the RMR and Q systems for rock mass classification are roughly analogous, a modified Q system is currently being used more extensively in the evaluation of stope stability (particularly in Australia). Note that the modified Tunneling Quality Index Q' assumes the joint water reduction factor (J_w) and the stress reduction factor (SRF) are set to unity, where:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

therefore

$$Q' = \frac{RQD}{J_n} \times \frac{J_r}{J_a}$$

For this mining study of the Avebury deposit the Q' system will be used. It was therefore necessary to convert the information in the McKeown geotechnical logs into a format suitable for creation of a Q' index.

Appendix C provides tabulations of the relevant geotechnical data, summaries of the resulting Q' indices and graphical analysis of the results. North lode hangingwall, South lode hangingwall, North lode ore & footwall and South lode ore & footwall are all treated as separate zones.

Due to considerable scatter and no obvious spatial correlation among the calculated Q' values, histograms were plotted for each zone. Typical values for Q', RQD and J_n were selected.

Table 3.2 Typical Geotechnical Parameters for Avebury Deposit

Zone	Q'	RQD	J_n
North Lode Hangingwall	6.0	90	4
North Lode Ore and Footwall	3.0	85	9
South Lode Hangingwall	4.0	85	4
South Lode Ore and Footwall	3.0	80	9

The implications of the geotechnical parameters tabulated above are further discussed in Appendix C.

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4. MINING METHOD SELECTION

A detailed comparison of underground vs. open-cut mining for the Avebury deposit is beyond the scope of this report. As indicated in the project scope, only a quick review of mining by open cut methods has been conducted, see Section 5.

Allegiance has expressed a strong preference towards an underground mining method for extraction of the North and South Avebury resources. This section discusses some of the possible underground extraction methods which could be applied too this deposit. The preferred extraction technique(s) are fully described and evaluated in the Section 7.

Alternative Mining Methods

Selection of a mining method is based mostly on the physical characteristics of the orebody in question although environmental and other considerations may have a major impact on the final decision.

Based on the current drilling results and understanding of the geology, the Avebury deposit has at least two distinct zones of mineralisation; namely North and the South zones. (Although the South zone has two regions of mineralisation this is all within the same serpentinite host and will be treated as one orebody.) These zones are horizontally separated by about 150 metres, which is sufficiently far that the two zones can be considered to be independent stoping areas. The key physical characteristics for selection of a mining method are listed for North Avebury and South Avebury in Table 4a.

Table 4a Characteristics of North & South Avebury orebodies

PARAMETER	NORTH	SOUTH
Dip	75°	60°
Average thickness	10 metres	25 metres
Top (RL)	2050	2050
Bottom (RL)	1800	1920
Hangingwall RMQ[e]	Fair	Poor-Fair
Footwall RMQ	Poor	Poor
Orebody RMQ	Poor	Poor
APPLICABLE MINING METHODS	**Open stoping (sublevel stoping), Benching, Cut & Fill.**	**Open stoping (sublevel stoping), Sub-level cave, (Open-cut, see Section 5).**

Each of the applicable mining methods has advantages and disadvantages and any one of these methods could be used at the Avebury deposit with varying degrees of success. A

[e] *Rock mass quality based on Q, after Grimstad and Barton, 1993.*

rigorous evaluation to determine optimal stoping method is not justified at the 'scoping study' stage.

Method Selection

Table 4b Stoping Methods arranged in order of most desirable to least desirable

Low up front development	Low operating cost per tonne	Minimal environ-mental impact	Application to poor ground conditions	Suitability for narrow orebodies	Least operator exposure to poor ground	Potentially highest production rate
Sub-level cave	Open stoping	Cut & Fill	Sub-level cave	Benching	Sub-level cave	Sub-level cave
Benching	Sub-level cave	Benching	Open stoping	Cut & Fill	Open stoping	Open stoping
Cut & Fill	Benching	Open stoping	Benching	Sub-level cave	Benching	Benching
Open stoping	Cut & Fill	Sub-level cave	Cut & Fill	Open stoping	Cut & Fill	Cut & Fill

Even though Sub-level cave receives the highest (best) ranking from the table above, Allegiance's requirement for "minimal environmental impact" rules out this mining method for the current mine scoping study. Sub-level cave by its nature would create an area of surface subsidence and result in an environmental management challenge. See Figure 5 for an indication of the likely region of surface subsidence resulting from sub-level cave mining.

Cut & Fill is the least desirable mining method as a consequence of higher risks to personnel and equipment in poor ground conditions, and higher operating costs.

Open stoping is generally used in thicker orebodies where the high up front capital development cost can be justified.

For all but very wide orebodies, bench stoping has become almost the universal choice (certainly within Australia) due to low development cost, simple filling, relatively low unit cost and medium productivity levels.

Based on the comments above and Table 4b this mine scoping study will assume respective underground mining method of:

- **Benching** for North Avebury and
- **Open stoping** for South Avebury.

Contractor vs. Owner Operator Mining

The advantages and disadvantages of owner mining vs. contractor mining is the subject of ongoing debate in the mining industry. Despite an overall trend towards contract mining, there have been a number of recent shifts back to owner mining (both in open cuts and underground mines). For small mines or short mine lives, it would be most likely that the

operation would be 100% contracted out. Conversely high production rates or very large resources tend to favour owner operators.

Ignoring the arbitrarily set 300,000 tpa production rate and based on the current knowledge of the deposit, Avebury is probably a mid sized resource which will have a relatively short mine life. As a consequence factors other than resources size and mine life will probably determine the final owner/contractor mix for the operation.

To provide a basis for mining costs calculated in this report it was necessary to make the initial assumptions that:

1. The Mining Lease owner (in this case Allegiance) would maintain management control of the underground operations. The Mining Manager and Underground Manager would be employees of the Owner.
2. All development and production processes would be contracted out as a single major Contract. With the exception of a number of service vehicles and light vehicles, the Contractor would own & supply all mobile equipment.
3. The Owner would carry out and supervise all fixed plant electrical and mechanical maintenance.
4. The Owner would maintain underground services. (Initial installation would be part of the Contract.)
5. Quality control aspects of the mining would be provided by the Owner; namely Survey, Mine Planning, Training and supply safety/rescue equipment.

It is the opinion of the author that this mix facilitates the Owner maintaining control while utilizing the benefits contractors can bring to an underground mine. Under the scenario above the necessary Owner workforce (mine department only) is tabulated below.

Table 4c Basic Owner Workforce (Mine department only)

POSITIONS	NUMBER of PERSONNEL
Surveyors & assistants	3
Mine engineers	3
Mine manager	1
Underground manager	1
Mine trainer	1
Maintenance foreman	1
Grader operator	1
Service crew	4
Tradesmen	3
TOTAL	18

The list above is considered to be a minimal Owner workforce with the remainder of the workforce supplied by the Contractor. At the other extreme is a totally owner operator mine with all work conducted by direct employees and all the equipment owned (or leased) directly by the Owner.

Section 10 provides costs for the Avebury deposit based on the owner/contractor mix proposed above **and** derives an estimated cost for owner operator mining.

5. OPEN CUT DESIGN

Suitability of the Method

South Avebury is much more likely to be selected for open cut mining than North Avebury for the following reasons:

- The top of both Avebury North and South orebodies are a similar distance below the surface at about 120 metres.
- South Avebury orebody is larger and has a higher tonnage per vertical metre.
- The North orebody is well suited to simple underground mining methods. Conversely extraction of the South orebody will involve a more complex underground mining method due its size and weak rock.
- An extension of the South mineralisation closer to surface is still a possibility, whereas this is unlikely for North Avebury.
- It is unlikely that the North & South orebodies could be extracted economically by one big open cut due to the 150-200 metre horizontal separation.

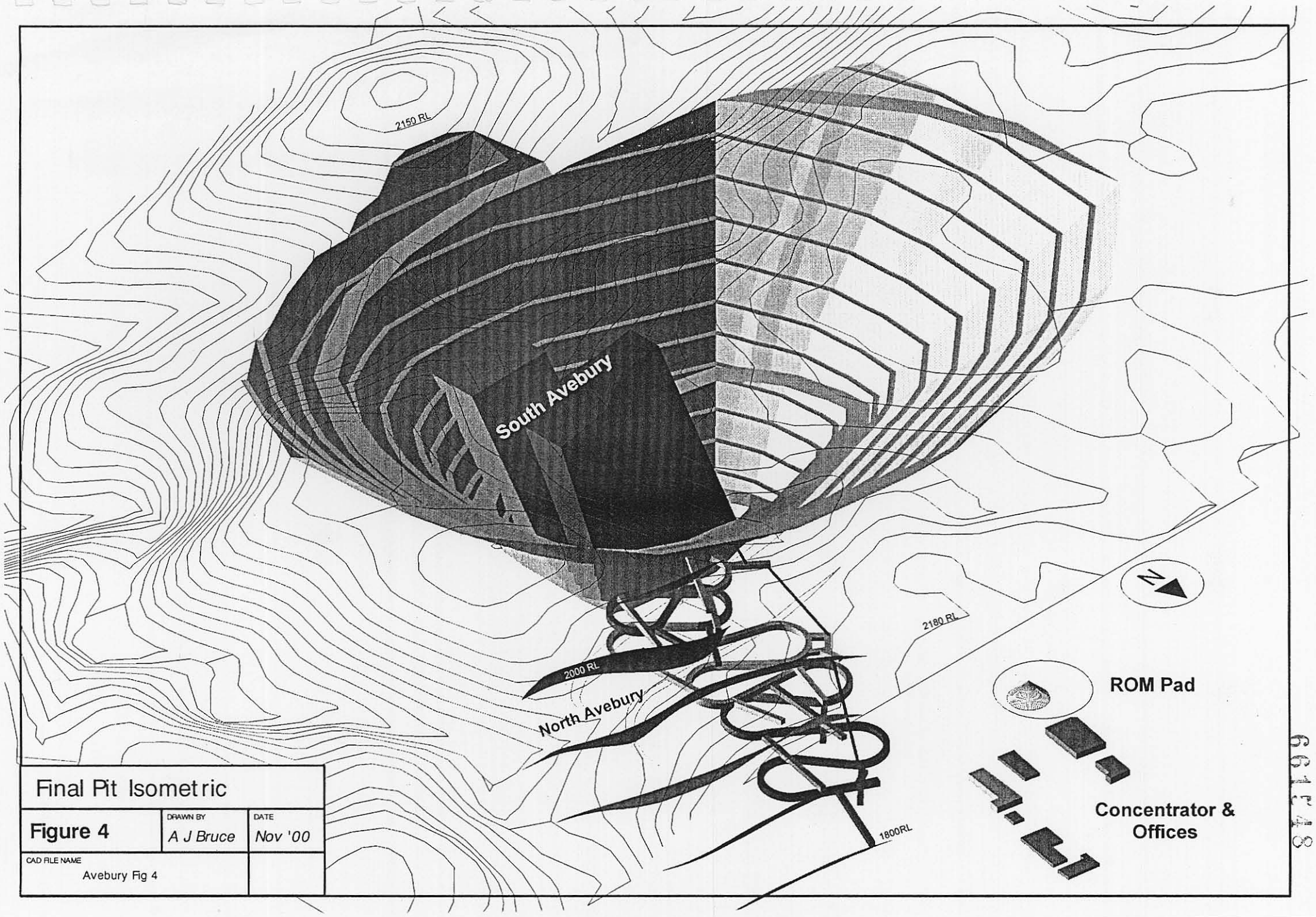
The following discussion of open cut mining consequently relates only to extraction of the South mineralisation with the assumption that the North orebody would be extracted by underground mining techniques.

Open Cut Design Parameters

The open cut design exercise conducted on South Avebury was intended to produce estimates for the size and cost of the necessary pit. A number of simplifying assumptions were made during the design largely as a result of the lack of information available on the deposit:

- Environmentally suitable dumping locations can be found nearby for disposal of the waste rock generated by the pit.
- With a pit, 100% of the South Resource could be extracted and milled profitably.
- Hemp Creek can be diverted to Comstock Creek and the pit would not be subject to excessive ground-water flows.
- The pit is to be mined by a Contractor.
- Pit wall angles were assumed to be 45° overall (with 40° above 2020 RL and 50° from 2020 RL to 1920 RL).
- The pit is stable at the wall angles above and no unusual geotechnical conditions are encountered.
- Pit and waste dump rehabilitation costs are excluded.

The 'final pit' shape and relationship to the underground workings for the North orebody are shown in isometric view in Figure 4.



Final Pit Isometric		
Figure 4	DRAWN BY: *A J Bruce*	DATE: *Nov '00*
CAD FILE NAME: Avebury Fig 4		

691748

Open Cut Conclusions

Open cut mining to extract all of the South Avebury Resource down to 1920 RL results in the following:

- Direct disturbance of 33 hectares of land, **excluding** the waste dumps.
- Movement of 74.4 million tonnes of waste.
- Extraction of 3.4 million tonnes of 'ore'.
- A total cost of about $105 million and,
- Average cost of production of $31[f] per tonne of ore.

The raw data and detailed calculations for extraction of South Avebury to 1920 RL are attached in Appendix D and Figure 5 shows the relationship of the proposed pit to the surface infrastructure.

Advantages of Open Cut Mining

Open cut mining provides an opportunity to mine extensions to a resource at marginal cost as the material is already being moved as waste. Additional Avebury South Resources (even low grade ones) could dramatically improve the economics of this mining alternative.

Another benefit from an open cut is the greatly reduced decline development and a portal location in the pit wall, which allows selection of good ground conditions prior to the start of the decline. The pit also provides excellent rock mass exposures eliminating much of the geotechnical risk.

Disadvantages of Open Cut Mining

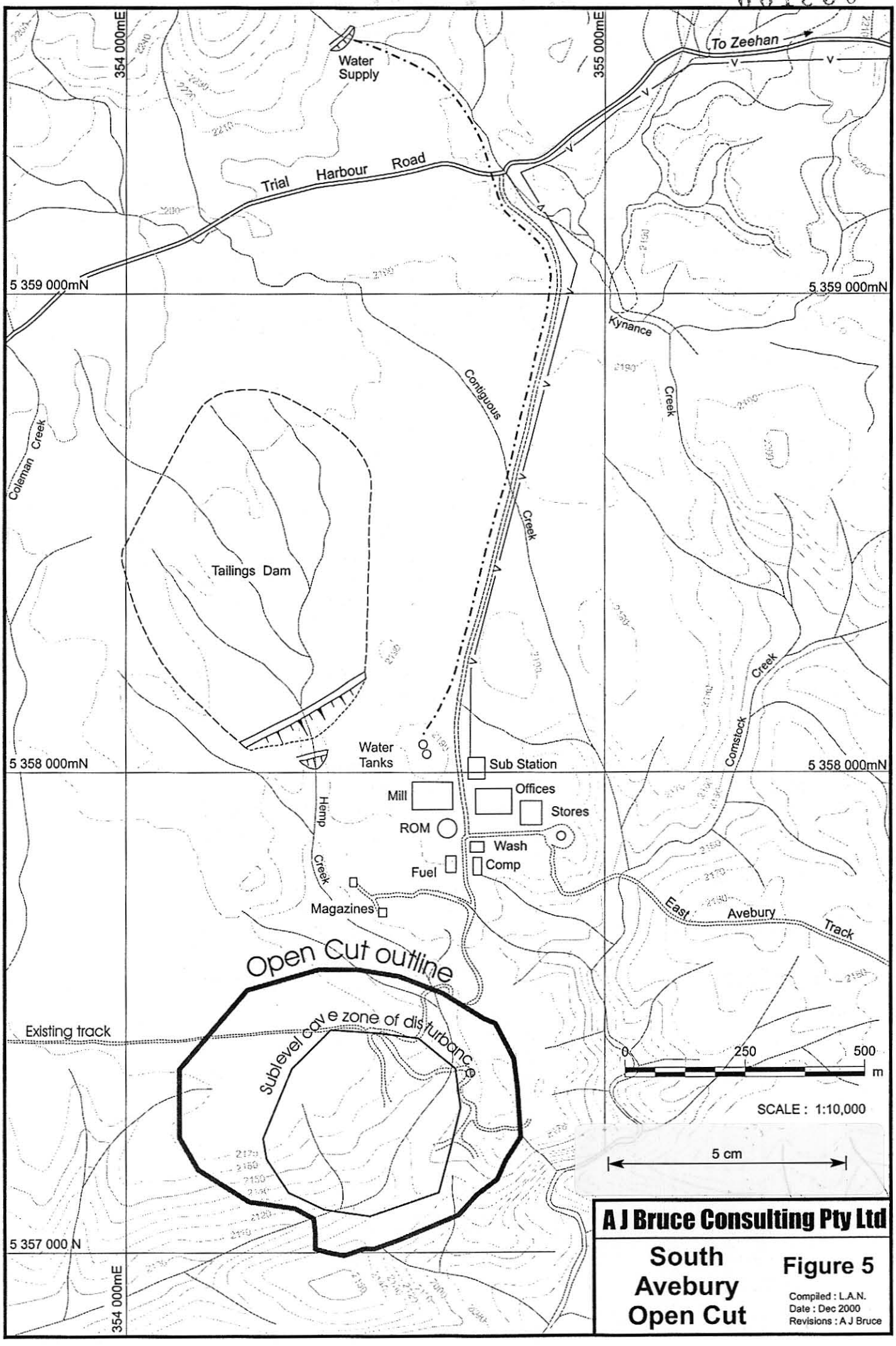
Prior to ore production large amounts of waste rock must be moved. This results in a large capital outlay prior to production.

It is unlikely that the North orebody can be mined until the pit is largely complete.

Finding suitable locations to stockpile the 74 million tonnes of waste will be an environmental challenge to say the least. Note that rehabilitation costs are not included above.

[f] *Includes the cost of moving the waste.*

661550



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South Avebury Open Cut

Figure 5

Compiled : L.A.N.
Date : Dec 2000
Revisions : A J Bruce

6. MINE DEVELOPMENT

6.1 Sublevel Intervals

Subsequent to the choice of a mining method comes the selection of a suitable sublevel interval (or in this case the selection of sublevel intervals for North and South Avebury). A sublevel interval must chosen prior to decline and stope access design. At the final design stage, determining an optimal sublevel interval can be an involved task depending on:

- Orebody geometry
- Geotechnical parameters
- Mining equipment (particularly production drills)
- Fill types
- Productivity requirements
- Orebody geometry
- Confidence in the geological interpretation

However, the Avebury deposit is at the stage where many of these factors are unknown or available as estimates only. Consequently the choice of sublevel intervals will be based more on 'industry standard' rather than an optimization study.

While open stope sublevel intervals vary widely, Australian bench stopes in steep dipping orebodies mostly fall in the range of 15 to 22 metres (floor to floor). For North Avebury a sublevel interval of 18 metres was selected as a good compromise. Maximum blast hole length is expected to be about 16 metres.

South Avebury will have a significantly higher sublevel as it is planned to mine this orebody by an open stoping technique. To ensure a single production drill would be suitable for both orebodies, a nominal 25 metre sublevel interval was chosen. Maximum production blast hole length will then be 21 metres.

Matching these sublevel intervals to the orebody geometry led to the extraction levels shown in Table 6.1. Note that detailed stoping and level development designs were only completed for those sublevels shown in **bold** text, this applies to Tables 6.3 b & c as well.

Table 6.1 Stope Extraction Levels (RL)

NORTH AVEBURY		SOUTH AVEBURY
2052	**1908**	**2050**
2034	1890	2025
2016	1872	**2000**
1998	1854	1975
1980	**1836**	**1950**
1962	1818	**1920**
1944	**1800**	
1926		

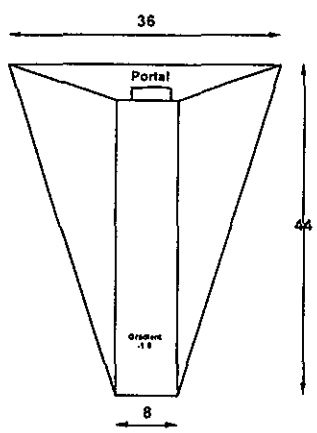
6.2 Decline Design Overview

Location

The weak nature of the serpentinite rocks associated with the Avebury orebodies effectively limits the location of any main declines to the hornfels or chert like rocks. Referring to Figure 2, the hornfels dominant sequence to the north of North Avebury tends to be the best rock geotechnically for a decline. However, to ensure that the main decline was reasonably close to both orebodies, it was positioned in the hornfels and chert sequences to the hangingwall of South Avebury. See Figures 6 and 7. Should the Avebury underground mine go ahead, the final decline location may be very different to that shown. As yet to be quantified geotechnical conditions are expected to play a major role in the final underground design.

Portal

As there is no known ore from surface (surface is about 2170 RL) to 2050 RL, the first 800 metres of declining can be arranged in a convenient layout. The design proposed provides the opportunity for underground diamond drilling and a portal location at a reasonably low surface elevation. Further optimization is possible. (Keeping the portal as low as possible minimises the overall development. Every metre lower for the portal saves about $21,000.)



Plan view of drop-cut excavation

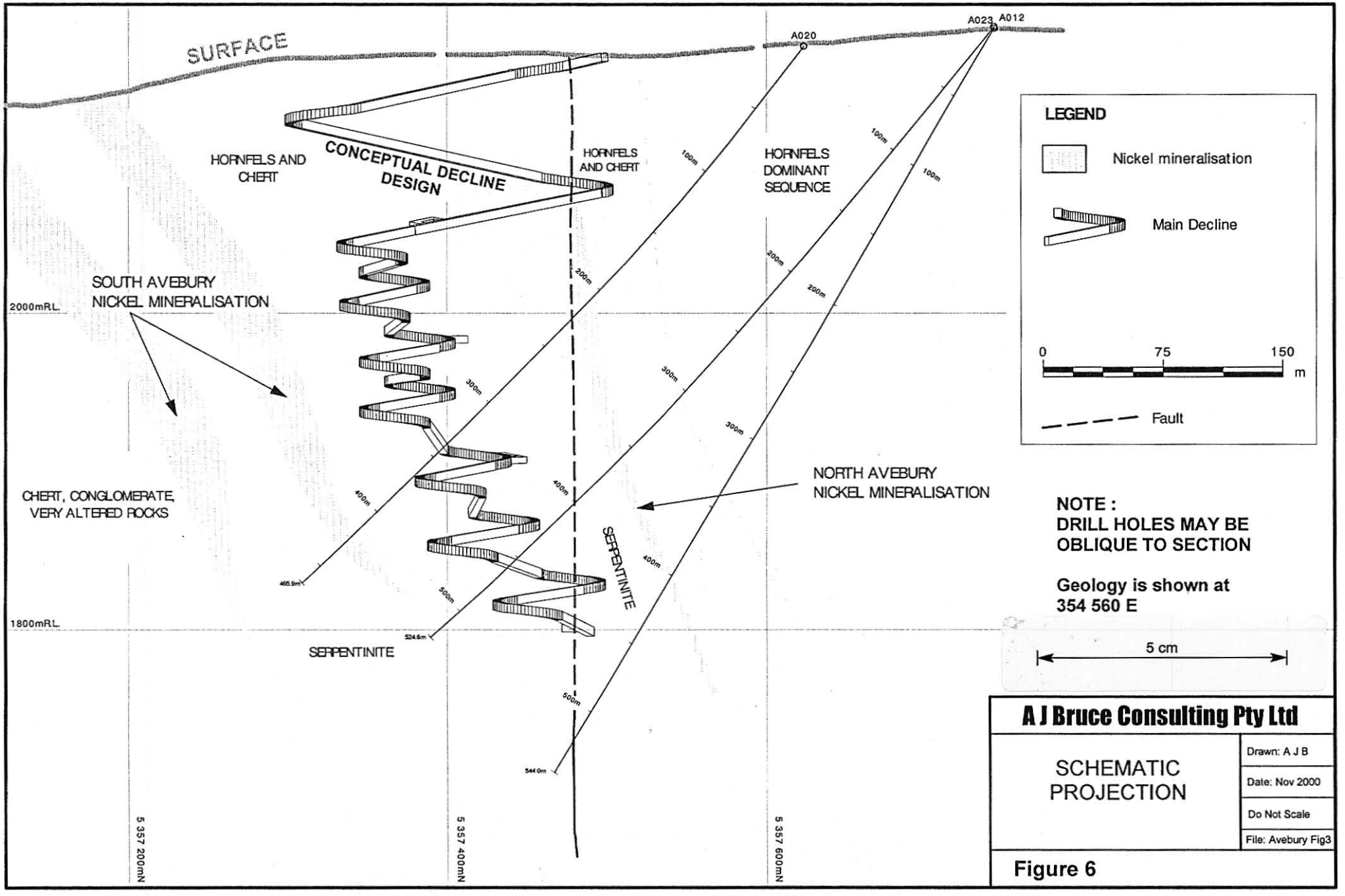
Due to the relatively shallow depth of weathering at the Avebury site only a relatively small 'drop-cut' is planned before the decline drill and blast starts. A local contractor would probably complete this work.

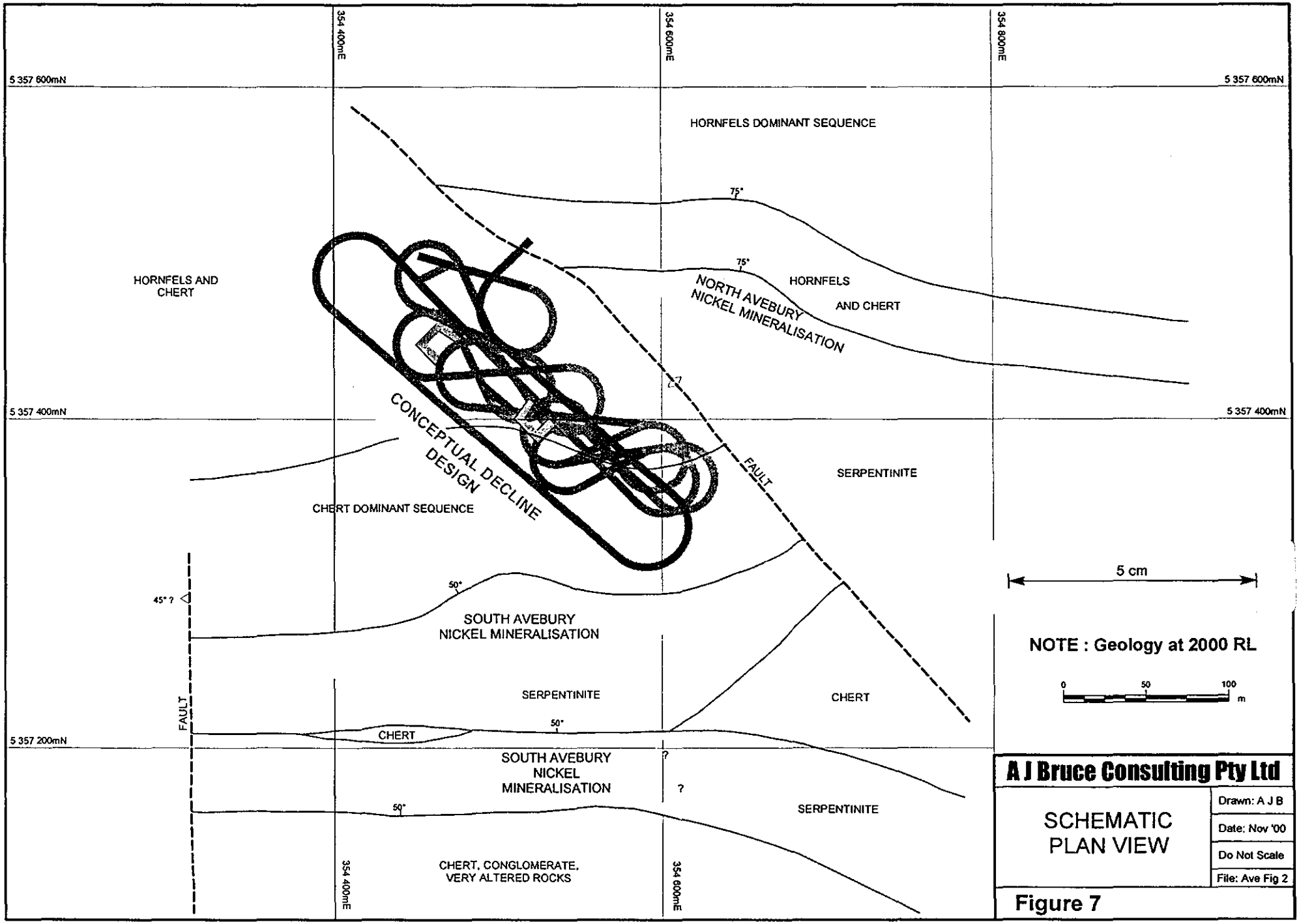
The total excavation should not exceed 4,700 bank cubic metres (bcm). $138,000 has be allowed for the drop cut excavation including ground support (cables, mesh and shotcrete).

An additional $2,500 per metre has been allowed for the first 20 metres of decline development. This provision should cover any extra ground support resulting from surface weathering effects.

Design Parameters

At a relatively modest production rate of 300,000 tpa the Avebury mine can afford to keep to mid sized equipment. Excavation sizes can consequently be minimised which will lead to more stable openings (and attendant lower ground support costs). Other than this


SURFACE
A023
A012
A020
HORNFELS AND CHERT
CONCEPTUAL DECLINE DESIGN
HORNFELS AND CHERT
HORNFELS DOMINANT SEQUENCE
SOUTH AVEBURY NICKEL MINERALISATION
2000mRL
1800mRL
CHERT, CONGLOMERATE, VERY ALTERED ROCKS
SERPENTINITE
NORTH AVEBURY NICKEL MINERALISATION
SERPENTINITE
100m
200m
300m
400m
500m
465.9m
524.6m
544.0m
5 357 200mN
5 357 400mN
5 357 600mN
LEGEND
Nickel mineralisation
Main Decline
0
75
150
m
Fault
NOTE :
DRILL HOLES MAY BE OBLIQUE TO SECTION
Geology is shown at 354 560 E
5 cm
A J Bruce Consulting Pty Ltd
SCHEMATIC PROJECTION
Drawn: A J B
Date: Nov 2000
Do Not Scale
File: Avebury Fig3
Figure 6


354 400mE
354 600mE
354 800mE
5 357 600mN
5 357 400mN
5 357 200mN
HORNFELS DOMINANT SEQUENCE
75°
75°
HORNFELS AND CHERT
NORTH AVEBURY NICKEL MINERALISATION
HORNFELS AND CHERT
CONCEPTUAL DECLINE DESIGN
FAULT
SERPENTINITE
CHERT DOMINANT SEQUENCE
50°
45° ?
SOUTH AVEBURY NICKEL MINERALISATION
SERPENTINITE
CHERT
FAULT
50°
CHERT
SOUTH AVEBURY NICKEL MINERALISATION
?
?
50°
SERPENTINITE
CHERT, CONGLOMERATE, VERY ALTERED ROCKS
5 cm
NOTE : Geology at 2000 RL
0
50
100
m
A J Bruce Consulting Pty Ltd
SCHEMATIC PLAN VIEW
Drawn: A J B
Date: Nov '00
Do Not Scale
File: Ave Fig 2
Figure 7

emphasis on smaller excavation size, the parameters for the decline design will be kept to those more or less standard within the Australian hard rock underground mining industry.

Table 6.2a Development Design Parameters

MAIN DECLINE:		**Comments**
Profile	Semi-arched	
Gradient in straights	-1:7	Vertical : Horizontal
Gradient in curves	-1:8	
Curve Radius	25 or 20 metres	On drive centre line
Height	5.0 metres	
Width	5.0 metres	Almost an industry standard
NORTH ACCESSES:		
Profile	Rectangular	Rounded corners
Gradient	variable +ve	To suit
Height	4.5 metres	About the minimum for 30 trucks
Width	5.0 metres	
SOUTH ACC & DRAW POINTS:		
Profile	Semi-arched	
Gradient	Generally +1:50	
Height	5.0 metres	Higher than Nth to suit extra auxiliary fans
Width	5.0 metres	

Level plans at 1:2000 scale for the proposed Avebury Decline are attached as Figures 8a to 8f.

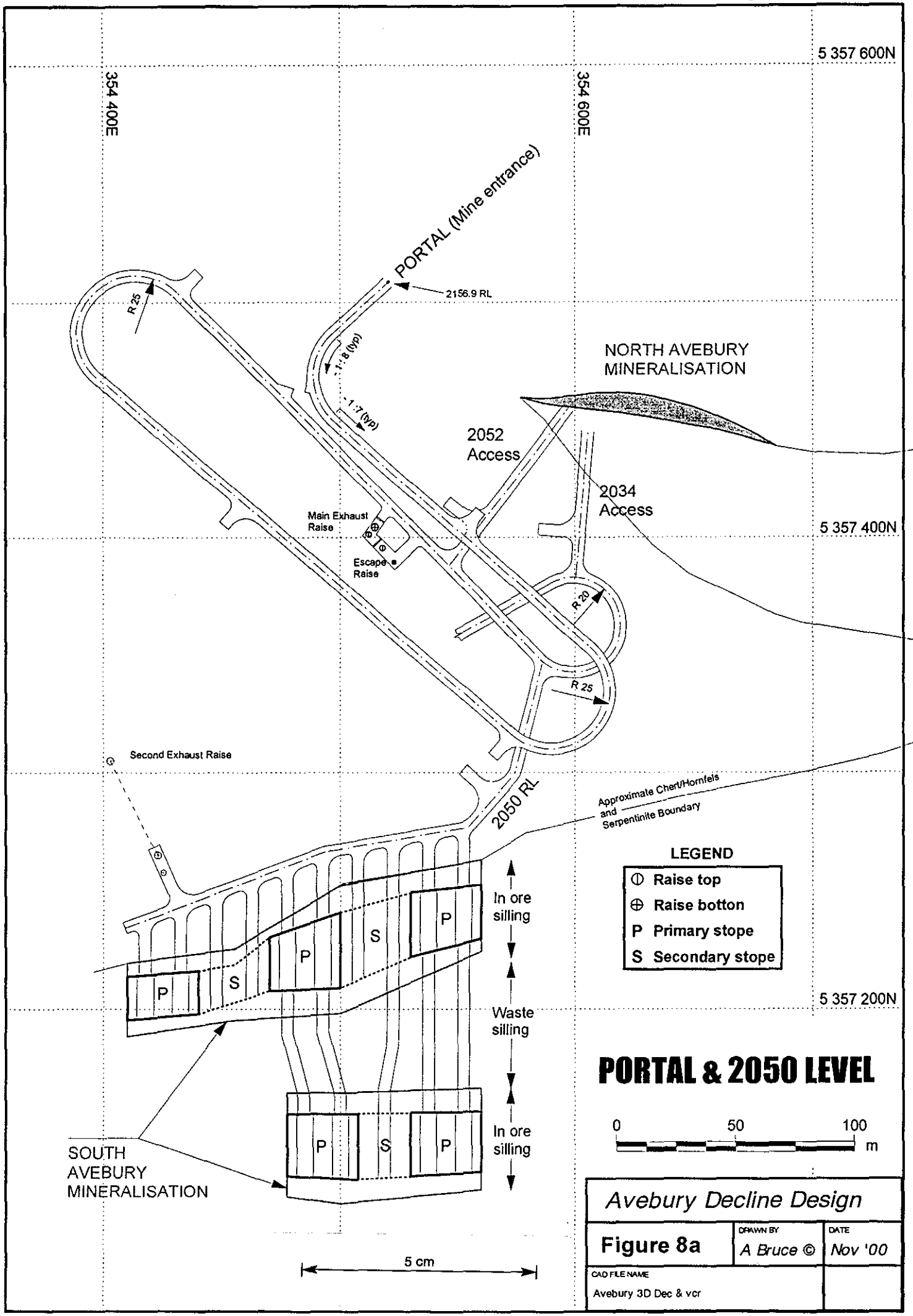
Development Ground Support

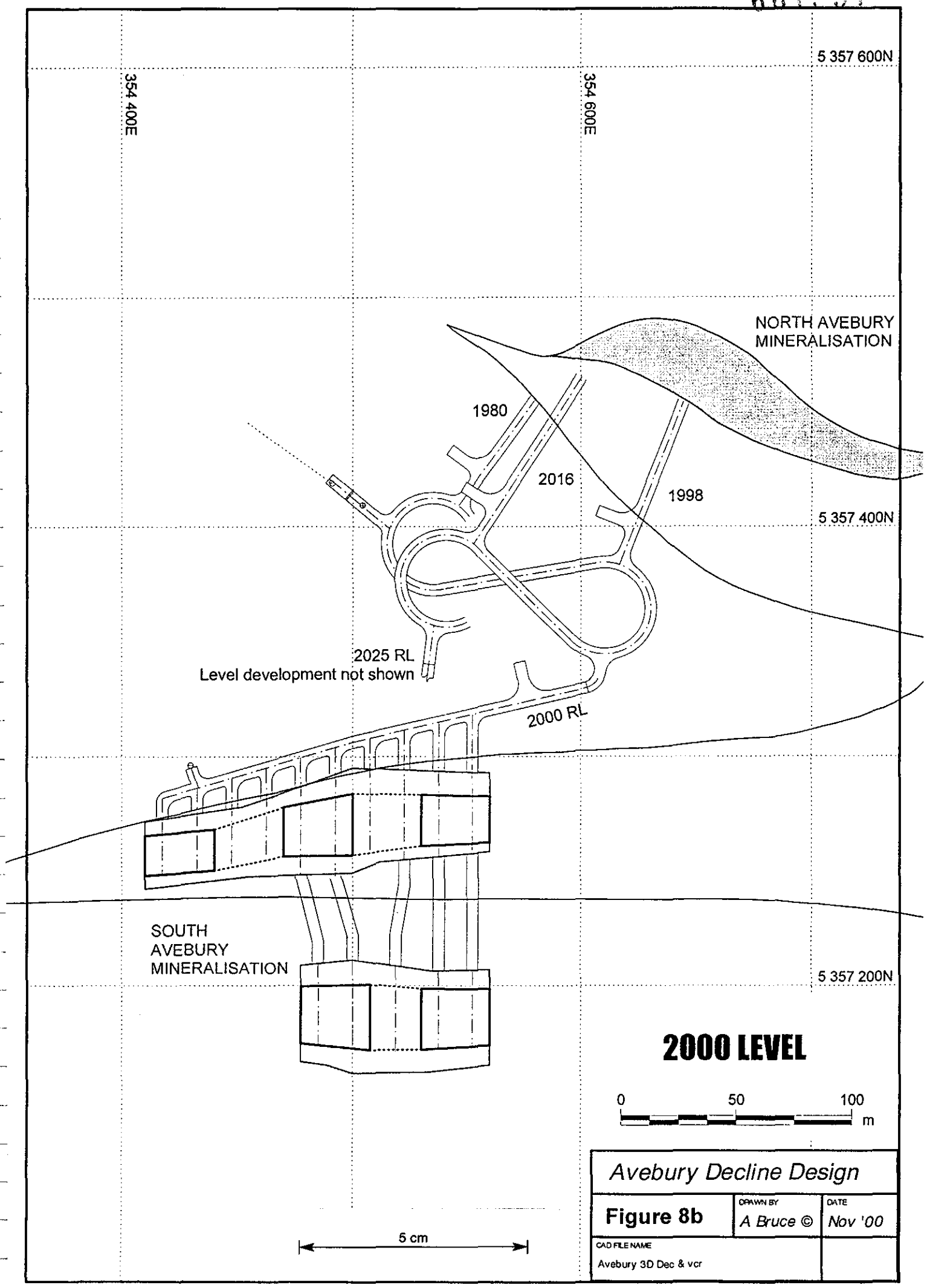
The amount ground support necessary for the Avebury underground mine is likely to be quite high due to relatively poor rock mass quality. The table below has four standards of ground support depending of the ultimate use of the development heading. A tick indicates that this item of support would be installed. The provision for ground support is considered to be generous. Underground exposures of the rock mass are necessary to determine the best level of support.

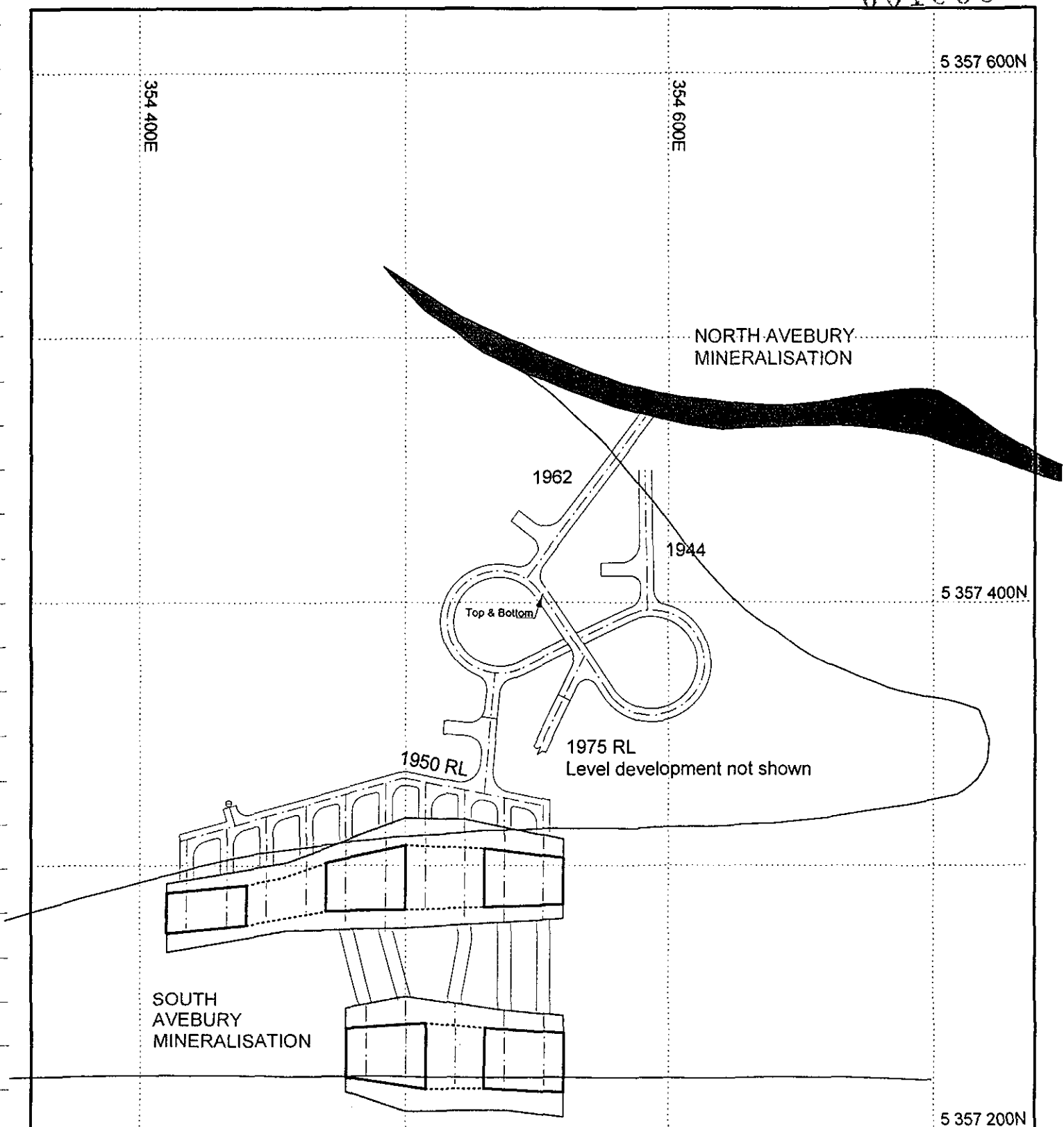
Table 6.2b Ground Support to be installed in development headings

ITEM OF GROUND SUPPORT	NORMAL STANDARD (CODE : N)	MINIMAL (CODE : M)	SILLS SUPPORT (CODE : S)	INTENSE (CODE : I)
Galvanised Split Set	√	√	√	√
Grouting of split set	√			
100x100 Galv. Mesh	√	√	√	√
8 metre Cable Bolt			√	
2.4 metre Resin Bolt				
Shotcrete Backs 50mm				√
Mesh Side Walls				√

NORMAL would apply to any heading being used regularly that has a long life – such as the main declines.
MINIMAL ground support would be used only in locations that do not serve as regular travelling ways.
SILL SUPPORT will be used in any ore development that is wider than five metres or may be stripped to a greater width during production.
INTENSE is not used in this report, but would probably be necessary in crosscuts for sub-level cave mining.


5 357 600N
354 400E
354 600E
PORTAL (Mine entrance)
2156.9 RL
R 25
1:8 (typ)
1:7 (typ)
NORTH AVEBURY
MINERALISATION
2052
Access
2034
Access
Main Exhaust
Raise
Escape
Raise
5 357 400N
R 20
R 25
Second Exhaust Raise
2050 RL
Approximate Chert/Hornfels
and
Serpentinite Boundary
LEGEND
Raise top
Raise botton
P Primary stope
S Secondary stope
In ore
silling
Waste
silling
In ore
silling
5 357 200N
SOUTH
AVEBURY
MINERALISATION
5 cm
PORTAL & 2050 LEVEL
0
50
100
m
Avebury Decline Design
Figure 8a
DRAWN BY
A Bruce ©
DATE
Nov '00
CAD FILE NAME
Avebury 3D Dec & vcr


5 357 600N
354 400E
354 600E
NORTH AVEBURY
MINERALISATION
1980
2016
1998
5 357 400N
2025 RL
Level development not shown
2000 RL
SOUTH
AVEBURY
MINERALISATION
5 357 200N
2000 LEVEL
0
50
100
m
5 cm
Avebury Decline Design
Figure 8b
DRAWN BY
A Bruce ©
DATE
Nov '00
CAD FILE NAME
Avebury 3D Dec & vcr

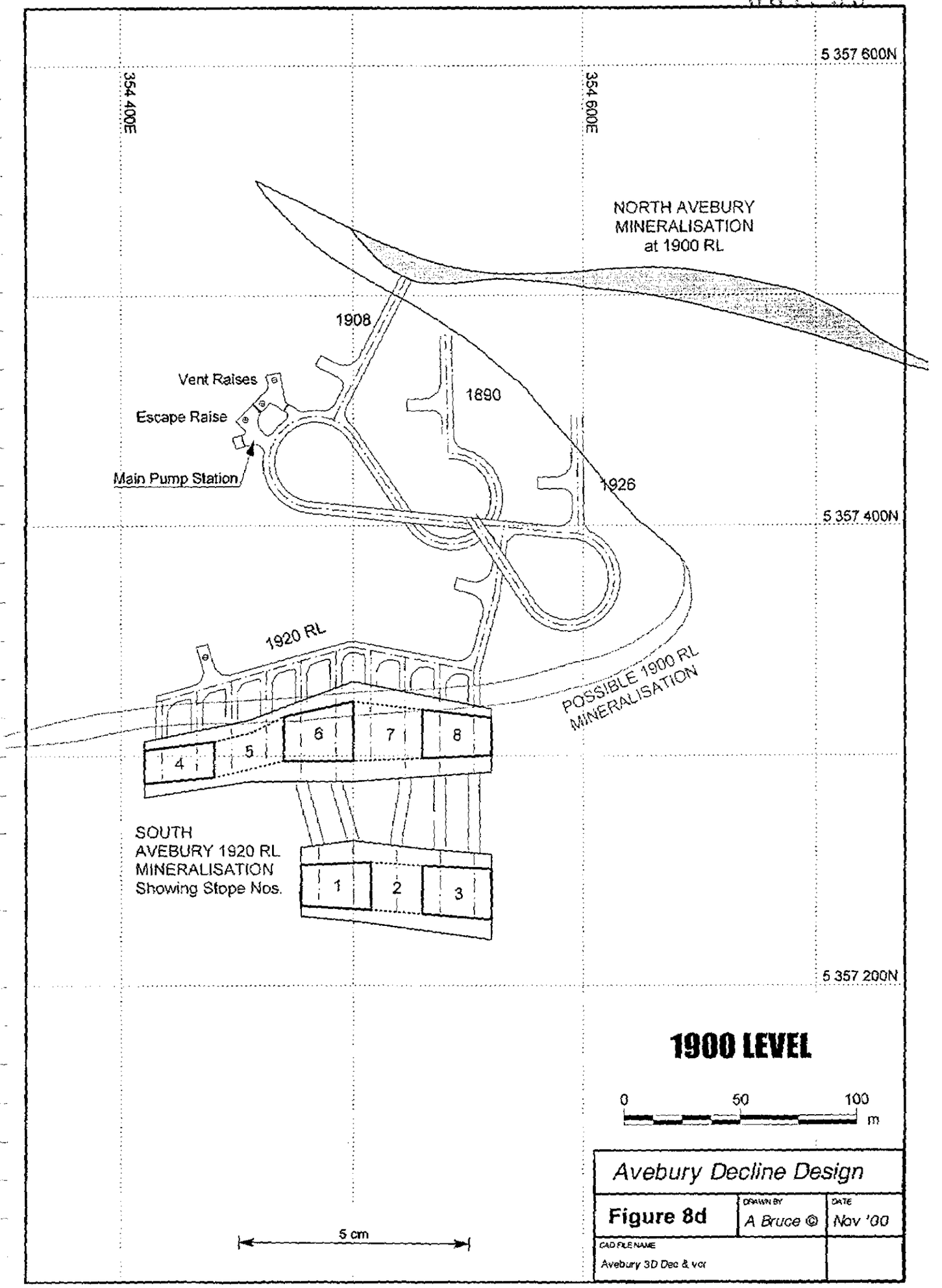


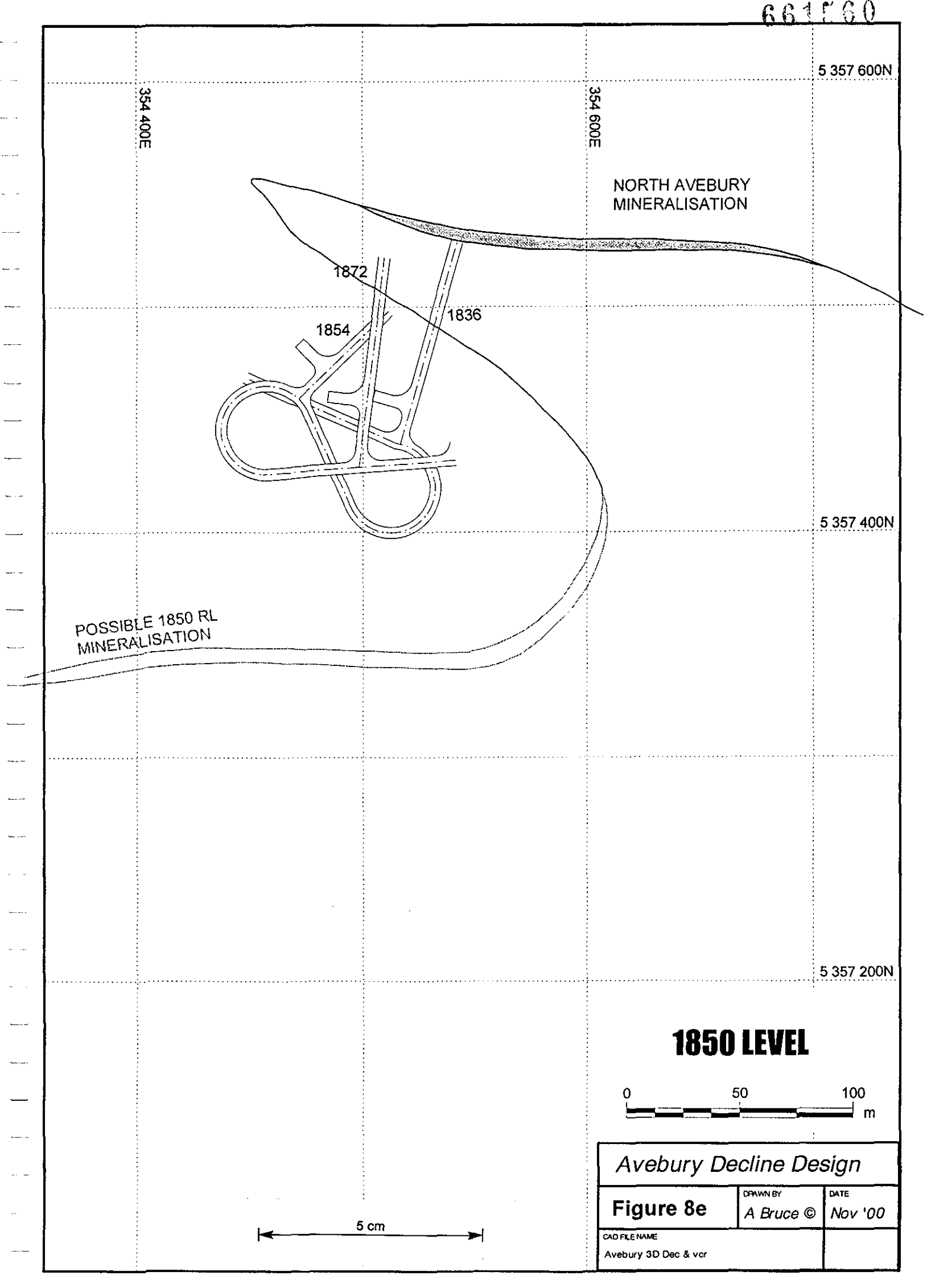
1950 LEVEL

0 50 100 m

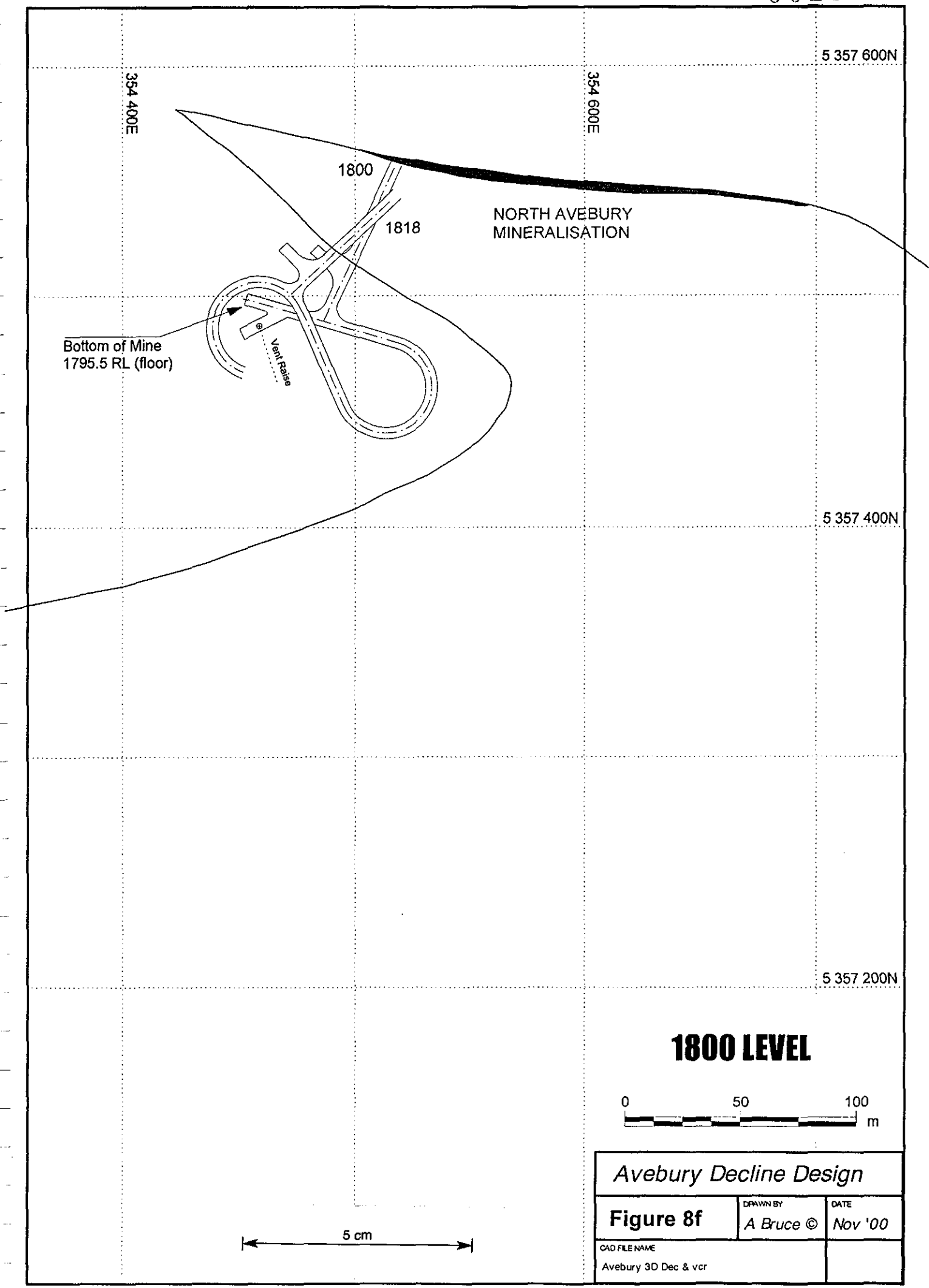
5 cm

Avebury Decline Design		
Figure 8c	DRAWN BY: A Bruce ©	DATE: Nov '00
CAD FILE NAME: Avebury 3D Dec & vcr		


5 357 600N
354 400E
354 600E
NORTH AVEBURY
MINERALISATION
at 1900 RL
1908
1890
1926
Vent Raises
Escape Raise
Main Pump Station
5 357 400N
1920 RL
POSSIBLE 1900 RL
MINERALISATION
4
5
6
7
8
SOUTH
AVEBURY 1920 RL
MINERALISATION
Showing Stope Nos.
1
2
3
5 357 200N
1900 LEVEL
0
50
100
m
5 cm
Avebury Decline Design
Figure 8d
DRAWN BY
A Bruce ©
DATE
Nov '00
CAD FILE NAME
Avebury 3D Dec & var



Avebury Decline Design		
Figure 8e	DRAWN BY *A Bruce ©*	DATE *Nov '00*
CAD FILE NAME Avebury 3D Dec & vcr		


5 357 600N
354 400E
354 600E
1800
1818
NORTH AVEBURY
MINERALISATION
Bottom of Mine
1795.5 RL (floor)
Vent Raise
5 357 400N
5 357 200N
1800 LEVEL
0
50
100
m
5 cm
Avebury Decline Design
Figure 8f
DRAWN BY
A Bruce ©
DATE
Nov '00
CAD FILE NAME
Avebury 3D Dec & vcr

Ground support is a cost major component in development cost.

Decline Services

Services costs make up only a minor component in the cost of development, but it is important to define exactly what is required particularly if contractors are used.

Table 6.2c Services to be installed in development headings

ITEM	TYPE	SERVICES TO A HIGH STANDARD (CODE : H)	NORMAL (CODE : N)	MINIMAL SERVICES (CODE : M)
Dewatering Line	150mm steel	√		
	110mm poly	√	√	
Comp. Air Line	150mm steel	√		
	75mm poly		√	
	50mm poly			√
Fresh Water Line	80mm steel	√		
	75mm poly		√	
	50mm poly			√
Electrical Cables	11kV	√		
	1000V	√	√	
Pipe Hangers		√	√	√

HIGH STANDARD. Services installation at the highest standard would only be installed in the main decline. The main decline is generally the conduit for collecting and pumping dirty water, distribution of fresh water, and main power distribution.
NORMAL. As used mostly in crosscuts to access ore.
MINIMAL SERVICES would only be used for temporary installations such as sill drives.

The proposed development profiles showing typical ground support and the layout of services are shown on Figure 9.

6.3 Lateral Development

The following tables summarise all of the lateral waste development necessary to extract the Avebury North and South orebodies based on the designs as shown on Figures 8a to 8f. Note that Figure 8a shows typical South Avebury ore development for the 2050 Level, as well as the waste development.

'Capital' and 'Operating' development have been separated, as each would be treated differently for taxation purposes.

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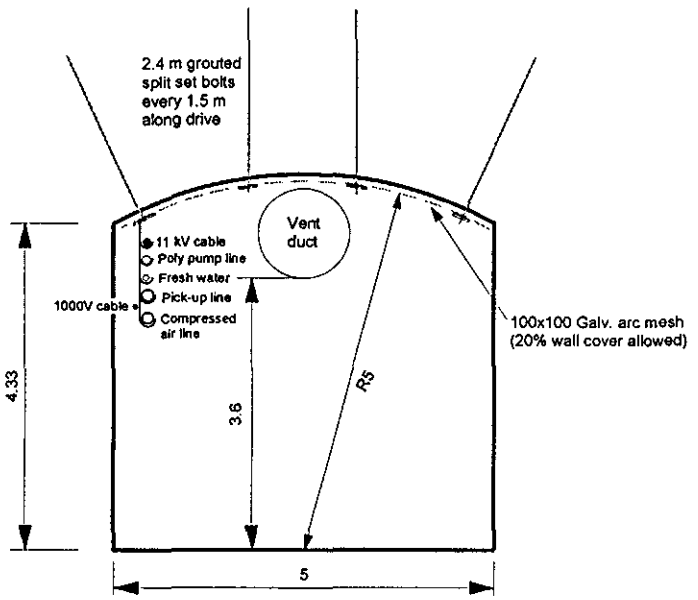
5 cm

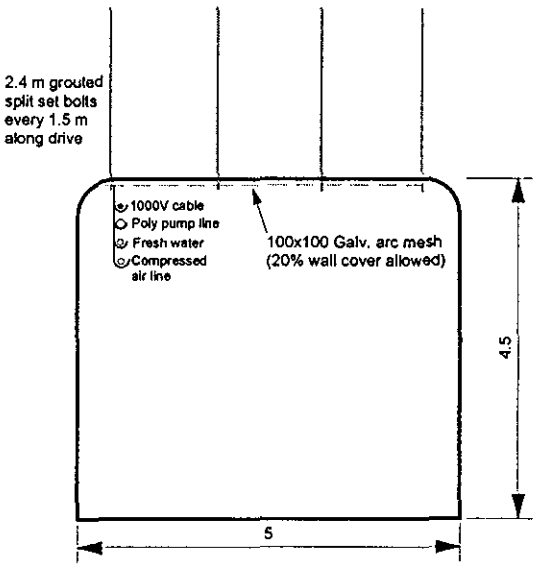
TITLE		
Development Profiles		
Figure 9	DRAWN BY: A Bruce ©	DATE: Nov '00
CAD FILE NAME: Decline Profile Figure 9		SCALE: 1 : 100

TYPICAL WASTE DEVELOPMENT



Declines with 'High' standard of services and 'Normal' ground support

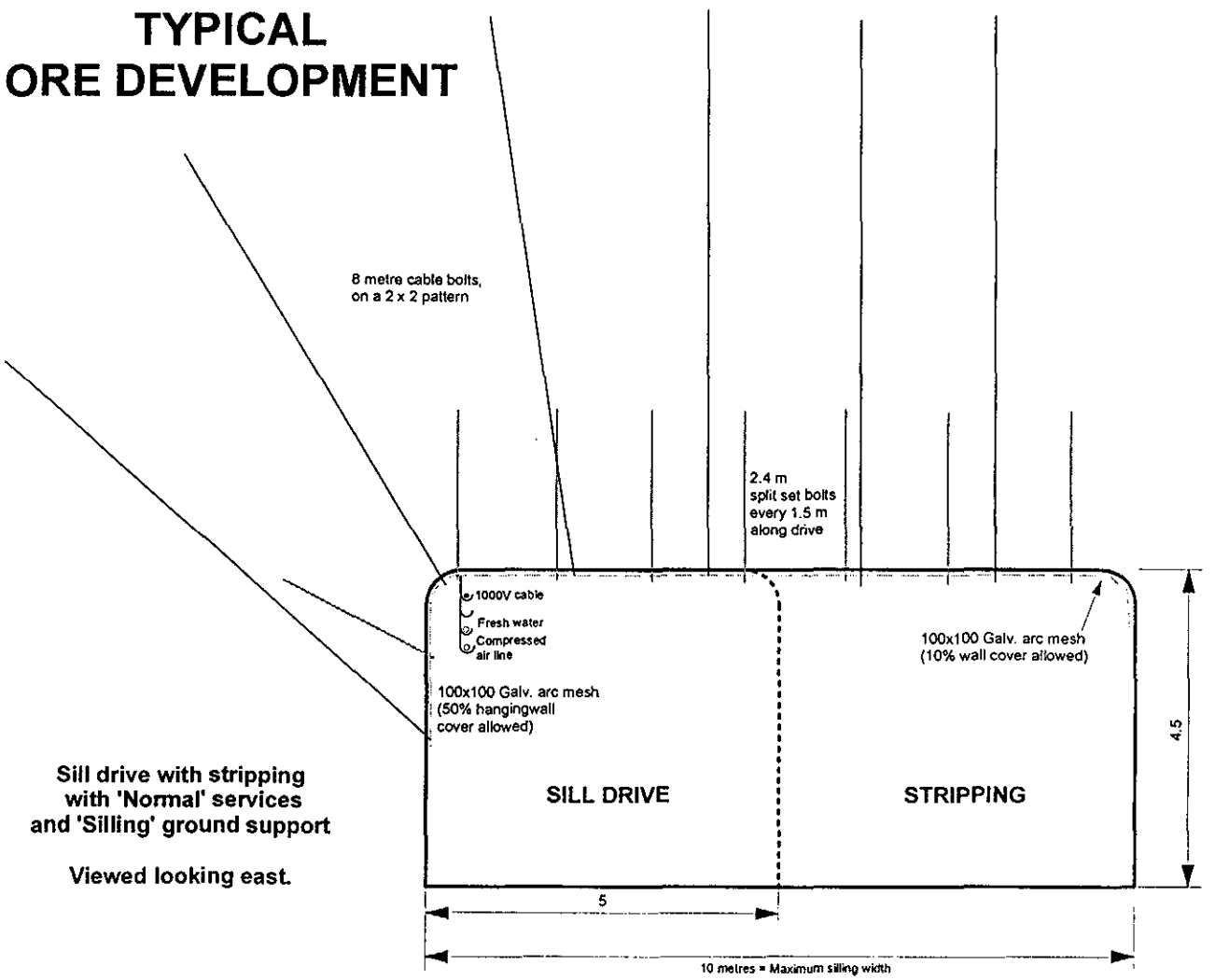
5.0 x 5 semi arch area = 23.9 m2



Crosscuts with 'Normal' services and 'Normal' ground support

4.5 x 5 rectangular = 22.5 m2

TYPICAL ORE DEVELOPMENT



Sill drive with stripping with 'Normal' services and 'Silling' ground support

Viewed looking east.

Table 6.3a CAPITAL DEVELOPMENT – Main Decline				
NOMINAL LEVEL	HAULAGE DISTANCE TO SURFACE	DECLINE	LOADING BAYS	INFRASTRUCTURE
Portal to 2050	**378**	755 m	40 m	40 m
2050	**859**	208 m	45 m	
2000	**1155**	383 m	75 m	25 m
1950	**1479**	265 m	60 m	
1950 sub-level	**1678**	133 m	15 m	
1900	**1904**	320 m	45 m	60 m
1850	**2250**	372 m	45 m	
1800	**2574**	275 m	30 m	17 m
	TOTALS	2711 m	355 m	142 m
	Height	5.0	6.0	4.5
	Width	5.0	5.0	5.0
	Services Code	H	M	N
	Ground Support Code	N	M	M

Notes on the Table above:

- The haulage distance to surface is not necessarily the distance the waste will be hauled as some will be used as stope fill. The distance from the portal to the surface waste dump is **not** included.
- Loading bays are 10 or 15 metres long. The first 15 metres of every access shown below is counted as loading bay and included above.
- Infrastructure development is related to either ventilation raises or pump stations.

Table 6.3b OPERATING DEVELOPMENT – North orebody				
NOMINAL LEVEL	HAULAGE DISTANCE TO SURFACE	ACCESS NAME	ACCESS LENGTH	LOAD BAY IN ACCESS
2050	**859**	**2052**	**65 m**	**15 m**
		2034	65 m	15 m
2000	**1155**	**2016**	**55 m**	**15 m**
		1998	55 m	15 m
		1980	55 m	15 m
1950	**1479**	**1962**	**55 m**	**15 m**
		1944	55 m	15 m
1950 sub-level	**1678**	1926	55 m	15 m
1900	**1904**	**1908**	**50 m**	**15 m**
		1890	50 m	15 m
1850	**2250**	1872	75 m	15 m
		1854	75 m	15 m
		1836	**75 m**	**15 m**
1800	**2574**	1818	58 m	15 m
		1800	**58 m**	**15 m**
TOTALS			901 m	225 m
	Height		4.5	5.0
	Width		5.0	5.0
	Services Code		N	M
	Ground Support Code		N	M

Table 6.3c OPERATING DEVELOPMENT – South orebody					
NOMINAL LEVEL	HAULAGE DISTANCE TO SURFACE	ACCESS NAME	ACCESS & FOOTWALL DRIVE	DRAW POINTS	WASTE SILLING (SERPENTINITE)
2050	859	2050	254 m	150 m	230 m
2000	1155	2025	216 m	150 m	222 m
		2000	216 m	150 m	222 m
1950	1479	1975	197 m	150 m	154 m
		1950	197 m	150 m	154 m
1900	1904	1920	221 m	150 m	152 m
TOTALS			1301 m	900 m	1134 m
	HEIGHT		5.0	4.5	4.5
	WIDTH		5.0	5.0	5.0
	SERVICES CODE		N	N	M
	GROUND SUPPORT CODE		N	N	N

Notes on Table 6.3c:

- Each level has 10 x 15m long draw points.
- The waste silling is in between South Avebury southern and northern lenses. As shown on Figures 8a to 8d.

6.4 Waste Development Costs

Calculations

Mine development is normally quoted as a cost per metre of linear excavation (generally measured horizontally). For this Avebury scoping study individual cost calculations were done for each distinct type of development allowing for appropriate ground support, installed services, development size and individually calculated haulage distance for the waste disposal. The results are summarised in Table 6.4. The unit rates used in the calculations are based on current contractor rates that are likely to apply in Western Tasmania.

Haulage distances were calculated after preparation of development and production schedules, with waste dumped underground directly into stopes where possible. In the case where waste went to surface an additional haul distance from the portal to a surface dump of 420 metres is added.

Each segment of the development may have a different haulage distance (depending on when it is mined and where it is dumped). Consequently it is not meaningful to quote haulage cost as a 'cost per metre'. Each case must be calculated individually, Table 6.4 reflects this.

Table 6.4 Waste Development Costs

	CAPITAL			NORTH OREBODY		SOUTH OREBODY		
	Decline	Loading Bays	Infra-structure	Access	Load Bay	Access[g]	Draw Points	Waste Silling
Basic Cost / metre	**$1,761**	**$2,030**	**$1,685**	**$1,685**	**$1,761**	**$1,761**	**$1,685**	**$1,685**
Service Code	H	M	N	N	M	N	N	M
⇒ **Cost / metre**	**$267**	**$67**	**$148**	**$148**	**$67**	**$148**	**$148**	**$67**
Ground support Code	N	M	M	N	M	N	N	N
⇒ **Cost / m2**	**$63**	**$56**	**$56**	**$63**	**$56**	**$63**	**$63**	**$63**
Sidewall support coverage	20%	10%	10%	10%	10%	20%	20%	10%
⇒ **Cost / metre**	**$441**	**$344**	**$327**	**$372**	**$333**	**$441**	**$428**	**$372**
Cost / metre excluding haulage	**$2,468**	**$2,442**	**$2,161**	**$2,205**	**$2,161**	**$2,350**	**$2,262**	**$2,124**
Total cost including haulage (Millions)	$7.4	$1.0	$0.3	$2.3	$0.6	$3.5	$2.3	$2.7
⇒ Total Cost / Metre	$2,733	$2,769	$2,421	$2,503	$2,479	$2,661	$2,557	$2,414

6.5 Ventilation

Based on the diesel equipment necessary to operate the Avebury mine, the minimum airflow to meet the statutory requirements is 125 m^3/s. As 125 m^3/s is a relatively low flow by Tasmanian standards (Tasmanian mines have historically had better ventilation than other States), a figure of 200 m^3/s was set to determine raise and ventilation fan sizes.

Table 6.5 Ventilation Raise Costs

PURPOSE	LENGTH (METRES)	DIAMETER (METRES)	TOTAL COST (SPECIALIST CONTRACTOR)
Main Vent	111	3.1	$227,000
Escape / Services to Surface	111	1.8	$155,000
1900 Vent	163	2.4	$256,000
1800 Vent	103	2.4	$180,000
Open Stope Primary Vent	122	2.4	$188,000
Open stope 1920-2050 Vent	149	1.8	$220,000
1960 Service	73	1.8	$133,000
1900 Service	84	1.8	$130,000
EQUIPING the Escape / Services Raises	268		$214,000
TOTAL			**$1,703,000**

[g] *Includes open stope tramming drive, a loading bay & vent crosscut.*

Ventilation simulations were run to determine the surface fan sizes and raise sizes to suit the 200 m^3/s airflow. The main surface exhaust fan will be about 200 kW and the second surface fan about 150 kW in total power. Estimated cost for purchase and installation of new fans is $255,000 and $128,000 respectively. It may be possible to locally source similar second-hand fans.

The fans and ventilation flows referred to above are adequate to service the mine through the development **and** production stages.

In addition to the cost of the fans and raises a total of $36,000 will be required for the construction of underground ventilation walls.

6.6 Development Summary

The total **waste** lateral development necessary to fully extract the known extent of Avebury North & South is 7,669 metres.

The total **waste** raising necessary to fully extract the known extent of Avebury North & South is 916 metres.

Figure 10 is a long projection of the mine looking North and showing key elements of mine development.

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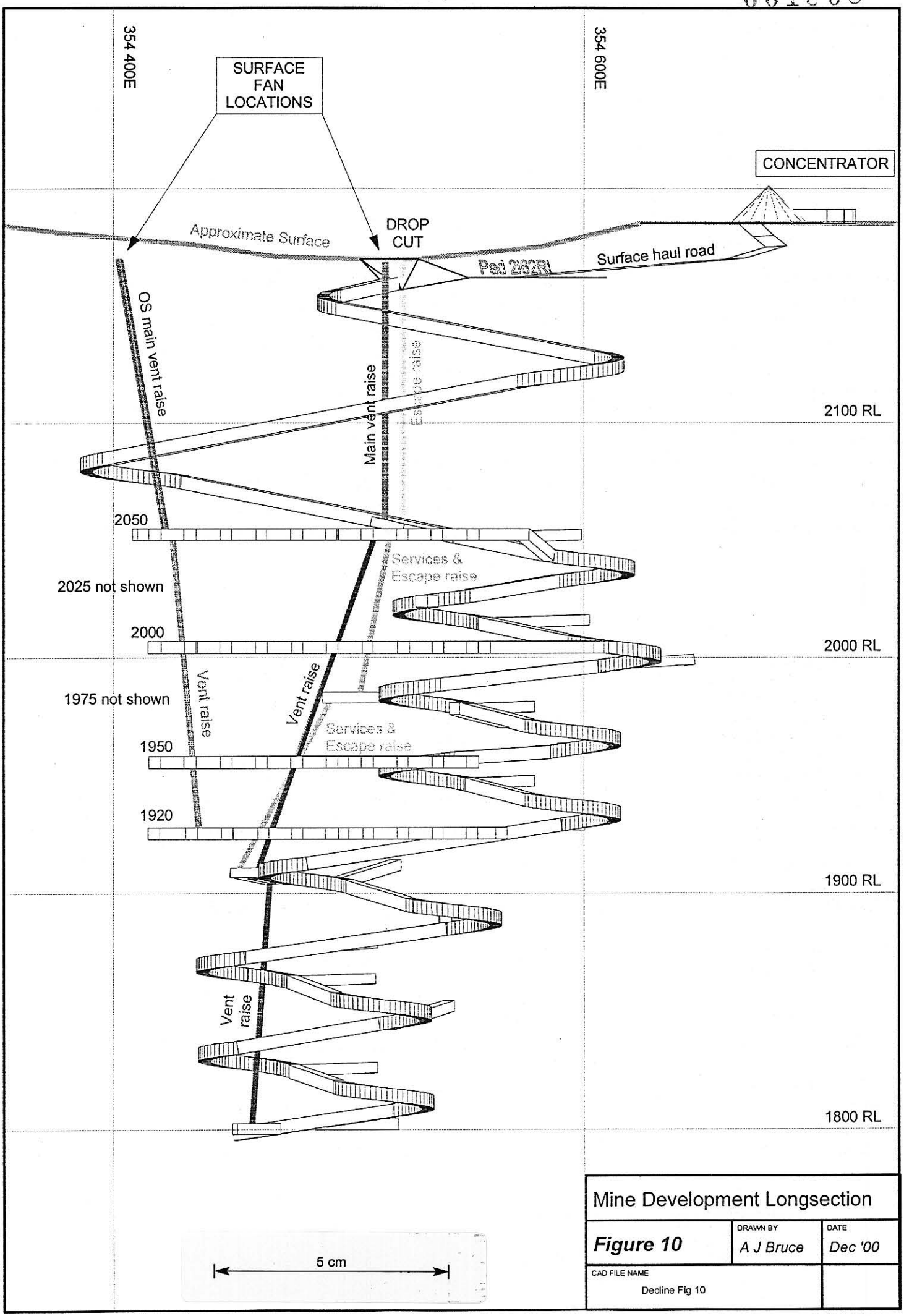

Mine Development Longsection		
Figure 10	DRAWN BY: A J Bruce	DATE: Dec '00
CAD FILE NAME: Decline Fig 10		

7. ORE PRODUCTION

7.1 Benching

Introduction

The proposed method of extraction for the North orebody is benching. This widely used mining method involves driving horizontal sills for the full strike length of the orebody. The sills are driven at vertical intervals equal to the sub-level interval (18 metres in this case). See Figure 11. The sills are stripped to the full orebody width if possible (the maximum stable width will depend on geotechnical conditions, 10 metres has been used for Avebury North). Once at least two sills are complete production can start.

The production sequence involves firing (usually vertical) rings of blast holes into a void created by a vertical cut-off raise between the sills. The bench face will retreat along strike until the maximum stable stope size is reached. At this point the stope must be filled with waste rock, a small pillar left and another cut-off raise mined. The cycle then repeats until the whole level is completed and filled. The next bench above can then start.

The following table gives the basic parameters for the North Avebury Bench stopes.

Table 7.1a Bench Stope Parameters

PARAMETER	VALUE	COMMENTS
Ore Density	3.0 t/m^3	Supplied by Allegiance
Sublevel Interval	18 m	Floor to Floor
Maximum Span	10 m	As calculated from geotechnical analysis
Stable Stope Length	18 m	As calculated from geotechnical analysis
Stope Pillar Size	4 m	Based on industry standards
Crown Pillar Losses	3 m	Likely losses after temporary crown extraction
Minimum Mining Width	4 m	Limited by equipment

Stope Summary

Table 7.1b North Avebury Extraction Reserve

SILL TONNES EXTRACTED	STOPE TONNES AVAILABLE	PILLAR LOSSES	STOPE TONNES BLASTED	NO. OF INDIVIDUAL STOPES
304,000	801,000	142,000	659,000	119

A summary of the extractable reserve for North Avebury using a Benching mining method is given in the table above. In addition, dilution will amount to about 52,000 tonnes or 8% of the blasted stope tonnes.

Total ore extraction from the North lens is 1,016,000 tonnes.

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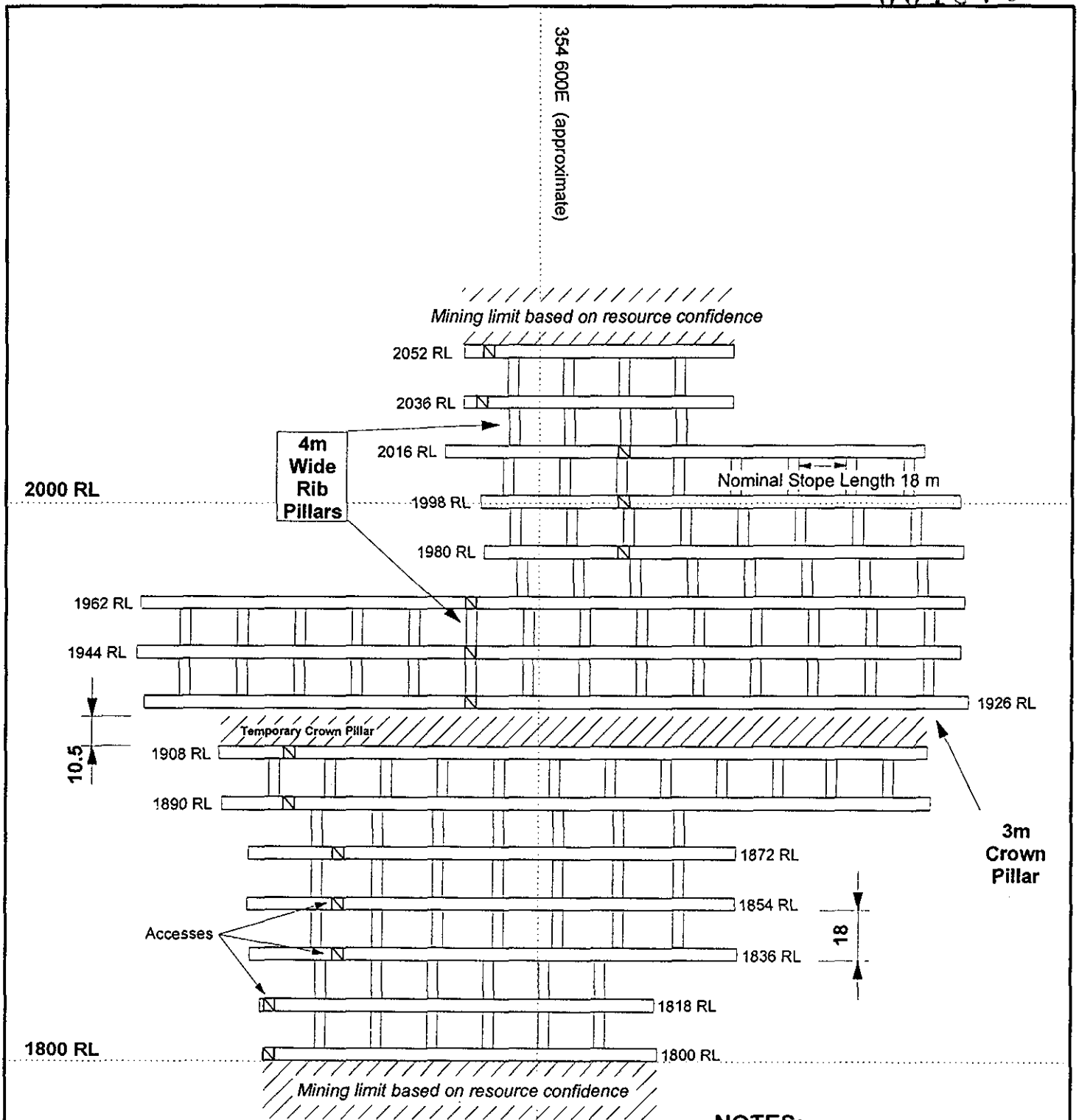


NOTES:

Silling 4.5h x 4w then strip to maximum stope width 10m.

Sublevel interval 18m.

Rib pillars & crown pillar will not be recoverable.

SCHEMATIC LONGITUDINAL PROJECTION

0 50 100 m

Scale 1:2000

5 cm

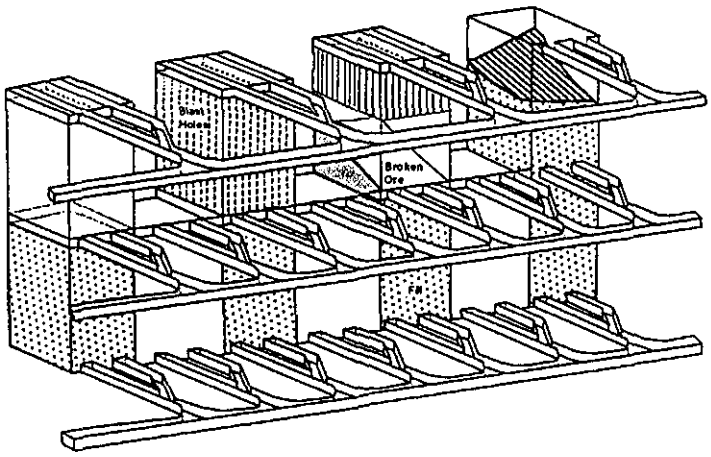
Nth Avebury Bench Stopes		
Figure 11	DRAWN BY *A Bruce ©*	DATE *Nov '00*
CAD FILE NAME Nth LongProj Stopes		

The calculation of an extractable reserve is based on the mining method, the parameters above and the sill geology as supplied[h] for the 1800, 1850, 1900, 1950, 2000 & 2050 RL's. Printouts of the spreadsheet calculations can be found in Appendix E.

7.2 Open Stoping

Introduction

The proposed extraction method for the South Avebury lenses is an open stoping technique best described as 'modified single lift vertical retreat stoping'. An idealized sketch of the method as discussed by Villaescusa[i] is shown here. In this report the method will just be referred to as open stoping (OS).



SINGLE LIFT VERTICAL RETREAT STOPING after Potvin

Extraction begins once development of at least two sublevels is complete.

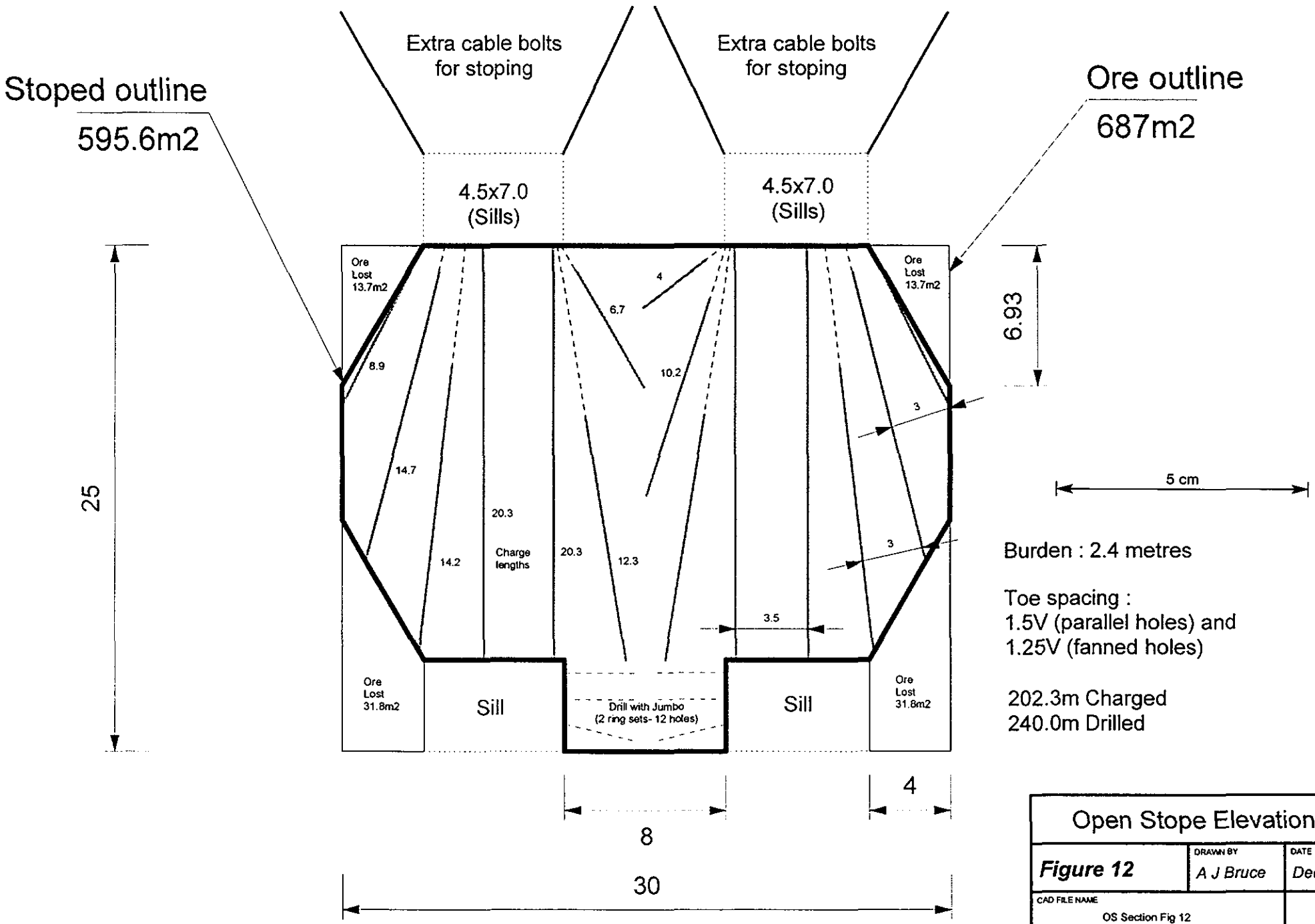
South Avebury was divided into five primary stopes and three secondary stopes as shown on Figure 8a. As each primary stope is completed it will be filled with a combination of waste rock and cemented mill tailings to create stable walls for the adjoining secondary stope(s).

Unlike the drawing it is planned too use fanned production drill holes rather than the parallel holes shown above. Avebury orebodies are not considered stable enough to strip the orebody to full width. A typical elevation, looking south, of a single lift is shown as Figure 12.

Table 7.2a Basic Open Stope Parameters

PARAMETER	VALUE	COMMENTS
Ore Density	**3.0 t/m^3**	**Supplied by Allegiance**
Sublevel Interval	**25 m**	**Used to match production drill hole length**
Maximum Span	**na**	**As calculated from geotechnical analysis**
Stable Stope Length	**30 m**	**As calculated from geotechnical analysis**
Stable Stope Width	**30 m**	**As calculated from geotechnical analysis**
Stope Pillar Size	**None**	**Cemented rock fill used**
Crown Pillar Losses	**None**	**Cemented rock fill used**
Maximum Sill Width	**7 m**	**As calculated from geotechnical analysis**

[h] *Avebury Ore RLs-R13.dxf, see Section 2.3*
[i] *See "A Review of Sublevel Stoping", E Villaescusa in Reference 9.*


Stoped outline
595.6m2
Extra cable bolts
for stoping
Extra cable bolts
for stoping
Ore outline
687m2
4.5x7.0
(Sills)
4.5x7.0
(Sills)
Ore
Lost
13.7m2
Ore
Lost
13.7m2
6.93
4
6.7
10.2
8.9
3
14.7
5 cm
20.3
Charge
lengths
20.3
12.3
14.2
3
3.5
25
Ore
Lost
31.8m2
Sill
Drill with Jumbo
(2 ring sets- 12 holes)
Sill
Ore
Lost
31.8m2
4
8
30
Burden : 2.4 metres
Toe spacing :
1.5V (parallel holes) and
1.25V (fanned holes)
202.3m Charged
240.0m Drilled
Open Stope Elevation
Figure 12
DRAWN BY
A J Bruce
DATE
Dec '00
CAD FILE NAME
OS Section Fig 12

Stope Summary

A summary of the extractable reserve for South Avebury using an Open Stope mining method is given in the table below. Dilution was calculated for each stope individually and is included in the table below.

Table 7.2b South Avebury Extraction Reserve

Sill Tonnes Extracted	Stope Tonnes Available	Pillar Losses	Low Grade Silling	Stope Tonnes Blasted	H/W, F/W & Fill Dilution	No. of Individual Stopes
153,000	1,846,000	244,000	68,000[1]	1,602,000[2]	203,000	40

[1]Note: includes 2050RL sills and 32,400 tonnes of low-grade silling.
[2]Note: includes the 96,000 tonnes of jumbo stripping

The calculation of an extractable reserve is based on the mining method, the parameters above and the 1920-2050 geology wireframes[j] as supplied. The calculations are much more complex than for the North Avebury Benches. Printouts of the spreadsheet calculations can be found in Appendix F.

Total ore extraction from the South lens is 2,026,000 tonnes.

7.3 Ore Production Costs

Sill Production

Orebody silling in preparation for stoping produces a significant proportion of the total life of mine production. Sill ore production always has a higher cost per tonne than stope production.

In a similar fashion to waste development, each area of sill production had the costs calculated individually. The costs include the jumbo development, service installation, ground support (including North stope hangingwalls) and haulage to the ROM pad at the concentrator. It should be noted that the development sizes are quite different for North and South Avebury.

The total cost per tonne, referred to below, is the cost per tonne of ore directly produced from the silling process.

[j] *E-mailed coordinates, see Section 2.3*

Table 7.3a Ore Sill Development Costs

OREBODY	NORTH			SOUTH	
PROCESS	SILLING	SILL STRIPPING	NORTHERN SILL	LOW GRADE	SOUTHERN SILL
UNITS:	METRES	M^3	METRES	METRES	METRES
QUANTITY:	3,035 (163,890t)	46,768 (140,306t)	1,261 (119,165t)	480 (32,400t)	736 (69,552t)
Unit Costs including haulage	$2,754	$82	$3,565	$2,379	$3,511
Total cost including haulage	$8.4 million	$3.8 million	$4.5 million	$1.1 million	$2.6 million
TOTAL COST / TONNE	**$51.00**	**$27.26**	**$37.73**	**$35.24**	**$37.16**

Stope Production Costs

Both benching and open stoping mining methods have similar production processes and for a scoping study there is no need to apply different unit costs.

Table 7.3b Stoping Unit Costs

ACTIVITY	UNIT COST	COMMENTS
Production Raises	$720/m	Bench raises 13.5 m OS raises 20.5 m
76mm Longhole drilling	$26.59/m	Assumes down holes
Charge-up & blast	$24.46/m of charge	Assumes ANFO
Secondary blasting	$0.06 per stope tonne	Estimate, higher for OS
Stope Bogging	$3.40/tonne	Includes remoting
Haulage to ROM	$4.47/tonne	Benches (average)
Haulage to ROM	$4.20/tonne	OS (smaller av. distance)
Rock fill ex development	$3.40/m3	Delivered to stope
Surface quarried fill	$8.96/m3	Delivered to stope
Cemented rock fill	$14.10/m3	Delivered to stope
Extra sill cables bolts	$340 each	Needed in OS only
Jumbo stripping for OS	$40/ m3	Pillar between the sills

With the exception of secondary blasting and cable bolting, each of the unit costs in Table 7.3b were calculated specifically for the Western Tasmanian situation and most are specific to the Avebury deposit and the proposed mining methods.

Table 7.3c Stoping Unit Costs

ACTIVITY	UNITS	UNIT COST	QUANTITY	TOTAL COST	OVERALL COST PER TONNE	COMMENTS
BENCH STOPING						
Pre - Production						
Sill Driving	t	$51.00	163,890	$8,358,390	$8.23	See Table 7.3a
Sill Stripping	t	$27.26	140,306	$3,824,742	$3.77	See Table 7.3a
Stope Production						
Rising	each	$9,720	119	$1,156,680	$1.14	13.5 x $720/m
Drilling	m	$26.59	54,883	$1,459,335	$1.44	12 t/metre drilled
Blasting	m	$24.48	41,162	$1,007,649	$0.99	16 t/metre drilled, pf=0.27
Stope Haulage Costs						
Dilution						52,000 tonnes
Secondary Blasting	t	$0.06	710,594	$42,636	$0.04	
Bogging	t	$3.20	710,594	$2,273,901	$2.24	
Average Haulage to ROM	t	$4.47	710,594	$3,176,355	$3.13	
Stope Filling						
Rock Fill ex Develop't	m3	$3.40	60,000	$204,000	$0.20	Placement cost only
Quarried Waste Rock Fill	m3	$8.96	217,064	$1,944,893	$1.92	
BENCH TOTAL				**$23,453,895**	**$23.10**	**for 1,015,000 t prod'n**
OPEN STOPING						
Pre - Production						
Sill Driving	t	$37.50	188,717	$7,076,888	$3.52	Av. unit cost for 2 lenses
Low Grade Silling	t	$35.24	32,400	$1,141,776	$0.57	See Table 7.3a
Stope Production						
Rising	m	$14,760	40	$590,400	$0.29	20.5 x $720
Drilling	m	$26.59	93,173	$2,477,481	$1.23	16 t/metre drilled
Blasting	m	$24.48	82,821	$2,027,453	$1.01	18 t/metre drilled, pf=0.24
Pillar Stripping	m3	$40.00	31,968	$1,278,720	$0.64	
Extra Cable Support	per 2 m	$680	833	$566,100	$0.28	2 x 8m cables every 2 m
Stope Haulage Costs						
Dilution						203,000 tonnes
Secondary Blasting	t	$0.08	1,586,678	$126,934	$0.06	Expected > than benching
Bogging	t	$3.20	1,789,883	$5,727,627	$2.85	
Average Haulage to ROM	t	$4.20	1,789,883	$7,517,510	$3.74	
Stope Filling						
Rock Fill ex Develop't	m3	$3.40	60,000	$204,000	$0.10	Placement cost only
Quarried Waste Rock Fill	m3	$8.96	185,735	$1,664,186	$0.83	
Cemented Rock Fill	m3	$14.10	415,055	$5,852,276	$2.91	
OS TOTAL				**$36,251,350**	**$18.03**	**for 2,011,000 t prod'n**

NOTES:
PRE-PRODUCTION SILLING & STRIPPING IS AS DERIVED IN TABLE 7.3A
DILUTION IS SHOWN AS IT AFFECTS THE BOGGING, HAULAGE & SECONDARY BLASTING COST.
THE ROCK FILL SPLIT BETWEEN OS AND BENCHING IS ARBRITRARY.

Due to a slightly different calculation method, the production tonnages in Table 7.3c do not agree exactly with the more accurate estimates provided in Tables 7.1b and 7.2b. The error is less than 1%. Production schedules are based on Tables 7.1b and 7.2b.

7.4 Production & Development Schedules

Once a commitment is made to extract the Avebury orebodies, it is reasonable to assume that development will proceed as quickly as possible to facilitate early production. On this basis it was assumed that for development and production scheduling purposes, a high-speed development contractor would be used until stope production was 'on line'. After that silling **and** waste development is done at a more modest rate presuming the use of a single jumbo. All mine ore & waste development could be completed just into year seven; but it may be desirable to slow this down to a 'just-in-time' scenario.

Ore from sill driving would become available early in the second year with stope production starting in the middle of the second year.

Table 7.4a provides a basic summary of the detailed production schedule attached as Table 7.4b. Detailed development schedules are included in Appendix G.

Table 7.4a Development & Production Schedule Summary

Year	Waste Develop't (m)	Ore Silling (m)	Sill Production (000's Tonnes)	Stope Production (000's Tonnes)	Total Production (000's Tonnes)	Est. Stope Void to Fill (000's m^3)
1	2831	0	0	0	0	0
2	1419	1401	142	108	250	17
3	929	1167	99	201	300	90
4	1164	803	86	214	300	100
5	951	1068	93	206	300	100
6	375	930	87	213	300	100
7		155	18	282	300	100
8				300	300	100
9				300	300	100
10				300	300	100
11				300	300	80
12				78	78	21

The nickel grades shown in Table 7.4b are preliminary estimates, based on an early grade tonnage model[k] for the South Avebury lenses. The grades resulting from this model were used to provide a feel for the likely effects of stope dilution. **These estimates are subject to further geological interpretation and should not be quoted outside this report.**

Waste generated by development will exceed the underground fill demand for Years one and two leading to a peak waste rock stockpile on surface of 256,000 tonnes in Year two. In Year five all of the surface excess waste rock will have been consumed and about 200,000 tpa of waste rock must to be sourced from a surface location. The cost of quarrying this rock locally is already included in the unit rates above.

[k] *Grade = $0.0692x^2$ - 0.7523x + 3.0071, where x = total Resource tonnage in millions of tonnes.*

A J Bruce Consulting Pty Ltd

Table 7.4b Life of Mine PRODUCTION & DEVELOPMENT SCHEDULE - Avebury Mine

17-Dec-00

ORE BODY	ORE BLOCK	Estimated Minable Grade	Minable Tonnes ooo's	Haulage to surf. Dist. (km)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
North																
BENCHES	2052 Sill	1.7	8.6	0.4						8.6						
80	*Development (metres)*									*80*						
	2034-2052 Bench	1.7	21.6	0.9								21.6				
	2034 Sill & Strip	1.7	8.6	0.9					8.6							
80	*Development (metres)*								*80*							
	2016-2034 Bench	1.7	21.6	0.9							21.6					
	2016 Sill & Strip	1.7	23.4	0.9					23.4							
70	*Development (metres)*							*70*								
	1998-2016 Bench	1.7	58.1	1.2							58.1					
	1998 Sill & Strip	1.7	23.4	1.2				23.4								
70	*Development (metres)*							*70*								
	1980-1998 Bench	1.7	58.1	1.2						58.1						
	1980 Sill & Strip	1.7	23.4	1.2				23.4								
70	*Development (metres)*							*70*								
	1962-1980 Bench	1.7	58.1	1.5					58.1							
	1962 Sill & Strip	1.7	37.2	1.5			37.2									
70	*Development (metres)*					*70*										
	1944-1962 Bench	1.7	91.8	1.5				91.8								
	1944 Sill & Strip	1.7	37.2	1.5		37.2										
70	*Development (metres)*				*70*											
	1926-1944 Bench	1.7	91.8	1.7			91.8									
	1926 Sill & Strip	1.7	37.2	1.7		37.2										
70	*Development (metres)*				*70*											
	1908-1926 Bench	1.7	53.2	1.9								53.2				
	1908 Sill & Strip	1.7	27.8	1.9						27.8						
65	*Development (metres)*									*65*						
	1890-1908 Bench	1.7	68.4	2.0								68.4				
	1890 Sill & Strip	1.7	27.8	2.0						27.8						
65	*Development (metres)*								*65*							
	1872-1890 Bench	1.7	28.8	2.1							28.8					
	1872 Sill & Strip	1.7	11.4	2.1					11.4							
90	*Development (metres)*								*90*							
	1854-1872 Bench	1.7	28.8	2.3						28.8						
	1854 Sill & Strip	1.7	11.4	2.3					11.4							
90	*Development (metres)*								*90*							
	1836-1854 Bench	1.7	28.8	2.3						28.8						
	1836 Sill & Strip	1.7	11.4	2.3			11.4									
90	*Development (metres)*						*90*									
	1818-1836 Bench	1.7	18.8	2.5					18.8							
	1818 Sill & Strip	1.7	7.6	2.5			7.6									
73	*Development (metres)*						*73*									
	1800-1818 Bench	1.7	18.8	2.6				18.8								
	1800 Sill & Strip	1.7	7.6	2.6			7.6									
73	*Development (metres)*						*73*									
17	*Development (metres)*						*17*									
South OPEN	2050 Sills	1.0	35.4	0.6						17.4	18					
STOPES	2050 Low grade Sills	0.8	5.4	0.6						5.4						
634	*Development (metres)*								*404*	*230*						
	2025 Sills	1.8	33.4	1.3					33.4							
	2025 Low grade Sills	0.8	5.4	1.3					5.4							
	2025-2050 Primary	1.6	238.2	1.3											238.2	
	2025-2050 Secondary	1.6	135.5	1.3											57.2	78.3
588	*Development (metres)*							*366*	*222*							
	2000 Sills	1.8	33.4	1.4				33.4								
	2000 Low grade Sills	0.8	5.4	1.4				5.4								
	2000-2025 Primary	1.6	231.4	1.4									59.2	172.2		
	2000-2025 Secondary	1.6	132.4	1.4										127.8	4.6	
588	*Development (metres)*							*588*								
	1975 Sills	1.8	29.4	1.5			29.4									
	1975 Low grade Sills	0.8	5.4	1.5			5.4									
	1975-2000 Primary	1.6	219.4	1.5								105.5	113.9			
	1975-2000 Secondary	1.6	126.9	1.5									126.9			
501	*Development (metres)*					*347*	*154*									
	1950 Sills	1.8	29.4	1.7		29.4										
	1950 Low grade Sills	0.8	5.4	1.7		5.4										
	1950-1975 Primary	1.6	204.7	1.7					32.3	88	84.4					
	1950-1975 Secondary	1.6	120.2	1.7							80.4	39.8				
501	*Development (metres)*				*347*	*154*										
	1920 Sills	1.8	27.8	1.9		27.8										
	1920 Low grade Sills	0.8	5.4	1.9		5.4										
	1920-1950 Primary	1.6	249.9	1.9		108	59.6	82								
	1920-1950 Secondary	1.6	146.4	1.9			42.7	13	91							
523	*Development (metres)*				*100*	*423*										
	Development (metres)															
	Miscellaneous			1.6												
8%	Bench Dilution	0.8	51.7	1.6			7.3	6.8	6.2	9.3	8.7	11.5				
	Operating Development															
3191	*Development (metres)*				*2244*	*425*	*522*									
	TOTALS:		3029	1.6 av.		250	300.0	300.0	300.0	300.0	300.0	300.0	300.0	300.0	300.0	78.3
	GRADE % Sn:		1.64%			1.65%	1.65%	1.65%	1.65%	1.60%	1.60%	1.64%	1.64%	1.64%	1.64%	1.64%
	RECOVERY:					90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%

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CONTAINED NICKEL	4141	4948	4942	4948	4793	4797	4909	4920	4920	4920	1284
Ni produced in MILL	3727	4453	4448	4453	4313	4317	4418	4428	4428	4428	1156

Days Per Year	250	250	250	250	250	250	250	250	250	250	250	250
Av. Ore Haul one way		1.777	1.772	1.624	1.661	1.623	1.504	1.669	1.480	1.400	1.302	1.300
Capital Development	*2244*	*425*	*539*									
Operating Development	*587.0*	*994.0*	*390.0*	*1164.0*	*951.0*	*375.0*						
Total Dev for YEAR	*2831.0*	*1419.0*	*929.0*	*1164.0*	*951.0*	*375.0*						
% Needing Fill	*20%*	*20%*	*90%*	*100%*	*100%*	*100%*	*100%*	*100%*	*100%*	*100%*	*80%*	*80%*
Void to Fill (m3)		*16,693*	*90,013*	*100,016*	*99,984*	*99,985*	*99,993*	*99,985*	*100,000*	*100,000*	*80,000*	*20,880*
Bulked Fill from Development (m3)	*99,085*	*45,692*	*29,914*	*37,481*	*30,522*	*12,075*						
Approx. FILL SURPLUS (m3)	*99085*	*28898*	*-60099*	*-62535*	*-69362*	*-87910*	*-99993*	*-99985*	*-100000*	*-100000*	*-80000*	*-20880*
NET SUFACE STOCKPILE (t)	*198170*	*256167*	*135968*	*10898*	*-127826*	*-303647*	*-503633*	*-703604*	*-903604*	*-1103604*	*-1083604*	*-1145384*

8. MOBILE EQUIPMENT and FIXED PLANT

Fixed Plant

Other than the surface fans which are covered in Section 6.5, the only significant fixed plant required for underground mine are a single underground pumpstation located at 1900RL and a air compressor which is best located close to the mine portal.

Table 8a Fixed Plant Belonging to the Mine Owner

ITEM	COST	COMMENTS
1900 Main Pumpstation	$180,000	2 x 50kW mono pumps
Surface Compressor	$133,000	160kW supplying 450l/s FAD

Mobile Equipment

Table 8b Mobile Equipment under the Contract Mining Scenario

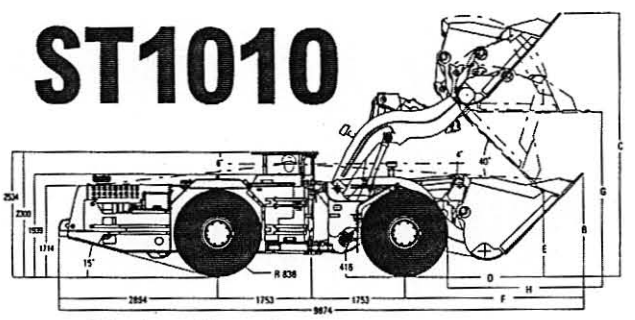
ITEM	UNIT COST	MINE SITE OWNED EACH	CONTRACTOR OWNS EACH	TOTAL REQUIRED EACH
MOBILE EQUIPMENT				
2 Boom Jumbo (2 x 45kW)	$840,000	0	1	1
Longhole Drill	$900,000	0	1	1
Loader 5m3	$745,000	0	2	2
Loader Remotes	$75,000	0	2	2
28 Tonne Truck	$829,000	0	2	2
Grader	$260,000	1	0	1
Light Vehicles t.s. U/G use	$80,000	5	4	9
Articulated Service Vehicles	$150,000	1	1	2
MISC EQUIPMENT				
Electric Face Pumps	$20,000	1	2	3
Air Pumps	$6,000	2	3	5
ANFO Loaders	$5,000	0	3	3
Mobile Toilets	$15,000	2	0	2
Self Rescuers & Cap Lamps	$600	40	0	40
Office Equipment & Software	$50,000	1	0	1
Miner Rescue Equipment	$100,000	1	0	1
Mains Firing Box	$5,000	1	0	1
Hand Held Rockdrills	$8,000	2	0	2
Fresh Air Refuge Chambers	$14,000	2	0	2
Dewatering Pump Tanks	$20,000	2	1	3
Mobile Substations 1000V	$180,000	2	1	3
Auxiliary Fans 55kW	$35,000	0	6	6
TOTAL COST		**$1,495,000**	**$5,991,000**	**$7,486,000**

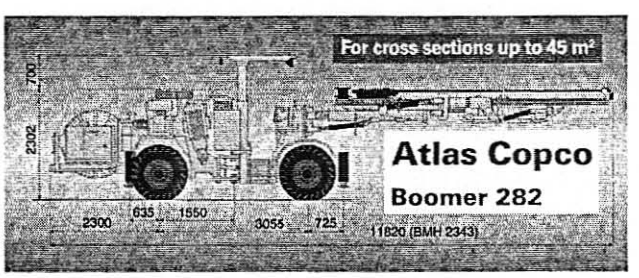
Should the mine Owner decide to operate the site under a totally owner operator scenario, they would need to spend the full $7,486,000 on the capital equipment listed in Table 8b.

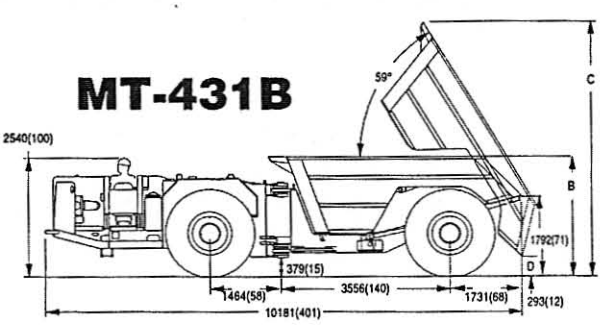
All of the equipment costs listed above are based on the purchase of new equipment. Some typical modern equipment, with the appropriate capabilities is shown below. Prices are current at October 2000 and are based on an $US exchange rate of 0.55 to the $A.

To **all** of the capital costs for this project, it is strongly recommended that an additional contingency provision of 8-10% be added to allow for items which are inevitably overlooked in costing exercises. This applies even at the scoping study stage.

The contingency is to allow for over sights and is added to the capital estimate, on top of this are the variances in the estimates which for a scoping study are taken as ±30%.







9. SERVICES and INTERFACES

The mining operation has been designed as a largely stand-alone operation supplying the concentrator with 300,000 tpa of ore. Despite this interdependence, there are a number of interfaces with other parts of the over-all mine. These interfaces are described below to enable **others** to allow for and cost these components in the Project as a whole.

9.1 Services

Power and fresh water supplies are assumed to have connection points at or near the decline portal.

Power

The underground mine workings will need a supply capacity peaking at 2 megawatts and a likely average demand of 1.4 megawatts for a monthly consumption of 1,008,000 kW hours. Included in the demand is the electric power for two surface fans and an air compressor. Each of these items would located in the vicinity of the portal.

Supply voltage should be 11kV to the underground workings with provision to supply 1000V, 415V and 240V at the portal pad. In addition the surface fans will require 415V supplies.

Mine Dirty Water Discharge

With no hydrological studies on the Avebury deposit it is difficult to quantify the likely mine dirty water discharge rate. Anecdotal comments that "no diamond drill holes have hit water" would imply a relatively dry mine. The mine design consequently allowed for a relatively small underground pump station rated at 13 litres per second average flow. Peak capacity will be about 24 l/s.

Fresh Water

Fresh (potable) water needs to be supplied to the mine portal. The storage system and pipeline must have a capacity to supply peak rates of 40 litres per second for up to one hour in duration. Average mine consumption will be about 4 litres per second.

9.2 Stope Fill

Where possible stopes will be filled with waste generated directly from mine development, however the primary open stopes require a 'cemented rock fill' (CRF) to ensure stable walls while the secondary stopes are extracted. The bench stopes will all be filled with waste rock fill (RF).

Cemented rock fill is a combination of RF and cemented hydraulic fill (CHF). The CHF is generated by mixing de-slimed concentrator tailings with 3-5% portland cement as a hardening agent (or an equally pozzolanic material).

The only useful fraction of mill tailings is that courser than 50 microns. 35% of Avebury tailings meet this specification according to Butler[1]. The quantity available is adequate for the peak mine requirements. Butler indicated that he would allow for the cyclones and storage tanks for the hydraulic fill in the Concentrator costs.

Peak use of portland cement will be 2,400 tonnes per annum at 4% by weight of the CHF requirement. Average usage will be about 2,000 tpa from Year 3 onwards.

9.3 Infrastructure

Assuming the underground mine is operated under the recommended contract system proposed in Section 4, most of the surfaces facilities will be owned and operated by the Contractor. A mobilisation and de-mobilisation cost of $400,000 will be incurred.

In addition to the facilities provided by the Contractor, the Owner needs to provide:

- Small mine offices for survey, mine planning & mine management.
- Change room facilities for about 30 (including geology) and cap lamp room.
- First aid facilities.
- Portal shed and lay-down area for ancillary equipment.
- Truck/loader wash down bay.
- A high standard all-weather haul road from the portal to ROM.
- Maintenance workshop for electrical & mechanical maintenance of mine infrastructure.
- Two surface magazines for explosives.

None of these items are included in the mining costs in this report.

9.4 Consumables

Explosives

Two surface magazines (one for high explosives and one for detonators) are required in a location at least 200 metres away from any habitable building. The magazines must be bunded with earth walls and in a cleared area no less than 400 m^2 (each).

Diesel Fuel

The mine is estimated to use 4,500 litre of diesel fuel per day.

An estimated 50-100 litres per day of oils and lubricants will be consumed in rock-drills and mobile equipment. Most of this will find its way into the mine dirty water discharge.

[1] *Mr. John Butler, Avebury Metallurgical Scoping Study consultant engineer.*

10. OVER-ALL MINING COSTS

Table 9a Contract Mining

ITEM	UNIT COST	UNITS	TOTAL
Infrastructure & Start-up	$4,604,655	each	$4,604,655
Waste Development	$20,050,992	each	$20,050,992
Bench Stoping	$23.10	per Bench tonne	$23,151,851
Open Stoping	$18.03	per OS tonne	$36,525,346
Services	$3.62	tonne	$10,963,532
Supervision	$3.81	tonne	$11,538,966
Total Mine Life expenditure			**$106,835,342**

A major mining Contractor would base their contract costs for a project such as the Avebury mine on:

- fully amortizing the mining equipment over project life,
- finance costs for their equipment at around 7-9% pa, and
- use a profit mark-up of 5-7%.

As a consequence of the assumptions[m] above the contractor profit mark-up can be estimated, for the purposes of this scoping study, to be around 19% of the unit rates.

Provided the Owner can maintain an equally efficient workforce, their **unit** operating cost should be about 84% of what a Contractor will charge. The following table estimates the reduction in operating costs should the Avebury owner decide to fully own and operate the underground mine.

Table 9b Owner Operator

ITEM	UNIT COST	UNITS	TOTAL
Infrastructure & Start-up	$10,595,655	each	$10,595,655
Waste Development	$18,290,560	each	$18,290,560
Bench Stoping	$19.37	per Bench tonne	$19,419,003
Open Stoping	$15.12	per OS tonne	$30,636,247
Services	$3.62	tonne	$10,963,532
Supervision	$3.81	tonne	$11,538,966
Total Mine Life expenditure			**$101,443,963**

[m] *Based on discussions with an experienced and reliable source from the Contracting industry.*

11. RISKS

From the mining point of view, the most significant threats are likely to result from poorer ground conditions than expected or excessive ground water. Both of these items will lead to high costs and in the worst case make the deposit uneconomic to mine.

As a consequence of the sparsity of geotechnical and hydrological data for the Avebury prospect, it is impossible to quantify the associated risks at this stage. A full assessment will have to wait until the Feasibility Study stage. By that time a detailed geotechnical program must have been completed to establish accurate estimates for the rock mass quality and identify any hydrological hazards.

Despite the comments in the paragraph above, it is considered unlikely that ground conditions or hydrological factors will have such a large effect on mining costs that the estimates provided in this report fall outside a 'scoping stage' level of accuracy. Quite a high standard of ground support has been provided for, and mine dewatering is rarely a major cost item.

12. FUTURE INVESTIGATIONS

Based on the current estimates for the size and geometry of the Avebury Resource, it would appear to be an attractive mining proposition. The orebody geometry lends itself to a number of efficient mining methods and operating costs should be reasonable (but not low mainly because of the difficult ground conditions). There is scope for reductions in the unit stoping costs should the resource size be increased. To move the project to a Feasibility Stage will involve the following key mining related steps.

- During the next phase of the diamond drilling program considerably more and higher quality recording of geotechnical data is necessary.
- Some hydrological studies are needed.
- The geotechnical data will need to be evaluated by external Geotechnical specialist.
- A mine feasibility study must be conducted fully integrating the hydrological and geotechnical studies.

Geotechnical

The first three points can be lumped together as they will all be closely related and will most economically be achieved by coordination through the one specialist consulting firm.

To gather the diamond drill data, triple tube drilling to give coverage at approximately 40m by 40m throughout the orebodies is recommended, at total of about 30 holes. McKeown indicated that allowance has already been made for the cost of the drilling in the next program. Logging of appropriate sections of this core will involve about 30 man-days. Hydrological fieldwork is likely to take 10 days and the Geotechnical/Hydrological

Table 13 Mine Life Expenditure Summary

COSTS ARE IN THOUSANDS of $'s		YEAR 0	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	MINE LIFE
			Year Total	Year Total	Year Total	Year Total	Year Total	Year Total	Year Total	Year Total	Year Total	Year Total	Year Total	Year Total	
DEVELOPMENT (m)	Capital		2,244.0	425.0	539.0										3,208.0
	Operating		587.0	994.0	390.0	1,164.0	951.0	375.0							4,461.0
PRODUCTION ('000s)	Bench Sill Tonnes			74	64	47	55	64							304.0
	Bench Tonnes				99	119	83	125	117	155					698.5
	TOTAL BENCH PRODUCTION			74	163	166	138	189	117	155					1,002.5
	OS Sill Tonnes			68	35	39	39	23	18						221.2
	OS Tonnes			108	102	95	123	88	165	145	300	300	300	78	1,805.0
	TOTAL OPEN STOPE PRODUCTION			176	137	134	162	111	183	145	300	300	300	78	2,026.2
	ANNUAL PRODUCTION			250.4	300.0	300.0	300.0	300.0	300.0	300.0	300.0	300.0	300.0	78.3	3,028.7
DEVELOPMENT COSTS	Capital Development Cost		$6086	$1153	$1462										$8700
	Operating Development Cost		$1566	$2651	$1040	$3104	$2536	$1000							$11897
STOPING COSTS	OVER-ALL Bench Cost			$1719	$3765	$3839	$3185	$4371	$2707	$3573					$23159
	OVER-ALL OS Cost			$3173	$2472	$2412	$2923	$1998	$3296	$2620	$5409	$5409	$5409	$1412	$36532
OWNER MANAGEMENT	Service Maintenance			$906	$1086	$1086	$1086	$1086	$1086	$1086	$1086	$1086	$1086	$283	$10964
	Mine Management			$954	$1143	$1143	$1143	$1143	$1143	$1143	$1143	$1143	$1143	$298	$11539
OTHER CAPITAL	SITE ESTABLISHMENT	$588		$88											
	VENT RAISES		$670	$885	$184										
	FIXED PLANT	$133	$255	$308											
	MOBILE EQUIPMENT	$204	$1189	$48	$54										
	CAPITAL DEVELOPMENT		$6086	$1153	$1462										$8700
	OTHER CAPITAL	$925	$2114	$1328	$238										$4605
	OPERATING EXPENDITURE		$1566	$9403	$9507	$11585	$10873	$9597	$8232	$8421	$7638	$7638	$7638	$1994	$94092
	ANNUAL TOTAL EXPENDITURE	$925	$9765	$11884	$11207	$11585	$10873	$9597	$8232	$8421	$7638	$7638	$7638	$1994	$107397

analysis and reporting a further 20 days. Core strength tests and allowance for some stress measurement may require up to $30,000. **The total estimated cost is $84,000 for a feasibility level Geotechnical Study.**

Mine Study

The mine feasibility study is estimated to take about 60 man-days. **The estimated total cost for a Mine Feasibility Study is $82,000** including some allowance for travel and research costs.

13. EXPENDITURE TIMETABLE

A set of detailed, mine related, cash flow calculations are included in Appendix H. Monthly data is supplied for the first six operational years and yearly totals are provided for years seven to twelve. A summary is attached as Table 13.

The following notes apply to Appendix H and Table 13:

- Calculation of stope production cost has been simplified by the use of the overall cost per tonne ($23.10 Bench and $18.03 Open Stope) rather than the individual activity costs (shown in Table 7.3c). This leads to a slight error in the totals (less than 1%).
- This method of calculation also leads to lower predicted operational costs in the early years and higher operational costs in the last six years. The averaging of the higher sill related costs over the total production causes the effect. More accurate calculation is not justified at this stage.
- A "Year 0" is shown for that expenditure which will occur before the decline development starts. The main items being – light vehicles, air compressor, establishment of the portal and contractor mobilisation.

The cash flows have been broken down into Capital and Operating to facilitate future financial modelling.

Andrew J Bruce

For and on behalf of:
A J Bruce Consulting Pty Ltd

14. REFERENCES

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15. Terms of Agreement for Professional Services

1.0 INTRODUCTION

These Terms of Agreement apply to work carried out by the Company and are deemed to incorporate the Guideline Terms of Agreement for Professional Services (Oct 1983 and later amendments) prepared and published by the Association of Consulting Engineers, Australia. In the event of any inconsistency between these Terms of Agreement and the Guideline Terms of Agreement published by the ACEA, these Terms shall prevail.

The 'Company' shall mean A J Bruce Consulting Pty Ltd.

The 'Client' shall mean the person or persons named in the Schedule hereto.

'Subcontractor' shall mean and include:
(a) any person, firm or company with whom the Company may arrange for any work to be performed for the Client; and

(b) any person who is now or hereafter a servant, agent or subcontractor of any of the persons referred to in (a) above.

1.1 SCOPE OF SERVICES

1.2 The Company and the Client hereby agree that the Company shall carry out the Services stated or referred to in the Schedule hereto for the benefit of the Client for the Price stated or referred to in the Schedule hereto and otherwise in accordance with these Terms of Agreement and the further documentation (if any) referred to in the Schedule hereto.

2.0 SCOPE OF RESPONSIBILITY

2.1 All goods or services provided by the Company are subject only to these Terms. Except where necessarily incorporated by statute, no other term, condition or warranty shall be incorporated herein unless specifically agreed in writing by the Company.

2.2 The Company undertakes to exercise reasonable care in accordance with standards ordinarily exercised by members of the profession generally who practice in the same locality and under similar conditions. There shall be no liability whatsoever in respect of any failure to exercise a degree or level of care beyond such reasonable care.

2.3 The Client will indemnify and hold the Company harmless against all claims, costs and demands (including in respect of negligence) by third parties in respect of the service concerned.

2.4 The Client acknowledges that in the event of the Company having failed to exercise reasonable care, the Company's liability, if any, shall be absolutely extinguished after the expiration of twelve months from the date of the Company's invoice in respect of the service.

2.5 In the event that the Company is liable to the Client for failure to exercise reasonable care, such liability shall be limited to the greater of:

(a) the amount of the fee actually paid by the Client to the Company for the service concerned (to a maximum of $25,000); or
(b) any other amount agreed in writing between the Client and the Company.

2.6 Despite any other Clause, where section 74 of the Trade Practices Act applies in respect of the service concerned, the Company's liability, if any, is limited to:

(a) the supplying of the service again; or
(b) the payment of the cost of having the service supplied again.

2.7 Under no circumstances, whether as a result of any act, neglect or default or otherwise howsoever, shall the Company have any liability for or in relation to any work, reports, information, plans, designs or specifications supplied or performed by any third party, including a third party engaged by or at the suggestion of the Company.

2.8 The Client acknowledges that:

(a) the Company does not supervise the work of others in the case of work undertaken by others pursuant to directions from the Company, the Company is responsible for exercising reasonable care in giving the directions, but is not responsible for the manner in which the directions are carried out;

(b) the copyright in all documents prepared by or at the request or direction of the Company shall be and remain the property of the Company;

(c) it will not supply or permit any report of the Company to be seen by or given to any third party without the express written consent of the Company unless that report has incorporated in or attached to it these Terms;

2.9 For the purposes of this Clause and without limiting Clause 2.8 the Client acknowledges that the Company is contracting as principal for and on behalf of its servants, agents and Subcontractors. The servants, agents and Subcontractors of the Company shall not be liable in respect of any loss or damage suffered by the Client as a result of goods or services provided by the Company, nor for any consequential loss or damage suffered by any third party. The loss or damage referred to shall mean and include loss or damage caused by the negligence or willful act or default of the Company, its servants, agents and Subcontractors or others whether or not such loss or damage is foreseeable or contemplated by the Company.

2.10 With the approval of the Client, the Company may use information regarding the project that is the subject of this Agreement in any promotional material it may prepare or use.

3.0 GENERAL

3.1 Disputes In the event that a dispute should arise relating to the performance of the services to be provided under this Agreement and should that dispute result in litigation it is agreed where the Company prevails that it shall be entitled to recover all reasonable costs incurred in the defence of the claim, including staff time, court costs, legal fees assessed on a solicitor and client basis and other claim-related expenses.

3.2 Termination and Suspension This Agreement may be terminated or suspended by either party as follows:

(a) The Agreement may only be terminated for a fundamental breach of the Agreement which shall include without limiting the same a failure to pay the Company the Price, or any part of the Price agreed to be paid by installments on performance of parts of the Services provided by the Company, within thirty (30) days of an invoice being posted or faxed by the Company to the Client;

(b) The party that wishes to terminate the Agreement for a fundamental breach shall give a written notice to the other party specifying the alleged breach or breaches and requiring that the breach of breaches be remedied within a specified period of time, which shall be a reasonable time in the circumstances and in no event shall be less than seven (7) days, and giving notice that the failure to remedy the breach or breaches within the time specified will result, without further notice, in the termination of this Agreement;

(c) At the expiration of the time stated in the notice from the day that the notice in (b) is given to the other party, subject to (d) hereof, without further notice this Agreement shall be terminated;

(d) Such termination shall not be effective if the breach or breaches stated in the notice in (b) has been remedied before expiration of the period specified in the written notice.

(e) In the event of termination the Company shall be paid for services performed to the termination notice date plus reasonable termination expenses.

(f) The Agreement may be suspended for a period not exceeding twelve months by the giving of a notice in writing by the party seeking to suspend this Agreement to the other party which notice shall state the Agreement is suspended and the period of suspension and the reasons for the suspension.

(g) The Agreement shall not be suspended and no notice shall be served purporting to suspend the Agreement on the ground of any alleged breach of the Agreement by either party but shall be confined to a suspension on the grounds of commercial considerations.

(h) Within seven (7) days of the receipt of the notice of suspension the party receiving the notice shall notify in writing the party giving the notice whether it accepts the suspension or not and

(i) in the event that the suspension is accepted then this Agreement and all obligations hereunder save as expressly provided otherwise and save for any obligations which have arisen from the performance of this Agreement as at the date of the acceptance of the suspension shall be suspended but this Agreement shall remain valid and binding;
(ii) in the event that the suspension is not accepted then the obligations under this Agreement shall continue as if the notice of suspension had not been given;

(f) In the event of termination, or suspension for more than three (3) months prior to completion of all reports contemplated by this Agreement, the Company may complete such analyses and records as are necessary to complete their files and may also complete a report on the Services performed to the date of notice of termination or suspension.

(g) The expenses of termination or suspension shall include all fees and costs of the Company in completing such analyses, records and reports.

(h) Notwithstanding anything herein contained, if the Company is prevented by circumstances beyond its direct control from performing the Services the Agreement shall end without prejudice to the accrued rights of either party.

3.3 Death or Insolvency The Agreement shall either survive the death or insolvency of the Client and his, her or its rights and obligations shall pass to his, her or its successors, receiver, administrator or liquidator or the Company may elect to immediately terminate the services and all fees and costs to the date of termination shall be due and payable as a secured creditor against the tangible assets of the Client.

4.0 SPECIAL CONDITIONS
None.

SCHEDULE

COMPANY:
A J Bruce Consulting Pty Ltd (ACN 085 961 977)

CLIENT:
Allegiance Mining N.L. (ACN 059 676 783)

SERVICES:
As per the proposal to conduct a Pre-feasibility Mining Study of the Avebury Prospect to Mr. Lindsay Newnham, dated 21 July 2000.

PRICE:
As per the proposal to conduct a Pre-feasibility Mining Study of the Avebury Prospect to Mr. Lindsay Newnham, dated 21 July 2000 and our standard Terms of Agreement for Professional Services.

FURTHER DOCUMENTATION:
Our letters to Mr. L Newnham dated 3 October 2000, 27 October 2000 and 18 December 2000.

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16. APPENDICIES

16.1 Appendix A: Geology Data received as an e-mail document

661691

11.0	1.0	354410.0	5357219.0	2050.0
11.0	2.0	354455.0	5357225.0	2050.0
11.0	3.0	354500.0	5357252.0	2050.0
11.0	4.0	354560.0	5357263.0	2050.0
11.0	5.0	354560.0	5357224.0	2050.0
11.0	6.0	354500.0	5357198.0	2050.0
11.0	7.0	354455.0	5357195.0	2050.0
11.0	8.0	354410.0	5357188.0	2050.0
3.0	1.0	354477.5	5357164.0	2050.0
3.0	2.0	354503.22	5357164.0	2050.0
3.0	3.0	354529.56	5357165.0	2050.0
3.0	4.0	354560.0	5357166.0	2050.0
3.0	5.0	354560.0	5357123.0	2050.0
3.0	6.0	354500.0	5357117.0	2050.0
3.0	7.0	354477.5	5357120.0	2050.0
11.0	1.0	354410.0	5357271.5	2000.0
11.0	2.0	354451.88	5357277.0	2000.0
11.0	3.0	354500.0	5357295.0	2000.0
11.0	4.0	354560.0	5357293.0	2000.0
11.0	5.0	354560.0	5357258.5	2000.0
11.0	6.0	354511.63	5357254.0	2000.0
11.0	7.0	354500.0	5357249.0	2000.0
11.0	8.0	354455.0	5357247.5	2000.0
11.0	9.0	354410.0	5357242.0	2000.0
3.0	1.0	354477.5	5357209.0	2000.0
3.0	2.0	354500.0	5357211.5	2000.0
3.0	3.0	354534.47	5357206.5	2000.0
3.0	4.0	354560.0	5357207.0	2000.0
3.0	5.0	354560.0	5357165.0	2000.0
3.0	6.0	354532.25	5357162.0	2000.0
3.0	7.0	354500.0	5357161.5	2000.0
3.0	8.0	354488.97	5357165.0	2000.0
3.0	9.0	354477.5	5357166.5	2000.0
11.0	1.0	354410.0	5357293.0	1950.0
11.0	2.0	354455.0	5357301.0	1950.0
11.0	3.0	354500.0	5357318.0	1950.0
11.0	4.0	354522.5	5357317.5	1950.0
11.0	5.0	354560.0	5357310.0	1950.0
11.0	6.0	354560.0	5357280.0	1950.0
11.0	7.0	354500.0	5357276.5	1950.0
11.0	8.0	354455.0	5357275.5	1950.0
11.0	9.0	354410.0	5357267.5	1950.0
3.0	1.0	354477.5	5357246.0	1950.0
3.0	2.0	354500.0	5357250.5	1950.0
3.0	3.0	354534.72	5357245.5	1950.0
3.0	4.0	354560.0	5357243.0	1950.0
3.0	5.0	354560.0	5357204.5	1950.0
3.0	6.0	354537.91	5357207.0	1950.0
3.0	7.0	354500.0	5357206.5	1950.0
3.0	8.0	354477.5	5357214.0	1950.0
11.0	1.0	354410.0	5357306.0	1920.0
11.0	2.0	354455.0	5357315.0	1920.0
11.0	3.0	354500.0	5357332.0	1920.0
11.0	4.0	354560.0	5357320.0	1920.0
11.0	5.0	354560.0	5357293.0	1920.0
11.0	6.0	354500.0	5357289.0	1920.0
11.0	7.0	354455.0	5357289.0	1920.0
11.0	8.0	354410.0	5357282.0	1920.0
3.0	1.0	354477.5	5357260.0	1920.0
3.0	2.0	354500.0	5357263.0	1920.0
3.0	3.0	354529.34	5357259.5	1921.0
3.0	4.0	354560.0	5357257.0	1920.0
3.0	5.0	354560.0	5357220.0	1920.0
3.0	6.0	354500.0	5357228.0	1920.0
3.0	7.0	354477.5	5357230.0	1920.0

APPENDIX B

16.2 Appendix B: Point Load Test Results

Date of Testing: 14-Sep-00
Piston Area, A: 3226 sq.mm
K factor K=14+0.175*D; Bieniawski (1975)

Hole Number	Rock Zone (see table below)	Hole Interval (m)	Rock Type	Vernier D2 mm	Vernier D1 mm	Core Diam, D (D2-D1)	PLT Reading P (psi)	PLT Reading P (kPa)	K factor	Is	Is(50)	UCS (MPa)	Test Comments
A018a	A	221.5	Serpentinite	74.4	26.9	47.50	139	958	22.3	1.4	1.3	30	
	A	221.5	Serpentinite	74.4	26.9	47.50	135	931	22.3	1.3	1.3	29	
A025	D	435.0	?	80.6	32.9	47.70	382	2634	22.3	3.7	3.7	82	
	D	435.0	?	80.6	32.9	47.70	317	2186	22.3	3.1	3.0	68	Partial break
A019	D	87.9	Hornfels	80.7	32.9	47.80	315	2172	22.4	3.1	3.0	67	
	D	90.3	Chert/Tourmaline	80.7	32.9	47.80	345	2379	22.4	3.4	3.3	74	
	D	84.8	Hornfels	80.7	32.9	47.80	327	2255	22.4	3.2	3.1	70	
	D	76.4	Hornfels	80.6	33.1	47.50	424	2923	22.3	4.2	4.1	91	Longitudinal break
	D	72.5	Hornfels	80.6	33.1	47.50	193	1331	22.3	1.9	1.9	41	
	D	66.2	Hornfels	80.5	33.2	47.30	353	2434	22.3	3.5	3.4	76	Diag break
	D	86.9	Hornfels	83.2	35.6	47.60	422	2910	22.3	4.1	4.1	90	
	D	131.9	Actinolite rock	83.2	35.6	47.60	457	3151	22.3	4.5	4.4	98	
	D	131.8	Actinolite rock	83.2	35.6	47.60	406	2799	22.3	4.0	3.9	87	
	D	129.0	Chert	83.1	35.5	47.60	578	3985	22.3	5.7	5.5	124	
	D	129.0	Chert	83.1	35.5	47.60	569	3923	22.3	5.6	5.5	122	
A023	B	423.9	Phlogophite/mica	83.2	35.7	47.50	158	1089	22.3	1.6	1.5	34	
	B	423.9	Phlogophite/mica	83.2	35.7	47.50	96	662	22.3	0.9	0.9	21	
A018a	A	228.3	Serpentinite/magnetite	83.2	35.9	47.30		0	22.3	0.0	0.0	0	Crap failure
	A	227.9	Sil rock-mt after serp	81.4	34.2	47.20	111	765	22.3	1.1	1.1	24	Surface break
	A	227.9	Sil rock-mt after serp	81.4	34.2	47.20	381	2627	22.3	3.8	3.7	83	Diagonal
	A	227.9	Sil rock-mt after serp	81.4	34.2	47.20	230	1586	22.3	2.3	2.2	50	
A025	A	534.0	Serp/Mag/Pe	83.2	35.8	47.40	104	717	22.3	1.0	1.0	22	
	A	534.0	Serp/Mag/Pe	83.2	35.8	47.40	150	1034	22.3	1.5	1.4	32	
	A	534.0	Serp/Mag/Pe	83.2	35.8	47.40	149	1027	22.3	1.5	1.4	32	
A014a	A	272.7	Sil rock (after serp) serp/pe	83.4	35.9	47.50	425	2930	22.3	4.2	4.1	91	Diagonal
	A	272.8	Sil rock (after serp) serp/pe	83.4	35.9	47.50	416	2868	22.3	4.1	4.0	89	
	A	268.2	Sil rock after serp	83.4	35.9	47.50	812	5599	22.3	8.0	7.8	175	No failure
	A	268.2	Sil rock after serp	83.2	35.8	47.40	562	3875	22.3	5.6	5.4	121	
	A	268.2	Sil rock after serp	83.2	35.8	47.40	651	4488	22.3	6.4	6.3	140	No failure
	C	253.8	actinolite school rock	83.3	35.7	47.60	327	2255	22.3	3.2	3.1	70	
	C	250.2	actinolite school rock (slightly weathered?)	83.3	35.7	47.60	180	1241	22.3	1.8	1.7	39	Diagonal
A018a	C	213.5	Actinolite/serp rock	83.1	35.9	47.20	117	807	22.3	1.2	1.1	25	
	C	212.1	phlogopite-qtz rock	83.1	35.7	47.40	176	1213	22.3	1.7	1.7	38	
	C	211.0	Chert	83.1	35.7	47.40	630	4344	22.3	6.2	6.1	136	Diagonal break
	A	224.9	Mt/serp rock	83	35.8	47.20	231	1593	22.3	2.3	2.2	50	
A019a	A	252.4	Serp/mt/pe	83	35.8	47.20	194	1338	22.3	1.9	1.9	42	

ROCK ZONE	DESCRIPTION	Average UCS (MPa)
A	Mineralisation	50
B	Central Avebury & South Avebury mineralisation	28
C	Immediate orebody[n] hangingwall	44
D	Hangingwall remote to orebody	86

[n] *The word orebody is used to describe a mining zone not a Resource status.*

APPENDIX C

16.3 Appendix C: Geotechnical Logging

A J Bruce Consulting Pty. Ltd.
A.C.N. 085 961 977
Telephone: (03) 63347080
Facsimile: (03) 63347079
E-mail: ajbruce@bigpond.com
P.O. Box 1170 Launceston Tas. 7250

	BENCHES			OPEN STOPES		
	Footwall	Backs	Hangingwall	Footwall	Backs	Hangingwall
Top RL	2050			2050		
Bottom RL	1800			1920		
Surface RL	2190			2190		
Average Depth	265			205		
σ_v	7.2			5.5		
σ_1	21.5			16.6		
Stress factors	0.6	1.3	0.6	0.5	1.5	0.5
σ Max	13	27	13	8	24	8
Average UCS	34	34	78	34	34	78
UCS/ σ max	2.6	1.3	6.1	4.1	1.4	9.4
Q'	3.0	3.0	6.0	3.0	3.0	4.0
A	0.3	0.1	0.6	0.4	0.1	0.9
B	0.3	1.0	0.3	0.3	1.0	0.3
C	4.0	2.0	6.0	6.0	2.0	4.0
N	1.1	0.6	6.5	1.9	0.6	4.3
Supported	Yes	Yes	Sill only	Yes	Yes	Sill only
>> S	4.5	3.0	5.0	7.0	4.0	5.0
Actual Stope Height/Width	22.5	10.0	18.0	20.5	7.0	20.5
Notes:	Floor to floor plus sill height	Desirable silling width after stripping	Excludes cabled sill hanging wall	Excludes cabled sill foot wall	Drive width only	Excludes cabled sill hanging wall
Stable Length*	15	15	23	44	Infinite	20
Typical RQD	85	85	90	80	80	85
Typical Jn	9	9	4	9	9	4
(RQD/Jn)/s	1.9	2.1		1.5	1.5	
Use Design Width/Height	22.5	10.0		20.0	26.0	
Use Design Length	18	100		30	22	
Actual S	5.0	4.5		6.0	6.0	
Entry	No	Yes		No	Remote loader	
Cable Density	0.1-0.15	0.25-0.35	NA	0.15-0.3	0.18-0.32	NA

*Note:
The stable stope size for benches was calculated as the average of the stable hangingwall and footwall lengths i.e. (15+23)/2 = 19 metres. Stope length used was 18 metres.
Maximum open stope dimension was calculated as the average of the stable hangingwall and footwall lengths i.e. (44+20)/2 = 32 metres. Max stope dimension used was 30 metres.

Q' , RQD and **Jn** were derived from the histogram calculations on the "Q Index Calcs_1" tables in this Appendix (Appendix C).

A J Bruce Consulting Pty. Ltd.
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SOUTH ORE SERPENTINITE Q, RQD Jn

Hole:	A008	rqd (m)	RQD	h/w at:		SL & fw No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
464.4	466.5	2.08	98	464.4	2.1	3	9	pr	1.5	s	4.0	1.1	4.1
466.5	469.5	2.35	78		3.0	4	15	ps	1.0	s	4.0	3.6	1.3
469.5	471.9	1.26	52		2.4	4	15	ps	1.0	s	4.0	4.2	0.9
471.9	474	1.90	90		2.1	3	9	ps	1.0	s	4.0	3.4	2.5
474	476.2	1.54	70		2.2	3	9	ps	1.0	s	4.0	3.2	1.9
476.2	484.4	2.07	25		8.2	2+	6	ps	1.0	s	4.0	6.3	1.0
478.3	481.8	2.97	84		3.5	3	9	ps	1.0	s	4.0	9.9	2.3
481.8	484.4	2.43	93		2.6	2+	6	ps	1.0	s	4.0	4.8	3.9
484.4	487.4	1.96	65		3.0	4	15	ps	1.0	s	4.0	4.1	1.1
487.4	488.8	0.94	67		1.4	3	9	ps	1.0	s	4.0	3.7	1.9
488.8	491.4	1.72	66		2.6	3+	12	ps	1.0	s	4.0	2.7	1.4
491.4	493.4	1.03	51		2.0	3	9	ps	1.0	s	4.0	3.6	1.4
493.4	495.9	1.91	76		2.5	3	9	ps	1.0	s	4.0	3.3	2.1
495.9	498.1	1.75	79		2.2	3	9	ps	1.0	s	4.0	3.6	2.2
498.1	499.7	1.60	100		1.6	1	2	pr	1.5	s	4.0	3.0	18.6

Hole:	A012	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
502.5	507.7	4.27	82	504.3	5.2	3	9	ps	1.0	s	4.0	2.6	2.3
507.7	509.2	0.44	29		1.5	3	9	ps	1.0	s	4.0	5.9	0.8
509.2	512	2.53	90		2.8	2	4	ps	1.0	s	4.0	2.9	5.6
512	515	2.74	91		3.0	2	4	ps	1.0	s	4.0	4.3	5.7
515	518.8	3.09	81		3.8	3	9	ps	1.0	s	4.0	4.9	2.3
518.8	520	0.91	75		1.2	3	9	ps	1.0	s	4.0	4.4	2.1
520	522	1.90	95		2.0	1	2	ps	1.0	s	4.0	2.2	11.9
522	524	2.00	100		2.0	1	2	ps	1.0	s	4.0	3.0	12.5
524	527	2.92	97		3.0	2	4	ps	1.0	s	4.0	3.5	6.1
527	530	2.73	91		3.0	3	9	ps	1.0	s	4.0	4.5	2.5
530	532.2	2.20	99		2.2	2	4	ps	1.0	s	4.0	4.1	6.2
532.2	534	1.80	100		1.8	1	2	ps	1.0	s	4.0	3.1	12.5
534	538.2	3.65	86		4.2	2	4	ps	1.0	s	4.0	3.9	5.4
538.2	541.5	3.17	96		3.3	1	2	ps	1.0	s	4.0	5.9	12.0
541.5	543.5	1.25	62		2.0	2	4	ps	1.0	s	4.0	4.3	3.9

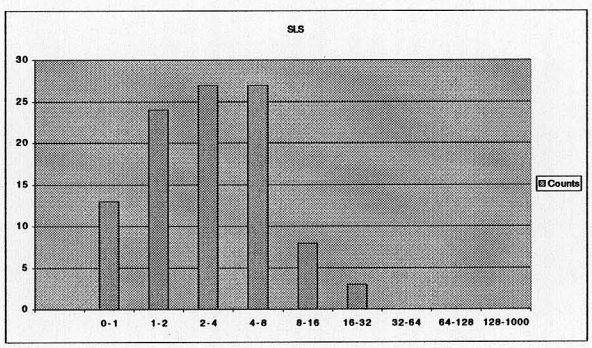
Hole:	A014	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
253.5	256.5	2.44	81	254.7	3.0	2	4	pr	1.5	s	4.0	1.5	7.8
256.5	259.5	2.26	75		3.0	2	4	ps	1.0	s	4.0	4.5	4.7
259.5	262.5	2.51	83		3.0	2	4	ps	1.0	s	4.0	4.5	5.2
262.5	265.5	2.64	88		3.0	1	2	ps	1.0	s	4.0	4.5	11.0
265.5	268.5	1.33	44		3.0	3	9	ps	1.0	s	4.0	4.5	1.2
268.5	270.2	0.15	10		1.7	4	15	ps	1.0	s	4.0	3.8	0.2
270.2	271.5	0.27	20		1.3	3	9	ps	1.0	s	4.0	2.3	0.6
271.5	271.9	0.17	42		0.4	3	9	ps	1.0	s	4.0	1.5	1.2
271.9	272.8	0.24	26		0.9	3	9	ps	1.0	s	4.0	0.8	0.7
272.8	274.5	1.20	70		1.7	3	9	ps	1.0	s	4.0	1.8	1.9
274.5	277	1.18	47		2.5	3	9	us	2.0	s	4.0	2.9	2.8
277	280.2	2.02	63		3.2	2	4	ps	1.0	s	4.0	4.1	3.9
280.2	283.2	2.00	66		3.0	3	9	ps	1.0	s	4.0	4.7	1.8
283.2	286.2	2.45	81		3.0	2	4	ps	1.0	s	4.0	7.7	5.1
286.2	286.6	0.25	62		0.4	1	2	ps	1.0	s	4.0	9.4	7.8

Hole:	A018	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
349	352	2.41	80	349.5	3.0	2	4	pr	1.5	s	4.0	1.5	7.5
352	355	2.54	84		3.0	2	4	pr	1.5	s	4.0	4.5	7.9
355	357.8	2.27	81		2.8	2	4	ps	1.0	s	4.0	4.4	5.1
357.8	360.9	0.46	14		3.1	3	9	ps	1.0	s	4.0	4.3	0.4
360.9	364	1.48	47		3.1	3	9	ps	1.0	s	4.0	7.4	1.3
364	367	3.00	100		3.0	3	9	ps	1.0	s	4.0	10.5	2.8
367	369.9	1.78	61		2.9	3	9	ps	1.0	s	4.0	13.5	1.7
369.9	373	2.28	73		3.1	2	4	ps	1.0	s	4.0	16.5	4.6
373	375.5	2.10	84		2.5	2	4	ps	1.0	s	4.0	19.3	5.3
375.5	378.5	2.22	74		3.0	2	4	ps	1.0	s	4.0	22.0	4.6
378.5	381.5	2.78	92		3.0	2	4	ps	1.0	s	4.0	25.0	5.8
381.5	384.6	2.94	94		3.1	3	9	ps	1.0	s	4.0	4.6	2.6
384.6	387.6	2.56	85		3.0	2	4	ps	1.0	s	4.0	4.6	5.3
387.6	390.7	2.94	94		3.1	2	4	ps	1.0	s	4.0	7.7	5.9
390.7	393.8	1.67	53		3.1	3	9	ps	1.0	s	4.0	10.8	1.5

Hole:	A026 SL	rqd (m)	RQD	"h/w" at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
289.8	296	2.59	41	293.8	6.2	3	9	ps	1.0	s	4.0	3.1	1.1
296	299	2.59	86		3.0	2	4	ps	1.0	s	4.0	7.7	5.4
299	302	2.54	84		3.0	2	4	pr	1.5	s	4.0	4.5	7.9
302	305	1.87	62		3.0	3	9	pr	1.5	s	4.0	4.5	2.6
305	308	2.94	98		3.0	2	4	pr	1.5	n	1.5	7.5	24.5
308	311	2.42	80		3.0	3	9	pr	1.5	s	4.0	10.5	3.3
311	314	1.69	56		3.0	3	9	pr	1.5	s	4.0	13.5	2.3
314	316.2	0.39	17		2.2	4+	15	pr	1.5	s	4.0	16.1	0.4
316.2	319.3	1.75	56		3.1	3	9	pr	1.5	s	4.0	18.8	2.3
319.3	322.4	2.47	79		3.1	3	9	pr	1.5	s	4.0	21.9	3.3
322.4	324.4	2.35	117		2.0	3	9	ps	1.0	s	4.0	24.4	3.3
324.4	325.5	0.86	78		1.1	2	4	pr	1.5	n	1.5	2.6	19.5
325.5	327.3	0.95	52		1.8	3	9	ps	1.0	s	4.0	2.0	1.4
327.3	330.3	2.30	76		3.0	3	9	ps	1.0	s	4.0	4.4	2.1
330.3	333.4	2.55	82		3.1	2	4	pr	1.5	s	4.0	7.5	7.7
333.4	336.5	3.00	96		3.1	2	4	pr	1.5	s	4.0	4.6	9.0
336.5	339.6	2.40	77		3.1	3	9	pr	1.5	s	4.0	7.8	3.2
339.6	342.7	2.67	92		3.1	2	4	ps	1.0	s	4.0	10.9	5.8
342.7	345.8	2.20	70		3.1	3	9	pr	1.5	s	4.0	14.0	2.9
345.8	348.9	2.41	77		3.1	2	4	pr	1.5	s	4.0	17.1	7.2
348.9	351.9	2.26	75		3.0	3	9	pr	1.5	n	1.5	20.1	8.3

Hole:	A026 NL	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
223.9	227	2.51	80	224.5	3.1	2	4	ps	1.0	s	4.0	1.6	5.0
227	230	1.44	48		3.0	4	15	ps	1.0	s	4.0	4.6	0.8
230	233.2	2.15	67		3.2	4	15	ps	1.0	s	4.0	4.6	1.1
233.2	235.7	2.03	81		2.5	3	9	ps	1.0	s	4.0	4.4	2.3
235.7	238.8	3.01	97		3.1	3	9	ps	1.0	s	4.0	7.3	2.7
238.8	241.5	1.70	62		2.7	3	9	ps	1.0	s	4.0	10.2	1.7
241.5	245.4	0.96	24		3.9	4	15	ps	1.0	s	4.0	13.5	0.4
245.4	247.5	0.10	10		2.1	3	9	ps	1.0	s	4.0	16.5	0.3
247.5	251	1.69	48		3.5	3	9	ps	1.0	s	4.0	19.3	1.3
251	253.5	0.90	36		2.5	3	9	ps	1.0	s	4.0	22.3	1.0
253.5	256.2	0.51	18		2.7	4+	15	ps	1.0	s	4.0	24.9	0.3
256.2	258.6	1.48	61		2.4	3	9	ps	1.0	s	4.0	3.9	1.7
258.6	261.2	2.45	94		2.6	3	9	pr	1.5	n	1.5	3.7	10.4
261.2	263	0.36	19		1.8	3	9	ps	1.0	s	4.0	5.9	0.5
263	265.3	0.96	41		2.3	3	9	ps	1.0	s	4.0	8.0	1.1
265.3	268.2	1.58	54		2.9	2	4	ps	1.0	s	4.0	3.8	3.4
268.2	271.3	1.01	32		3.1	4	9	pr	1.5	s	4.0	6.8	1.3
271.3	274	1.66	61		2.7	2	4	pr	1.5	s	4.0	9.7	5.7
274	274.6	0.24	39		0.6	3	9	pr	1.5	s	4.0	11.3	1.6
274.6	277.2	1.24	47		2.6	3	9	pr	1.5	s	4.0	12.9	2.0
277.2	279.7	1.93	77		2.5	3	9	ps	1.0	s	4.0	15.5	2.1

Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
1.0	0-1	13	13
2.0	1-2	24	37
4.0	2-4	27	64
8.0	4-8	27	91
16.0	8-16	8	99
32.0	16-32	3	102
64.0	32-64	0	102
128.0	64-128	0	102
1000.0	128-1000	0	102



Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
10.0	0-10	2	2
20.0	10-20	5	7
30.0	20-30	4	11
40.0	30-40	3	14
50.0	40-50	9	23
60.0	50-60	7	30
70.0	60-70	16	46
80.0	70-80	17	63
90.0	80-90	18	81
100.0	90-100	20	101

Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
2.0	0-10	7	7
4.0	10-20	31	38
9.0	20-30	54	92
15.0	30-40	10	102
50.0	40-50	0	102

SOUTH LODE HANGINGWALL

Q Index Calcs_1

Hole:	A008	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
445.3	448.5	1.34	41		3.2	3	9	ps	1.0	s	4.0	17.5	1.1
448.5	449.8	1.3	99		1.3	1	2	ur	3.0	n	1.5	15.3	99.0
449.8	452.2	2.15	89		2.4	1	2	ur	3.0	n	1.5	13.4	89.0
452.2	454.5	2.26	98		2.3	1	2	ur	3.0	n	1.5	11.1	98.0
454.5	457.5	2.8	93		3.0	1	2	ur	3.0	n	1.5	8.4	93.0
457.5	460.5	3	100		3.0	2	4	pr	1.5	n	1.5	5.4	25.0
460.5	463.5	3	100		3.0	1	2	pr	1.5	s	4.0	2.4	18.8
463.5	464.4	0.9	100		0.9	0	1	up	4.0	n	1.5	0.4	266.7
464.4	466.5	2.06	98	464.4	0.0	3	9	pr	1.5	s	4.0	0.0	4.1

Hole:	A012	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
479.1	482.7	3.25	90		3.6	2	4	ps	1.0	s	4.0	23.4	5.6
482.7	485.0	1.74	75		2.3	3	9	ps	1.0	s	4.0	20.5	2.1
485.0	488.0	3.00	100		3.0	2	4	ps	1.0	s	4.0	17.8	6.3
488.0	491.0	3.00	100		3.0	2	4	ps	1.0	s	4.0	14.8	6.3
491.0	494.0	2.86	95		3.0	2	4	ps	1.0	s	4.0	11.8	5.9
494.0	497.0	3.00	100		3.0	1	2	ps	1.0	s	4.0	8.8	12.5
497.0	499.5	1.81	72		2.5	3	9	ps	1.0	s	4.0	6.1	2.0
499.5	502.5	2.67	89		3.0	3+?	12	ps	1.0	s	4.0	3.3	1.9
502.5	505.5	2.81	93	504.3	1.8	4	15	ps	1.0	s	4.0	0.9	1.6

Hole:	A014	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
229.5	232.5	3.00	100		3.0	2	4	us	2.0	s	4.0	23.7	12.5
232.5	235.5	3.00	100		3.0	2	4	pr	1.5	s	4.0	20.7	9.4
235.5	238.5	2.80	93		3.0	3	9	ur	3.0	s	4.0	17.7	7.8
238.5	241.5	2.32	77		3.0	3	9	pr	1.5	s	4.0	14.7	3.2
241.5	244.5	1.83	61		3.0	3	9	pr	1.5	s	4.0	11.7	2.5
244.5	247.5	2.85	95		3.0	3	9	pr	1.5	s	4.0	8.7	4.0
247.5	250.5	2.59	86		3.0	3	9	pr	1.5	s	4.0	5.7	3.6
250.5	253.5	1.94	64		3.0	3	9	pr	1.5	s	4.0	2.7	2.7
253.5	256.5	2.44	81	254.7	1.2	2	4	pr	1.5	s	4.0	0.6	7.6

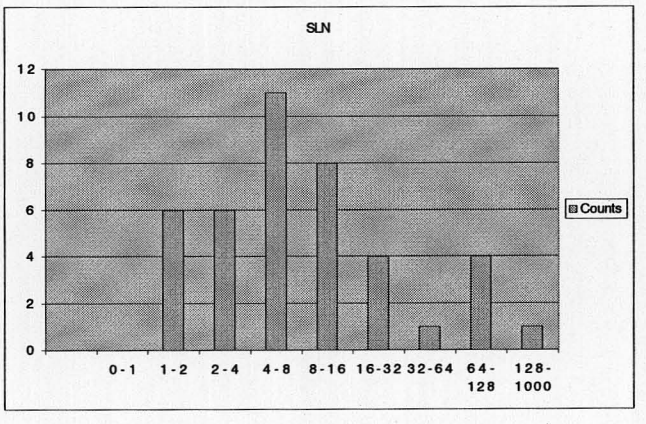
Hole:	A018	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
328.0	331.0	2.12	70		3.0	Rubble	15	ps	1.0	s	4.0	20.0	1.2
331.0	334.0	1.80	60		3.0	3	9	ps	1.0	s	4.0	17.0	1.7
334.0	336.9	2.67	92		2.9	2	4	pr	1.5	s	4.0	14.1	8.6
336.9	339.2	1.53	66		2.3	2	4	pr	1.5	s	4.0	11.5	6.2
339.2	342.2	2.51	83		3.0	2	4	us	2.0	s	4.0	8.8	10.4
342.2	344.3	1.69	80		2.1	2	4	pr	1.5	s	4.0	6.3	7.5
344.3	346.0	1.49	87		1.7	2	4	pr	1.5	s	4.0	4.3	8.2
346.0	349.0	2.71	90		3.0	1	2	pr	1.5	s	4.0	2.0	18.9
349.0	352.0	2.41	80	349.5	0.5	2	4	pr	1.5	s	4.0	0.3	7.5

Hole:	A026	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
No info													
No info													
No info													
No info													
211.4	214.6	1.85	57		3.2	2	4	ps	1.0	n	1.5	11.5	9.5
214.6	217.7	3.00	96		3.1	1	2	ps	1.0	n	1.5	8.4	32.0
217.7	220.8	3.10	99		3.1	1	2	pr	1.5	n	1.5	5.3	49.5
220.8	223.9	2.77	89		3.1	3	9	pr	1.5	n	1.5	2.1	9.9
223.9	227.0	2.51	80	224.5	0.6	2	4	ps	1.0	s	4.0	0.3	5.0

Q Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
1.0	0-1	0	0
2.0	1-2	6	6
4.0	2-4	6	12
8.0	4-8	11	23
16.0	8-16	8	31
32.0	16-32	4	35
64.0	32-64	1	36
128.0	64-128	4	40
1000.0	128-1000	1	41

RQD Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
10.0	0-10	0	0
20.0	10-20	0	0
30.0	20-30	0	0
40.0	30-40	0	0
50.0	40-50	1	1
60.0	50-60	2	3
70.0	60-70	4	7
80.0	70-80	6	13
90.0	80-90	9	22
100.0	90-100	19	41

Jn Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
2.0	0-10	10	10
4.0	10-20	16	26
9.0	20-30	12	38
15.0	30-40	3	41
50.0	40-50	0	41



NORTH ORE SERPENTINITE: Q, RQD, Jn

NL & fw

Hole:	A008	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
355.3	358.3	1.05	34	357.3	3.0	3	9	ps	1.0	s	4.0	1.5	0.9
358.3	359.7	0.66	47		1.4	2+	6	ur	3.0	n	1.5	3.7	15.7
359.7	362.9	1.70	53		3.2	3+	12	ur	3.0	s	4.0	6.0	3.3
362.9	364.3	0.83	59		1.4	3+	12	ur	3.0	s	4.0	8.3	3.7
364.3	367.5	3.20	100		3.2	1	2	pr	1.5	s	4.0	10.6	18.8
367.5	370.5	3.00	100		3.0	1+	3	pr	1.5	s	4.0	13.7	12.5
370.5	373.7	1.98	61		3.2	2+	6	pr	1.5	s	4.0	16.8	3.8
373.7	376.5	2.50	89		2.8	2	4	pr	1.5	s	4.0	19.8	8.3
376.5	379.5	2.97	99		3.0	1	1	pr	1.5	s	4.0	22.7	37.1
379.5	382.1	1.75	67		2.6	3	9	pr	1.5	s	4.0	25.5	2.8
382.1	385.5	2.34	68		3.4	3	9	pr	1.5	s	4.0	28.5	2.8
385.5	388.3	2.05	73		2.8	3	9	pr	1.5	s	4.0	31.6	3.0
388.3	391.3	2.94	98		3.0	3	9	ps	1.0	s	4.0	34.5	2.7
391.3	394.3	2.38	79		3.0	3+	12	ps	1.0	s	4.0	37.5	1.6
394.3	397.5	2.06	64		3.2	3	9	ps	1.0	s	4.0	40.6	1.8

Hole:	A010		RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
205.8	208.6	2.74	97	208.4	2.8	2	4	ps	1.0	s	4.0	1.4	6.1
202.7	205.8	3.06	98		3.1	2	4	ps	1.0	s	4.0	4.3	6.1
199.8	202.7	2.05	70		2.9	3	9	ps	1.0	s	4.0	7.3	1.9
196.8	199.8	3.00	100		3.0	0	1	up	4.0	s	4.0	10.3	100.0
193.8	196.8	2.81	93		3.0	1+	3	ps	1.0	s	4.0	13.3	7.6
190.8	193.8	1.84	61		3.0	1+	3	ps	1.0	s	4.0	16.3	5.1
188.2	190.8	1.43	54		2.6	1+	3	ps	1.0	s	4.0	19.1	4.5
187	188.2	0.20	16		1.2	3	9	ps	1.0	s	4.0	21.0	9.4
184.3	187	0.27	10		2.7	3	9	ps	1.0	s	4.0	23.0	9.3
181.8	184.3	1.77	70		2.5	3	9	ps	1.0	s	4.0	25.8	1.9
178.8	181.8	2.75	91		3.0	3	9	ps	1.0	s	4.0	28.3	2.5
175.8	178.8	1.22	40		3.0	3	9	ps	1.0	s	4.0	31.3	1.1
172.8	175.8	2.46	82		3.0	3	9	ps	1.0	s	4.0	34.3	2.3
169.8	172.8	3.00	100		3.0	2	2	ps	1.0	s	4.0	37.3	12.5
166.8	169.8	2.70	90		3.0	2	2	ps	1.0	s	4.0	40.3	11.3

Hole:	A012		RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
390.3	392	1.03	60	391.9	1.7	3	9	us	2.0	s	4.0	0.8	3.3
392	395	2.74	91		3.0	3	9	us	2.0	s	4.0	3.2	5.1
395	397.2	1.36	61		2.2	3	9	us	2.0	s	4.0	5.8	5.4
397.2	400.5	1.92	58		3.3	3	9	ps	1.0	s	4.0	8.5	1.8
400.5	402.5	0.76	38		2.0	3	9	ps	1.0	s	4.0	11.2	1.1
402.5	404.2	0.99	58		1.7	2	4	ps	1.0	s	4.0	13.1	3.6
404.2	407	1.55	55		2.8	2	4	ps	1.0	s	4.0	15.3	3.4
407	408.9	0.68	35		1.9	2	4	ps	1.0	s	4.0	17.7	2.2
408.9	410	0.25	22		1.1	3	9	ps	1.0	s	4.0	19.2	0.6
410	411.5	1.29	86		1.5	2	4	ps	1.0	s	4.0	20.5	5.4
411.5	413.6	0.42	19		2.1	3	9	ps	1.0	s	4.0	22.3	0.5
413.6	415.3	0.93	54		1.7	3	9	ps	1.0	s	4.0	24.2	1.5
415.3	418.7	2.56	75		3.4	2	4	ps	1.0	s	4.0	26.7	4.7
418.7	421.8	2.38	76		3.1	3	9	ps	1.0	s	4.0	30.0	2.1
421.8	424.8	2.93	97		3.0	2	4	ps	1.0	s	4.0	33.0	6.1

Hole:	A013		RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
389	392	2.84	94	391.9	3.0	2	4	ur	3.0	s	4.0	1.5	17.8
392	395	1.80	60		3.0	3	9	ps	1.0	s	4.0	4.5	1.7
395	398	2.29	76		3.0	4	15	ur	3.0	s	4.0	7.5	3.8
398	401	1.49	49		3.0	3	9	us	2.0	s	4.0	10.5	2.7
401	403.8	0.91	32		2.8	3	9	us	2.0	s	4.0	13.4	1.8
403.8	406.6	1.55	55		2.8	3	9	ps	1.0	s	4.0	16.2	1.5
406.6	409.6	2.53	84		3.0	1	2	ps	1.0	s	4.0	19.1	10.5
409.6	412.7	2.30	74		3.1	2	4	ps	1.0	s	4.0	22.2	4.6
412.7	415.8	2.64	85		3.1	2	4	ps	1.0	s	4.0	25.3	5.3
415.8	418.9	2.30	74		3.1	2	4	ps	1.0	s	4.0	28.4	4.6
418.9	421.9	2.66	88		3.0	2	4	ps	1.0	s	4.0	31.4	5.5
421.9	424.9	2.83	94		3.0	2	4	ps	1.0	s	4.0	34.4	5.9
424.9	428	3.03	97		3.1	1	2	ps	1.0	s	4.0	37.5	12.1
428	431	2.46	82		3.0	1	2	ps	1.0	s	4.0	40.5	10.3
431	433.5	2.49	99		2.5	1	2	ps	1.0	s	4.0	43.3	12.4

Hole:	A018		RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
214	217	2.26	75	214.4	3.0	2	4	ps	1.0	s	4.0	1.5	4.7
217	219.7	1.70	62		2.7	3	9	ps	1.0	s	4.0	4.3	1.7
219.7	221.7	0.49	24		2.0	3	9	ps	1.0	s	4.0	6.7	0.7
221.7	224.8	2.47	79		3.1	2	2	ps	1.0	s	4.0	9.3	9.9
224.8	227.9	1.84	59		3.1	3	9	ps	1.0	s	4.0	12.4	1.8
227.9	229	0.29	26		1.1	4+	15	ps	1.0	s	4.0	14.5	0.4
229	232	1.67	55		3.0	4+	15	ps	1.0	s	4.0	16.5	0.9
232	235	2.91	97		3.0	1	2	ps	1.0	s	4.0	19.5	12.1
235	238	2.98	99		3.0	1	2	ps	1.0	s	4.0	22.5	12.4
238	241	3.00	100		3.0	2	4	ps	1.0	s	4.0	25.5	6.3
241	244	3.00	100		3.0	2	4	ps	1.0	s	4.0	28.5	6.3
244	246.9	2.96	102		2.9	2	4	ps	1.0	s	4.0	31.5	6.4
246.9	249.9	2.47	82		3.0	3	9	ps	1.0	s	4.0	34.4	2.5
249.9	253	2.84	91		3.1	2	4	ps	1.0	s	4.0	37.5	5.7
253	256	2.79	93		3.0	2	4	ps	1.0	s	4.0	40.5	5.8

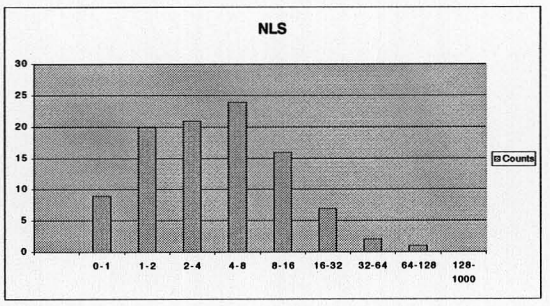
Hole:	A019		RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
241.4	244.2	2.32	82	241.8	2.4	2	4	pr	1.5	s	4.0	1.2	7.7
244.2	247.3	3.09	99		3.1	2	4	pr	1.5	n	1.5	3.9	24.8
247.3	250.4	2.88	92		3.1	1	2	pr	1.5	n	1.5	7.1	46.0
250.4	252.2	1.38	75		1.8	2	4	pr	1.5	n	1.5	9.5	18.8
252.2	254	1.56	86		1.8	2	4	pr	1.5	s	4.0	11.3	8.1
254	257	2.38	79		3.0	3	9	pr	1.5	n	1.5	13.7	6.8
257	259.7	1.86	68		2.7	3	9	pr	1.5	n	1.5	16.6	7.8
259.7	261.7	1.68	84		2.0	2	4	pr	1.5	n	1.5	18.9	21.0
261.7	263.2	1.19	79		1.5	3	9	pr	1.5	s	4.0	20.7	3.3
263.2	266	1.63	58		2.8	3	9	pr	1.5	s	4.0	22.8	2.4
266	269	2.33	77		3.0	4	15	pr	1.5	s	4.0	25.7	1.9
269	272.4	1.96	57		3.4	4	15	pr	1.5	s	4.0	28.9	1.3
272.4	274.9	1.21	48		2.5	4	15	pr	1.5	s	4.0	31.9	1.2
274.9	278	1.44	46		3.1	4	15	pr	1.5	s	4.0	34.7	1.2
278	280.3	0.44	19		2.3	4+	15	ps	1.0	s	4.0	37.4	0.3

Hole:	A020		RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid distance to F/W	Q1
224	227	1.80	60	225.8	1.2	4	15	us	2.0	s	4.0	0.6	2.0
227	229.5	1.81	72		2.5	4	15	pr	1.5	n	1.5	2.4	4.8
229.5	232.6	2.81	90		3.1	2	4	pr	1.5	n	1.5	5.2	22.5
232.6	235.6	2.70	90		3.0	3	9	pr	1.5	n	1.5	8.3	10.0
235.6	238.7	2.71	87		3.1	3	9	us	2.0	s	4.0	11.4	4.8
238.7	241.8	2.32	74		3.1	3	9	pr	1.5	n	1.5	14.5	8.2
241.8	243.9	1.46	69		2.1	3	9	pr	1.5	s	4.0	17.1	2.9
243.9	245	0.97	88		1.1	2	4	pr	1.5	n	1.5	18.7	22.0
245	249.4	3.53	80		4.4	4	15	pr	1.5	s	4.0	21.4	2.0
249.4	251	0.95	59		1.6	3	9	ps	1.0	s	4.0	24.4	1.6

Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
1.0	0-1	9	9
2.0	1-2	20	29
4.0	2-4	21	50
8.0	4-8	24	74
16.0	8-16	16	90
32.0	16-32	7	97
64.0	32-64	2	99
128.0	64-128	1	100
1000.0	128-1000	0	100

Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
10.0	0-10	1	1
20.0	10-20	3	4
30.0	20-30	3	7
40.0	30-40	5	12
50.0	40-50	4	16
60.0	50-60	16	32
70.0	60-70	11	43
80.0	70-80	16	59
90.0	80-90	16	75
100.0	90-100	24	99

Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
2.0	0-10	13	13
4.0	10-20	32	45
9.0	20-30	41	86
15.0	30-40	14	100
50.0	40-50	0	100



Hole:	A008	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
331.5	334.5	2.86	95		3	2	4	ps	1.0	s	4.0	24.3	6
334.5	337.5	3	100		3	2	4	ps	1.0	n	1.5	21.3	17
337.5	340.5	3	100		3	1	2	ps	1.0	n	1.5	18.3	33
340.5	342.8	2.17	94		2.3	2	4	ps	1.0	n	1.5	15.65	16
342.8	345.7	2.59	89		2.9	3	9	ps	1.0	n	1.5	13.05	7
345.7	348.6	2.53	87		2.9	3	9	ps	1.0	n	1.5	10.15	6
348.6	352.5	3.71	95		3.9	2	4	pr	1.5	n	1.5	6.75	24
352.5	355.3	1.4	49		2.8	3	9	ur	3.0	s	4.0	3.4	4
355.3	358.3	1.03	34	357.3	2	3	9	ps	1.0	s	4.0	1	1

Hole:	A010	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
229.8	232.8	2.89	96		3	1+	3	ps	1.0	s	4.0	25.3	8
226.8	229.8	2.91	97		3	3	9	ps	1.0	s	4.0	22.3	3
223.8	226.8	2.36	78		3	3	9	ps	1.0	s	4.0	19.3	2
220.7	223.8	2.18	70		3.1	4	15	ps	1.0	s	4.0	16.25	1
217.8	220.7	2.74	94		2.9	1+	3	ps	1.0	s	4.0	13.25	8
214.8	217.8	2.86	95		3	1	2	ps	1.0	s	4.0	10.3	12
211.8	214.8	2.97	99		3	2	4	ps	1.0	s	4.0	7.3	6
208.6	211.8	2.70	84		3.2	2	4	ps	1.0	s	4.0	4.2	5
205.8	208.6	2.74	97	208.4	2.8	2	4	ps	1.0	s	4.0	1.3	6

Hole drilled south to north

Hole:	A012	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
365.7	368	1.64	71		2.3	2	4	ps	1.0	s	4.0	25.05	4
368	371	2.46	82		3	2	4	ps	1.0	s	4.0	22.4	5
371	374.2	3.08	96		3.2	3	9	ps	1.0	s	4.0	19.3	3
374.2	378.8	4.08	88		4.6	2	4	ps	1.0	s	4.0	15.4	6
378.8	380.6	1.8	99		1.8	2	4	ps	1.0	s	4.0	12.2	6
380.6	384.5	3.67	94		3.9	2	4	ps	1.0	s	4.0	9.35	6
384.5	387.6	3.03	97		3.1	2	4	ps	1.0	s	4.0	5.85	6
387.6	390.3	1.44	53		2.7	2	4	ps	1.0	s	4.0	2.95	3
390.3	392	1.03	60	391.9	1.6	3	9	us	2.0	s	4.0	0.8	3

Hole:	A013	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
365.0	368.0	3.00	100		3	1	2	pr	1.5	n	1.5	25.4	50
368.0	371.0	2.91	97		3	2	4	pr	1.5	s	4.0	22.4	9
371.0	374.0	3.00	100		3	2	4	pr	1.5	n	1.5	19.4	25
374.0	377.0	3.00	100		3	2	4	pr	1.5	n	1.5	16.4	25
377.0	380.0	3.00	100		3	2	4	pr	1.5	n	1.5	13.4	25
380.0	383.0	2.83	94		3	1	2	pr	1.5	s	4.0	10.4	16
383.0	386.0	2.71	90		3	2	4	pr	1.5	s	4.0	7.4	8
386.0	389.0	2.92	97		3	2	4	pr	1.5	s	4.0	4.4	9
389.0	392.0	2.84	94	391.9	2.9	2	4	ur	3.0	s	4.0	1.45	18

Hole:	A015	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
395.0	398.0	2.43	81		3	3	9	ps	1.0	s	4.0	24.4	2
398.0	401.0	3.00	100		3	2	4	pr	1.5	s	4.0	21.4	9
401.0	404.0	2.88	96		3	2	4	ps	1.0	s	4.0	18.4	6
404.0	407.0	2.64	88		3	2	4	ps	1.0	s	4.0	15.4	6
407.0	410.0	2.92	97		3	2	4	ps	1.0	s	4.0	12.4	6
410.0	413.0	2.87	95		3	3	9	ps	1.0	s	4.0	9.4	3
413.0	416.0	3.00	100		3	2	4	ps	1.0	s	4.0	6.4	6
416.0	419.0	3.00	100		3	2	4	pr	1.5	s	4.0	3.4	9
419.0	422.0	2.90	96	420.9	1.9	1	2	pr	1.5	s	4.0	0.95	18

Hole:	A018	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
194.8	195.2	0.00	10		0.4	3	9	ps	1.0	s	4.0	19.4	0.3
195.2	196.0	0.26	32		0.8	3	9	ps	1.0	s	4.0	18.8	1
196.0	199.0	2.36	78		3	3	9	ps	1.0	s	4.0	16.9	2
199.0	202.0	2.73	91		3	3	9	ps	1.0	s	4.0	13.9	3
202.0	205.0	2.62	87		3	3	9	pr	1.5	n	1.5	10.9	10
205.0	208.0	1.96	65		3	3	9	pr	1.5	n	1.5	7.9	7
208.0	211.0	2.89	96		3	2	4	pr	1.5	n	1.5	4.9	24
211.0	214.0	2.84	94		3	2	4	ps	1.0	s	4.0	1.9	6
214.0	217.0	2.26	75	214.4	0.4	2	4	ps	1.0	s	4.0	0.2	5

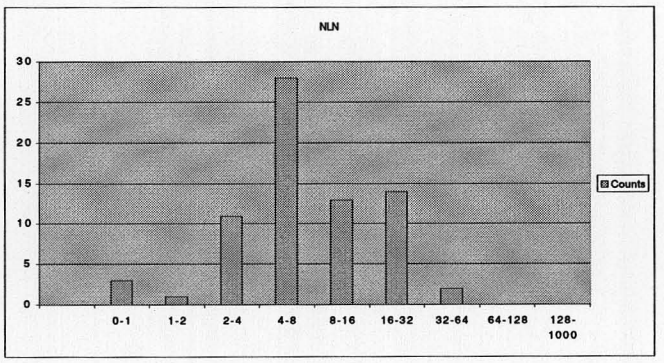
Hole:	A019	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
218.0	221.0	2.67	89		3	2	4	pr	1.5	n	1.5	22.3	22
221.0	224.0	2.77	92		3	2	4	pr	1.5	n	1.5	19.3	23
224.0	227.0	2.86	95		3	3	9	pr	1.5	n	1.5	16.3	11
227.0	230.0	2.91	97		3	3	9	pr	1.5	n	1.5	13.3	11
230.0	233.0	2.81	93		3	2	4	pr	1.5	n	1.5	10.3	23
233.0	235.3	2.23	96		2.3	2	4	pr	1.5	s	4.0	7.65	9
235.3	238.4	2.69	86		3.1	2	4	pr	1.5	n	1.5	4.95	22
238.4	241.4	2.91	97		3	2	4	pr	1.5	n	1.5	1.9	24
241.4	244.2	2.32	82	241.8	0.4	2	4	pr	1.5	s	4.0	0.2	8

Hole:	A020	rqd (m)	RQD	h/w at:		No J Sets	Jn	Rough	Jr	Fill	Ja	Centroid Dist to H/W	Q1
203.0	206.0	2.46	82		3	3	4	ps	1.0	s	4.0	21.3	5
206.0	209.0	2.88	96		3	2	4	ps	1.0	s	4.0	18.3	6
209.0	212.0	2.92	97		3	2	9	us	2.0	s	4.0	15.3	5
212.0	215.0	2.76	92		3	2	9	ps	1.0	s	4.0	12.3	3
215.0	216.9	1.17	61		1.9	2	4	ps	1.0	s	4.0	9.85	4
216.9	218.2	0.50	38		1.3	3	4	us	2.0	s	4.0	8.25	5
218.2	221.0	2.42	86		2.8	3	4	us	2.0	s	4.0	6.2	11
221.0	224.0	2.57	85		3	3	4	us	2.0	s	4.0	3.3	11
224.0	227.0	1.80	60	225.8	1.8	4	4	us	2.0	s	4.0	0.9	8

Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
1.0	0-1	3	3
2.0	1-2	1	4
4.0	2-4	11	15
8.0	4-8	28	43
16.0	8-16	13	56
32.0	16-32	14	70
64.0	32-64	2	72
128.0	64-128	0	72
1000.0	128-1000	0	72

Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
10.0	0-10	1	1
20.0	10-20	0	1
30.0	20-30	0	1
40.0	30-40	3	4
50.0	40-50	1	5
60.0	50-60	3	8
70.0	60-70	3	11
80.0	70-80	4	15
90.0	80-90	15	30
100.0	90-100	42	72

Histogram Intervals	Histogram Intervals	Counts	
0.0		0	0
2.0	0-10	5	5
4.0	10-20	46	51
9.0	20-30	20	71
15.0	30-40	1	72
50.0	40-50	0	72



MCKEOWN MINING - BASIC GEOTECHNICAL LOG

PROJECT AVEBURY **DATE** FEB 00

HOLE NO A012 **LOGGED BY** MICK McKEOWN

from m	to m	recovery m	rqd m	rock type	weatherg	rock strength	no of defects	no of sets of defects	shape & roughness	fill type	rubble flag	comments
390.3	392.0	1.7	1.03	391.9 NLN	5	4	14	3	US	S	–	
392.0	395.0	3.0	2.74		5	3	12	3	US	S		
395.0	397.2	2.2	1.36	396.9 NLS	5	3	21	3	US	S	–	
397.2	400.5	3.3	1.92		5	3	15	3	PS	S	–	
400.5	402.5	2.0	0.76		5	3	21	3	PS	S	–	
402.5	404.2	1.7	0.99		5	3	12	2	PS	S	–	
404.2	407.0	2.8	1.55		5	3	23	2	PS	S	–	
407.0	408.9	1.9	0.68		5	3	22	2	PS	S	–	
408.9	410.0	1.1	0.25		5	3	20	3	PS	S	–	
410.0	411.5	1.5	1.29		5	3	7	2	PS	S	–	
411.5	413.6	2.1	0.42		5	3	30	3	PS	S	–	
413.6	415.3	1.3	0.93		5	3	18	3	PS	S	–	
415.3	418.7	3.4	2.56		5	3	18	2	PS	S	–	
418.7	421.8	3.1	2.38		5	3	18	3	PS	S	–	
421.8	424.8	3.0	2.93		5	3	15	2	PS	S	–	
424.8	426.5	1.7	0.64		5	3	25	3	PS	S	–	
426.5	427.6	1.1	0.10		5	3	-	-	PS	S	N	joint induced rubble
427.6	428.8	0.2	0.00		5	3	-	-	PS	S	D	driller rubble

APPENDIX D

16.4 Appendix D: Open Cut Calculations

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OPEN CUT CALCULATIONS

Ore Density	3.0	
Waste Density	2.8	
Surface RL	2180	RL of the ROM
Ramp Grade	1:9	
Dist to Mill	650	not the same as the Portal to ROM distance

Bench RL	Bench Area	RL Range	Total Volume	Volume Mineralised	Waste	Ore	Waste Cost	Ore Cost	Cumulative $	Cumulative Ore	Cumulative Cost /t Ore
m	m2		bcm	bcm	tonnes	tonnes	Millions	Millions	Millions		
2180	34,605	2170-	346,050		968,940	0	$0.42		$0.4	0	
2160	156,783	2150-2170	3,135,660		8,779,848	0	$5.40		$5.8	0	
2140	206,205	2130-2150	4,124,100		11,547,480	0	$9.20		$15.0	0	
2120	227,207	2110-2130	4,544,140		12,723,592	0	$12.45		$27.5	0	
2100	195,226	2090-2110	3,904,520		10,932,656	0	$12.68		$40.2	0	
2080	156,265	2070-2090	3,125,300		8,750,840	0	$11.74		$51.9	0	
2060	117,452	2050-2070	2,349,040		6,577,312	0	$10.02		$61.9	0	
2040	85,764	2030-2050	1,715,280	188,860	4,273,976	566,580	$8.18	$3.15	$73.2	566,580	$129
2020	70,059	2010-2030	1,401,180	183,880	3,408,440	551,640	$7.40	$3.36	$84.0	1,118,220	$75
2000	55,517	1990-2010	1,110,340	178,400	2,609,432	535,200	$6.43	$3.55	$94.0	1,653,420	$57
1980	42,306	1970-1990	846,120	172,920	1,884,960	518,760	$5.33	$3.72	$103.0	2,172,180	$47
1960	30,207	1950-1970	604,140	161,580	1,239,168	484,740	$4.11	$3.73	$110.9	2,656,920	$42
1940	19,351	1930-1950	387,020	157,300	643,216	471,900	$2.83	$3.89	$117.6	3,128,820	$38
1920	11,675	1920-1930	116,750	78,650	106,680	235,950	$0.91	$2.07	$120.6	3,364,770	$36
			27,709,640	**1,121,590**	**74,446,540**	**3,364,770**	**$97.1 M**	**$23.5 M**			**$36**

66604

APPENDIX E

16.5 Appendix E: Extraction Reserve Calculations – North Avebury

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NORTH BLOCK

Sill Height	4.5
Sill Drive Width	4.0
Sublevel Interval	18
NOMINAL Stope Length	18
Rib Pillars	4
Ore s.g.	3.0
Waste s.g.	2.9

Actual Sill RL	Typical of Level:	Access Length m	Silling m	Sill Stripping m2	Sill Height m	Ore Block Height m	Sill Tonnes	Total Bench Tonnes	Crown Loss m	No. of Stopes / Raises	No of Pillars	Actual Stope Length	TONNES RECOVERED	PILLAR LOSSES	Rock Fill Required m3
2052	2050	80	100	236	4.5		8,586								
						13.5		25,758		5	4	16.8	21,637	4,121	0
2034		80	100	236	4.5		8,586								
						13.5		25,758		5	4	16.8	21,637	4,121	15,024
2016	2000	70	185	996	4.5		23,436								
						13.5		70,308		9	8	17.0	58,147	12,161	27,194
1998		70	185	996	4.5		23,436								
						13.5		70,308		9	8	17.0	58,147	12,161	27,194
1980		70	185	996	4.5		23,436								
						13.5		70,308		9	8	17.0	58,147	12,161	31,789
1962	1950	70	315	1497	4.5		37,220								
						13.5		111,659		15	14	17.3	91,808	19,850	43,009
1944		70	315	1497	4.5		37,220								
						13.5		111,659		15	14	17.3	91,808	19,850	43,009
1926		70	315	1497	4.5		37,220								
						13.5		83,511	3.0	1	0	265.0	64,953	18,558	0
1908	1900	65	265	1002	4.5		27,837								
						13.5		83,511		13	12	16.7	68,384	15,127	32,074
1890		65	265	1002	4.5		27,837								
						13.5		34,263		8	7	18.4	28,781	5,482	13,401
1872	1850	90	175	146	4.5		11,421								
						13.5		34,263		8	7	18.4	28,781	5,482	13,401
1854		90	175	146	4.5		11,421								
						13.5		34,263		8	7	18.4	28,781	5,482	13,401
1836		90	175	146	4.5		11,421								
						13.5		22,680		7	6	16.6	18,792	3,888	8,784
1818	1800	73	140	0	4.5		7,560								
						13.5		22,680		7	6	16.6	18,792	3,888	8,784
1800		73	140	0	4.5		7,560								
							304,196	800,928		119			658,594	142,334	277,064 m3
TOTALS		1126.0 m	3035.0 m	10393 m3		Blocks=	1,105,124 tonnes					Mined=	962,789 t	excludes dilution	
												Recovery=	87%		

Stope Summaries

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APPENDIX F

16.6 Appendix F: Extraction Reserve Calculations – South Avebury

SOUTH BLOCK

			Stope No.	Strike Length	No of Sills	Pillar Area	Sill Area	Fill Dilution
				m		m2	m2	tonnes
Sill Height	4.5	m	1	30.0	2	91.4	63.0	
Sill Drive Width	7.0	m	2	22.5	1	91.4	31.5	1605
Rib Pillars	0	m	3	30.0	2	91.4	63.0	
Ore s.g.	3.0	t/m3	4	30.0	2	91.4	63.0	
Waste s.g.	2.9	t/m3	5	30.0	2	91.4	63.0	1605
h/w Fall-off	1.5	0.80%	6	30.0	2	91.4	63.0	
f/w Fall-off	1	0.00%	7	30.0	2	91.4	63.0	1605
Fill Fall-off	0.5	0.00%	8	30.0	2	91.4	63.0	
Pillar Area	91.4	m2						

Actual Sill RL	Typical of Level:	Stope No	Stope Area	Stope Type	Top Area Method	End Area Method 1.78% Ni	Silling	Total Silling		Pillar Losses 1.78% Ni	h/w Dilution 0.80% Ni	f/w Dilution (waste) 0.20% Ni	Fill Dilution 0.00% Ni	Sill Extraction 1.78% Ni	Stope Extraction	Primary	Secondary	Cemented Fill Required	Rock Fill Required
			m2			tonnes	m	m		tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	m3	m3
2050	2050	1	801	P			26.5	53						5,009					
		2	607	S			26.6	27						2,552					
		3	779	P			26.5	53						5,009					
		4	543	P			23.7	48						4,536					
		5	536	S			24.1	49						4,631					
		6	815	P			23.7	48						4,536					
		7	889	S			24.1	49						4,631					
		8	757	P			23.7	48	375.0					4,536					
														35,438	0				
2025	2000	1	808	P	60,338	59,625	26.5	53		7,266	2,680	1,727		5,009	51,758	51,758		18,922	
		2	624	S	46,163	45,478	27.3	28		7,390	1,978	1,275	1,605	2,646	40,301		40,301		0
		3	772	P	58,163	59,625	26.5	53		7,266	2,680	1,727		5,009	51,758	51,758		18,922	
		4	536	P	40,463	50,738	21.4	43		6,183	2,680	1,727		4,064	44,898	44,898		16,321	
		5	595	S	42,413	52,200	22.3	45		6,361	2,680	1,727	1,605	4,253	47,599		47,599		0
		6	731	P	57,975	50,738	21.4	43		6,183	2,680	1,727		4,064	44,898	44,898		16,321	
		7	746	S	61,313	52,200	22.3	45		6,361	2,680	1,727	1,605	4,253	47,599		47,599		0
		8	645	P	52,575	50,738	21.4	43	353.0	6,183	2,680	1,727		4,064	44,898	44,898		16,321	
														33,359	373,708	238,210	135,498	86,806	0
2000	2000	1	808	P	60,600	59,625	26.5	53		7,266	2,680	1,727		5,009	51,758	51,758		18,922	
		2	624	S	46,800	46,069	27.3	28		7,486	1,978	1,275	1,605	2,646	40,795		40,795		15,015
		3	772	P	57,900	59,625	26.5	53		7,266	2,680	1,727		5,009	51,758	51,758		18,922	
		4	536	P	40,200	48,150	21.4	43		5,868	2,680	1,727		4,064	42,626	42,626		15,563	
		5	595	S	44,625	50,175	22.3	45		6,115	2,680	1,727	1,605	4,253	45,820		45,820		17,226
		6	731	P	54,825	48,150	21.4	43		5,868	2,680	1,727		4,064	42,626	42,626		15,563	
		7	746	S	55,950	50,175	22.3	45		6,115	2,680	1,727	1,605	4,253	45,820		45,820		17,226
		8	645	P	48,375	48,150	21.4	43	353.0	5,868	2,680	1,727		4,064	42,626	42,626		15,563	
														33,359	363,830	231,393	132,436	84,534	49,468
1975	1950	1	663	P	55,163	55,463	22.8	46		6,759	2,680	1,727		4,347	48,764	48,764		17,704	
		2	541	S	43,688	43,453	24.2	25		7,061	1,978	1,275	1,605	2,363	38,888		38,888		14,285
		3	694	P	54,975	55,463	22.8	46		6,759	2,680	1,727		4,347	48,764	48,764		17,704	
		4	452	P	37,050	45,338	18.9	38		5,525	2,680	1,727		3,591	40,629	40,629		14,740	
		5	483	S	40,425	47,588	20.0	40		5,799	2,680	1,727	1,605	3,780	44,021		44,021		16,469
		6	641	P	51,450	45,338	18.9	38		5,525	2,680	1,727		3,591	40,629	40,629		14,740	
		7	702	S	54,300	47,588	20.0	40		5,799	2,680	1,727	1,605	3,780	44,021		44,021		16,469
		8	600	P	46,688	45,338	18.9	38	311.0	5,525	2,680	1,727		3,591	40,629	40,629		14,740	
														29,390	346,344	219,414	126,930	79,627	47,223
1950	1950	1	663	P	49,725	51,300	22.8	46		6,252	2,680	1,727		4,347	45,109	45,109		16,485	
		2	541	S	40,575	40,838	24.2	25		6,636	1,978	1,275	1,605	2,363	36,698		36,698		13,555
		3	694	P	52,050	51,300	22.8	46		6,252	2,680	1,727		4,347	45,109	45,109		16,485	
		4	452	P	33,900	42,525	18.9	38		5,182	2,680	1,727		3,591	38,159	38,159		13,917	
		5	483	S	36,225	45,000	20.0	40		5,484	2,680	1,727	1,605	3,780	41,749		41,749		15,711
		6	641	P	48,075	42,525	18.9	38		5,182	2,680	1,727		3,591	38,159	38,159		13,917	
		7	702	S	52,650	45,000	20.0	40		5,484	2,680	1,727	1,605	3,780	41,749		41,749		15,711
		8	600	P	45,000	42,525	18.9	38	311.0	5,182	2,680	1,727		3,591	38,159	38,159		13,917	
														29,390	324,889	204,695	120,195	74,721	44,978
1920	1920	1	577	P	55,800	58,320	20.4	41		5,923	3,355	2,162		3,875	54,040	54,040		19,305	
		2	469	S	45,450	45,056	20.3	21		6,101	2,484	1,601	1,605	1,985	42,661		42,661		15,417
		3	642	P	60,120	58,320	20.4	41		5,923	3,355	2,162		3,875	54,040	54,040		19,305	
		4	437	P	40,005	50,355	18.4	37		5,114	3,355	2,162		3,497	47,262	47,262		16,920	
		5	479	S	43,290	54,000	20.0	40		5,484	3,355	2,162	1,605	3,780	51,859		51,859		19,081
		6	661	P	58,590	50,355	18.4	37		5,114	3,355	2,162		3,497	47,262	47,262		16,920	
		7	691	S	62,685	54,000	20.0	40		5,484	3,355	2,162	1,605	3,780	51,859		51,859		19,081
		8	550	P	51,750	50,355	18.4	37	294.0	5,114	3,355	2,162		3,497	47,262	47,262		16,920	
														27,783	396,245	249,867	146,379	89,368	53,580
			30699	Primary	1,271,753	1,279,980		1,997		243,709	108,929	70,199	24,077	188,717	1,805,016			415,055	195,247
				Secondary	716,550	718,819				1.78% Ni	0.80% Ni	0.20% Ni	0.00% Ni	1.78% Ni	1.64% Ni				
TOTALS					1,988,303	1,998,799													

Stope Summaries

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APPENDIX G

16.7 Appendix G: Development Scheduling

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DEVELOPMENT by PERIOD

YEAR 1

22-Dec-00

LOCATION	Type	Crew	Grand	To be	Contractor Days	31 Days	31 Days	30 Days	31 Days	30 Days	27 Days	31 Days	28 Days	31 Days	30 Days	31 Days	30 Days	
					Mine Days	23 Days	21 Days	22 Days	22 Days	21 Days	22 Days	21 Days	20 Days	23 Days	21 Days	21 Days	22 Days	
	Cap. or Op.		Total	Done		Period1 July 23 Days	Period2 August 21 Days	Period3 September 22 Days	Period4 October 22 Days	Period5 November 21 Days	Period6 December 22 Days	Period7 January 21 Days	Period8 February 20 Days	Period9 March 23 Days	Period10 April 21 Days	Period11 May 21 Days	Period12 June 22 Days	TOTALS 259 Days
Portal to 2050	C	Contract	795	795.0 m		237	237	229	92									795
2060 Vent Connections	C	Contract	40	40.0 m					40									40
2050	C	Contract	253	253.0 m					101	152								253
2000	C	Contract	458	458.0 m						67	182	209						458
2000 Vent Connections	C	Contract	25	25.0 m							12	13						25
1950	C	Contract	325	325.0 m									212	113				325
1950 sub-level	C	Contract	148	148.0 m											74	74		148
1900	C	Contract	365	200.0 m													200	200
1900 Pump Stn & Vent	C	Contract	60	60.0 m														
1850	C	Contract	417	417.0 m														
1800	C	Contract	305	305.0 m														
1800 Vent x-cut	C	Contract	17	17.0 m														
2052 Access	Op	Contract	80	80.0 m														
2052 Sill	Ore		100	100.0 m														
2034 Access	Op	Contract	80	80.0 m														
2034 Sill	Ore		100	100.0 m														
2016 Access	Op	Contract	70	70.0 m														
2016 Sill	Ore		185	185.0 m														
1998 Access	Op	Contract	70	70.0 m														
1998 Sill	Ore		185	185.0 m														
1980 Access	Op	Contract	70	70.0 m														
1980 Sill	Ore		185	185.0 m														
1962 Access	Op	Contract	70	70.0 m														
1962 Sill	Ore		315	315.0 m														
1944 Access	Op	Contract	70	70.0 m											47	23		70
1944 Sill	Ore		315	315.0 m														
1926 Access	Op	Contract	70	70.0 m												70		70
1926 Sill	Ore		315	315.0 m														
1908 Access	Op	Contract	65	65.0 m														
1908 Sill	Ore		265	265.0 m														
1890 Access	Op	Contract	65	65.0 m														
1890 Sill	Ore		265	265.0 m														
1872 Access	Op	Contract	90	90.0 m														
1872 Sill	Ore		175	175.0 m														
1854 Access	Op	Contract	90	90.0 m														
1854 Sill	Ore		175	175.0 m														
1836 Access	Op	Contract	90	90.0 m														
1836 Sill	Ore		175	175.0 m														
1818 Access	Op	Contract	73	73.0 m														
1818 Sill	Ore		140	140.0 m														
1800 Access	Op	Contract	73	73.0 m														
1800 Sill	Ore		140	140.0 m														
2050 Level Dev	Op	Contract	404	404.0 m														
2050 Ore Drives	Ore		455	455.0 m														
2050 Waste Silling	Op		230	230.0 m														
2025 Level Dev	Op	Contract	366	366.0 m														
2025 Ore Drives	Ore		433	433.0 m														
2025 Waste Silling	Op		222	222.0 m														
2000 Level Dev	Op	Contract	366	366.0 m														
2000 Ore Drives	Ore		433	433.0 m														
2000 Waste Silling	Op		222	222.0 m														
1975 Level Dev	Op	Contract	347	347.0 m														
1975 Ore Drives	Ore		397	397.0 m														
1975 Waste Silling	Op		154	154.0 m														
1950 Level Dev	Op	Contract	347	347.0 m										117	113	117		347
1950 Ore Drives	Ore		397	397.0 m														
1950 Waste Silling	Op		154	154.0 m														
1920 Level Dev	Op	Contract	371	100.0 m													100	100
1920 Ore Drives	Ore		374	374.0 m														
1920 Waste Silling	Op		152	152.0 m														
		Waste only total:	7669.0 m	7233.0 m		237.0	237.0	229.3	232.9	219.2	193.5	222.2	211.6	230.3	233.8	284.3	300.0	2831.0
					Per 'Mine' Day	10.3m/d	11.3m/d	10.4m/d	10.6m/d	10.4m/d	8.8m/d	10.6m/d	10.6m/d	10.0m/d	11.1m/d	13.5m/d	13.6m/d	10.9m/d
					Capital	237.0	237.0	229.3	232.9	219.2	193.5	222.2	211.6	113.4	74.0	74.0	200.0	2244.0
					Operating									116.9	159.8	210.3	100.0	587.0
					Ore Rate													
BENCH Ore tonnes / metres	*100.2*				*Silling Tonnes*													
OS Ore tonnes / metres	*89.3*				*Contract rate*	*7.6*	*7.6*	*7.6*	*7.5*	*7.3*	*7.2*	*7.2*	*7.6*	*7.4*	*7.6*	*9.2*	*10.0*	*10.9*
					Contract	237.0	237.0	229.3	232.9	219.2	193.5	222.2	211.6	230.3	233.8	284.3	300.0	2831.0
					Avebury rate													
					Avebury crew													
					Operating									116.9	159.8	210.3	100.0	587.0
					TOTAL									116.9	159.8	210.3	100.0	587.0

YEAR 1
22 Dec 00 19:49

YEAR 2

LOCATION	Type	Crew	Grand	To be	Contractor Days	31 Days	31 Days	30 Days	31 Days	30 Days	27 Days	31 Days	28 Days	31 Days	30 Days	31 Days	30 Days	TOTALS
					Mine Days	23 Days	21 Days	22 Days	22 Days	21 Days	22 Days	21 Days	20 Days	23 Days	21 Days	21 Days	22 Days	
			Total	Done		Period1	Period2	Period3	Period4	Period5	Period6	Period7	Period8	Period9	Period10	Period11	Period12	
	Cap. or Op.		Remaining			July	August	September	October	November	December	January	February	March	April	May	June	
						23 Days	21 Days	22 Days	22 Days	21 Days	22 Days	21 Days	20 Days	23 Days	21 Days	21 Days	22 Days	259 Days
Portal to 2050	C																	
2060 Vent Connections	C																	
2050	C																	
2000	C																	
2000 Vent Connections	C																	
1950	C																	
1950 sub-level	C																	
1900	C	Contract	165	165.0 m		165												165
1900 Pump Stn & Vent	C	Contract	60	60.0 m			60											60
1850	C		417	200.0 m									27	27	27	58	60	200
1800	C		305	305.0 m														
1800 Vent x-cut	C		17	17.0 m														
2052 Access	Op		80	80.0 m														
2052 Sill	Ore		100	100.0 m														
2034 Access	Op		80	80.0 m														
2034 Sill	Ore		100	100.0 m														
2016 Access	Op		70	70.0 m														
2016 Sill	Ore		185	185.0 m														
1998 Access	Op		70	70.0 m														
1998 Sill	Ore		185	185.0 m														
1980 Access	Op		70	70.0 m														
1980 Sill	Ore		185	185.0 m														
1962 Access	Op	Contract	70	70.0 m			70											70
1962 Sill	Ore		315	315.0 m														
1944 Access	Op																	
1944 Sill	Ore		315	315.0 m								89	85	98	43			315
1926 Access	Op																	
1926 Sill	Ore		315	315.0 m		134	123	58										315
1908 Access	Op		65	65.0 m														
1908 Sill	Ore		265	265.0 m														
1890 Access	Op		65	65.0 m														
1890 Sill	Ore		265	265.0 m														
1872 Access	Op		90	90.0 m														
1872 Sill	Ore		175	175.0 m														
1854 Access	Op		90	90.0 m														
1854 Sill	Ore		175	175.0 m														
1836 Access	Op		90	90.0 m														
1836 Sill	Ore		175	175.0 m														
1818 Access	Op		73	73.0 m														
1818 Sill	Ore		140	140.0 m														
1800 Access	Op		73	73.0 m														
1800 Sill	Ore		140	140.0 m														
2050 Level Dev	Op		404	404.0 m														
2050 Ore Drives	Ore		455	455.0 m														
2050 Waste Silling	Op		230	230.0 m														
2025 Level Dev	Op		366	366.0 m														
2025 Ore Drives	Ore		433	433.0 m														
2025 Waste Silling	Op		222	222.0 m														
2000 Level Dev	Op		366	366.0 m														
2000 Ore Drives	Ore		433	433.0 m														
2000 Waste Silling	Op		222	222.0 m														
1975 Level Dev	Op		347	347.0 m										66	92	92	97	347
1975 Ore Drives	Ore		397	397.0 m														
1975 Waste Silling	Op		154	154.0 m														
1950 Level Dev	Op																	
1950 Ore Drives	Ore		397	397.0 m					42	88	92	92	84					397
1950 Waste Silling	Op		154	154.0 m						37	81	37						154
1920 Level Dev	Op	Contract	271	271.0 m		136	136											271
1920 Ore Drives	Ore		374	374.0 m		79	72	75	75	72								374
1920 Waste Silling	Op		152	152.0 m				76	76									152
			4838.0 m	4621.0 m		300.5	265.5	76.0	76.0	36.7	80.7	36.7	27.4	93.3	119.6	149.8	156.9	1419.0
						13.1m/d	12.6m/d	3.5m/d	3.5m/d	1.7m/d	3.7m/d	1.7m/d	1.4m/d	4.1m/d	5.7m/d	7.1m/d	7.1m/d	5.5m/d
					Capital	165.0	60.0						27.4	27.4	27.4	57.5	60.3	425.6
					Operating	135.5	205.5	76.0	76.0	36.7	80.7	36.7		65.9	92.2	92.2	96.6	994.0
					Ore Rate	*7.0*	*6.4*	*4.4*	*3.6*	*5.0*	*3.2*	*5.8*	*6.0*	*3.3*	*1.5*			
					Ore Tonnes	*20491*	*18709*	*12586*	*10473*	*14271*	*8210*	*17167*	*15994*	*9810*	*4265*			*131976*
					H Speed rate	*9.7*	*8.6*											2.2
					H Speed crew	300.5	265.5											566.0
					Avebury rate	7.0	6.4	7.9	7.0	6.8	6.9	7.6	7.4	7.4	7.2	7.1	7.1	7.1
					Avebury crew	160.5	133.8	173.4	154.4	142.1	151.3	159.4	148.0	170.1	151.1	149.8	156.9	1850.7
					Operating	135.5	205.5	76.0	76.0	36.7	80.7	36.7		65.9	92.2	92.2	96.6	994.0
					TOTAL	135.5	205.5	76.0	76.0	36.7	80.7	36.7		65.9	92.2	92.2	96.6	994.0

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YEAR 3

LOCATION	Type	Crew	Grand Total	To be Done	Contractor Days / Mine Days	31 Days / 23 Days	31 Days / 21 Days	30 Days / 22 Days	31 Days / 22 Days	30 Days / 21 Days	27 Days / 22 Days	31 Days / 21 Days	28 Days / 20 Days	31 Days / 23 Days	30 Days / 21 Days	31 Days / 21 Days	30 Days / 22 Days	TOTALS
	Cap. or Op.		Remaining			Period1 July 23 Days	Period2 August 21 Days	Period3 September 22 Days	Period4 October 22 Days	Period5 November 21 Days	Period6 December 22 Days	Period7 January 21 Days	Period8 February 20 Days	Period9 March 23 Days	Period10 April 21 Days	Period11 May 21 Days	Period12 June 22 Days	259 Days
Portal to 2050	C																	
2060 Vent Connections	C																	
2050	C																	
2000	C																	
2000 Vent Connections	C																	
1950	C																	
1950 sub-level	C																	
1900	C	Contract																
1900 Pump Stn & Vent	C	Contract																
1850	C		217	217.0 m		92	84	40										217
1800	C		305	305.0 m				36	79	75	79	36						305
1800 Vent x-cut	C		17	17.0 m								17						17
2052 Access	Op		80	80.0 m														
2052 Sill	Ore		100	100.0 m														
2034 Access	Op		80	80.0 m														
2034 Sill	Ore		100	100.0 m														
2016 Access	Op		70	70.0 m														
2016 Sill	Ore		185	185.0 m														
1998 Access	Op		70	70.0 m														
1998 Sill	Ore		185	185.0 m														
1980 Access	Op		70	70.0 m														
1980 Sill	Ore		185	185.0 m														
1962 Access	Op																	
1962 Sill	Ore		315	315.0 m		82	75	79	79									315
1944 Access	Op																	
1944 Sill	Ore																	
1926 Access	Op																	
1926 Sill	Ore																	
1908 Access	Op		65	65.0 m														
1908 Sill	Ore		265	265.0 m														
1890 Access	Op		65	65.0 m														
1890 Sill	Ore		265	265.0 m														
1872 Access	Op		90	90.0 m														
1872 Sill	Ore		175	175.0 m														
1854 Access	Op		90	90.0 m														
1854 Sill	Ore		175	175.0 m														
1836 Access	Op		90	90.0 m				23	23	22	23							90
1836 Sill	Ore		175	175.0 m											61	56	58	175
1818 Access	Op		73	73.0 m						36	37							73
1818 Sill	Ore		140	140.0 m								44	44	51				140
1800 Access	Op		73	73.0 m								49	24					73
1800 Sill	Ore		140	140.0 m									42	98				140
2050 Level Dev	Op		404	404.0 m														
2050 Ore Drives	Ore		455	455.0 m														
2050 Waste Silling	Op		230	230.0 m														
2025 Level Dev	Op		366	366.0 m														
2025 Ore Drives	Ore		433	433.0 m														
2025 Waste Silling	Op		222	222.0 m														
2000 Level Dev	Op		366	366.0 m														
2000 Ore Drives	Ore		433	433.0 m														
2000 Waste Silling	Op		222	222.0 m														
1975 Level Dev	Op																	
1975 Ore Drives	Ore		397	397.0 m									74	85	78	78	82	397
1975 Waste Silling	Op		154	154.0 m											51	51	53	154
1950 Level Dev	Op																	
1950 Ore Drives	Ore																	
1950 Waste Silling	Op																	
1920 Level Dev	Op	Contract																
1920 Ore Drives	Ore																	
1920 Waste Silling	Op																	
			3419.0 m	3419.0 m		92.4	84.4	98.8	101.7	132.7	139.0	102.3	23.5		50.5	50.5	52.9	929.0
						4.0m/d	4.0m/d	4.5m/d	4.6m/d	6.3m/d	6.3m/d	4.9m/d	1.2m/d		2.4m/d	2.4m/d	2.4m/d	3.6m/d
					Capital	92.4	84.4	76.1	78.9	75.4	78.9	52.9						539.0
					Operating			22.8	22.8	57.4	60.1	49.5	23.5		50.5	50.5	52.9	390.0
					Ore Rate	*2.8*	*2.6*	*2.8*	*2.7*			*1.5*	*5.8*	*7.7*	*4.6*	*4.3*	*4.6*	
					Ore Tonnes	*8249*	*7532*	*7891*	*7891*			*4453*	*15331*	*22519*	*13069*	*12537*	*13134*	*112606*
					H Speed rate													
					H Speed crew													
					Avebury rate	6.8	6.6	7.3	7.3	6.3	6.3	6.4	7.0	7.7	7.0	6.7	7.0	6.9
					Avebury crew	157.0	138.2	159.9	160.8	132.7	139.0	134.2	139.1	176.3	147.1	140.2	154.6	1779.2
					Operating			22.8	22.8	57.4	60.1	49.5	23.5		50.5	50.5	52.9	390.0
					TOTAL			22.8	22.8	57.4	60.1	49.5	23.5		50.5	50.5	52.9	390.0

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YEAR 4

				Contractor Days	31 Days	31 Days	30 Days	31 Days	30 Days	27 Days	31 Days	28 Days	31 Days	30 Days	31 Days	30 Days	
				Mine Days	23 Days	21 Days	22 Days	22 Days	21 Days	22 Days	21 Days	20 Days	23 Days	21 Days	21 Days	22 Days	
LOCATION	Type	Crew	Grand	To be	Period1	Period2	Period3	Period4	Period5	Period6	Period7	Period8	Period9	Period10	Period11	Period12	TOTALS
	Cap. or Op.		Total Remaining	Done	July 23 Days	August 21 Days	September 22 Days	October 22 Days	November 21 Days	December 22 Days	January 21 Days	February 20 Days	March 23 Days	April 21 Days	May 21 Days	June 22 Days	259 Days
Portal to 2050	C																
2060 Vent Connections	C																
2050	C																
2000	C																
2000 Vent Connections	C																
1950	C																
1950 sub-level	C																
1900	C	Contract															
1900 Pump Stn & Vent	C	Contract															
1850	C																
1800	C																
1800 Vent x-cut	C																
2052 Access	Op		80	80.0 m													
2052 Sill	Ore		100	100.0 m													
2034 Access	Op		80	80.0 m													
2034 Sill	Ore		100	100.0 m													
2016 Access	Op		70	70.0 m											34	36	70
2016 Sill	Ore		185	185.0 m													
1998 Access	Op		70	70.0 m							23	22	25				70
1998 Sill	Ore		185	185.0 m										61	61	64	185
1980 Access	Op		70	70.0 m	24	22	23										70
1980 Sill	Ore		185	185.0 m										61	61	64	185
1962 Access	Op																
1962 Sill	Ore																
1944 Access	Op																
1944 Sill	Ore																
1926 Access	Op																
1926 Sill	Ore																
1908 Access	Op		65	65.0 m													
1908 Sill	Ore		265	265.0 m													
1890 Access	Op		65	65.0 m													
1890 Sill	Ore		265	265.0 m													
1872 Access	Op		90	90.0 m													
1872 Sill	Ore		175	175.0 m													
1854 Access	Op		90	90.0 m													
1854 Sill	Ore		175	175.0 m													
1836 Access	Op																
1836 Sill	Ore																
1818 Access	Op																
1818 Sill	Ore																
1800 Access	Op																
1800 Sill	Ore																
2050 Level Dev	Op		404	404.0 m													
2050 Ore Drives	Ore		455	455.0 m													
2050 Waste Silling	Op		230	230.0 m													
2025 Level Dev	Op		366	366.0 m	64	58	61	61	58	64							366
2025 Ore Drives	Ore		433	433.0 m													
2025 Waste Silling	Op		222	222.0 m													
2000 Level Dev	Op		366	366.0 m	77	71	74	74	71								366
2000 Ore Drives	Ore		433	433.0 m						74	71	68	78	71	71		433
2000 Waste Silling	Op		222	222.0 m							73	69	80				222
1975 Level Dev	Op																
1975 Ore Drives	Ore																
1975 Waste Silling	Op																
1950 Level Dev	Op																
1950 Ore Drives	Ore																
1950 Waste Silling	Op																
1920 Level Dev	Op	Contract															
1920 Ore Drives	Ore																
1920 Waste Silling	Op																
			2490.0 m	2490.0 m	165.4	151.0	158.2	134.9	128.7	63.8	95.8	91.3	104.9		34.2	35.8	1164.0
					7.2m/d	7.2m/d	7.2m/d	6.1m/d	6.1m/d	2.9m/d	4.6m/d	4.6m/d	4.6m/d		1.6m/d	1.6m/d	4.5m/d
				Capital													
				Operating	165.4	151.0	158.2	134.9	128.7	63.8	95.8	91.3	104.9		34.2	35.8	1164.0
				Ore Rate						2.6	2.2	2.3	2.4	6.5	6.3	4.5	
				Ore Tonnes						6646	6344	6042	6948	18509	18509	12744	75741
				H Speed rate													
				H Speed crew													
				Avebury rate	7.2	7.2	7.2	6.1	6.1	5.5	6.7	6.8	6.9	6.5	7.9	6.1	6.7
YEAR 4				Avebury crew	165.4	151.0	158.2	134.9	128.7	120.9	141.2	136.8	159.3	136.7	166.5	134.4	1734.2
22 Dec 00 19:49				Operating	165.4	151.0	158.2	134.9	128.7	63.8	95.8	91.3	104.9		34.2	35.8	1164.0
				TOTAL	165.4	151.0	158.2	134.9	128.7	63.8	95.8	91.3	104.9		34.2	35.8	1164.0

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YEAR 5

				Contractor Days	31 Days	31 Days	30 Days	31 Days	30 Days	27 Days	31 Days	28 Days	31 Days	30 Days	31 Days	30 Days	
				Mine Days	23 Days	21 Days	22 Days	22 Days	21 Days	22 Days	21 Days	20 Days	23 Days	21 Days	21 Days	22 Days	
LOCATION	Type	Crew	Grand Total	To be Done	Period1 July	Period2 August	Period3 September	Period4 October	Period5 November	Period6 December	Period7 January	Period8 February	Period9 March	Period10 April	Period11 May	Period12 June	TOTALS
	Cap. or Op.		Remaining		23 Days	21 Days	22 Days	22 Days	21 Days	22 Days	21 Days	20 Days	23 Days	21 Days	21 Days	22 Days	259 Days
Portal to 2050	C																
2060 Vent Connections	C																
2050	C																
2000	C																
2000 Vent Connections	C																
1950	C																
1950 sub-level	C																
1900	C	Contract															
1900 Pump Stn & Vent	C	Contract															
1850	C																
1800	C																
1800 Vent x-cut	C																
2052 Access	Op		80	80.0 m													
2052 Sill	Ore		100	100.0 m													
2034 Access	Op		80	80.0 m				27	26	27							80
2034 Sill	Ore		100	100.0 m							34	32	34				100
2016 Access	Op																
2016 Sill	Ore		185	185.0 m	64	59	62										185
1998 Access	Op																
1998 Sill	Ore																
1980 Access	Op																
1980 Sill	Ore																
1962 Access	Op																
1962 Sill	Ore																
1944 Access	Op																
1944 Sill	Ore																
1926 Access	Op																
1926 Sill	Ore																
1908 Access	Op		65	65.0 m													
1908 Sill	Ore		265	265.0 m													
1890 Access	Op		65	65.0 m											32	33	65
1890 Sill	Ore		265	265.0 m													
1872 Access	Op		90	90.0 m						31	30	29					90
1872 Sill	Ore		175	175.0 m									81	94			175
1854 Access	Op		90	90.0 m	47	43											90
1854 Sill	Ore		175	175.0 m											85	90	175
1836 Access	Op																
1836 Sill	Ore																
1818 Access	Op																
1818 Sill	Ore																
1800 Access	Op																
1800 Sill	Ore																
2050 Level Dev	Op		404	404.0 m							65	71	71	65	65	68	404
2050 Ore Drives	Ore		455	455.0 m													
2050 Waste Silling	Op		230	230.0 m													
2025 Level Dev	Op																
2025 Ore Drives	Ore		433	433.0 m	66	60	63	60	60	63	60						433
2025 Waste Silling	Op		222	222.0 m			57	54	57	54							222
2000 Level Dev	Op																
2000 Ore Drives	Ore																
2000 Waste Silling	Op																
1975 Level Dev	Op																
1975 Ore Drives	Ore																
1975 Waste Silling	Op																
1950 Level Dev	Op																
1950 Ore Drives	Ore																
1950 Waste Silling	Op																
1920 Level Dev	Op	Contract															
1920 Ore Drives	Ore																
1920 Waste Silling	Op																
			1326.0 m	1326.0 m	47.0	43.0	56.8	81.3	82.6	112.7	94.8	99.5	70.9	64.8	96.5	101.1	951.0
					2.0m/d	2.0m/d	2.6m/d	3.7m/d	3.9m/d	5.1m/d	4.5m/d	5.0m/d	3.1m/d	3.1m/d	4.6m/d	4.6m/d	3.7m/d
				Capital													
				Operating	47.0	43.0	56.8	81.3	82.6	112.7	94.8	99.5	70.9	64.8	96.5	101.1	951.0
				Ore Rate	*4.2*	*3.8*	*4.2*	*1.8*	*1.9*	*2.2*	*3.0*	*1.2*	*3.9*	*3.3*	*2.9*	*3.2*	
				Ore Tonnes	*12350*	*11276*	*11813*	*5378*	*5378*	*5634*	*8771*	*3232*	*11550*	*9379*	*8564*	*8971*	*102294*
				H Speed rate													
				H Speed crew													
				Avebury rate	6.2	5.9	6.7	5.5	5.8	7.3	7.5	6.2	7.0	6.4	7.5	7.8	6.7
				Avebury crew	143.7	123.6	148.2	121.6	122.4	161.2	157.5	123.9	161.4	134.1	157.7	170.5	1725.7
				Operating	47.0	43.0	56.8	81.3	82.6	112.7	94.8	99.5	70.9	64.8	96.5	101.1	951.0
				TOTAL	47.0	43.0	56.8	81.3	82.6	112.7	94.8	99.5	70.9	64.8	96.5	101.1	951.0

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YEAR 6

LOCATION	Type	Crew	Grand Total	To be Done	Period1 July	Period2 August	Period3 September	Period4 October	Period5 November	Period6 December	Period7 January	Period8 February	Period9 March	Period10 April	Period11 May	Period12 June	TOTALS
	Cap. or Op.		Remaining		23 Days	21 Days	22 Days	22 Days	21 Days	22 Days	21 Days	20 Days	23 Days	21 Days	21 Days	22 Days	259 Days
Contractor Days					31 Days	31 Days	30 Days	31 Days	30 Days	27 Days	31 Days	28 Days	31 Days	30 Days	31 Days	30 Days	
Mine Days					23 Days	21 Days	22 Days	22 Days	21 Days	22 Days	21 Days	20 Days	23 Days	21 Days	21 Days	22 Days	
Portal to 2050	C																
2060 Vent Connections	C																
2050	C																
2000	C																
2000 Vent Connections	C																
1950	C																
1950 sub-level	C																
1900	C	Contract															
1900 Pump Stn & Vent	C	Contract															
1850	C																
1800	C																
1800 Vent x-cut	C																
2052 Access	Op		80	80.0 m								25	29	26			80
2052 Sill	Ore		100	100.0 m											49	51	100
2034 Access	Op																
2034 Sill	Ore																
2016 Access	Op																
2016 Sill	Ore																
1998 Access	Op																
1998 Sill	Ore																
1980 Access	Op																
1980 Sill	Ore																
1962 Access	Op																
1962 Sill	Ore																
1944 Access	Op																
1944 Sill	Ore																
1926 Access	Op																
1926 Sill	Ore																
1908 Access	Op		65	65.0 m	23	21	22										65
1908 Sill	Ore		265	265.0 m									70	64	64	67	265
1890 Access	Op																
1890 Sill	Ore		265	265.0 m	47	42	45	45	42	45							265
1872 Access	Op																
1872 Sill	Ore																
1854 Access	Op																
1854 Sill	Ore																
1836 Access	Op																
1836 Sill	Ore																
1818 Access	Op																
1818 Sill	Ore																
1800 Access	Op																
1800 Sill	Ore																
2050 Level Dev	Op																
2050 Ore Drives	Ore		455	455.0 m	54	49	51	51	49	51	49	47	54				455
2050 Waste Silling	Op		230	230.0 m				48	46	48	46	43					230
2025 Level Dev	Op																
2025 Ore Drives	Ore																
2025 Waste Silling	Op																
2000 Level Dev	Op																
2000 Ore Drives	Ore																
2000 Waste Silling	Op																
1975 Level Dev	Op																
1975 Ore Drives	Ore																
1975 Waste Silling	Op																
1950 Level Dev	Op																
1950 Ore Drives	Ore																
1950 Waste Silling	Op																
1920 Level Dev	Op	Contract															
1920 Ore Drives	Ore																
1920 Waste Silling	Op																
			375.0 m	375.0 m	22.7	20.7	21.7	47.7	45.6	47.7	45.6	68.4	28.8	26.3			375.0
					1.0m/d	1.0m/d	1.0m/d	2.2m/d	2.2m/d	2.2m/d	2.2m/d	3.4m/d	1.3m/d	1.3m/d			1.4m/d
				Capital													
				Operating	22.7	20.7	21.7	47.7	45.6	47.7	45.6	68.4	28.8	26.3			375.0
				Ore Rate	*3.2*	*2.9*	*3.2*	*3.1*	*3.0*	*3.5*	*1.5*	*1.6*	*4.0*	*2.3*	*3.8*	*4.2*	
				Ore Tonnes	*9454*	*8632*	*9043*	*9043*	*8632*	*9043*	*4376*	*4167*	*11812*	*6409*	*11303*	*11841*	*103758*
				H Speed rate													
				H Speed crew													
				Avebury rate	4.2	3.9	4.2	5.2	5.2	5.7	3.7	5.0	5.3	3.5	3.8	4.2	4.5
				Avebury crew	96.7	82.4	91.7	115.5	109.3	125.5	76.9	99.8	121.2	73.6	80.8	91.6	1165.0
				Operating	22.7	20.7	21.7	47.7	45.6	47.7	45.6	68.4	28.8	26.3			375.0
				TOTAL	22.7	20.7	21.7	47.7	45.6	47.7	45.6	68.4	28.8	26.3			375.0

YEAR 6
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APPENDIX H

16.8 Appendix H: Detailed Capital & Operating Expenditure Schedule

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			Year 0
DEVELOPMENT (m)	Capital		
	Operating		
PRODUCTION ('000s)	Bench Sill Tonnes		
	Bench Tonnes		
	TOTAL BENCH PRODUCTION		
	OS Sill Tonnes		
	OS Tonnes		
	TOTAL OPEN STOPE PRODUCTION		
	PERIOD PRODUCTION		
		UNIT RATES	
	Capital Development Cost	$2,712	
	Operating Development Cost	$2,667	
	OVER-ALL Bench Cost	$23.10	
	OVER-ALL OS Cost	$18.03	
OWNER MANAGEMENT	Service Maintenance	$3.62	
	Mine Management	$3.81	
	SITE ESTABLISHMENT		$587,520
	VENT RAISES		$0
	FIXED PLANT		$133,000
	MOBILE EQUIPMENT		$204,000
	CAPITAL DEVELOPMENT		$0
	OTHER CAPITAL		$924,520
	OPERATING EXPENDITURE		$0
	PERIOD TOTAL		$924,520

			Year 1 Period1	Year 1 Period2	Year 1 Period3	Year 1 Period4	Year 1 Period5	Year 1 Period6	Year 1 Period7	Year 1 Period8	Year 1 Period9	Year 1 Period10	Year 1 Period11	Year 1 Period12	Year Total
DEVELOPMENT (m)	Capital		237.0	237.0	229.3	232.9	219.2	193.5	222.2	211.6	113.4	74.0	74.0	200.0	2,244.0
	Operating		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	116.9	159.8	210.3	100.0	587.0
PRODUCTION ('000s)	Bench Sill Tonnes														0
	Bench Tonnes														0
TOTAL BENCH PRODUCTION			0	0	0	0	0	0	0	0	0	0	0	0	0
	OS Sill Tonnes														0
	OS Tonnes														0
TOTAL OPEN STOPE PRODUCTION			0	0	0	0	0	0	0	0	0	0	0	0	0
PERIOD PRODUCTION															
		UNIT RATES													
Capital Development Cost		$2,712	$642,666	$642,666	$621,935	$631,708	$594,343	$524,747	$602,488	$573,935	$307,465	$200,688	$200,688	$542,400	$6,085,728
Operating Development Cost		$2,667	$0	$0	$0	$0	$0	$0	$0	$0	$311,836	$426,237	$560,756	$266,700	$1,565,529
OVER-ALL Bench Cost		$23.10	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
OVER-ALL OS Cost		$18.03	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
OWNER MANAGEMENT	Service Maintenance	$3.62	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Mine Management	$3.81	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
SITE ESTABLISHMENT			$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
VENT RAISES			$0	$0	$0	$231,000	$159,000	$71,333	$0	$0	$137,000	$71,333	$0	$0	$669,666
FIXED PLANT			$0	$0	$0	$0	$255,000	$0	$0	$0	$0	$0	$0	$0	$255,000
MOBILE EQUIPMENT			$120,000	$270,000	$350,000	$90,000	$0	$6,000	$180,000	$0	$0	$0	$0	$173,000	$1,189,000
CAPITAL DEVELOPMENT			$642,666	$642,666	$621,935	$631,708	$594,343	$524,747	$602,488	$573,935	$307,465	$200,688	$200,688	$542,400	$6,085,728
OTHER CAPITAL			$120,000	$270,000	$350,000	$321,000	$414,000	$77,333	$180,000	$0	$137,000	$71,333	$0	$173,000	$2,113,666
OPERATING EXPENDITURE			$0	$0	$0	$0	$0	$0	$0	$0	$311,836	$426,237	$560,756	$266,700	$1,565,529
PERIOD TOTAL			$762,666	$912,666	$971,935	$952,708	$1,008,343	$602,080	$782,488	$573,935	$756,301	$698,258	$761,444	$982,100	$9,764,923

			Year 2 Period1	Year 2 Period2	Year 2 Period3	Year 2 Period4	Year 2 Period5	Year 2 Period6	Year 2 Period7	Year 2 Period8	Year 2 Period9	Year 2 Period10	Year 2 Period11	Year 2 Period12	Year Total
DEVELOPMENT (m)	Capital		165.0	60.0	0.0	0.0	0.0	0.0	0.0	27.4	27.4	27.4	57.5	60.3	425.0
	Operating		135.5	205.5	76.0	76.0	36.7	80.7	36.7	0.0	65.9	92.2	92.2	96.6	994.0
PRODUCTION ('000s)	Bench Sill Tonnes		11.6	10.5	7.1	5.9	8.0	4.6	9.7	9.0	5.5	2.4	0.0	0	74
	Bench Tonnes														0
	TOTAL BENCH PRODUCTION		11.6	10.5	7.1	5.9	8.0	4.6	9.7	9.0	5.5	2.4	0.0	0.0	74.4
	OS Sill Tonnes		10.6	9.6	6.5	5.4	7.4	4.2	8.8	8.2	5.1	2.2	0.0	0	68
	OS Tonnes									8	25	25	25	25	108
	TOTAL OPEN STOPE PRODUCTION		10.6	9.6	6.5	5.4	7.4	4.2	8.8	16.2	30.1	27.2	25.0	25.0	176.0
	PERIOD PRODUCTION		22.1	20.2	13.6	11.3	15.4	8.9	18.5	25.3	35.6	29.6	25.0	25.0	250.4
		UNIT RATES													
	Capital Development Cost	$2,712	$447,480	$162,720	$0	$0	$0	$0	$0	$74,301	$74,301	$74,301	$156,033	$163,463	$1,152,600
	Operating Development Cost	$2,667	$361,379	$548,069	$202,692	$202,692	$97,790	$215,138	$97,790	$0	$175,718	$246,005	$246,005	$257,720	$2,650,998
	OVER-ALL Bench Cost	$23.10	$266,838	$243,635	$163,898	$136,379	$185,846	$106,913	$223,555	$208,281	$127,751	$55,544	$0	$0	$1,718,640
	OVER-ALL OS Cost	$18.03	$190,357	$173,804	$116,921	$97,290	$132,578	$76,269	$159,479	$292,823	$541,885	$490,374	$450,750	$450,750	$3,173,280
OWNER MANAGEMENT	Service Maintenance	$3.62	$80,035	$73,076	$49,160	$40,906	$55,742	$32,067	$67,053	$91,432	$128,817	$107,160	$90,500	$90,500	$906,448
	Mine Management	$3.81	$84,236	$76,911	$51,740	$43,053	$58,668	$33,750	$70,572	$96,231	$135,579	$112,784	$95,250	$95,250	$954,024
	SITE ESTABLISHMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$44,400	$43,200	$0	$87,600
	VENT RAISES		$256,000	$134,000	$75,333	$4,000	$188,000	$224,000	$4,000	$0	$0	$0	$0	$0	$885,333
	FIXED PLANT		$0	$0	$0	$180,000	$0	$127,500	$0	$0	$0	$0	$0	$0	$307,500
	MOBILE EQUIPMENT		$14,000	$0	$14,000	$20,000	$0	$0	$0	$0	$0	$0	$0	$0	$48,000
	CAPITAL DEVELOPMENT		$447,480	$162,720	$0	$0	$0	$0	$0	$74,301	$74,301	$74,301	$156,033	$163,463	$1,152,600
	OTHER CAPITAL		$270,000	$134,000	$89,333	$204,000	$188,000	$351,500	$4,000	$0	$0	$44,400	$43,200	$0	$1,328,433
	OPERATING EXPENDITURE		$982,845	$1,115,495	$584,411	$520,319	$530,624	$464,138	$618,449	$688,767	$1,109,750	$1,011,867	$882,505	$894,220	$9,403,390
	PERIOD TOTAL		$1,700,325	$1,412,215	$673,744	$724,319	$718,624	$815,638	$622,449	$763,068	$1,184,051	$1,130,568	$1,081,738	$1,057,683	$11,884,423

			Year 3 Period1	Year 3 Period2	Year 3 Period3	Year 3 Period4	Year 3 Period5	Year 3 Period6	Year 3 Period7	Year 3 Period8	Year 3 Period9	Year 3 Period10	Year 3 Period11	Year 3 Period12	Year Total
DEVELOPMENT (m)	Capital		92.4	84.4	76.1	78.9	75.4	78.9	52.9	0.0	0.0	0.0	0.0	0.0	539.0
	Operating		0.0	0.0	22.8	22.8	57.4	60.1	49.5	23.5	0.0	50.5	50.5	52.9	390.0
PRODUCTION ('000s)	Bench Sill Tonnes		4.7	4.3	4.5	4.5	0.0	0.0	2.5	8.7	12.8	7.4	7.1	7.4	63.8
	Bench Tonnes						12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	99.1
	TOTAL BENCH PRODUCTION		4.7	4.3	4.5	4.5	12.4	12.4	14.9	21.1	25.2	19.8	19.5	19.8	162.9
	OS Sill Tonnes		2.5	2.3	2.4	2.4	0.0	0.0	1.4	4.7	7.0	4.0	3.9	4.1	34.8
	OS Tonnes		15	20	20	15	10	10	10	2.3					102.3
	TOTAL OPEN STOPE PRODUCTION		17.5	22.3	22.4	17.4	10.0	10.0	11.4	7.0	7.0	4.0	3.9	4.1	137.1
	PERIOD PRODUCTION		22.2	26.6	26.9	21.9	22.4	22.4	26.3	28.1	32.1	23.8	23.4	23.9	300.0
		UNIT RATES													
	Capital Development Cost	$2,712	$250,659	$228,863	$206,295	$214,088	$204,357	$214,088	$143,417	$0	$0	$0	$0	$0	$1,461,768
	Operating Development Cost	$2,667	$0	$0	$60,697	$60,697	$153,020	$160,307	$131,887	$62,804	$0	$134,767	$134,767	$141,184	$1,040,130
	OVER-ALL Bench Cost	$23.10	$107,968	$98,579	$103,274	$103,274	$286,440	$286,440	$344,725	$487,088	$581,167	$457,480	$450,526	$458,340	$3,765,300
	OVER-ALL OS Cost	$18.03	$316,416	$402,569	$404,567	$314,417	$180,300	$180,300	$205,114	$126,893	$125,476	$72,818	$69,858	$73,184	$2,471,913
OWNER MANAGEMENT	Service Maintenance	$3.62	$80,449	$96,275	$97,412	$79,312	$81,088	$81,088	$95,204	$101,809	$116,267	$86,312	$84,628	$86,520	$1,086,362
	Mine Management	$3.81	$84,671	$101,328	$102,524	$83,474	$85,344	$85,344	$100,201	$107,152	$122,370	$90,842	$89,069	$91,061	$1,143,381
	SITE ESTABLISHMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	VENT RAISES		$0	$0	$0	$0	$0	$184,000	$0	$0	$0	$0	$0	$0	$184,000
	FIXED PLANT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	MOBILE EQUIPMENT		$14,000	$0	$0	$0	$0	$40,000	$0	$0	$0	$0	$0	$0	$54,000
	CAPITAL DEVELOPMENT		$250,659	$228,863	$206,295	$214,088	$204,357	$214,088	$143,417	$0	$0	$0	$0	$0	$1,461,768
	OTHER CAPITAL		$14,000	$0	$0	$0	$0	$224,000	$0	$0	$0	$0	$0	$0	$238,000
	OPERATING EXPENDITURE		$589,503	$698,751	$768,474	$641,174	$786,192	$793,479	$877,131	$885,745	$945,280	$842,220	$828,848	$850,289	$9,507,086
	PERIOD TOTAL		$854,162	$927,613	$974,769	$855,263	$990,549	$1,231,567	$1,020,548	$885,745	$945,280	$842,220	$828,848	$850,289	$11,206,854

			Year 4 Period1	Year 4 Period2	Year 4 Period3	Year 4 Period4	Year 4 Period5	Year 4 Period6	Year 4 Period7	Year 4 Period8	Year 4 Period9	Year 4 Period10	Year 4 Period11	Year 4 Period12	Year Total
DEVELOPMENT (m)	Capital		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Operating		165.4	151.0	158.2	134.9	128.7	63.8	95.8	91.3	104.9	0.0	34.2	35.8	1,164.0
PRODUCTION ('000s)	Bench Sill Tonnes		0.0	0.0	0.0	0.0	0.0	4.1	3.9	3.7	4.3	11.4	11.4	7.9	46.8
	Bench Tonnes		10	10	10	10	10	10	10	10	10	10	10	9.4	119.4
	TOTAL BENCH PRODUCTION		10.0	10.0	10.0	10.0	10.0	14.1	13.9	13.7	14.3	21.4	21.4	17.3	166.2
	OS Sill Tonnes		0.0	0.0	0.0	0.0	0.0	3.4	3.2	3.1	3.6	9.5	9.5	6.5	38.8
	OS Tonnes		15	15	15	15	15	10	10						95.0
	TOTAL OPEN STOPE PRODUCTION		15.0	15.0	15.0	15.0	15.0	13.4	13.2	3.1	3.6	9.5	9.5	6.5	133.8
	PERIOD PRODUCTION		25.0	25.0	25.0	25.0	25.0	27.5	27.2	16.8	17.9	30.9	30.9	23.8	300.0
		UNIT RATES													
	Capital Development Cost	$2,712	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Operating Development Cost	$2,667	$441,111	$402,754	$421,932	$359,702	$343,352	$170,082	$255,532	$243,364	$279,868	$0	$91,174	$95,516	$3,104,388
	OVER-ALL Bench Cost	$23.10	$231,000	$231,000	$231,000	$231,000	$231,000	$325,859	$321,547	$317,236	$330,171	$495,182	$495,182	$399,043	$3,839,220
	OVER-ALL OS Cost	$18.03	$270,450	$270,450	$270,450	$270,450	$270,450	$241,683	$238,893	$55,803	$64,173	$170,951	$170,951	$117,709	$2,412,414
OWNER MANAGEMENT	Service Maintenance	$3.62	$90,500	$90,500	$90,500	$90,500	$90,500	$99,590	$98,354	$60,918	$64,626	$111,923	$111,923	$86,167	$1,086,000
	Mine Management	$3.81	$95,250	$95,250	$95,250	$95,250	$95,250	$104,817	$103,516	$64,115	$68,018	$117,797	$117,797	$90,690	$1,143,000
	SITE ESTABLISHMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	VENT RAISES		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	FIXED PLANT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	MOBILE EQUIPMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	CAPITAL DEVELOPMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	OTHER CAPITAL		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	OPERATING EXPENDITURE		$1,128,311	$1,089,954	$1,109,132	$1,046,902	$1,030,552	$942,031	$1,017,842	$741,435	$806,856	$895,854	$987,028	$789,124	$11,585,022
	PERIOD TOTAL		$1,128,311	$1,089,954	$1,109,132	$1,046,902	$1,030,552	$942,031	$1,017,842	$741,435	$806,856	$895,854	$987,028	$789,124	$11,585,022

			Year 5 Period1	Year 5 Period2	Year 5 Period3	Year 5 Period4	Year 5 Period5	Year 5 Period6	Year 5 Period7	Year 5 Period8	Year 5 Period9	Year 5 Period10	Year 5 Period11	Year 5 Period12	Year Total
DEVELOPMENT (m)	Capital		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Operating		47.0	43.0	56.8	81.3	82.6	112.7	94.8	99.5	70.9	64.8	96.5	101.1	951.0
PRODUCTION ('000s)	Bench Sill Tonnes		6.6	6.0	6.3	2.9	2.9	3.0	4.7	1.7	6.2	5.0	4.6	4.8	54.8
	Bench Tonnes		3	5	5	10	10	10	5	10		10	10	5.1	83.1
	TOTAL BENCH PRODUCTION		9.6	11.0	11.3	12.9	12.9	13.0	9.7	11.7	6.2	15.0	14.6	9.9	137.9
	OS Sill Tonnes		4.7	4.3	4.5	2.0	2.0	2.1	3.3	1.2	4.4	3.6	3.2	3.4	38.8
	OS Tonnes		10	10	10	10	10	10	10	10	10	10	10	13.3	123.3
	TOTAL OPEN STOPE PRODUCTION		14.7	14.3	14.5	12.0	12.0	12.1	13.3	11.2	14.4	13.6	13.2	16.7	162.1
	PERIOD PRODUCTION		24.3	25.3	25.8	24.9	24.9	25.2	23.0	23.0	20.6	28.6	27.8	26.6	300.0
		UNIT RATES													
	Capital Development Cost	$2,712	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Operating Development Cost	$2,667	$125,470	$114,560	$151,461	$216,790	$220,392	$300,610	$252,734	$265,374	$189,174	$172,724	$257,386	$269,642	$2,536,317
	OVER-ALL Bench Cost	$23.10	$222,125	$255,035	$261,680	$297,546	$297,546	$300,715	$224,045	$270,999	$142,927	$347,067	$336,974	$228,830	$3,185,490
	OVER-ALL OS Cost	$18.03	$264,755	$257,412	$261,084	$217,076	$217,076	$218,827	$240,285	$202,405	$259,286	$244,442	$238,864	$301,152	$2,922,663
OWNER MANAGEMENT	Service Maintenance	$3.62	$87,966	$91,649	$93,427	$90,212	$90,212	$91,060	$83,354	$83,106	$74,456	$103,467	$100,766	$96,324	$1,086,000
	Mine Management	$3.81	$92,583	$96,459	$98,331	$94,947	$94,947	$95,840	$87,729	$87,468	$78,364	$108,898	$106,054	$101,380	$1,143,000
	SITE ESTABLISHMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	VENT RAISES		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	FIXED PLANT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	MOBILE EQUIPMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	CAPITAL DEVELOPMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	OTHER CAPITAL		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	OPERATING EXPENDITURE		$792,899	$815,115	$865,982	$916,572	$920,174	$1,007,053	$888,147	$909,352	$744,207	$976,597	$1,040,044	$997,329	$10,873,470
	PERIOD TOTAL		$792,899	$815,115	$865,982	$916,572	$920,174	$1,007,053	$888,147	$909,352	$744,207	$976,597	$1,040,044	$997,329	$10,873,470

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			Year 6 Period1	Year 6 Period2	Year 6 Period3	Year 6 Period4	Year 6 Period5	Year 6 Period6	Year 6 Period7	Year 6 Period8	Year 6 Period9	Year 6 Period10	Year 6 Period11	Year 6 Period12	Year Total
DEVELOPMENT (m)	Capital		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Operating		22.7	20.7	21.7	47.7	45.6	47.7	45.6	68.4	28.8	26.3	0.0	0.0	375.0
PRODUCTION ('000s)	Bench Sill Tonnes		5.8	5.3	5.6	5.6	5.3	5.6	2.7	2.6	7.3	4.0	7.0	7.3	64.2
	Bench Tonnes		10	10	10	10	10	10	10	10	10	10	10	15	125.0
	TOTAL BENCH PRODUCTION		15.8	15.3	15.6	15.6	15.3	15.6	12.7	12.6	17.3	14.0	17.0	22.3	189.2
	OS Sill Tonnes		2.1	1.9	2.0	2.0	1.9	2.0	1.0	0.9	2.6	1.4	2.5	2.6	22.8
	OS Tonnes		10	10	10	10	10	10	10	10		8			88.0
	TOTAL OPEN STOPE PRODUCTION		12.1	11.9	12.0	12.0	11.9	12.0	11.0	10.9	2.6	9.4	2.5	2.6	110.8
	PERIOD PRODUCTION		27.9	27.2	27.6	27.6	27.2	27.6	23.7	23.5	19.9	23.4	19.5	24.9	300.0
		UNIT RATES													
	Capital Development Cost	$2,712	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Operating Development Cost	$2,667	$60,412	$55,158	$57,785	$127,312	$121,525	$127,312	$121,525	$182,413	$76,676	$70,009	$0	$0	$1,000,125
	OVER-ALL Bench Cost	$23.10	$366,133	$354,382	$360,258	$360,258	$354,382	$360,258	$293,542	$290,564	$399,833	$322,610	$392,553	$515,746	$4,370,520
	OVER-ALL OS Cost	$18.03	$217,758	$214,501	$216,129	$216,129	$214,501	$216,129	$197,636	$196,811	$46,800	$169,634	$44,781	$46,914	$1,997,724
OWNER MANAGEMENT	Service Maintenance	$3.62	$101,097	$98,602	$99,850	$99,850	$98,602	$99,850	$85,682	$85,049	$72,054	$84,615	$70,508	$90,242	$1,086,000
	Mine Management	$3.81	$106,404	$103,777	$105,090	$105,090	$103,777	$105,090	$90,179	$89,513	$75,836	$89,056	$74,209	$94,978	$1,143,000
	SITE ESTABLISHMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	VENT RAISES		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	FIXED PLANT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	MOBILE EQUIPMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	CAPITAL DEVELOPMENT		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	OTHER CAPITAL		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	OPERATING EXPENDITURE		$851,804	$826,421	$839,112	$908,639	$892,787	$908,639	$788,564	$844,350	$671,199	$735,922	$582,051	$747,880	$9,597,369
	PERIOD TOTAL		$851,804	$826,421	$839,112	$908,639	$892,787	$908,639	$788,564	$844,350	$671,199	$735,922	$582,051	$747,880	$9,597,369

661C2099

			Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
			Year Total	Year Total	Year Total	Year Total	Year Total	Year Total
DEVELOPMENT (m)	Capital							
	Operating							
PRODUCTION ('000s)	Bench Sill Tonnes		0.0	0.0	0.0	0.0	0.0	0.0
	Bench Tonnes		117.2	154.7	0.0	0.0	0.0	0.0
	TOTAL BENCH PRODUCTION		117.2	154.7	0.0	0.0	0.0	0.0
	OS Sill Tonnes		18.0	0.0	0.0	0.0	0.0	0.0
	OS Tonnes		164.8	145.3	300.0	300.0	300.0	78.3
	TOTAL OPEN STOPE PRODUCTION		182.8	145.3	300.0	300.0	300.0	78.3
	PERIOD PRODUCTION		300.0	300.0	300.0	300.0	300.0	78.3
		UNIT RATES						
	Capital Development Cost	$2,712	$0	$0	$0	$0	$0	$0
	Operating Development Cost	$2,667	$0	$0	$0	$0	$0	$0
	OVER-ALL Bench Cost	$23.10	$2,706,858	$3,572,554	$0	$0	$0	$0
	OVER-ALL OS Cost	$18.03	$3,295,884	$2,619,759	$5,409,000	$5,409,000	$5,409,000	$1,411,749
OWNER MANAGEMENT	Service Maintenance	$3.62	$1,085,928	$1,085,841	$1,086,000	$1,086,000	$1,086,000	$283,446
	Mine Management	$3.81	$1,142,924	$1,142,832	$1,143,000	$1,143,000	$1,143,000	$298,323
	SITE ESTABLISHMENT		$0	$0	$0	$0	$0	$0
	VENT RAISES		$0	$0	$0	$0	$0	$0
	FIXED PLANT		$0	$0	$0	$0	$0	$0
	MOBILE EQUIPMENT		$0	$0	$0	$0	$0	$0
	CAPITAL DEVELOPMENT		$0	$0	$0	$0	$0	$0
	OTHER CAPITAL		$0	$0	$0	$0	$0	$0
	OPERATING EXPENDITURE		$8,231,593	$8,420,986	$7,638,000	$7,638,000	$7,638,000	$1,993,518
	PERIOD TOTAL		$8,231,593	$8,420,986	$7,638,000	$7,638,000	$7,638,000	$1,993,518

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MINERAL RESOURCES
14 FEB 2001
TASMANIA

AVEBURY NICKEL PROJECT
METALLURGICAL SCOPING STUDY

FOR

MINERAL RESOURCES
FILE REF: EL28/88 PT5.
16 FEB 2001
DOC REF:
OFFICER | FOR ACTION | FOR INFO
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RESUBMIT TO | DATE

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01_4526 (I)
Vol 1 of 2
Scoping Study - Avebury Nickel Project - Summary Report - EL28/1988
Allegiance Mining NL*; Newmham Exploration and Mini Newmham, L.A.
EL28/1988

REPORT 00900
G. P. SHELDON
DECEMBER 2000

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SUMMARY

A scoping study metallurgical test program was completed on drill core samples from the Avebury nickel prospect.

The drill core was from primary ore, containing nickel mainly as coarse grained pentlandite, with some finer grains associated with magnetite. Pentlandite was the dominant sulphide.

The drill core used for most of the testwork was whole core from holes A018A, A019A and A014A. Composites were prepared from these samples representing the total ore supplied from each hole, and combinations of the core from the holes based on the weight of ore received from each hole. The composites and head assays were :

Composite Head Assays

Composite Name	Ni %	Co ppm	Fe %	S %
Hole Composites				
A018A WC	1.35	310	16.0	1.25
A019A WC	3.48	840	15.1	4.19
A014A WC	2.04	413	8.87	3.38
Mixed hole composites				
18/19 WC composite	2.63	615	15.2	3.18
14/18/19WC composite	2.60	598	13.3	3.25

A Bond Ball Mill Work Index test was completed on a composite of equal weights of core from Hole A018A and A019A. The work index result was 18.4 kWh/tonne.

The ore was not amenable to upgrading by heavy media preconcentration before grinding.

Magnetic separation tests on ore ground to 80% passing 75µm indicated a high content of magnetite. Tests were completed with the A018A WC and A019A WC hole composites. However there was a significant content of nickel recovered to these magnetic products, and hence could not be discarded without incurring significant nickel losses. Results are summarised below :

Composite	% Wt	Assays of Magnetics			Metal Distrib. to Mags (%)		
	Mags	Ni %	Co ppm	Fe %	Ni	Co	Fe
A018A WC	30.6	1.50	395	43.7	34.5	37.6	81.6
A019A WC	16.8	2.15	555	47.3	11.3	12.0	54.4

The ore was very amenable to conventional flotation processes. Using the 14/18/19WC composite, an open circuit batch test (test 31) produced a concentrate grading 22.6% Ni, 5090 ppm Co with recoveries of 88.6% nickel and 87.1% cobalt. This test included grinding to 80% passing 75µm and two stages of cleaning of the rougher concentrate. Higher recoveries would be expected with closed circuit cleaning. A closed circuit cycle test (test 30) on the same composite which included regrinding of the rougher concentrate before

cleaning produced a final concentrate grading 20.4% Ni and 4670 ppm Co with recoveries of 93.3% nickel and 91.8% cobalt.

Multi element analysis was completed on the cleaner concentrates and final tails from a closed circuit cycle test. The only elements present in the concentrate likely to be of economic value were nickel and cobalt.

Comments on the flotation flowsheet and variables tested are as follows :

- Primary grinding to 80% passing 75µm produced higher nickel and cobalt recoveries than grinding to 80% passing 106µm, of the order of 2% to 4%.
- Copper sulphate additions of up to 400 g/t improved the nickel flotation rates and recoveries. It is possible smaller additions could be used.
- The collector sodium isobutyl xanthate (SIBX) was used for all the testwork for nickel flotation. Total additions were typically less than 90 g/t.
- Carboxy Methyl Cellulose (CMC) additions of 110 g/t to the cleaners may assist in rejection of slimes and gangue during cleaning, resulting in higher grade concentrates and with lower MgO levels.
- Regrinding of the rougher concentrate before cleaning was effective at reducing the magnetite content in the cleaner concentrates, most likely by improving liberation of the magnetite - pentlandite composites. With further development of this process option there is potential to improve concentrate grades without loss in recovery.

There is considerable scope to develop the flowsheet and reagent scheme, with optimisation of grinding, reagent dosages and flowsheet design with the potential benefits of improving grades and recoveries of nickel and cobalt to the concentrates.

1 INTRODUCTION

A preliminary metallurgical test program was undertaken on samples from the Avebury nickel deposit as part of a scoping study. The ore is of a primary nature, containing nickel as nickel sulphides. No metallurgical testwork had been completed on the deposit before.

Several samples of crushed drill core and whole core from three different holes were available for the testwork. The metallurgical testwork included

- a grinding work index on a composite sample,
- preliminary flotation tests to indicate a likely flotation flowsheet, concentrate grades and recoveries, and produce final concentrates
- some development of flotation conditions to indicate reagent requirements and grind fineness
- multi element analysis on the final concentrates and tails
- sizings on the final concentrate and tails

Mineralogical work on samples of drill core was completed prior to this metallurgical study. The dominant sulphide in the ore was pentlandite, occurring mainly as coarse grains >200µm and coarse crystals to 2mm, but also as 5µm to 100µm grains in magnetite. There were traces of other sulphides, namely pyrrhotite, pyrite, chalcopyrite and gersdorffite. Magnetite was also present in significant amounts.

2 SAMPLES RECEIVED AND PREPARATION

2.1 Samples Received

The samples were received in two separate consignments for testwork. The details are shown in Tables 1 and 2 .

Table 1 Samples Received on 8th August 2000

Hole No	Sample	From (m)	To (m)	Wt kg	Comment
A018A	1	213.9	217.0	15.7	Whole NQ Core
A018A	2	217.0	221.5	19.7	Whole NQ Core
A019A	1	240.5	244.4	21.9	Whole NQ Core
A019A	2	244.4	247.4	17.8	Whole NQ Core
A019A	3	247.4	250.4	17.6	Whole NQ Core
					CRUSHED CORE:
A018	-	215.4	216.4	1.2	nominally -12mm
A018	-	216.4	217.4	1.3	nominally -12mm
A018	-	218.2	219.2	1.4	nominally -12mm
A018	-	219.2	220.2	0.7	nominally -12mm
A018	-	220.2	221.2	1.0	nominally -12mm
					CRUSHED CORE
A019	-	243	244	1.0	nominally -12mm
A019	-	245	246	1.3	nominally -12mm
A019	-	246	247	1.5	nominally -12mm
A019	-	247	248	1.0	nominally -12mm
A019	-	248	249	1.4	nominally -12mm

The whole core was damp as received, and was dried overnight at 55°C. The crushed samples were dry.

Table 2 Samples Received on 7th September 2000

Hole No	Sample	From (m)	To (m)	Wt kg	Comment
A014A	-	254.7	267.3	65	Whole NQ core

Samples were dry when received.

2.2 Sample Preparation

Crushed Core Samples

The crushed samples received (five samples for each of the two holes) were combined in total to produce two composites based on the hole numbers. The composites were identified as follows :

A018 CC (from A018 -12mm crushed core samples received)

A019 CC (from A019 -12mm crushed core samples received)

These composites were crushed to -2mm and prepared by rotary riffling to provide 1kg portions for metallurgical testwork. Head assay samples were prepared from randomly selected 1kg portions. Head assays are shown in Table 4 in section 3.

These composites were used for some initial scouting metallurgical tests, before proceeding with tests with composites prepared from whole core.

Whole Core Samples from Holes A018A and A019A

For the initial sample preparation with the five whole NQ core samples listed in Table 1 all the samples were kept separate and crushed to -12.5 mm. A small portion was riffled out and prepared for assay. The assay results are shown in Table 4 in section 3. Once the assays were known the samples were prepared to produce composites for metallurgical testwork.

The whole core samples were individually crushed to -12.7mm and subdivided to provide 2 kg of each for combining into a composite for Bond Ball Mill Work Index testing, and remaining portions for preparation into composites for each hole for flotation and other metallurgical tests. The hole composites were identified as follows:

A018A WC (from the A018A whole core samples received)

A019A WC (from the A019A whole core samples received).

A 2kg portion at -12.5mm of each of the above hole composites was retained for possible future use.

Details of the weight portions split out and crush sizes etc are shown in Table 3. Portions were prepared into 1kg lots at -2mm for testwork and head assays, and for preparation of other composites (18/19WC composite).

Composite 18/19WC

A blended composite called 18/19WC composite was then prepared by combining weighted portions of the A018A WC and A019A WC composites together. The weighted portions were also based on the total weights of sample received from each hole. The composites prepared for flotation and associated testwork were crushed to -2mm and prepared by rotary riffling to provide 1kg portions for metallurgical testwork. Weights used for the sample and composite preparation are shown in Table 3. This composite was prepared for metallurgical testwork before the A014AWC core (see below) was available.

Whole Core Sample from Hole A014A

This sample was received only after the A018AWC and A019Awc composites were prepared and flotation testwork was well underway. The core from this hole was received as a bulk sample over the entire interval length (as detailed in Table 2 and not in separate drill length intervals). The total sample was crushed to -12.5 mm, and 20 kg was removed and stored at this size. The remainder was crushed to -2mm for composite preparation and rotary split into 1 kg bags, and a portion prepared for head assays. Another portion was used to prepare another composite (14/18/19 composite). Details on the quantities and portions prepared are shown in Table 3.

Composite 14/18/19WC

This composite was prepared by combining weighted portions of -2mm ore from each of the whole core hole composites. The weight ratio used for compositing was based on the original weights received from the total core from each of the holes. Details are shown in Table 3. A total of 24 kg of the new composite was prepared, and was split by rotary riffle into 24 x 1kg portions.

Table 3 Sample and Composite Preparation Weights and Details

Hole	Sample	Total Wt kg available	Wt kg for Bond WI -12.5mm	Wt kg for Hole Composite
A018A	1	15.7	2.0	13.7
A018A	2	19.7	2.0	17.7
A019A	1	21.9	2.0	19.9
A019A	2	17.8	2.0	15.8
A019A	3	17.6	2.0	15.6
	Total Sample for Bond WI Tests =		10.0	
	Total sample for Hole Composite A018A WC =			31.4
	Total sample for Hole Composite A019A WC =			51.3

Hole Composite	Total kg Available	Preserved kg @ -12.5mm	kg @ -2mm for testwork	kg @ -2mm for 18/19WC comp
A018A WC	31.4	2.0	19.9	9.1
A019A WC	51.3	2.0	33.8	14.9
	Total wt kf of 18/19WC composite (@ -2mm) =			24.0

Hole	Sample	Total Wt kg available	Stored kg @ -12.5mm	Wt kg for -2mm : Hole Composite
A014A	all	65	20.0	45.0

Preparation of 14/18/19 WC Composite :			
	Original total wts received		
	kg per hole	% of total core	Wt kg for comp.
kg of A018A WC -2mm ore	35.4	22.4	7.0
kg of A019A WC -2mm ore	57.3	36.3	11.3
kg of A014A WC -2mm ore	65.0	41.2	13.0
Total of all core	157.7	99.9	
Total wt 14/18/19 WC comp.			31.3

3 HEAD ASSAYS

Head assays were completed on all the composites. The samples submitted for head assays were prepared by hand riffling approximately 200g of sample from a randomly selected 1kg portion of -2mm ore. The sample was pulverised at Metcon and submitted to Analabs Laboratories. Head assays are shown in Table 4. All assays were by AAS, except for S which was by Leco furnace. Throughout the testwork all routine assays from testwork were submitted to Analabs. Specific gravity (SG) determinations were also carried out on separate portions of -2mm ore, using a pycnometer. The SG determinations are true values of the ore and are not bulk density determinations. The SG values are also shown in Table 4.

The 18/19WC composite was also submitted for a multi element ICP scan. The scan was completed by ALS in Brisbane. The ICP scan results are shown in Table 5.

Table 4 Head Assays and Specific Gravity Results

Hole	Sample Type	Ni %	Ni % duplicate	Ni % average	Fe %	S %	Mg %	Cu ppm	Co ppm	SG
A018CC	composite	1.21	1.18	**1.20**	18.1	0.92		30	255	-
A019CC	composite	2.55	2.55	**2.55**	13.7	2.81		125	595	-
A018AWC	sample 1	1.43	1.42	**1.43**	17	1.38		48	270	-
A018AWC	sample 2	1.30	1.27	**1.29**	15.7	0.968		32	365	-
A019A WC	sample 1	3.10	3.10	**3.10**	13.8	4.34		340	805	-
A019A WC	sample 2	3.55	3.45	**3.50**	15.8	4.24		200	845	-
A019A WC	sample 3	3.45	3.60	**3.53**	15.2	4.06		220	825	-
A018A WC	composite	1.31	1.39	**1.35**	16.0	1.25		-	310	2.93
A019A WC	composite	3.45	3.50	**3.48**	15.1	4.19		-	840	3.24
A014A WC	composite	1.96	1.92	**1.94**	9.00	3.34	14.2	195	395	2.92
A014A WC	resample *	2.25	2.15	**2.20**	8.80	3.35	-	210	445	-
A014A WC	repeat	1.98	2.00	**1.99**	8.80	3.46	-	180	400	
		overall average		**2.04**	**8.87**	**3.38**	**14.2**	**195**	**413**	
18/19 WC	composite	2.60	2.65	**2.63**	15.2	3.18	-	-	615	3.10
14/18/19WC	composite	2.90	2.60	**2.75**	13.3	3.26	-	237	638	3.01
	repeat	2.45	2.45	**2.45**	13.2	3.24	-	203	557	
		overall average		**2.60**	13.3	3.25	-	220	598	

* Repeat assays on the A014A WC composite were carried out on another sub sample riffled from another 1 kg portion of -2mm ore and prepared for the repeat analysis.

Table 5 ICP Scan Results on the 18./19 WC Composite

Ag	ppm	<1
Al	%	0.6
As	ppm	660
Ba	ppm	36
Bi	ppm	32
Ca	%	2.32
Cd	ppm	<5
Cr	ppm	491
K	%	0.27
Mg	%	16.3
Mn	%	0.26
Mo	ppm	<5
Na	%	0.02
P	%	0.02
Pb	ppm	28
Sb	ppm	<10
Si	%	16.8
Sr	ppm	<10
Ti	%	0.03
V	ppm	<10
Zn	ppm	80
Zr	ppm	11

4 TRIAL GRINDS

Trial grinds were completed with the whole core composites to determine the relationship between the laboratory grind times and the product fineness. All grinding for the trial grinds and throughout the testwork program was completed in a sealed stainless steel laboratory rod mill, using 1 kg of -2mm ore sample and Sydney tap water, at 60% solids.

Trial grinds were carried out for different times and the products sized. The relationship between grind time and fineness was plotted. From this data the grind time required to achieve a desired fineness could be determined. Details of all trial grinds are shown in Appendix 1. Summarised results with the grind times used in the testwork are shown in Table 6.

Table 6 Grind Times Used for Flotation Testwork

Composite	Grind time in minutes to achieve 80% passing :		
	45µm	75µm	106µm
A018A WC		35.5	29.0
A019A WC		21.0	16.0
18/19 WC	36.0	25.0	20.5
A014A WC		20.5	17.0
14/18/19 WC		25.0	18.5

5 BOND BALL MILL WORK INDEX

A Bond Ball Mill Work Index test was completed with the work index composite. This composite was prepared from whole core from holes A018A and A019A, as detailed in section 2.2, Table 3.

The test was completed following the standard Bond procedure. Details of the test are included in Appendix 2.

Results were :

Bond Ball Mill Work Index = 18.4 kWh/tonne, for a product of 80% passing 97μm.

6 HEAVY MEDIA PRECONCENTRATION TEST

One test was completed with the A019A WC composite to examine the potential of using a heavy medium preconcentration cyclone for rejection of some of the gangue, and consequently some upgrading of the nickel and cobalt values into a preconcentrate, before grinding and flotation. A heavy liquid separation test was completed with a crushed and screened sample of the composite to simulate the separations that would occur in a heavy media cyclone.

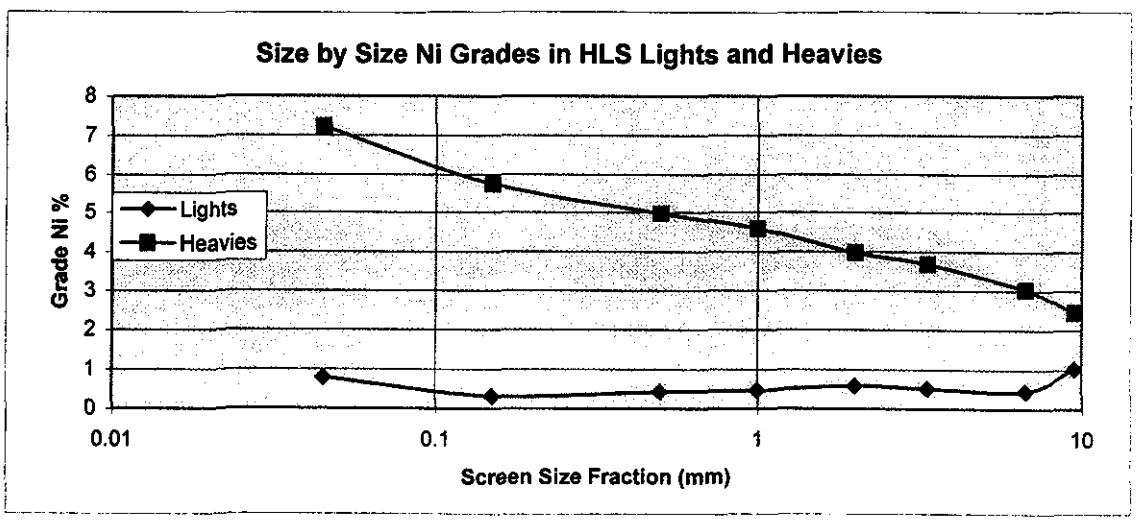
The sample used for the testwork was -12.7 mm (refer section 2.2 for sample preparation details), and the slimes were removed by wet screening on 45µm. The -12.7mm +45µm material was separated in tetrabromoethane (TBE) which has an SG of 2.92. The ore sample was added to the TBE in a 3 litre beaker, and mixed. The phases were allowed to separate, and re stirred gently to allow release of any entrained misplaced particles. The stirring and separation sequence was repeated several times. The separated ore and gangue phases were removed from the solution, washed with acetone and dried. The lights (gangue) and heavies (preconcentrate) were weighed, and dry screened over a range of screens, and the fractions assayed. Complete detailed results are shown in Appendix 3.

Summarised results are shown in Table 7 and Figure 1.

Table 7 Overall Results for the Heavy Liquid Separation Test

Product	Wt %	Assays				Distribution			
		Ni %	Co ppm	Fe %	S %	Ni	Co	Fe	S
Total Lights	9.3	0.60	156	7.2	0.70	1.8	1.9	4.6	1.6
Total Heavies	87.6	3.37	809	15.4	4.39	94.2	94.3	92.6	94.5
Slimes (-45µm)	3.1	4.05	900	13.0	5.06	4.0	3.7	2.8	3.9
Calc Slimes + Heavies	90.7	3.39	812.12	15.3	4.41	98.2	98.0	95.4	98.4
Calc Total	100.0	3.14	751	14.5	4.07	100.0	100.0	100.0	100.0
Actual head		3.48	840	15.1	4.19	-	-	-	-

Figure 1



5 cm

The results from the heavy liquid separation test show only a low weight recovery to the lights (rejects) of 9.3%. While the nickel content of the lights were low, the low weight recovery and low degree of nickel upgrading into the heavies (preconcentrate) suggests there would be little or no advantage in utilising a preconcentration step before grinding and flotation.

7 MAGNETIC SEPARATION RESULTS

Magnetic separations were carried out on various flotation products including flotation feed and concentrates. All magnetic separations were completed using a Davis Tube. The Davis Tube is a wet magnetic separator, and uses wash water to remove any non magnetics entrained in the magnetics, and hence produces very clean products.

Results from all magnetic separations are included in Appendix 4. Results from tests with flotation concentrates and tails are discussed in later sections of this report as they apply to the flotation tests.

Magnetic separation tests were also completed on ground ore (flotation feed) from composites A018CC, A018AWC, A019CC and A019AWC to determine the magnetic contents and the distributions of metals and sulphur in the products. It was considered that if a clean magnetic product could be produced with low nickel content, it may be advantageous to remove the magnetite from the circuit before flotation, thereby increasing the flotation feed grade for nickel and possibly assisting in selectivity during flotation.

Result in Table 8 show grades and recoveries to the magnetics, after testing at a moderate magnetic field strength of 0.8A (equivalent to approximately 4500 gauss).

Table 8 Results to the Magnetic Product of Davis Tube Tests with Ground Ore

Composite	Grind	% Wt	Assays						Recovery %					
	% -75µm		Ni %	Co pp	Fe %	S %	Cu ppm	Mg %	Ni	Co	Fe	S	Cu	Mg
A018CC	57	35.3	1.00	220	45.4	0.80	26	9.75	29.1	30.2	86.3	24.3	13.9	19.3
A018A WC	80	30.6	1.50	395	43.7	1.43	70	-	34.5	37.6	81.6	35.3	24.8	-
A019CC	91	14.8	1.12	270	54.4	1.08	225	6.20	6.2	6.4	57.6	5.3	21.6	5.4
A019A WC	80	16.8	2.15	555	47.3	3.54	415	-	11.3	12	54.4	14.1	23.4	-

The results show approximately twice as much magnetics (magnetite) in both the 018 composites than in the 019 composites.

The magnetics also contained a significant content of nickel and hence the magnetics could not be considered as tailings without incurring significant losses in nickel recovery. The nickel content of the magnetite also suggests some association of nickel with magnetite, possibly as unliberated grains of nickel sulphides with magnetite, for the grind finesses tested (shown in Table 8).

8 FLOTATION

8.1 Procedures and Summary of Flotation Results

All flotation tests were carried out using 1 kg portions of ground ore, in a Denver D12 laboratory flotation cell. For rougher / scavenger flotation the operating cell volume was nominally 2.5 litres. In tests where cleaning was carried out, the operating cell volume was nominally 1.3 litres.

In some tests the rougher concentrate was reground before cleaning. All regrinding was carried out using a small sealed stainless steel rod mill, with rod diameters of 5 to 10mm.

The fineness of the reground products were checked by laser sizing after the grind.

Reagents used in the testwork were :

- Sodium isobutyl xanthate (SIBX) as collector. No other collectors were tested.
- Methyl isobutyl Carbinol (MIBC) as frother.
- Copper sulphate (CuSO4 5H2O) as a sulphide activator.
- Carboxy Methyl Cellulose (CMC) as a gangue depressant / dispersant. The CMC grades used were Finfix BDA and Finfix 300, supplied in Australia by Henkel Australia.
- Dextrin for gangue depression
- Sodium silicate (Na2SiO3) as a slime dispersant. The Na2SiO3 was N Grade, supplied by PQ Australia.
- Products from all flotation tests were dried, weighed and prepared for assay at Metcon, then dispatched to Analabs for analysis.

Details of all flotation tests and results are shown in Appendix 5.

Summarised results from all batch flotation tests are shown in Tables 9 and 10.

Table 9 **Summarised Results From Staged Rougher Flotation Tests**

Test No	Grind P80 µm	Calculated Head Ni %	Co ppm	S %	Total Rougher Conc Grades Ni %	Co ppm	S %	Recoveries % Ni	Co	S	Ni Grades in Rougher Concentrates : @ 90% Ni Rec	@ 80% Ni Rec	@ 70% Ni Rec	@ 60% Ni Rec	CuSO4 Added	Comments / Flotation Conditions
A018CC Composite																
1	-	1.20	252	1.21	6.14	1247	6.19	37.6	36.3	37.5	-	-	-	-	no	Grind 57% -75µm. SIBX and MIBC only
3	-	1.24	264	1.51	5.0	997	5.14	62.2	58.6	52.7	-	-	-	-	no	Grind 93% -75µm. SIBX and MIBC only
A019CC Composite																
2	-	2.70	642	2.84	20.4	4435	20	78.8	72.1	73.4	-	-	22.8	24.5	no	Grind 91% -75µm. SIBX and MIBC only
A018A WC Composite																
4	75	1.34	331	1.34	6.51	1395	6.98	66.0	57.4	71.1	-	-	-	7.8	nil	Prefloat at start. SIBX only.
5	75	1.35	334	1.30	6.47	1462	6.62	75.1	68.5	79.6	-	-	6.5	7.3	scav	No prefloat. CMC to depress gangue.
18	75	1.40	340	1.29	5.8	1401	5.4	90.6	90.1	92.3	6.2	9.5	12.0	-	start	SIBX, no prefloat, no CMC
A019A WC Composite																
6	75	3.45	856	4.42	22.0	5025	26.1	80.0	73.8	73.6	-	22.0	24.0	26.6	nil	Prefloat at start. SIBX only.
7	75	3.50	867	4.59	19.8	4795	25.8	92.3	90.3	91.9	20.3	22.2	24.3	26.9	scav	No prefloat. CMC to depress gangue.
17	75	3.73	907	4.63	19.9	4741	24.7	94.6	92.8	94.8	23.4	25.4	-	-	start	SIBX, no prefloat, no CMC
A014A WC Composite																
16	75	2.64	519	3.46	15.9	3079	19.4	94.3	92.9	87.9	22.0	25.5	-	-	start	SIBX, no prefloat, no CMC
18/19 WC Composite																
8	75	2.62	648	3.17	15.5	3730	19.0	91.2	88.8	92.6	16.5	19.0	19.4	20.2	scav	Prefloat at start. SIBX only.
9	75	2.66	652	3.37	15.6	3745	20.3	92.8	90.8	95.2	17.3	18.8	20.0	20.2	scav	No prefloat. CMC to depress gangue.
10	75	2.64	651	3.44	14.7	3530	19.7	92.3	90.0	95.0	17.4	19.2	21.0	-	start	SIBX, no prefloat, no CMC
11	45	2.56	630	3.23	11.8	2839	15.2	94.3	92.4	96.4	15.7	17.6	19.6	-	start	As for test 10 at a finer grind
19	106	2.67	650	3.22	15.7	3717	19.2	92.0	89.7	93.5	18.1	20.4	22.5	-	start	As for test 10 at a coarser grind
14/18/19 WC Composite																
21	106	2.36	547	3.00	16.2	3662	20	91.4	89.3	88.9	17.5	20.2	21.7	-	start	As for test 10 at a coarser grind

Table 10 **Summarised Results From Cleaner Type Flotation Tests**

Test No	Grind P80 µm	Calculated Head Ni %	Co ppm	S %	Total Rougher Conc Grades Ni %	Co ppm	S %	Recoveries % Ni	Co	S	Ni Grade in Rghr / Clnr Concentrates : @ 90% Ni Rec	@ 80% Ni Rec	@ 70% Ni Rec	@ 60% Ni Rec	CuSO4 Added	Comments / Flotation Conditions
A018A WC Composite																
23	106	1.26	306	1.25	8	1884	8.3	84.7	82.3	88.9	-	18.5	24.8	25.5	start	Roughing as for test 18, at coarser grind
27	106	1.29	342	1.26	5.5	1424	5.8	84.4	82.9	91.5	-	10.0	21.0	21.2	start	As per test 23 with regrind of rougher conc
A019A WC Composite																
24	106	3.42	842	4.35	20.4	4932	25.9	91.7	89.9	91.8	24.0	25.7	25.8	-	start	Roughing as for test 17, at a coarser grind
28	106	3.64	976	4.58	19.0	5092	24.2	89.2	89.1	90.4	-	24.6	25.8	26.2	start	As per test 24 with regrind of rougher conc
A014A WC Composite																
25	106	2.05	411	3.17	13.9	2727	19.0	90.1	88.3	78.8	13.9	22.1	22.8	-	start	Roughing as for test 16, at a coarser grind
29	106	2.09	471	3.29	12.5	2838	19.3	88.4	88.8	86.3	-	20.6	22.3	23.0	start	As per test 25 with regrind of rougher conc
18/19 WC Composite																
20	75	2.75	667	2.97	18.3	4333	19.9	92.8	90.7	93.7	25.4	27.1	27.7	28.1	start	Roughing based on test 10.
14/18/19 WC Composite																
22	106	2.40	547	3.09	17.6	3955	21.9	88.6	86.6	87.5	18.2	24.0	24.8	25.3	start	Roughing based on test 21
26	106	2.33	591	3.12	14.6	3670	19.6	90.8	89.9	91.1	15.5	25.2	25.8	26.2	start	As per test 22 with regrind of rougher conc
31	75	2.48	569	3.13	13.1	2949	16.5	93.6	91.9	93.8	21.7	-	-	-	start	No regrind, but with CMC added to the cleaners

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8.2 Preliminary Rougher Tests with the Crushed Core Composites

Three preliminary rougher tests were completed with the two composites of the crushed core samples (A018CC and A019CC).

These tests involved a simple reagent system, using only SIBX collector and MIBC frother, at natural pH. These tests were simple scouting tests to observe the flotation response. Trial grinds were not completed with these composites, as they were not planned for use for the flotation development. Therefore, the grinds for flotation were estimates only, and sample of ground flotation feed was taken and screened on 75µm to indicate the grind fineness.

At the start of the flotation tests (before adding collector) it was noticed there was some free floating slime gangue, with an appearance of talc. This material was removed before adding collector, using a prefloat stage with frother only. There was a greater weight recovery of gangue in the prefloat from the A018CC composite than from the A019CC composite. Mg assays on the prefloat concentrate were reasonably high, between 15% and 20% Mg, possibly indicating the presence of talc. Mineralogical examinations would be needed for confirmation.

Nickel flotation was slow for the A018CC composite and only low grade concentrates were produced. Conversely, flotation was much faster for the A019CC composite, and high rougher concentrate grades were produced.

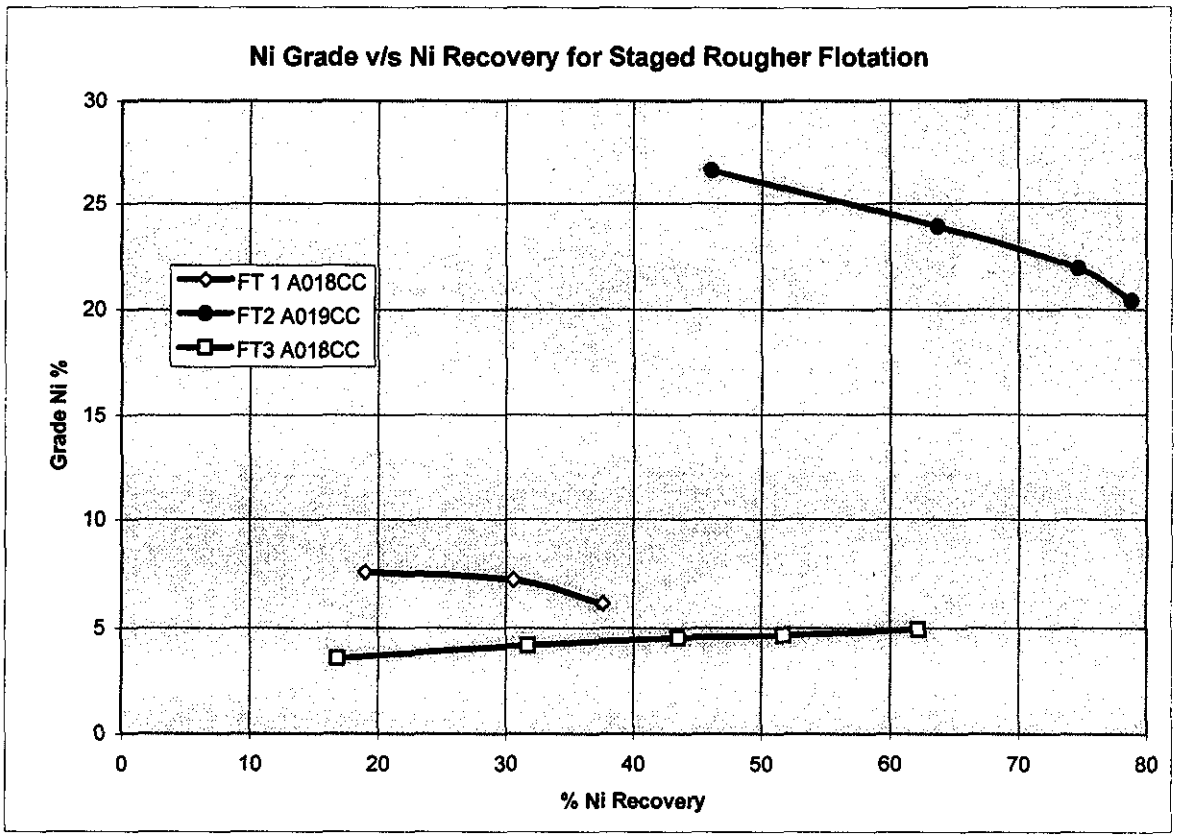
Tests 3 and 4 were with the A018CC composite, similar to test 1, except additional flotation stages with additional collector to attempt to increase the nickel recoveries, and grind times adjusted to be closer to the target of 80% passing 75µm.

Summarised results for preflotation are shown in Table 11, and nickel flotation grade recovery curves (after prefloation) are shown in Figure 2.

Table 11 Results for Preflotation

Test	Composite	Prefloat Concentrate Assays							Recoveries %					
		Wt %	Ni %	Co ppm	Fe %	S %	Cu ppm	Mg %	Ni	Co	Fe	S	Cu	Mg
1	A018CC	8.72	1.22	255	12.5	1.22	74	19.9	8.9	8.8	6.1	8.8	15.8	10.1
3	A018CC	12.1	1.17	245	14.5	1.14	56	19.3	11.4	11.2	9.3	9.1	14.9	13.0
2	A019CC	2.36	5.3	1120	10.3	5.32	705	15.3	4.6	4.1	1.7	4.4	11.6	2.2

Figure 2 Nickel Flotation from the Crushed Core Composites



8.3 Staged Rougher Flotation Tests A018AWC Composite

Three tests were completed (tests 4, 5 and 18), all with grinds to 80% passing 75µm. This grind fineness was selected as a starting point. Some mineralogical information was available on the mineral grain sizes, and had indicated most of the pentlandite was relatively coarse at >100µm, (from Central Mineralogical Report 00/7/8, Avebury Drill Holes A018, A019; which was completed before this metallurgical investigation). The mineralogical report also noted some 5 to 100µm grains of pentlandite in magnetite and gangue.

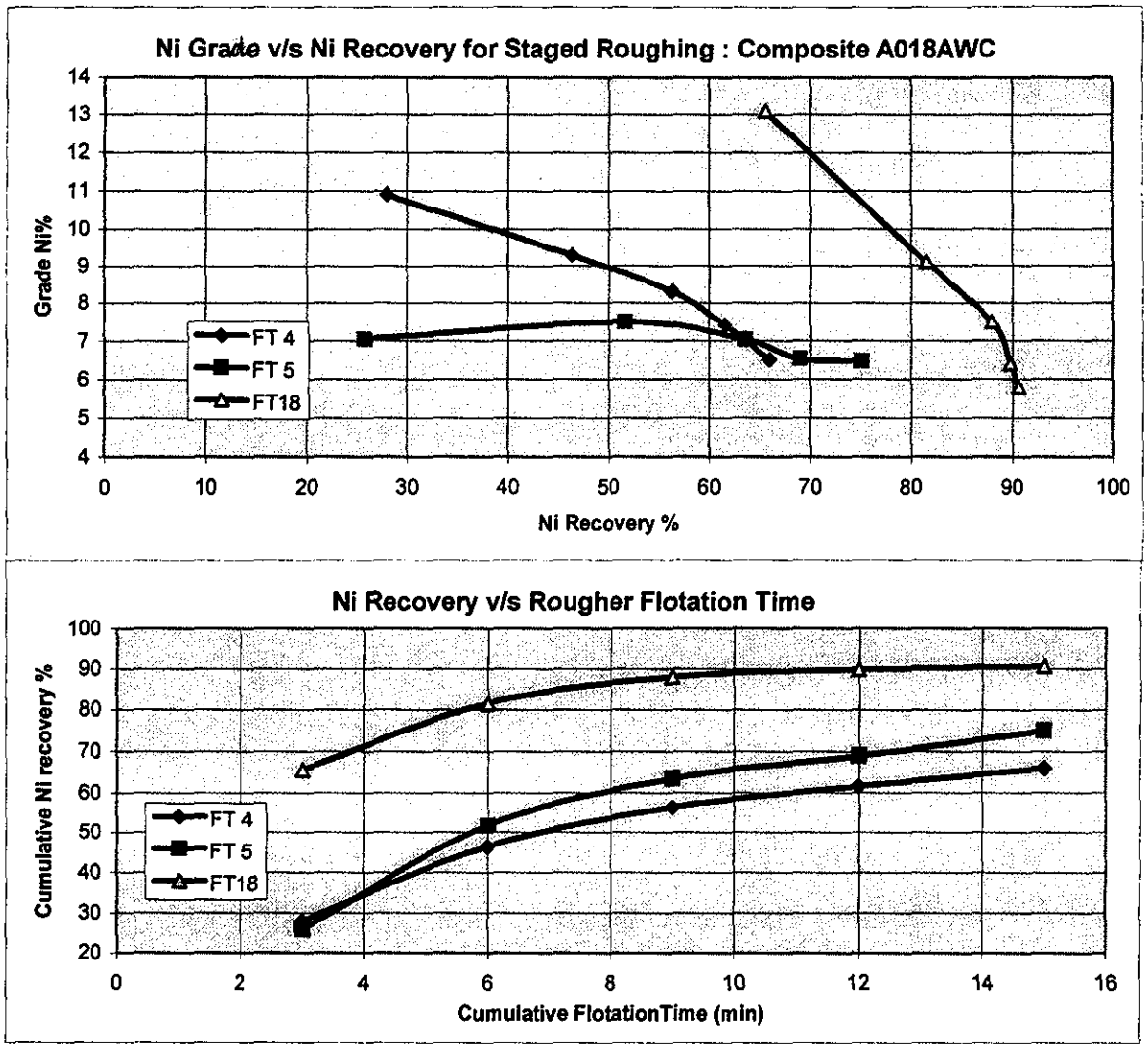
Flotation conditions for test 4 were initially based on tests 1 to 3 with the crushed core composites (section 8.2).

For test 5 the prefloat stage was deleted, as the losses of nickel to the prefloat concentrate were considered excessive. Rather than remove the gangue in the prefloat the approach was to attempt to depress it with CMC. The last scavenger stage included the addition of CuSO4, to examine if the activator would give an increase in the overall nickel recovery.

Test 18 was aimed at maximising nickel recovery. The prefloat was not included, and the CMC was deleted. Copper sulphate was added at the start of flotation to maximise activation and nickel flotation rates from the start.

Grade and recovery curves, and nickel recovery versus flotation time are shown in Figure 3.

Figure 3



Test 5, without the prefloat and with CMC gave improved overall nickel recovery, as the nickel otherwise recovered to the prefloat was now reporting to the rougher concentrate. However the CMC seemed to have little effect on depressing the gangue, particularly in the initial rougher stages, and nickel grades in the concentrates were low. The addition of CuSO4 to the last scavenger stage was effective, and there was a significant increase in sulphides in the froth. The grade in the concentrate with the CuSO4 was higher than in the preceding concentrate (5.85% Ni versus 3.60% Ni) confirming an increase in nickel flotation rate.

As the copper sulphate was effective at promoting nickel flotation it was added from the start in test 18. As the CMC seemed to be ineffective in test 5, it was not added in test 18. Test 18 gave a very significant improvement in flotation response, with big improvements in both nickel grade and recovery, and flotation rates. Final recoveries were 90.6% for nickel, 90.1% for cobalt and 92.3% for sulphur.

A small sample of flotation tails from test 4 were passed through the Davis Tube to recover magnetite. There was 31.3% weight recovery to the magnetics, with 40.2% nickel recovery (with respect to the nickel in the total tails), at a grade of 0.62% Ni, 46.2% Fe

and 0.47% S. This indicated that 40% of the nickel in the tails was associated with magnetite. Complete results from the magnetic separation test are shown in Appendix 4.

8.4 Staged Rougher Flotation Tests A019AWC Composite

Three tests were completed (tests 6, 7 and 17). Conditions were as for the staged rougher tests with A018A WC composite (refer section 8.3). These included :

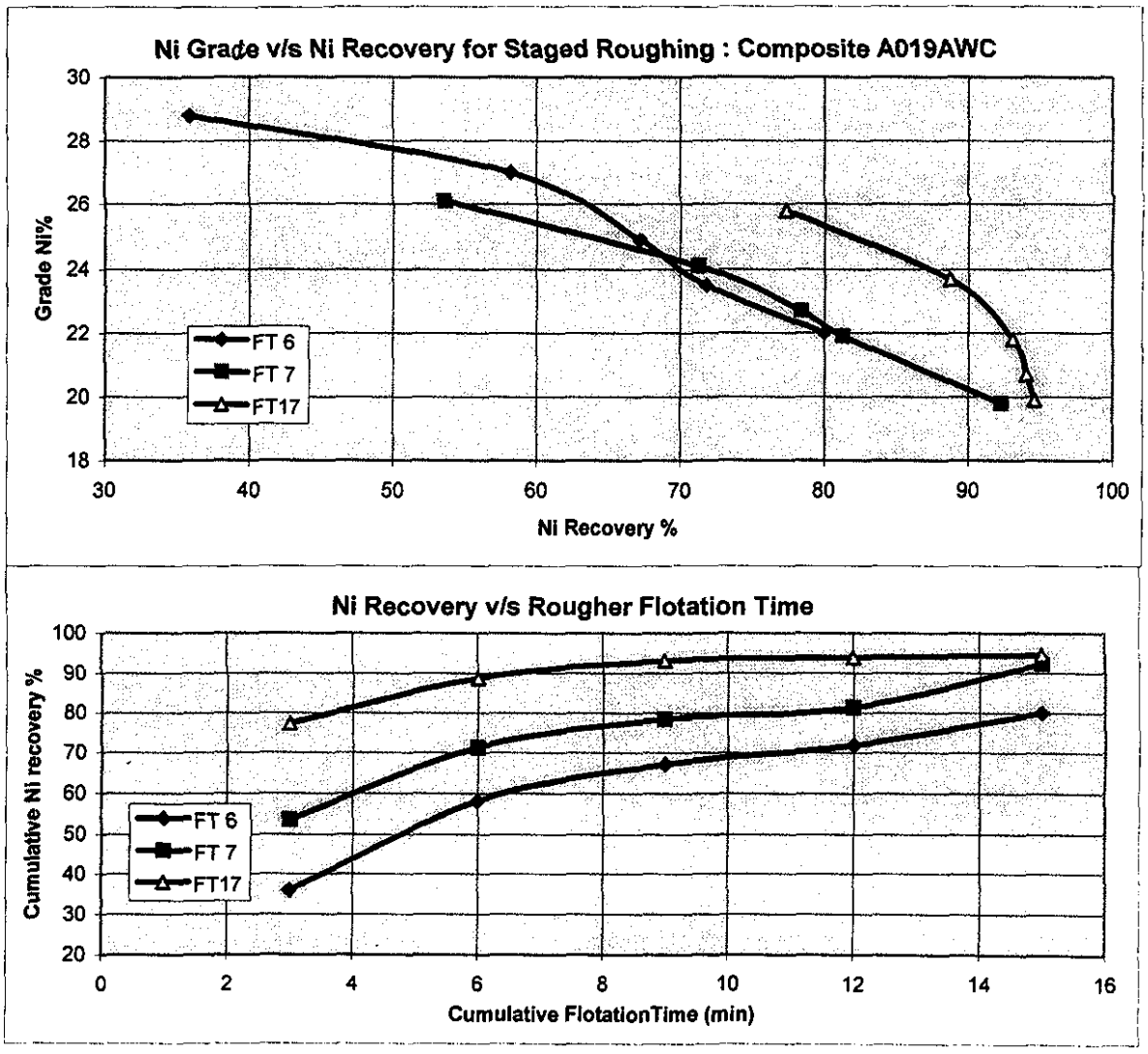
Test 6 with a prefloat, and without CuSO4 or CMC.

Test 7 without the prefloat, and with CMC andCuSO4 added to the last scavenger stage.

Test 17, without the prefloat or CMC, and CuSO4 added from the start.

Nickel flotation results are shown by the grade - recovery, and recovery versus time curves in Figure 4.

Figure 4



The effects of these test variables were as for the tests with A018A WC composite., although all the results were significantly better.

The deletion of the prefloat gave an improvement in nickel recovery. The CMC may have been of some benefit in depressing gangue, as the grades in the concentrate were not

adversely affected, indicating the gangue recovered in the prefloat in test 6 was not recovered in the nickel concentrate in test 7 where the prefloat was deleted. Again, the addition of CuSO4 in the last stage of test 7 visually gave an increase in sulphides in the concentrate, and the results confirm the observations, with an increase in rougher concentrate grade from 10.8% Ni in concentrate 4 to 11.7 % Ni in concentrate 5 after the CuSO4. Nickel recoveries were significantly increased, from a stage recovery of 2.9% in concentrate 4 to 11.0% in concentrate 5.

In test 7 the sulphur and iron grades and recoveries were also increased after the CuSO4 addition to stage 5, indicating the increased recovery of pyrite as well as pentlandite.

Test 17, with the CuSO4 added from the start, and without the prefloat or CMC, gave the best flotation response. Nickel flotation rates were fast., with 77.% nickel recovery in the first 2 minutes. Total recoveries were 94.6% for nickel, 92.8% for cobalt and 94.8% for sulphur, to a rougher concentrate grade of 19.9% Ni.

A small sample of tails from test 6 was also passed trough the Davis Tube to remove magnetite. The recoveries to magnetics were 17.0% weight, 24.8% nickel and 63.4% iron relative the total metal contents in the tails. Grades of the magnetics were 1.03% Ni, 2.16% S and 49.0% Fe. Details are shown in Appendix 4.

8.5 Staged Rougher Flotation Tests A014AWC Composite

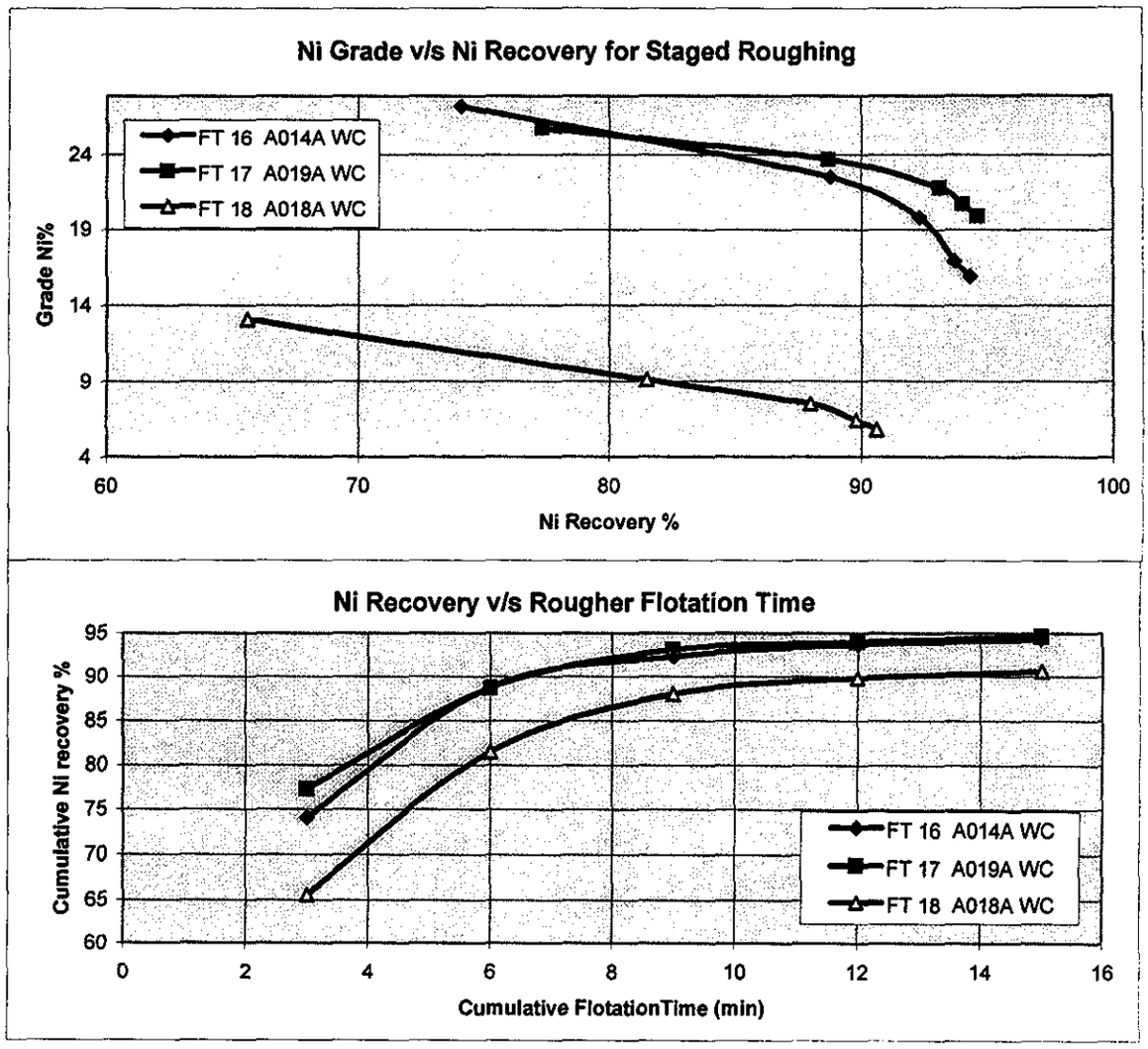
As this composite was only available mid way through the program, and flotation conditions had already been partly established for the A018AWC and A019AWC composites, only one flotation test was completed, using the best conditions established for the other composites at the time. These were as for tests 17 and 18 with the other composites, and included :

- Grinding to 80% passing 75µm
- No preflotation
- No CMC
- CuSO4 added from the start.

Results for nickel flotation are summarised in Figure 5. Results for tests 16 and 17 with these conditions for the other composites are shown for comparison.

The results show the flotation performance of the A014AWC composite generally resembles the good performance of the A019AWC composite (test 17), and was significantly better than for A018AWC composite (test 18). The position of the nickel grade - recovery curve is very similar to that for the A019A WC composite (test 17). The slightly lower concentrate grades are likely to be due to the lower nickel head grade in the A014AWC composite of 2.04% Ni, compared to A019AWC at 3.48% Ni.

Figure 5



8.6 Staged Rougher Flotation Tests 18/19 WC Composite

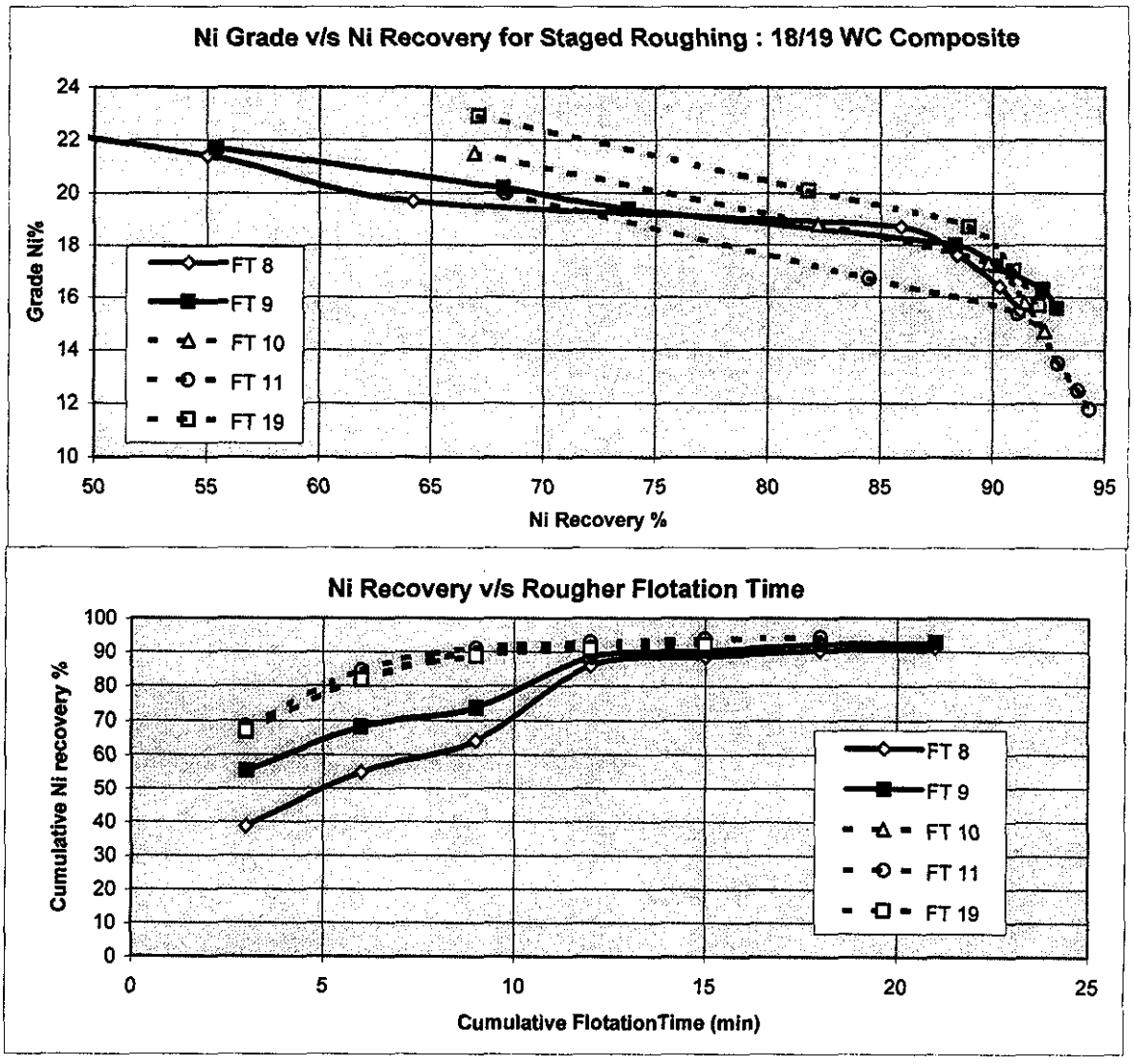
Five staged rougher flotation tests were completed with this composite. The test variables included those used for the individual hole composites (refer sections 8.3 and 8.4), followed by two tests using the best reagent conditions, with a finer and a coarser grind.

The flotation variables tested were as follows :

Test No	Grind P80 in µm	Prefloat included	CuSO4 additions	CMC additions
8	75	Yes	to Scavengers	No
9	75	No	to Scavengers	Yes
10	75	No	at start	No
11	45	No	at start	No
19	106	No	at start	No

Tests 8 and 9 included extended scavenger flotation time with extra collector additions to examine if nickel recoveries could be increased. Nickel flotation results are summarised in Figure 6.

Figure 6



The preflotation stage in test 8 recovered only 2.22% weight with elemental recoveries of 2.1% nickel, 2.0% cobalt and 2.3% sulphur. Tests 8 and 9 gave generally similar results. Test 9 without the prefloat and with CMC gave faster initial nickel recoveries, however results at the end of scavenging were almost identical for these tests. Test 8 with the prefloat achieved 91.2% nickel recovery, and test 9 without the prefloat achieved 92.8% nickel recovery.

In both tests 8 and 9, the addition of CuSO4 to the scavengers (at the start of rougher stage 4) gave a significant increase in the metal recoveries (Ni, Co, Fe, Cu and S). As for A019AWC composite (test 7) there was a larger increase in the S and Fe grades than Ni, indicating recovery of some pyrite as well as pentlandite.

Test 10 with CuSO4 added from the start and without CMC or a prefloat gave the best results compared to tests 8 and 9. This was consistent with the trends for the hole composites A018AWC and A019AWC. The use of CuSO4 from the start gave fast initial nickel flotation response.

Tests 11 and 19 with the finer and coarser grinds respectively, gave almost identical nickel flotation rates to test 10. However there was a trend for slightly slower flotation rates with the coarser grinds. There was also a more significant trend for higher concentrate nickel grades with coarser grinding. Results highlighting these trends and final grades and recoveries to the total rougher concentrate are shown in Table 12.

Table 12 Effects of Primary Grind Fineness on Grades and Recoveries to the Total Rougher Concentrate

Test	Grind	Grades			Recoveries %		
	P80 µm	Ni %	Co ppm	S %	Ni	Co	S
11	45	11.8	2839	15.2	94.3	92.4	96.4
10	75	14.7	3530	19.7	92.3	90.0	95.0
19	106	15.7	3717	19.2	92.0	89.7	93.5

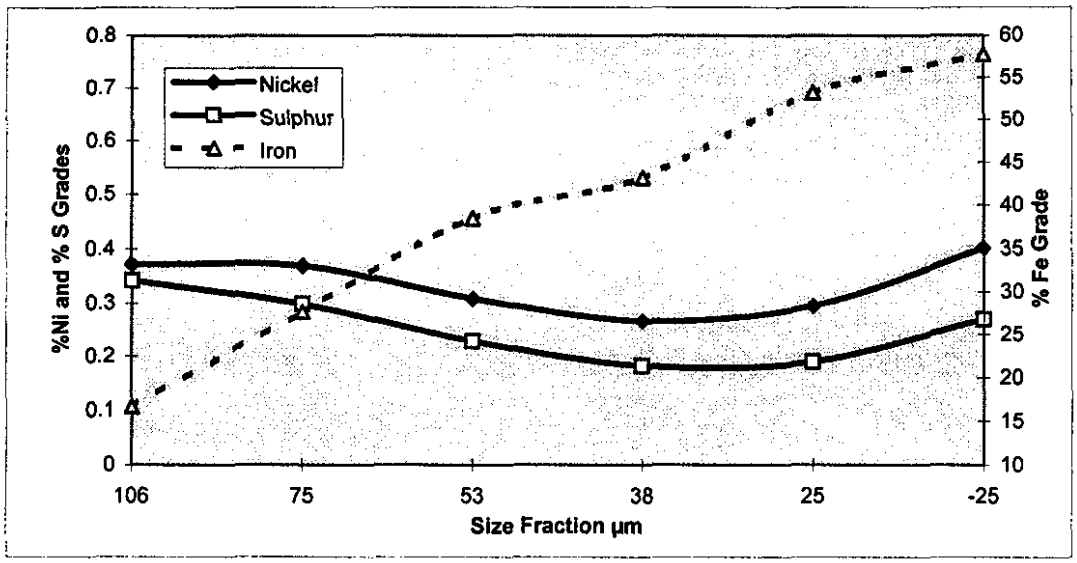
As the differences in nickel recoveries between the 75µm and the 106µm grinds was only 0.3%, and there was a 1% increase in nickel grade of the concentrate, it was concluded that for this composite the optimum grind was in the vicinity of 106µm.

(Note that later in the test program with the 14/18/19WC composite, there was a more significant decrease in recoveries with the 106µm grind compared to the 75µm grind, and the 106µm grind was not considered optimum).

Magnetic separations were also carried out with the tail for Test 8, and the mags and non mags were sized and the fractions assayed. The objective was to examine if there was any trend for increases in the nickel content of fractions of the magnetics. Results are detailed in Appendix 4.

There was no significant variation in the nickel (and sulphur) grades in the magnetic fractions. The lowest grade of 0.266 % Ni was in the -53 +38µm fraction and the highest grade of 0.401% Ni was in the -25µm fraction. There was however a significant increase in the iron grades into the finer fractions. These trends are shown in Figure 7. The relatively flat curves for nickel and sulphur, compared to the steep curve for iron suggests the coarser magnetite is associated with non sulphide gangue, and not with any increase in nickel content. It also suggests the liberation of magnetite is poor for sizes coarser than 38µm.

Figure 7 Size by Size Assays of the Magnetics from FT8 Flotation Tails



A small series of 500 g scale tests were completed to examine the effects of gangue and slime depressants and dispersants in the rougher stages, particularly the first rougher stages. The procedure involved 1000 g grinds to 80% passing 75µm, followed by splitting the pulp into two halves to provide 2 x 500g flotation feed samples. Four flotation tests were completed (tests 12 to 15) from 2 x 1000g grinds. Flotation was limited to 3 x 1 minute stages of roughing to evaluate the effects of the depressants. Flotation was carried out in the 1.3 litre cell so pulp densities would be similar to the 1000 g tests normally carried out in the 2.5 litre cell.

Copper sulphate was added from the start. The variables were as follows :

Test 12 no depressants added

Test 13 80 g/t dextrin added from the start.

Test 14 600 g/t Na2SiO3 added from the start

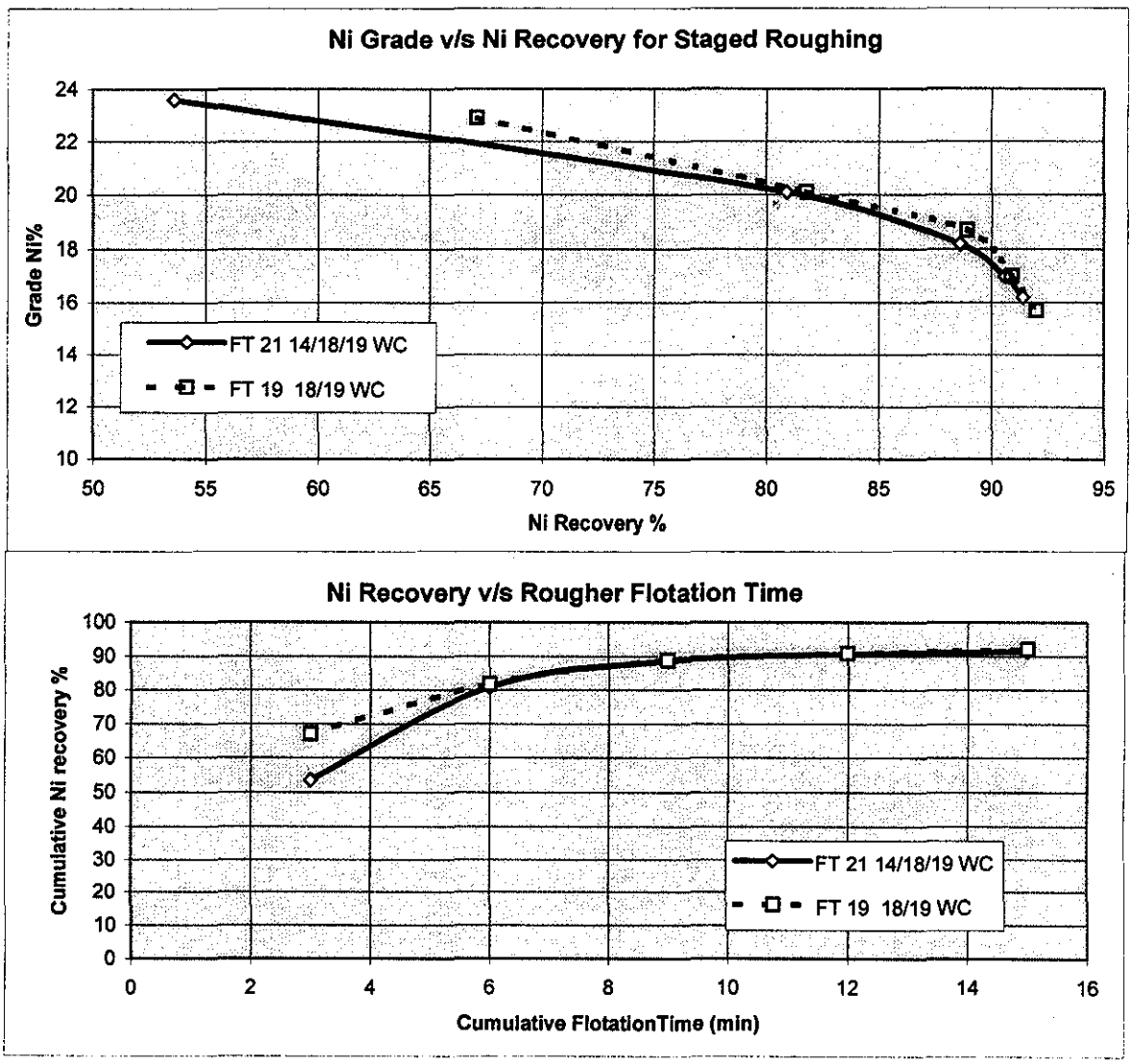
Test 15 200 g/t Na2SiO3 added from the start.

Detailed results are included in Appendix 5. The results suggest there was no benefit from using either dextrin or Na2SiO3, as the best results were achieved from test 12 without any depressants added.

8.7 Staged Rougher Flotation Tests 14/18/19 WC Composite

One staged rougher test, test 21, was completed with 14/18/19 WC composite using the conditions considered to be near optimum at the time. These included a grind to 80% passing 106 µm, no prefloat, and CuSO4 added from the start. These conditions were as for test 19 with the 18/19WC composite. Results for test 21 are shown in Figure 8, and results from test 19 are also shown for comparison.

Figure 8



The results show the flotation response of the 14/18/19WC composite was very similar to the 18/19WC composite. Total recoveries from test 21 were 91.4% for nickel, 89.3% for cobalt and 88.9% for sulphur.

8.8 Cleaner Flotation Tests with A018AWC, A019AWC and A014AWC Composites

Two tests were completed with each composite, all with primary grinds to 80% passing 106µm. Rougher flotation conditions were the same for all tests and included no preflotation, CuSO4 added from the start and no depressants.

For each composite one test involved direct cleaning and recleaning (1st cleaner and 2nd cleaner) of the rougher concentrate, and one test involved regrinding the rougher concentrate before cleaning. Regrind fineness was an estimate only, however the products were laser sized and gave P80 values between 28µm and 37µm. After regrinding, extra CuSO4 and SIBX was added to activate and promote flotation of the reground and fresh sulphide surfaces. The doses added were estimated as sufficient, although no comparative tests were completed to ensure doses were sufficient or near optimum. The 2nd cleaner concentrate was removed in three separate sequential stages for separate analysis to give grade recovery data for this cleaner stage.

Results for nickel flotation are shown in Figure 9. Grade and recovery data is summarised in Table 10, section 8.1. Final concentrate results are summarised in Table 13.

Figure 9

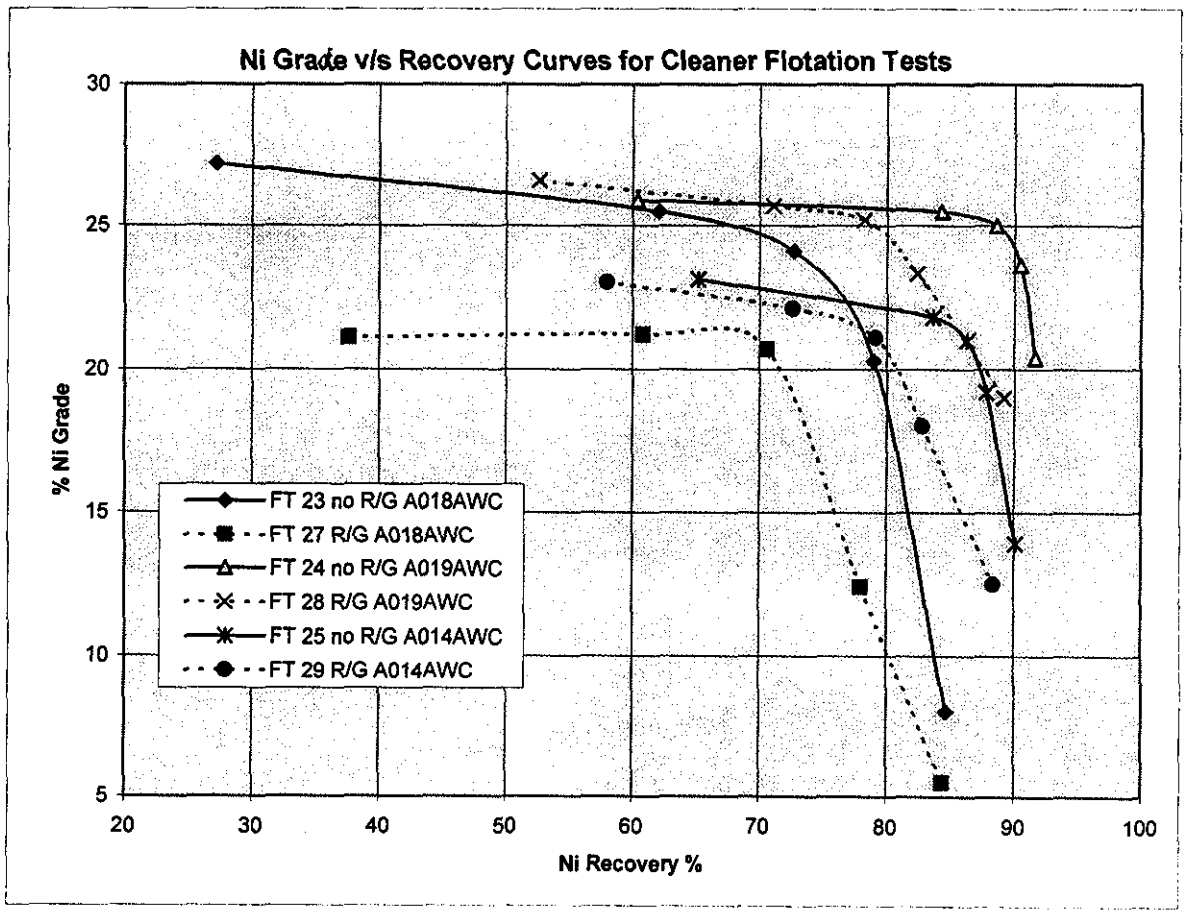


Table 13 Results of the 2nd Cleaner Concentrate

Test	Composite	Rghr Conc R/G	Ni Rec % Rghr Conc	2nd Cleaner Conc Assays Ni %	Co ppm	Fe %	S %	Recoveries % Ni	Co	Fe	S
23	A018A WC	No	84.7	24.1	5349	29.6	25.6	72.7	66.3	7.1	78.1
27	A018A WC	Yes	84.4	20.7	5336	23.6	21.8	70.6	68.5	5.9	76.3
24	A019A WC	No	91.7	25.0	6052	32.0	31.7	88.6	87.1	26.7	88.5
28	A019A WC	Yes	89.2	25.2	6759	29.7	31.2	78.2	78.1	19.3	76.9
25	A014A WC	No	90.1	21.0	4117	33.0	27.9	86.3	84.6	31.8	74.3
29	A014A WC	Yes	88.4	21.1	4842	32.7	29.6	79.1	80.6	26.1	70.7

For all composites the best results were achieved without regrinding. After regrinding, there was either a reduction in the final concentrate nickel grade or lower nickel recovery. It is possible that this effect was due to inadequate doses of reagents after the regrind, giving a slower flotation response.

There was also a relationship between concentrate grade at a given recovery and head grade for nickel. With decreasing head grades the concentrate grades were also lower, as shown below for a concentrate recovery of 80%, for the tests without the regrinds:

Composite	Test	Head Grade Ni %	% Ni in Conc at 80% Rec
A019AWC	24	3.48	25.7
A014AWC	25	2.04	22.1
A018AWC	23	1.35	18.5

8.8.1 Magnetic Separations of Final Concentrates

Final concentrates from these flotation tests (tests 23 o 25 and 27 to 29) were also passed through the Davis Tube magnetic separator. Results are detailed in Appendix 4. The first series of tests were completed with the concentrates from the tests without the regrind before cleaning (tests 23 to 25). Results showed high weight recoveries to the magnetics (up to 66.6% for test 23 with A018A WC), and the magnetics contained a high nickel content. It was considered that there was a significant component of pentlandite / magnetite composites in the flotation concentrates, and that regrinding could be of benefit to improve liberation and rejection of magnetite, hence potentially improving final concentrate nickel grades.

However, as discussed earlier in this section, the results from the tests with regrinding before cleaning gave generally inferior results compared to the ~~test~~ tests without regrinding. The magnetic separations on the final concentrates from the tests with the regrinds gave very low weight recoveries to magnetics, indicating the regrind had been effective at improving liberation and allowing the magnetite to be rejected to the cleaner tails. However , even with the better rejection of magnetite after the regrind, there was no improvement in concentrate nickel grades compared to the tests without the regrind, as shown in Table 13 in section 8.8. There was some reduction in iron grades, and a significant reduction in iron recovery in the final concentrate. It is possible that while

some of the iron, as magnetite was rejected after the regrind, there could have been an increase in other slime non sulphides being recovered to the concentrate, thereby limiting any increase in the nickel grade of the concentrate.

It is possible that the use of the regrind, followed by an optimised reagent scheme before cleaning, and including additions of slime depressants or dispersants, may give the best concentrate nickel grades and recoveries.

8.9 Cleaner Flotation Tests with 18/19WC and 14/18/19WC Composites

One flotation test was completed with 18/19 WC composite without a regrind, and after a primary grind to 80% passing 75µm. Other conditions were as for the cleaner tests with the hole composites discussed in section 8.8.

Three tests were completed with 14/18/19 composite. Variables tested included primary grind fineness, regrinding versus no regrinding before cleaning, and with CMC added to the cleaners to aid depression of talc and gangue, with the specific target of reducing the Mg content in the final concentrate.

Test numbers and variables are summarised in Table 14.

Table 14 Variables Tested in Cleaner Flotation Tests

Test No	Composite	Primary Grind P80 size in µm	Regrind before cleaning	CMC added to cleaners
20	18/19 WC	75	No	No
22	14/18/19 WC	106	No	No
26	14/18/19 WC	106	Yes	No
31	14/18/19 WC	75	No	Yes

Results are summarised in Figure 10 and Table 15.

Figure 10

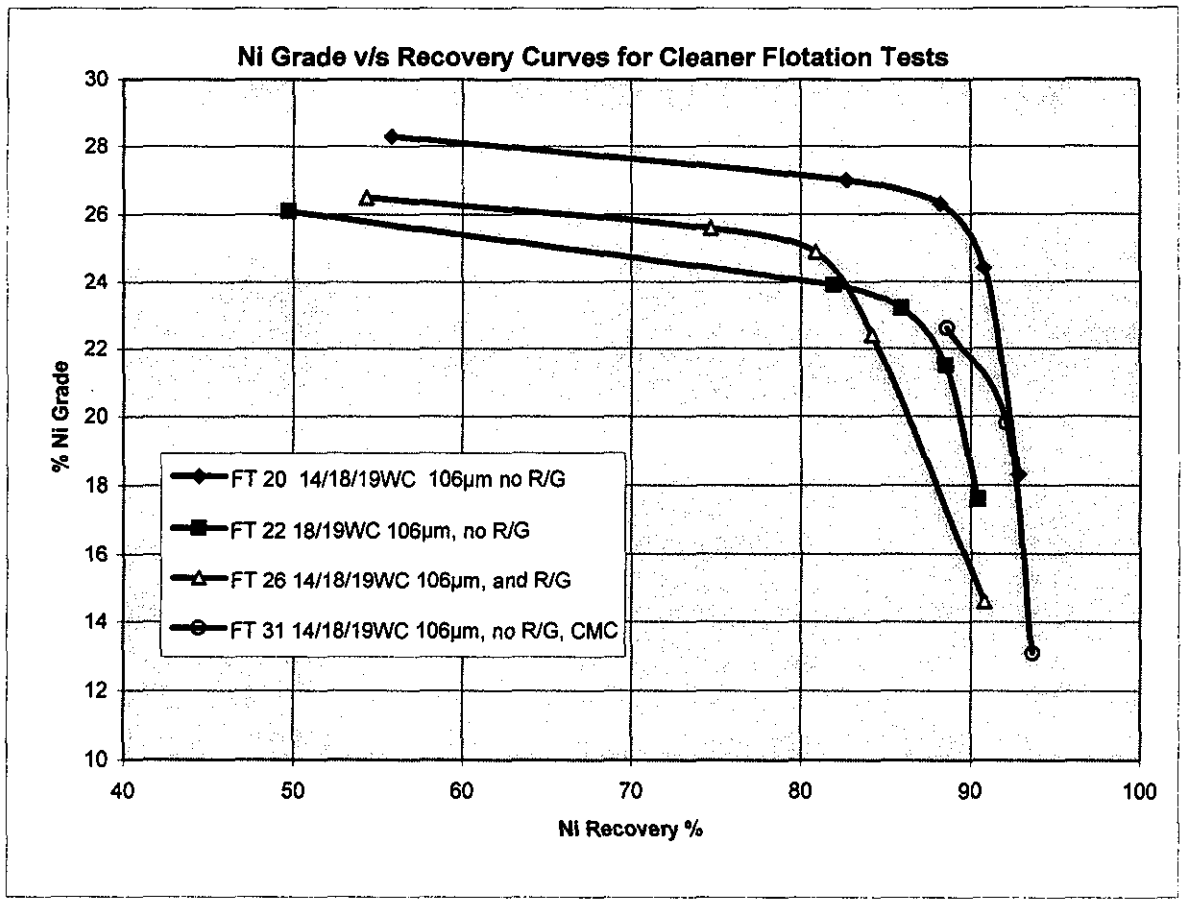


Table 15 Results of the 2nd Cleaner Concentrate

Test	Composite	Test No	Rghr Conc R/G	Ni Recov % Rghr Conc	2nd Clnr Conc Assays Ni %	Co ppm	S %	Recoveries % Ni	Co	S
20	18/19 WC	20	No	92.8	26.3	6216	28.6	88.2	85.9	88.6
22	14/18/19WC	22	No	88.6	23.2	5182	28.5	85.9	84.2	82.2
26	14/18/19WC	26	Yes	90.8	24.9	6297	30.9	80.9	80.6	75.0
31	14/18/19WC	31	No	93.6	22.6	5090	27.9	88.6	87.1	87.0

The results from the 18/19WC composite showed good upgrading with cleaning, to produce a high grade concentrate of 26.3% Ni..

With the 14/18/19WC composite, the 106µm grind gave lower nickel recoveries to the rougher concentrate than the finer 75µm grind. With both rougher and cleaner tests with the 106µm grind, the recoveries were from 88.6% to 91.4% nickel recovery. For test 31 with the 75µm grind the nickel recovery to the rougher concentrate was 93.6%.

Regrinding the rougher concentrate before cleaning (test 23) again gave lower nickel recovery in the final concentrate compared to when there was no regrind (test22).

However as for the other cleaner tests with the hole composites, this effect may be due to unoptimised reagent additions to the pulp after the regrind.

Best flotation results were achieved with test 31, with the 75µm grind, without the regrind, and with CMC additions to the cleaners. The CMC was specifically targeted at reducing the magnesium level in the final concentrate. Magnesium in concentrate was noted as a possible concern after multi element analysis of final concentrates from Cycle test 30 (discussed in the next section, 8.10). In the cycle test the Mg levels (assayed as Mg but reported as MgO) had been assayed at levels approaching possible penalty levels. Hence, test 31 was completed with the CMC in the cleaner only, and the products assayed for Mg. Results are shown in Table 16.

Table 16 MgO and Other Values in FT 31 Products and Cycle Test 30 Concentrate

	MgO %	SiO2 %	Al2O3 %	CaO %
FT 30 Cycle 3, 2nd Cleaner Conc.	5.77	8.80	0.12	1.05
FT 30 Cycle 5 2nd Cleaner Conc	6.58	9.59	0.32	1.09
FT 31 2nd Cleaner Conc	4.32	6.17	0.12	0.66
FT 31 2nd Cleaner Tail	21.04	34.36	0.68	4.71
FT 31 1st Cleaner Tail	26.72	44.54	1.01	6.50
FT 31 Tail	29.16	44.30	1.10	5.74

As Cycle test 30 included the regrinding of the rougher concentrate, and batch test 31 did not, and the cycle test included recirculation of cleaner tails, and batch test 30 did not, it cannot be concluded if the lower Mg assay for concentrate from test 30 was due to the effects of the CMC or the other variables. However, test 30 has demonstrated that Mg grades in the final concentrate can be reduced from those in the cycle test.

8.9.1 Magnetic Separations of Final Concentrates

The final concentrate from tests 22 and 26 (with and without the regrind) were subjected to magnetic separations with the Davis Tube. As for the magnetic separations discussed in section 8.8.1, there was a high weight recovery to the magnetics from the test 22 without the regrind (50.7% wt to the magnetics), and a low weight recovery from test 26 (2.2% wt) with the regrind. Comments and discussion on this response are dealt with in section 8.8.1. Full results from the magnetic separation tests are included in Appendix 4.

8.10 Cycle Test 30 with 14/18/19WC Composite

A closed cycle flotation test (test 30) was completed with the 14/18/19 WC composite to examine the effects of circulating loads from the cleaner tails) on the final concentrate grades and recoveries. The flotation circuit tested was based on batch test 26 except with a finer primary grind, and included :

- A primary grind to 80% passing 75 µm
- CuSO4 added from the start
- Regrinding of the total rougher/scavenger concentrate before cleaning, to a target of 80% passing 30µm. Regrind fineness checked by laser sizing.
- Reground product was cleaned and recleaned.
- The 1st cleaner tail was recycled back to the rougher feed (before the CuSO4 addition), of the next cycle.
- The 2nd cleaner tail was recycled to the 1st cleaner feed on the next cycle.
- Total of 6 cycles, each with a 1kg composite portion.
- A final concentrate and tail sample was sized and cyclosized, and another portion was assayed for a comprehensive multi element scan. The multi element scan results are shown in Section 9.

Details and results from the flotation test are included in Appendix 6.

Summarised results are shown in Table 17.

Table 17 Cycle Test 30 Flotation Results

	Wt	Ni	Co	Fe	S	Recoveries %			
	%	%	ppm	%	%	Ni	Co	Fe	S
Final Concentrates (2nd Cleaner Concentrates)									
Cycle 1	9.9	23.8	5470	30.2	30.0	-	-	-	-
Cycle 2	10.4	22.5	5150	30.2	27.9	92.2	90.9	24.1	90.4
Cycle 3	11.1	23.6	5390	32.5	28.3	93.6	92.3	27.1	89.7
Cycle 4	11.2	20.0	4560	28.8	27.8	93.0	91.4	25.6	93.2
Cycle 5	10.1	21.6	4980	28.9	27.8	91.1	89.8	22.7	89.0
Final Cycle 6 Products									
2nd Cleaner Conc.	11.5	20.4	4670	29.8	27.5	93.3	91.8	27.5	92.1
Tail	88.5	0.19	54	10.2	0.31	6.7	8.2	72.5	7.9
Circulating Loads									
2nd Cleaner Tail	5.5	2.05	460	13.5	5.21	4.5	4.3	6.0	8.3
1st Cleaner Tail	14.0	0.43	175	10.2	1.84	2.4	4.2	11.4	7.5

Full sizings and cylosizings were completed on Cycle 5 concentrate and tails. Results are shown in Table 18. The cyclosizer separation sizes were calculated using the specific

gravity of the concentrate and tails. These specific gravities were measured, and were determined to be :

SG of 2nd cleaner concentrate = 3.98 g/cm^3

SG of the Tails = 2.82 g/cm^3

The sizing s are shown in Table 18.

Table 18 Sizings of Final Concentrate and Tails from Cycle 5.

2nd Cleaner Concentrate

screen size µm	Wt Ret. g	Wt Ret. %	Wt Passing %
150	0.0	0.0	100.0
106	0.2	0.4	99.6
90	0.3	0.6	99.1
75	0.6	1.1	98.0
63	0.8	1.5	96.5
53	2.5	4.6	91.8
45	1.7	3.2	88.7
32	9.1	16.9	71.8
22	6.3	11.6	60.2
16	6.9	12.7	47.5
11	5.7	10.6	36.9
8	4.1	7.7	29.2
-8	15.8	29.2	-
Total	53.9	100.0	

Tail

screen size µm	Wt Ret. g	Wt Ret. %	Wt Passing %
150	0.3	0.2	99.8
106	3.0	2.3	97.4
90	7.2	5.6	91.8
75	10.2	8.0	83.8
63	11.9	9.3	74.5
53	10.4	8.1	66.4
45	4.3	3.4	63.1
41	5.0	3.9	59.2
29	9.0	7.0	52.2
21	12.5	9.8	42.4
15	11.4	8.9	33.5
11	9.1	7.1	26.4
-11	33.8	26.4	-
Total	128.1	100.0	

The cycle test results showed a reduction in concentrate nickel grade with recirculation of the cleaner tails, from Cycle 1 through to Cycle 6. However there was an increase in recoveries to the 2nd cleaner concentrate compared to batch tests 26 and 31. In batch test 31, the 2nd cleaner concentrate assayed 22.6% Ni with 88.6% recovery. This compares to Cycle 5 from the cycle test with a grade of 21.6% Ni with 91.1% recovery.

9 MULTI ELEMENT ANALYSIS OF FLOTATION CONCENTRATE AND TAILS

The final concentrate and tails from Cycle test 30, Cycle5 were submitted to ALS in Brisbane for multi element analysis. Results are shown in Table 19. The MgO analysis of the concentrate was 6.58%, and was higher than desirable. Hence, other concentrates of higher nickel grades were also analysed, as discussed in section 8.9, and shown by Table 16.

Table 19 Multi Element Scan Results

Element	Units	Method	FT30 Cyc 5 Conc	FT 30 Cyc 5 Tail
Ni	%	AAS	21.6	0.24
Co	ppm	AAS	4980	64
S	%	Leco	27.83	0.389
Fe	%	AAS	28.9	11.1
Cu	ppm	AAS	1500	210
Au	ppb	Fire / ICP	1821	162
Pt	ppb	Fire A.	<2	<2
Pd	ppb	Fire A.	20	<2
Hg	ppm	C.V. AAS	0.2	0.1
Cl	%	Titration	0.03	<0.01
F	ppm	ISE	220	670
SiO2	%	ICP	9.59	42.98
Al2O3	%	ICP	0.32	1.09
CaO	%	ICP	1.09	5.67
MgO	%	ICP	6.58	29.61
TiO2	%	ICP	0.01	0.04
Na2O	%	ICP	0.01	0.04
K2O	%	ICP	0.05	0.33
P2O5	%	ICP	0.03	0.04
MnO	%	ICP	0.11	0.32

Element	Units	Method	FT30 Cyc 5 Conc	FT 30 Cyc 5 Tail
Ag	ppm	ICP	<1	1
As	ppm	ICP	4640	99
Ba	ppm	ICP	13	32
Bi	ppm	ICP	616	13
Cd	ppm	ICP	<5	<5
Co	ppm	ICP	3890	57
Cr	ppm	ICP	1310	1750
Mo	ppm	ICP	75	34
Pb	ppm	ICP	397	17
Sb	ppm	ICP	160	<10
Sr	ppm	ICP	<10	<10
V	ppm	ICP	16	19
Zn	ppm	ICP	537	70
Zr	ppm	ICP	22	10
B	ppm	ICP MS	26	-
Nb	ppm	ICP MS	0.9	1.0
Sc	ppm	ICP MS	0.9	7.2
Se	ppm	ICP MS	17.8	<5
Sn	ppm	ICP MS	1.2	2.5
Ta	ppm	ICP MS	0.7	0.2
Te	ppm	ICP MS	1.8	<0.2
W	ppm	ICP MS	9.1	11.3

Some other concentrates were selected and also assayed for boron to confirm the result, as boron levels in nickel concentrates can be significant for marketing purposes. Concentrates and boron results were as follows :

FT 30 Cycle 3 2nd Cleaner Conc 18 ppm B

FT31 2nd Cleaner Concentrate 16 ppm B

10 CONCLUSIONS

1. There was variability in the grinding hardness of the three drill hole composites used for the testwork. The A018A WC composites was the hardest to grind, and A019A WC the easiest the grind. A composite prepared from equal weights of A019A and A018A WC material had a Bond Ball Mill Work Index of 18.4 kWh/tonne.

2. The ore samples tested were not amenable to heavy media preconcentration before grinding and flotation. Testing of crushed A019AWC ore with heavy liquids gave only a low weight rejection to the light fraction and little upgrading to the heavies (preconcentrate).

3. All samples tested were amenable to flotation for production of a high grade nickel concentrate, with high recoveries, using a conventional flotation flowsheet.

4. Flotation reagents used were CuSO4 for sulphide activation, SIBX as collector, and MIBC as frother. Reagent doses required for good flotation response were low to moderate, and considered normal. A gangue dispersant / depressant such as CMC maybe of benefit to aid rejection of slime gangue, increasing nickel grades of the concentrates and reducing the MgO contents of the concentrates. Reagent doses were not optimised, but gave satisfactory results at the doses used.

5. There is scope to improve the flotation response and possibly reduce overall reagent consumption with optimisation of the flowsheet and reagent system.

6. The natural pH of the ore was slightly alkaline (pH around 10.0), which did not require any additional pH modification.

7. Satisfactory results were achieved with a primary grind fineness of 80% passing 75µm. Nickel recoveries to the rougher concentrate of greater than 90% were achieved for all whole core composites, and as high as 93.6% for the 14/18/19WC composite. Coarser grinding to 80% passing 106µm gave lower recoveries to the rougher concentrate.

8. A simple flotation circuit consisting of roughing with two stages of cleaning produced cleaner concentrates of greater than 20% nickel grade. These concentrates contained a high proportion of magnetics, (magnetite) and the magnetics contained high nickel values. The high nickel values of the magnetics indicated the grains are composites of pentlandite (major) with magnetite (minor).

9. Regrinding of the rougher concentrates before cleaning was successful for reducing the magnetic content on the cleaner concentrate, however the reagent doses used after the regrind were not adequate and need optimisation. It is considered that an optimised circuit with regrinding of the rougher concentrates before cleaning will assist in producing higher grade concentrates without reducing nickel recoveries compared to a flowsheet without the regrind.

10. Indications were that the nickel grades in the final concentrate can be improved with small additions of CMC to the cleaners to reject gangue for the cleaner concentrates. The effect may also be to reduce the MgO values in the final concentrate.

11. Cobalt recoveries to the nickel concentrates were generally similar to, or only slightly lower than the nickel recoveries.

12. From batch test 31 with the 14/18/19 WC composite, the 2nd cleaner concentrate assayed 22.6% Ni, 5090 ppm Co, with recoveries of 88.6 % for nickel and 87.1% from cobalt. This test included a 75 µm grind, no regrinding, and with CMC added to the

cleaners, and open circuit cleaning. In cycle test 30 with the same composite, with closed circuit cleaning, the recoveries were 93.3% for nickel and 91.8% for cobalt, at 20.4% nickel grade. The cycle test included a regrind before roughing and without CMC.

13. Although settling and filtration tests with flotation products were not included in this test program, no particular problems were evident when handling the flotation concentrates and tails. Flotation concentrates filtered satisfactorily through the laboratory vacuum filter through filter papers, and flotation tails in buckets settled with a clear supernatant in a reasonable time (without flocculant).

APPENDIX 1

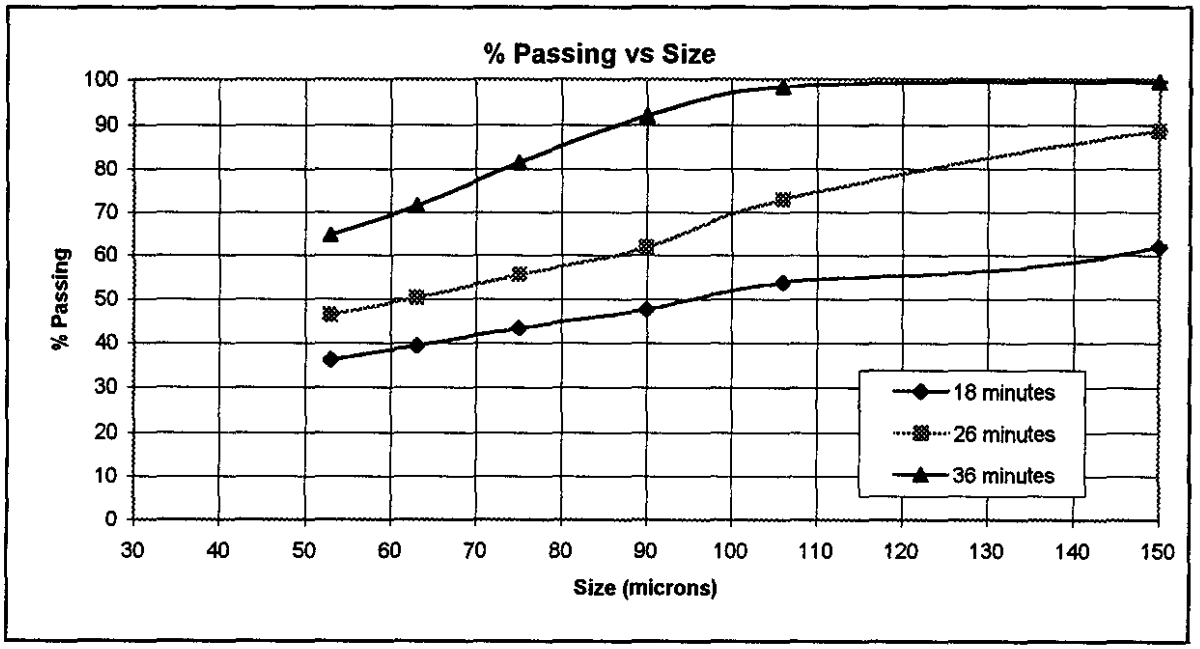
TRIAL GRINDS

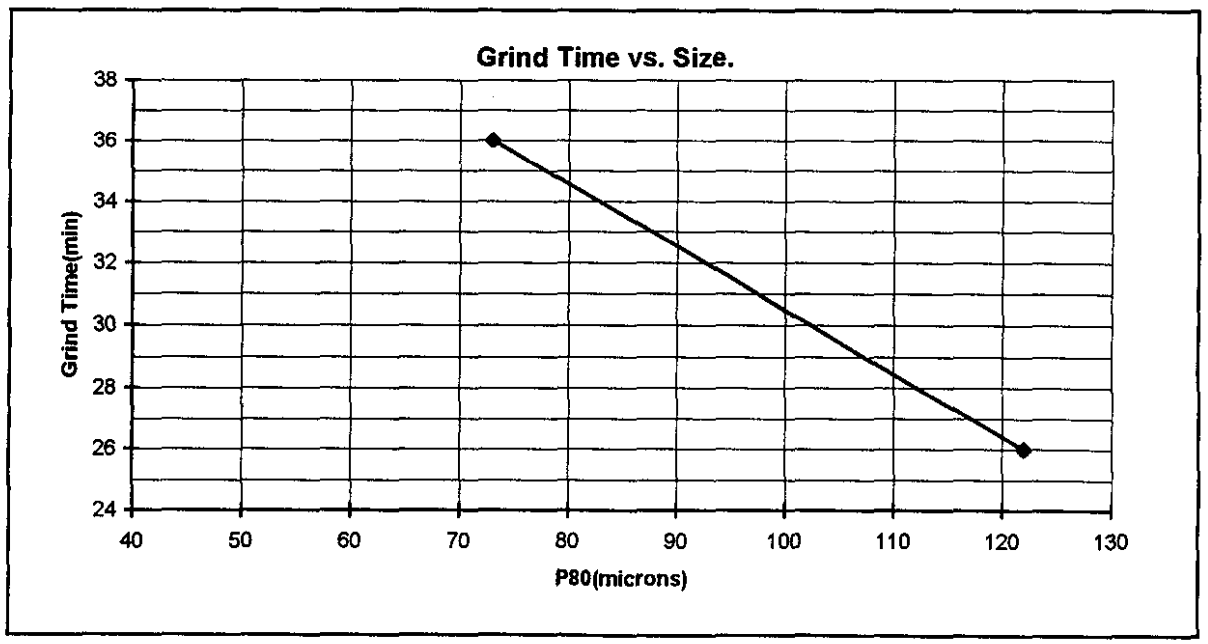
00900 AVEBURY

018A Composite

Stainless steel rod mill.
(Yellow Rod *Mill*)
1kg / 60% Solids

Trial grind 1		Trial grind 2		Trial grind 3	
18 minutes		26 minutes		36 minutes	
Size (µm)	**% Pass**	**Size (µm)**	**% Pass**	**Size (µm)**	**% Pass**
150	62.0	150	88.6	150	99.6
106	53.8	106	72.9	106	98.4
90	47.6	90	61.8	90	92.1
75	43.3	75	55.6	75	81.5
63	39.3	63	50.4	63	71.4
53	36.0	53	46.2	53	64.6



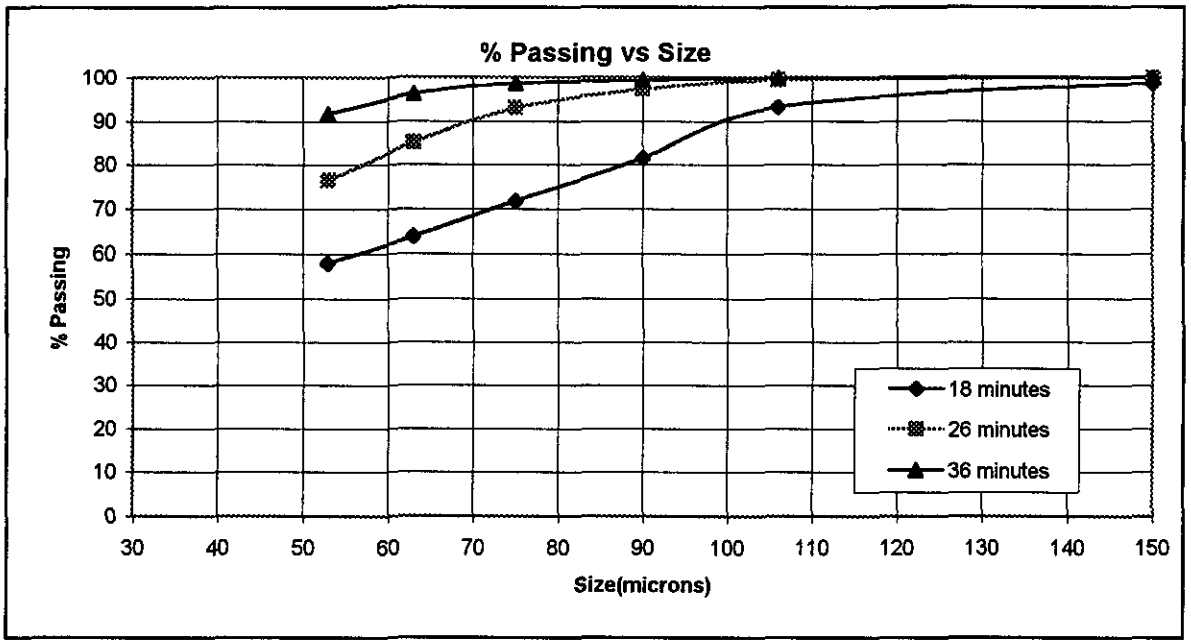


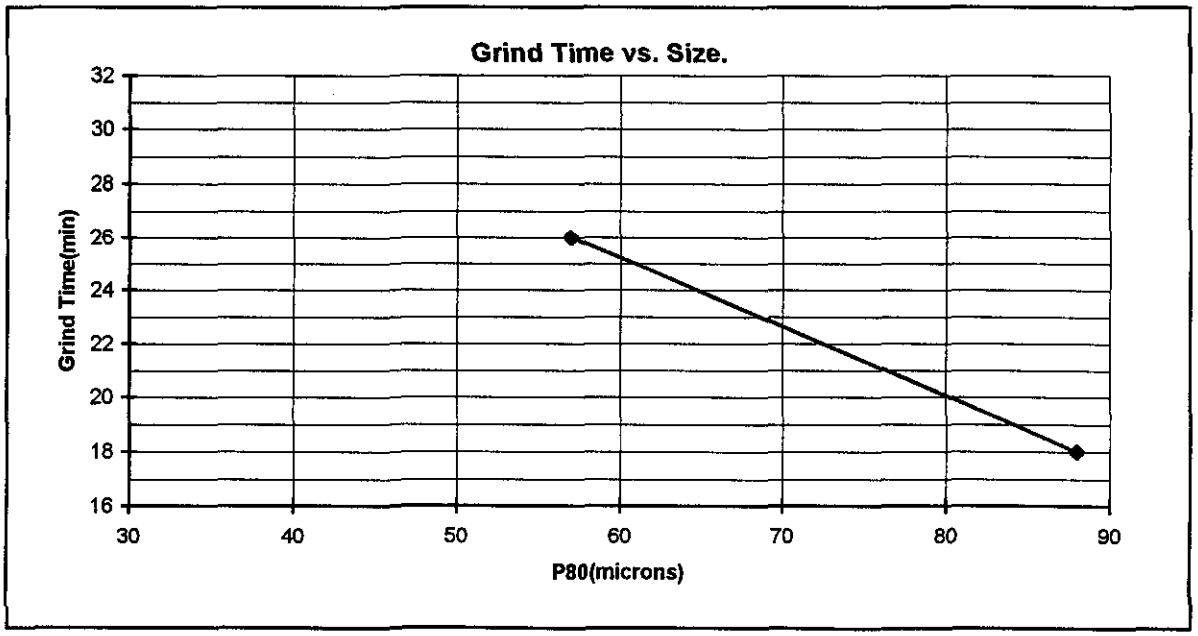
00900 AVEBURY

019A Composite

Stainless steel rod mill.
(Yellow Rod *Mill*)
1kg / 60% Solids

Trial grind 1 18 minutes		**Trial grind 2** 26 minutes		**Trial grind 3** 36 minutes	
Size (µm)	**% Pass**	**Size (µm)**	**% Pass**	**Size (µm)**	**% Pass**
150	98.7	150	99.8	150	100.0
106	93.2	106	99.4	106	99.8
90	81.5	90	97.4	90	99.4
75	71.8	75	93.0	75	98.6
63	63.8	63	85.1	63	96.3
53	58.0	53	76.5	53	91.7



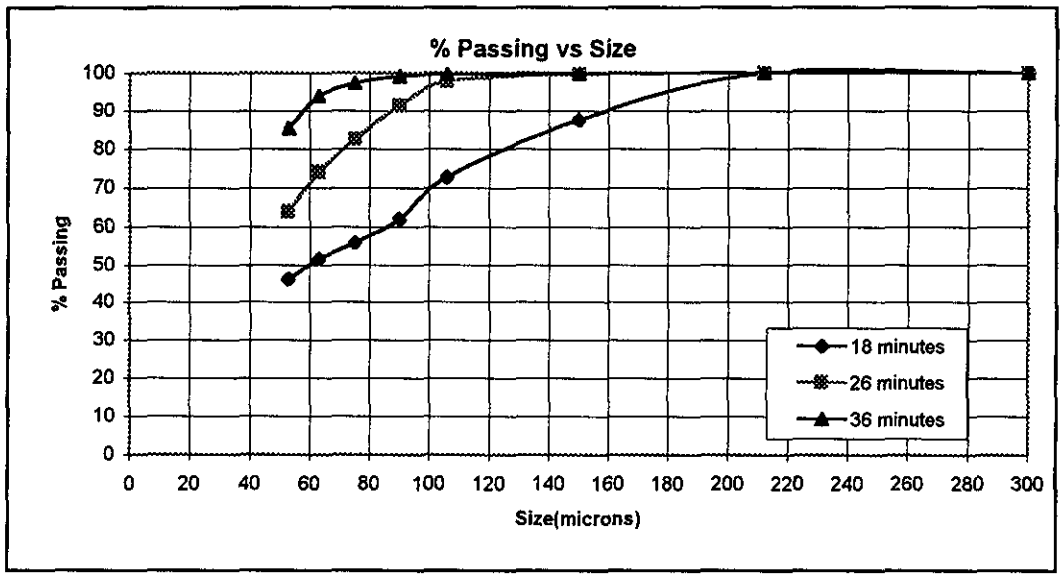


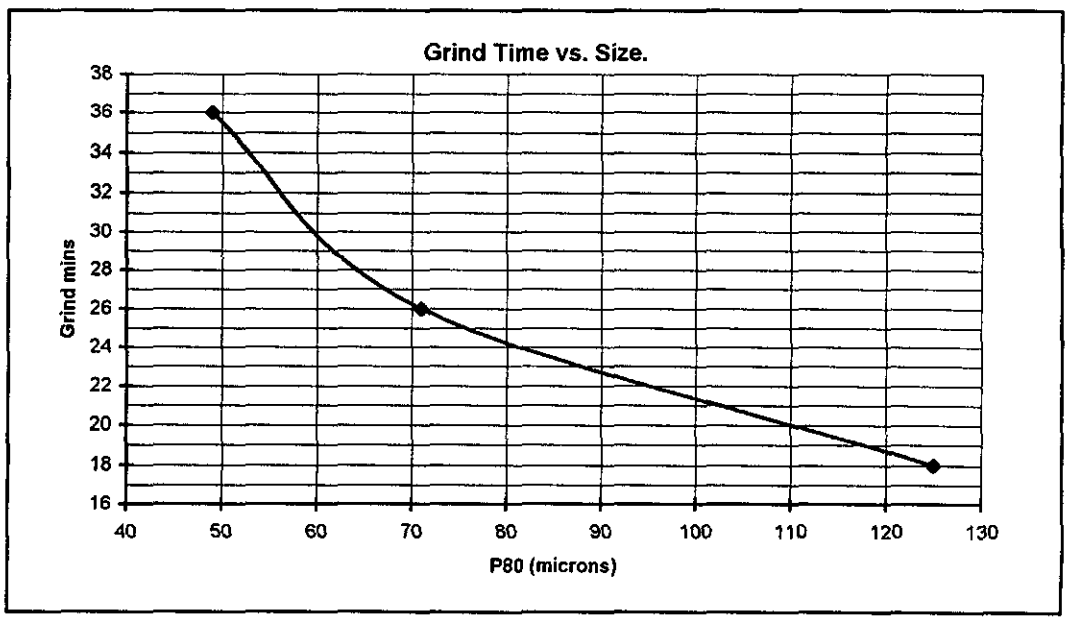
00900 AVEBURY

18/19 Composite

Stainless steel rod mill.
(Yellow Rod *Mill*)
1kg / 60% Solids

Trial grind 1		Trial grind 2		Trial grind 3	
18 minutes		26 minutes		36 minutes	
Size (µm)	**% Pass**	**Size (µm)**	**% Pass**	**Size (µm)**	**% Pass**
150	87.6	150	99.7	150	99.9
106	72.7	106	97.8	106	99.7
90	62.0	90	91.3	90	99.1
75	55.8	75	82.5	75	97.3
63	51.6	63	74.0	63	94.0
53	46.1	53	63.8	53	85.5





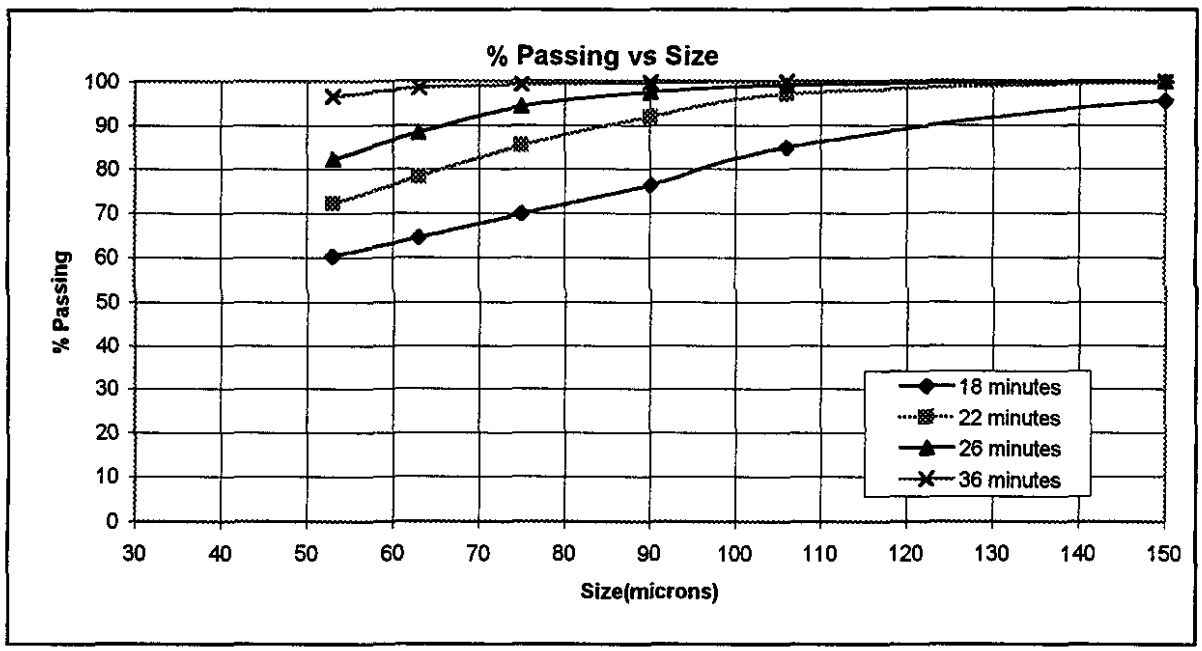
METCON LABORATORIES PTY LIMITED

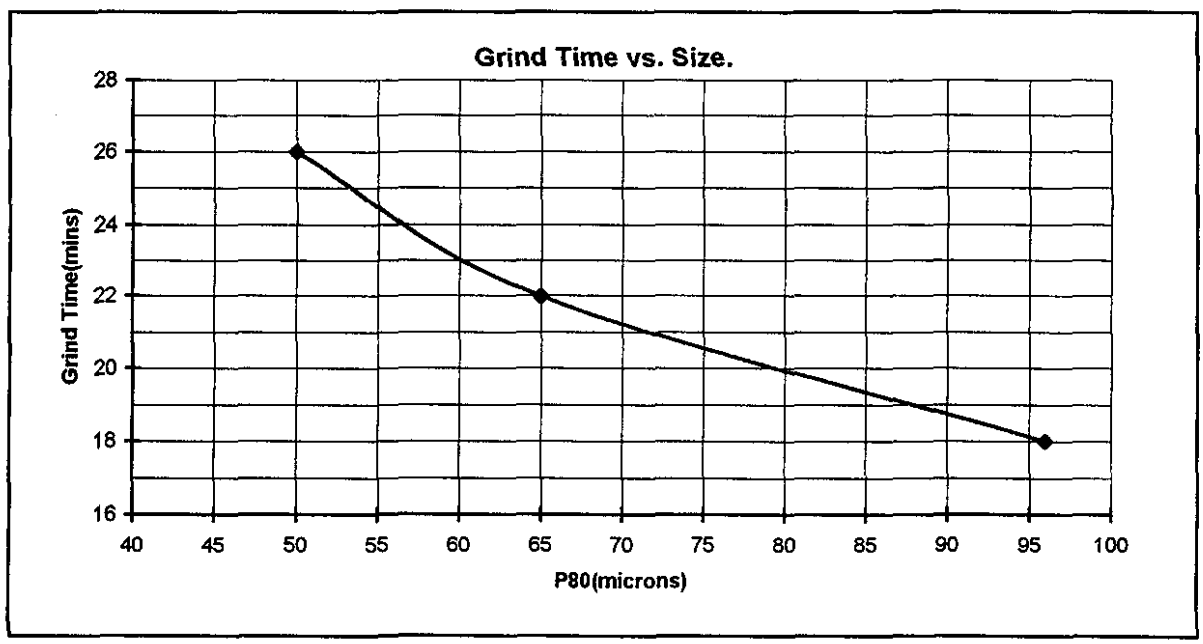
00900 AVEBURY

A014AWC

Stainless steel rod mill.
(Yellow Rod *Mill*)
1kg / 60% Solids

Trial grind 1		Trial grind 2		Trial grind 3		Trial grind 4	
18 minutes		22 minutes		26 minutes		36 minutes	
Size (µm)	**% Pass**	**Size (µm)**	**% Pass**	**Size (µm)**	**% Pass**	**Size (µm)**	**% Pass**
150	95.6	150	99.8	150	99.9	150	99.9
106	84.9	106	97.0	106	99.2	106	99.8
90	76.4	90	91.8	90	97.5	90	99.7
75	69.9	75	85.4	75	94.3	75	99.3
63	64.5	63	78.3	63	88.4	63	98.6
53	60.3	53	72.2	53	82.2	53	96.5



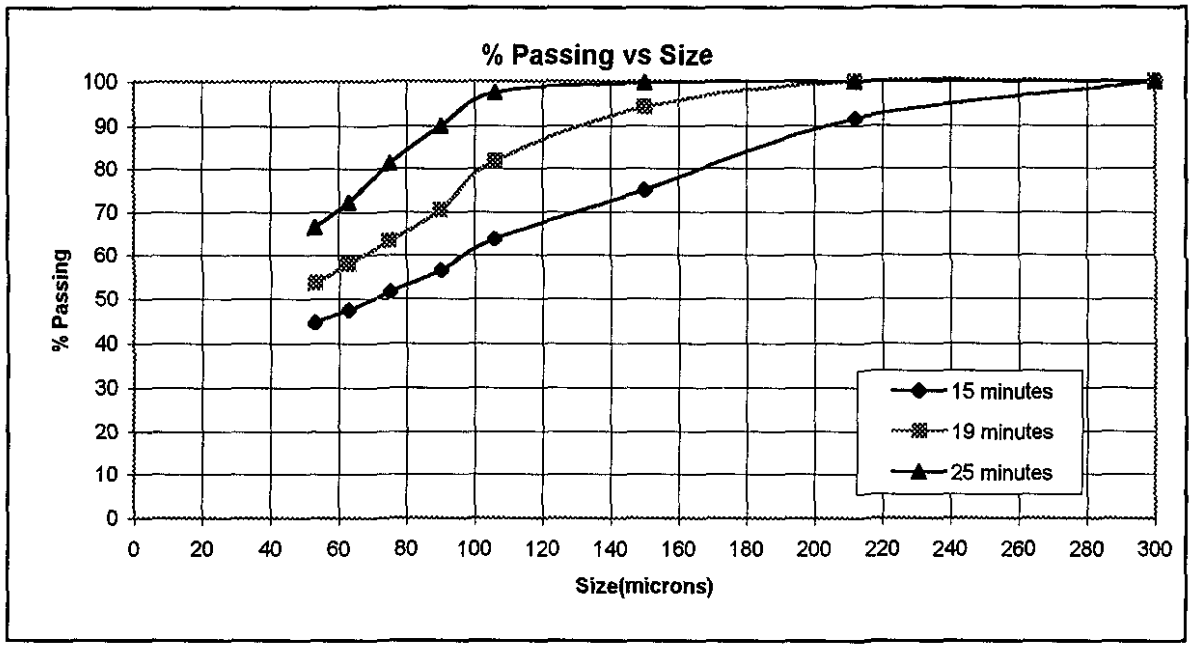


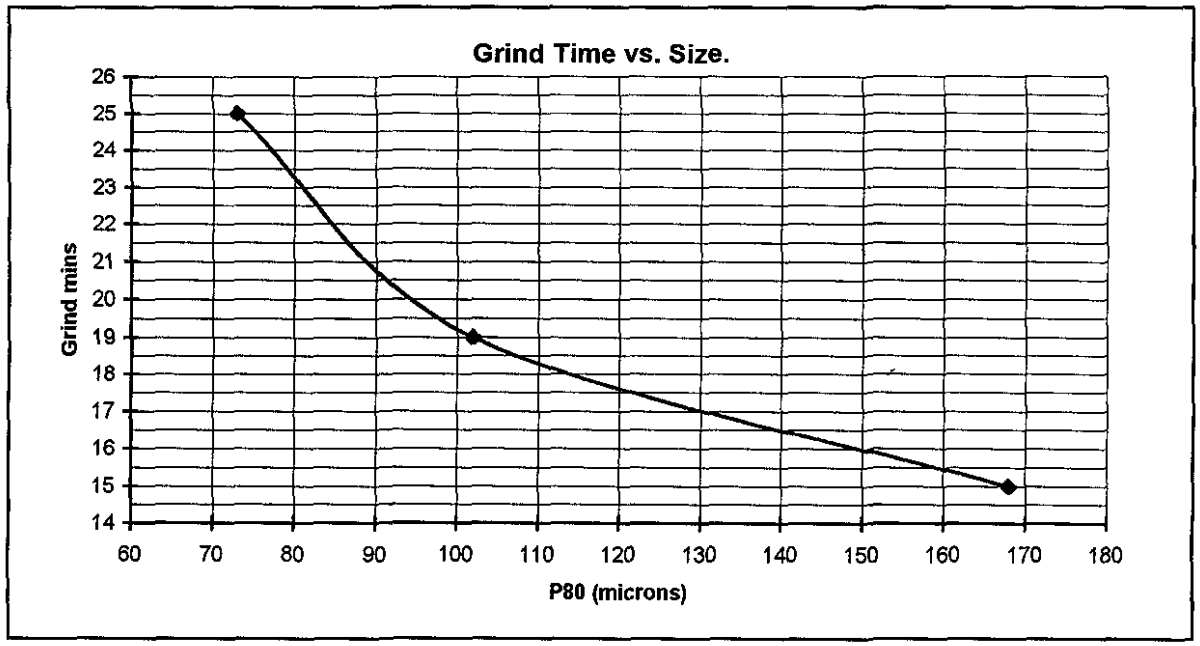
00900 AVEBURY

14/18/19 WC Composite

Stainless steel rod mill.
(Yellow Rod *Mill*)
1kg / 60% Solids

Trial grind 1		Trial grind 2		Trial grind 3	
15 minutes		19 minutes		25 minutes	
Size (µm)	**% Pass**	**Size (µm)**	**% Pass**	**Size (µm)**	**% Pass**
150	75.0	150	94.3	150	99.7
106	63.8	106	81.9	106	97.5
90	56.5	90	70.4	90	89.9
75	51.6	75	63.4	75	81.1
63	47.5	63	58.0	63	72.2
53	44.7	53	53.7	53	66.6





APPENDIX 2

BOND BALL MILL WORK INDEX TEST DETAILS AND RESULTS

REPORT 00900

BOND BALL MILL WORK INDEX RESULTS
SAMPLE : W. I. Composite

Unit Weight (g) :	1571
Feed P80 (µm)	2550
Product P80 (µm)	97
Product g/rev	1.04
Product screen size (µm)	150
Ideal product wt (g)	449

Cycle	revs	g product	g fines in feed	g fines produced	g/rev produced
1	100	290	220	70	0.70
2	583	585	41	544	0.93
3	395	483	82	401	1.02
4	374	459	68	391	1.05
5	367	445	64	381	1.04

$$\text{WI [kWh/tonne]} = \frac{44.5 * 1.102}{(Pu)^{0.23}(G)^{0.82}\left[\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right]}$$

Pu = Product screen size
G = Final g/rev produced
P = 80% passing size of product
F = 80% passing size of feed

WORK INDEX (kWh/tonne) : 18.4

The test was completed using the Bond Ball Mill Work Index test procedure as developed by F.C.Bond (Crushing and Grinding Calculations by F.C.Bond(1961) British Chemical Engineering, Vol 6, No's 6,8).

METCON LABORATORIES PTY LTD

BOND BALL MILL TEST SHEET

REPORT 00900
SAMPLE W. I. Composite

UNIT WEIGHTS

	Gross wt g	Tare wt g	Net wt g	Volume ml	g/ml	g/700 ml
1st measurement	1907.0	246.7	1660.3	740	2.24	1571
2nd measurement	1851.5	246.7	1604.8	715	2.24	1571
					average	1571

REQUIRED PRODUCT SCREEN = 150 µm

FEED SIZINGS (Crushed to 100% passing 3.35mm)

size µm	Wt g	% Wt ret	% pass
2800	39.2	12.8	87.2
2360	39.4	12.9	74.3
1700	59.1	19.3	55.1
1180	39.5	12.9	42.2
600	42.8	14.0	28.2
300	26.9	8.8	19.4
150	16.7	5.5	14.0
-150	42.8	14.0	-
Total	306.4		

PRODUCT SIZING (5th CYCLE)

size µm	Wt g	% Wt ret	% pass
125	6.3	4.1	95.9
106	14.1	9.3	86.6
90	17.8	11.7	74.9
75	12.7	8.3	66.6
63	12.6	8.3	58.3
53	10.3	6.8	51.5
45	4.7	3.1	48.4
38	8.3	5.5	43.0
-38	65.4	43.0	-
Total	152.2	100.0	

APPENDIX 3

HEAVY LIQUID SEPARATION TEST RESULTS

Heavy Media Separation Test

Feed :

A019WC Composite, crushed to nominally -12.7mm

Procedure :

Approximately 1.0 kg of sample was deslimed by wet screeniing at 45µm.
The +45µm fraction was separated in TBE (heavy liquid) at SG2.92
The floats (lights) and sinks (heavies) were each sized, and the size fractions assayed.

Results :

	Size	Wt	Wt	Assays					Distribution %				
	mm	g	%	Ni%	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Lights	9.5	15.2	1.61	1.03	225	10.3	1.15	200	0.5	0.5	1.1	0.5	1.1
	6.7	16.9	1.79	0.459	125	7.45	0.49	120	0.3	0.3	0.9	0.2	0.7
	3.35	15.7	1.67	0.525	145	6.8	0.64	100	0.3	0.3	0.8	0.3	0.5
	2.00	8	0.85	0.591	175	6.9	0.75	120	0.2	0.2	0.4	0.2	0.3
	1.00	8.7	0.92	0.473	135	5.9	0.59	140	0.1	0.2	0.4	0.1	0.4
	0.50	6.2	0.66	0.430	120	5.4	0.49	175	0.1	0.1	0.2	0.1	0.4
	0.15	8.3	0.88	0.323	94	4.6	0.40	185	0.1	0.1	0.3	0.1	0.5
	0.045	8.7	0.92	0.793	200	6.9	0.98	215	0.2	0.2	0.4	0.2	0.6
	Total Lights		9.30	0.60	156	7.2	0.70	152	1.8	1.9	4.6	1.6	4.6
Heavies	9.5	275.9	29.27	2.45	585	14.4	3.24	240	22.9	22.8	29.0	23.3	22.9
	6.7	195.5	20.74	3.03	720	14.1	3.97	270	20.0	19.9	20.1	20.2	18.3
	3.35	162.3	17.22	3.68	895	16.1	4.67	390	20.2	20.5	19.1	19.8	21.9
	2.00	67.3	7.14	3.97	960	16.1	5.14	295	9.0	9.1	7.9	9.0	6.9
	1.00	53.8	5.71	4.58	1090	16.9	6.40	360	8.3	8.3	6.6	9.0	6.7
	0.50	28.6	3.03	4.98	1210	17.7	6.36	400	4.8	4.9	3.7	4.7	4.0
	0.15	27.8	2.95	5.75	1390	18.5	7.27	420	5.4	5.5	3.8	5.3	4.0
	0.045	14.4	1.53	7.23	1690	22.6	8.58	430	3.5	3.4	2.4	3.2	2.1
	Total Heavies		87.59	3.37	809	15.4	4.39	304	94.2	94.3	92.6	94.5	86.8
Slimes	-0.045	29.3	3.11	4.05	900	13	5.06	850	4.0	3.7	2.8	3.9	8.6
	Calc Total	942.6	100.0	3.14	751	14.5	4.07	307	100.0	100.0	100.0	100.0	100.0
	Actual Head			3.48	840	15.1	4.19	-					

Size by Size Recovery to Lights

Size	Wt g	Wt g	% Rec	Elemental Recoveries to Lights (%)				
mm	Lights	Heavies	Lights	Ni	Co	Fe	S	Cu
9.5	15.2	275.9	5.2	2.3	2.1	3.8	1.9	4.4
6.7	16.9	195.5	8.0	1.3	1.5	4.4	1.1	3.7
3.35	15.7	162.3	8.8	1.4	1.5	3.9	1.3	2.4
2.00	8	67.3	10.6	1.7	2.1	4.8	1.7	4.6
1.00	8.7	53.8	13.9	1.6	2.0	5.3	1.5	5.9
0.50	6.2	28.6	17.8	1.8	2.1	6.2	1.6	8.7
0.15	8.3	27.8	23.0	1.6	2.0	6.9	1.6	11.6
0.045	8.7	14.4	37.7	6.2	6.7	15.6	6.4	23.2
Total	87.7	825.6	9.6	18	20	51	17	65

Size by Size Results

Size mm	Product	Wt g	Wt %	Assays Ni%	Co ppm	Fe %	S %	Cu ppm	Distribution % Ni	Co	Fe	S	Cu
9.5	Lights	15.2	5.2	1.03	225	10.3	1.15	200	2.3	2.1	3.8	1.9	4.4
	Heavies	275.9	94.8	2.45	585	14.4	3.24	240	97.7	97.9	96.2	98.1	95.6
	Total	291.1	100.0	2.38	566	14.2	3.13	238	100.0	100	100	100	100
6.7	Lights	16.9	8.0	0.459	125	7.45	0.49	120	1.3	1.5	4.4	1.1	3.7
	Heavies	195.5	92.0	3.025	720	14.1	3.97	270	98.7	98.5	95.6	98.9	96.3
	Total	212.4	100.0	2.82	673	13.6	3.69	258	100.0	100.0	100.0	100.0	100.0
3.35	Lights	15.7	8.8	0.525	145	6.8	0.635	100	1.4	1.5	3.9	1.3	2.4
	Heavies	162.3	91.2	3.68	895	16.1	4.67	390	98.6	98.5	96.1	98.7	97.6
	Total	178	100.0	3.40	829	15.3	4.31	364	100.0	100.0	100.0	100.0	100.0
2	Lights	8	10.6	0.591	175	6.9	0.745	120	1.7	2.1	4.8	1.7	4.6
	Heavies	67.3	89.4	3.97	960	16.1	5.14	295	98.3	97.9	95.2	98.3	95.4
	Total	75.3	100.0	3.61	877	15.1	4.67	276	100.0	100.0	100.0	100.0	100.0
1	Lights	8.7	13.9	0.473	135	5.9	0.588	140	1.6	2.0	5.3	1.5	5.9
	Heavies	53.8	86.1	4.575	1090	16.9	6.4	360	98.4	98.0	94.7	98.5	94.1
	Total	62.5	100.0	4.00	957	15.4	5.59	329	100.0	100.0	100.0	100.0	100.0
0.5	Lights	6.2	17.8	0.43	120	5.4	0.488	175	1.8	2.1	6.2	1.6	8.7
	Heavies	28.6	82.2	4.98	1210	17.7	6.36	400	98.2	97.9	93.8	98.4	91.3
	Total	34.8	100.0	4.17	1016	15.5	5.31	360	100.0	100.0	100.0	100.0	100.0
0.15	Lights	8.3	23.0	0.323	94	4.6	0.399	185	1.6	2.0	6.9	1.6	11.6
	Heavies	27.8	77.0	5.75	1390	18.5	7.27	420	98.4	98.0	93.1	98.4	88.4
	Total	36.1	100.0	4.50	1092	15.3	5.69	366	100.0	100.0	100.0	100.0	100.0
0.045	Lights	8.7	37.7	0.793	200	6.9	0.979	215	6.2	6.7	15.6	6.4	23.2
	Heavies	14.4	62.3	7.23	1690	22.6	8.58	430	93.8	93.3	84.4	93.6	76.8
	Total	23.1	100.0	4.81	1129	16.7	5.72	349	100.0	100.0	100.0	100.0	100.0

APPENDIX 4

MAGNETIC SEPARATION TEST RESULTS

DAVIS TUBE MAGNETIC SEPARATION TESTS

Procedure :

Samples of flotation feed were taken as pulps after the grind.
The samples were run through the Davis Tube at medium power (0.8A) to recover magnetite. The magnetics were removed.
The non mags were re-run through the Davis Tube, as a scavenger, with the Davis Tube at full power (1.5A).
All products were dried and assayed.

Davis Tube Magnetic Separation with Float Test FT1 Float Feed
Sample : A018CC
Fineness : 57.1% -75µm

Product	Wt	Wt	Assays						Distribution %					
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Mg %	Ni	Co	Fe	S	Cu	Mg
Magnetics (@ 0.8A)	18.6	35.3	1.00	220	45.4	0.8	26	9.75	29.1	30.2	86.3	24.3	13.9	19.3
Mag Scav (@ 1.5 A)	5.9	11.2	2.65	530	7.7	2.76	76	21.2	24.5	23.1	4.6	26.7	12.9	13.3
Non Mags	28.2	53.5	1.05	225	3.15	1.06	90	22.5	46.4	46.8	9.1	49.0	73.1	67.4
Calc head	52.7	100.0	1.21	257	18.6	1.16	66	17.9	100.0	100.0	100.0	100.0	100.0	100.0
Actual Head			1.20	255	18.1	0.92	30	-						
Magnetics (@ 0.8A)		35.3	1.00	220	45.4	0.8	26	9.8	29.1	30.2	86.3	24.3	13.9	19.3
Mags + Mag Scav		46.5	1.40	295	36.3	1.3	38	12.5	53.6	53.2	90.9	51.0	26.9	32.6

Davis Tube Magnetic Separation with Float Test FT2 Float Feed
Sample A019CC
Fineness : 90.8% -75µm

Product	Wt	Wt	Assays						Distribution %					
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Mg %	Ni	Co	Fe	S	Cu	Mg
Magnetics (@ 0.8A)	9.3	14.8	1.12	270	54.4	1.08	225	6.2	6.2	6.4	57.6	5.3	21.6	5.4
Mag Scav (@ 1.5 A)	2.5	4.0	2.70	710	12	3.57	175	22.2	4.0	4.6	3.4	4.8	4.5	5.2
Non Mags	51.0	81.2	2.95	680	6.7	3.31	140	18.7	89.8	89.0	38.9	89.9	73.8	89.4
Calc head	62.8	100.0	2.67	620	14.0	2.99	154	17.0	100.0	100.0	100.0	100.0	100.0	100.0
Actual Head			2.55	595	13.7	2.81	125	-						
Magnetics (@ 0.8A)		14.8	1.12	270	54.4	1.1	225	6.2	6.2	6.4	57.6	5.3	21.6	5.4
Mags + Mag Scav		18.8	1.45	363	45.4	1.6	214	9.6	10.2	11.0	61.1	10.1	26.2	10.6

DAVIS TUBE MAGNETIC SEPARATION TESTS

Procedure :

Samples of flotation feed and tails were taken as pulps from the flotation cell
The samples were run through the Davis Tube at medium power (0.8A) to recover magnetite.
All products were dried and assayed.

Davis Tube Magnetic Separation with Float Test FT4 Float Feed and Tails
Sample : A018A WC
Fineness : 80% -75µm

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
FLOAT FEED												
Magnetics	18.6	30.6	1.50	395	43.7	1.43	70	34.5	37.6	81.6	35.3	24.8
Non Mags	42.1	69.4	1.26	290	4.35	1.16	94	65.5	62.4	18.4	64.7	75.2
Calc head	60.7	100.0	1.33	322	16.4	1.24	87	100.0	100.0	100.0	100.0	100.0
Actual Head			1.35	310	16.0	1.25	-					
FLOAT TAILS												
Magnetics	17.1	31.3	0.62	225	46.2	0.47	66	40.3	46.0	87.7	39.3	30.0
Non Mags	37.6	68.7	0.42	120	2.95	0.33	70	59.7	54.0	12.3	60.7	70.0
Calc head	54.7	100.0	0.48	153	16.5	0.37	69	100.0	100.0	100.0	100.0	100.0
Actual Head			0.48	155	16.2	0.39	56					

Davis Tube Magnetic Separation with Float Test FT6 Float Feed and Tails
Sample : A019A WC
Fineness : 80% -75µm

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
FLOAT FEED												
Magnetics	14.2	16.8	2.15	555	47.3	3.54	415	11.3	12.0	54.4	14.1	23.4
Non Mags	70.3	83.2	3.40	825	8.0	4.36	275	88.7	88.0	45.6	85.9	76.6
Calc head	84.5	100.0	3.19	780	14.6	4.22	299	100.0	100.0	100.0	100.0	100.0
Actual Head			3.48	840	15.1	4.19	-					
FLOAT TAILS												
Magnetics	11.4	17.0	1.03	295	49.0	2.16	320	24.8	21.6	63.4	29.5	23.0
Non Mags	55.6	83.0	0.64	220	5.8	1.06	220	75.2	78.4	36.6	70.5	77.0
Calc head	67.0	100.0	0.71	233	13.2	1.25	237	100.0	100.0	100.0	100.0	100.0
Actual Head			0.76	250	13.7	1.30	200					

FT 8 Tail : Davis Tube Magnetics

Size	Wt	Wt	Assays of Fractions					Distribution %				
µm	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
106	8.7	4.9	0.372	130	16.7	0.342	155	5.4	6.3	2.1	6.7	4.5
75	52.2	29.3	0.369	120	27.7	0.298	155	32.2	34.8	20.4	35.0	27.0
53	43.9	24.6	0.308	98	38.5	0.228	130	22.6	23.9	23.9	22.5	19.1
38	23.9	13.4	0.266	82	43.2	0.182	135	10.6	10.9	14.6	9.8	10.8
25	22	12.3	0.294	80	53.2	0.189	150	10.8	9.8	16.6	9.4	11.0
-25	27.5	15.4	0.401	94	57.7	0.268	300	18.4	14.4	22.4	16.6	27.6
Total	178.2	100	0.336	101.02	39.7	0.2493	168	100	100	100	100	100

FT 8 Tail : Davis Tube Non Magnetics

Size	Wt	Wt	Assays of Fractions					Distribution %				
µm	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
106	15.7	3.7	0.193	58	3.35	0.195	125	4.6	3.9	3.0	4.6	1.8
75	87.7	20.4	0.178	60	4.10	0.195	110	23.8	22.3	20.4	25.6	8.7
53	65.2	15.2	0.151	60	4.65	0.136	115	15.0	16.6	17.2	13.3	6.7
38	32.1	7.5	0.141	60	5.00	0.117	145	6.9	8.2	9.1	5.6	4.2
25	42.5	9.9	0.119	52	4.40	0.118	150	7.7	9.4	10.6	7.5	5.7
-25	186.6	43.4	0.148	50	3.75	0.155	435	42.0	39.6	39.7	43.3	72.9
Total	429.8	100	0.1528	54.794	4.10	0.1552	259	100	100	100	100	100

FT 8 Tail : Calculated Total Mags + Non Mags

Size	Wt	Wt	Assays of Fractions					Distribution %				
µm	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
106	24.4	4.0	0.2568	84	8.1	0.2474	136	5.0	4.9	2.2	5.4	2.3
75	139.9	23.0	0.2493	82	12.9	0.2334	127	27.8	27.7	20.4	29.4	12.6
53	109.1	17.9	0.2142	75	18.3	0.173	121	18.6	19.8	22.6	17.0	9.3
38	56.0	9.2	0.1943	69	21.3	0.1447	141	8.7	9.4	13.5	7.3	5.6
25	64.5	10.6	0.1787	62	21.0	0.1422	150	9.2	9.6	15.4	8.3	6.9
-25	214.1	35.2	0.180	56	10.7	0.1695	418	30.8	28.7	25.9	32.7	63.3
Total	608	100	0.2065	68	14.5	0.1828	232	100	100	100	100	100

FT 8 Tail : Distribution of Elements to Mags and Non Mags per Size Fraction Relative to Total Fraction

Size	Mags	Non Mags	Distribution to Magnetics					Distribution to Non Magnetics				
µm	Wt %	Wt %	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
106	35.7	64.3	51.6	55.4	73.4	49.3	40.7	48.4	44.6	26.6	50.7	59.3
75	37.3	62.7	55.2	54.3	80.1	47.6	45.6	44.8	45.7	19.9	52.4	54.4
53	40.2	59.8	57.9	52.4	84.8	53.0	43.2	42.1	47.6	15.2	47.0	56.8
38	42.7	57.3	58.4	50.4	86.5	53.7	40.9	41.6	49.6	13.5	46.3	59.1
25	34.1	65.9	56.1	44.3	86.2	45.3	34.1	43.9	55.7	13.8	54.7	65.9
-25	12.8	87.2	28.5	21.7	69.4	20.3	9.2	71.5	78.3	30.6	79.7	90.8
Total	29.3	70.7	47.7	43.3	80.0	40.0	21.2	52.3	56.7	20.0	60.0	78.8

Results from Magnetic Separation Tests on Cleaner Concentrates.

Notes :

The 2nd cleaner concentrates from the tests indicated below were prepared by recombining weighted portions from the 2nd cleaner concentrate stages 1, 2 and 3, before pulverising for assay. These portions were passed through the Davis Tube at moderate field strength (0.8A). The magnetics and non magnetics were assayed.

Float Test No	Composite	Product	Total 2nd Cleaner Concentrate							Distribution %				
			Wt g	Wt %	Assays Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
22	14/18/19WC	Mags	8.71	50.7	14.6	3110	43.2	29.9	1530	34.5	34.0	59.5	51.9	51.7
		Non Mags	8.47	49.3	28.5	6200	30.2	28.6	1470	65.5	66.0	40.5	48.1	48.3
		Total (Conc)	17.2	100	21.5	4633	36.8	29.3	1500	100	100	100	100	100
		Actual Conc			23.2	5182	33.0	28.5	1357					
23	A018A WC	Mags	4.97	66.6	24.0	5340	32.7	25.3	435	65.3	65.7	72.9	64.2	57.0
		Non Mags	2.49	33.4	25.5	5560	24.3	28.2	655	34.7	34.3	27.1	35.8	43.0
		Total (Conc)	7.46	100	24.5	5413	29.9	26.3	508	100	100	100	100	100
		Actual Conc			24.1	5349	29.6	25.6	511					
24	A019A WC	Mags	4.82	23.4	18.1	4750	40.2	28.4	1210	16.5	17.9	28.9	22.0	20.9
		Non Mags	15.8	76.6	27.9	6640	30.1	30.6	1400	83.5	82.1	71.1	78.0	79.1
		Total (Conc)	20.6	100	25.6	6198	32.5	30.1	1356	100	100	100	100	100
		Actual Conc			25.0	6052	32.0	31.7	1331					
25	A014A WC	Mags	6.11	39.2	19.3	4500	36.7	29.4	1190	32.2	36.9	46.0	38.8	30.8
		Non Mags	9.46	60.8	26.2	4960	27.8	29.9	1730	67.8	63.1	54.0	61.2	69.2
		Total (Conc)	15.6	100	23.5	4779	31.3	29.7	1518	100	100	100	100	100
		Actual Conc			21.0	4117	33.0	27.9	1653					

661678

Davis Tube Magnetic Separations of Cleaner Concentrates from Flotation Tests

The tests were completed on the total 2nd cleaner concentrate from the flotation tests.

The concentrate for the magnetic separation tests was prepared by recombining weighted portions of each of the staged 2nd cleaner concentrates

The non mags were submitted for analysis. There was insufficient of magnetics for analysis. Hence, the assays of the non mags ware compared with the total concentrate assays from the flotation test results.

Flotation Test No.	Composite		Wt g	Wt %	Assays Ni %	Co ppm	Fe %	Cu ppm
26	14/18/19 WC	Mags	0.16	2.2	-	-	-	-
		Non Mags	7.03	97.8	25.4	5780	29.8	1900
		Calc Total	7.19	100.0	-	-	-	-
		Actual Total Conc assays			24.9	6297	31.6	1888
27	A018A WC	Mags	0.25	5.4	-	-	-	-
		Non Mags	4.37	94.6	21.2	4890	21.4	770
		Calc Total	4.62	100.0	-	-	-	-
		Actual Total Conc assays			20.7	5336	23.6	676
28	A019A WC	Mags	0.25	2.2	-	-	-	-
		Non Mags	11.01	97.8	27.4	6690	29.6	1530
		Calc Total	11.26	100.0	-	-	-	-
		Actual Total Conc assays			25.2	6759	29.7	1446
29	A014A WC	Mags	0.33	4.1	-	-	-	-
		Non Mags	7.78	95.9	22.5	4480	31.1	1970
		Calc Total	8.11	100.0	-	-	-	-
		Actual Total Conc assays			21.1	4842	32.7	1922

APPENDIX 5

BATCH FLOTATION TEST DETAILS AND RESULTS

FLOTATION TEST DATA SHEET

TEST NO.	FT 1	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	57.1% -75µm
DATE :	18/8/00	RPM	60
SAMPLE :	A018CC	MINUTES	25
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Preliminary scouting staged rougher test.
Float feed samples were taken for sizing and for magnetic separation tests.

	REAGENTS			Eh	pH	COND	FLOAT
		SIBX	MIBC	mV		(min)	(min)
Cond			10		9.7	1	
Prefloat	Good froth. Some talc and slimes floating						5
Cond		20		124	9.8	2	
Rougher Stage 1	Pentlandite, some lightly flocced slimes floating						3
Cond		20		114	9.7	2	
Rougher Stage 2	Fine pentlandite, silicates? and less slimes						3
Cond		40	10	19	9.7	2	
Rougher Stage 3	Not much of anything floating						3

Product	Wt	Wt	Assays						Distribution %					
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Mg %	Ni	Co	Fe	S	Cu	Mg
Prefloat	79.0	8.72	1.22	255	12.5	1.22	74	19.9	8.9	8.8	6.1	8.8	15.8	10.1
Rougher Conc 1	27.3	3.01	7.55	1550	16.7	7.67	140	15.8	19.0	18.5	2.8	19.1	10.4	2.8
Rougher Conc 2	18.7	2.06	6.75	1360	17.6	6.74	130	15.7	11.6	11.1	2.0	11.5	6.6	1.9
Rougher Conc 3	20.5	2.26	3.70	740	15.3	3.73	98	17.7	7.0	6.6	1.9	7.0	5.4	2.3
Tail	760.6	83.94	0.763	165	18.7	0.776	30	16.9	53.5	54.9	87.2	53.7	61.8	82.9
Calc head	906.1	100.0	1.20	252	18.0	1.212	41	17.1	100.0	100.0	100.0	100.0	100.0	100.0
Actual Head			1.20	255	18.1	0.92	30	-						
Rougher Conc 1		3.01	7.55	1550	16.7	7.67	140	15.8	19.0	18.5	2.8	19.1	10.4	2.8
Rougher Cons 1 + 2		5.08	7.22	1473	17.1	7.29	136	15.8	30.6	29.6	4.8	30.5	16.9	4.7
Rougher Cons 1 to 3		7.34	6.14	1247	16.5	6.19	124	16.4	37.6	36.3	6.7	37.5	22.4	7.0

FLOTATION TEST DATA SHEET

TEST NO.	FT 2
PROJECT NO.	900
DATE :	18/8/00
SAMPLE :	A019CC
WT (g)	1000

GRINDING MILL :	Stainless Steel Rod
Product fineness	90.8% -75µm
RPM	60
MINUTES	25
% SOLIDS	60

OBJECTIVE : Preliminary scouting staged rougher test.
Float feed samples were taken for sizing and for magnetic separation tests.

	REAGENTS										Eh	pH	COND	FLOAT
			SIBX		MIBC						mV		(min)	(min)
Cond					10						80	10	1	
Prefloat	Talc and slimes at start. Very barren at end.													5
Cond			20								46	9.8	2	
Rougher Stage 1	Lots of pentlandite, good froth													3
Cond			10		10						77	9.8	2	
Rougher Stage 2	Good froth													3
Cond			10		10						62	9.8	2	
Rougher Stage 3														3
Cond			10		10						76	9.7	2	
Rougher Stage 4														3

Product	Wt	Wt	Assays						Distribution %					
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Mg %	Ni	Co	Fe	S	Cu	Mg
Prefloat	20.6	2.36	5.30	1120	10.3	5.32	705	15.3	4.6	4.1	1.7	4.4	11.6	2.2
Rougher Conc 1	40.8	4.68	26.6	5600	26.4	24.2	425	4.0	46.1	40.8	8.7	40.0	13.8	1.1
Rougher Conc 2	21.9	2.51	18.9	4120	21.4	18.7	380	7.1	17.6	16.1	3.8	16.6	6.6	1.1
Rougher Conc 3	17.5	2.01	14.6	3430	20.3	16.8	390	9.2	10.9	10.7	2.9	11.9	5.4	1.1
Rougher Conc 4	10.8	1.24	9.20	2300	16.2	11.5	375	11.9	4.2	4.4	1.4	5.0	3.2	0.9
Tail	760.6	87.20	0.513	175	13.3	0.72	98	17.7	16.6	23.8	81.6	22.1	59.3	93.6
Calc head	872.2	100.0	2.70	642	14.2	2.84	144	16.5	100.0	100.0	100.0	100.0	100.0	100.0
Actual Head			2.55	595	13.7	2.81	125	-						
Rougher Conc 1		4.68	26.6	5600	26.4	24.2	425	4.0	46.1	40.8	8.7	40.0	13.8	1.1
Rougher Cons 1 + 2		7.19	23.9	5083	24.7	22.3	409	5.0	63.7	56.9	12.5	56.5	20.4	2.2
Rougher Cons 1 to 3		9.20	21.9	4722	23.7	21.1	405	5.9	74.6	67.7	15.3	68.4	25.9	3.3
Rougher Cons 1 to 4		10.43	20.4	4435	22.8	20.0	402	6.6	78.8	72.1	16.7	73.4	29.1	4.2

FLOTATION TEST DATA SHEET

TEST NO.	FT 3
PROJECT NO.	900
DATE :	21/8/00
SAMPLE :	A018CC
WT (g)	1000

GRINDING MILL :	Stainless Steel Rod
Product fineness	93% -75µm
RPM	60
MINUTES	40
% SOLIDS	60

OBJECTIVE : Based on test 1, but with a finer grind and extra collector.
Float feed samples were taken for sizing.

	REAGENTS										Eh	pH	COND	FLOAT
			SIBX		MIBC						mV		(min)	(min)
Cond					10						93	9.8	1	
Prefloat	Good froth. Some talc and slimes floating													5
Cond			20								60	9.7	2	
Rougher Stage 1	Pentlandite, some lightly flocced slimes floating													3
Cond			20										2	
Rougher Stage 2														3
Cond			20								60	9.6	2	
Rougher Stage 3														3
Cond			20										2	
Rougher Stage 4														3
Cond			40										2	
Rougher Stage 5														3

Product	Wt	Wt	Assays						Distribution %					
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Mg %	Ni	Co	Fe	S	Cu	Mg
Prefloat	112.4	12.05	1.17	245	14.5	1.14	56	19.3	11.4	11.2	9.3	9.1	14.9	13.0
Rougher Conc 1	53.9	5.78	3.60	725	16	3.93	78	17.5	16.8	15.9	4.9	15.0	10.0	5.6
Rougher Conc 2	33.0	3.54	5.20	1050	16.4	5.57	100	16.3	14.9	14.1	3.1	13.0	7.8	3.2
Rougher Conc 3	23.5	2.52	5.80	1170	17.5	5.53	120	16.6	11.8	11.2	2.4	9.2	6.7	2.3
Rougher Conc 4	17.0	1.82	5.55	1110	18.1	5.68	115	15.9	8.2	7.7	1.8	6.8	4.6	1.6
Rougher Conc 5	17.3	1.86	6.95	1400	19	7.03	115	15.3	10.4	9.8	1.9	8.6	4.7	1.6
Tail	675.3	72.43	0.451	110	19.8	0.799	32	18	26.4	30.2	76.6	38.2	51.2	72.7
Calc head	932.4	100.0	1.24	264	18.7	1.514	45	17.9	100.0	100.0	100.0	100.0	100.0	100.0
Actual Head			1.20	255	18.1	0.92	30	-						
Rougher Conc 1		5.78	3.60	725	16	3.93	78	17.5	16.8	15.9	4.9	15.0	10.0	5.6
Rougher Cons 1 + 2		9.32	4.21	848.4	16.2	4.55	86	17.0	31.7	30.0	8.0	28.0	17.8	8.9
Rougher Cons 1 to 3		11.84	4.55	916.9	16.4	4.76	94	16.9	43.5	41.1	10.4	37.2	24.5	11.2
Rougher Cons 1 to 4		13.66	4.68	942.6	16.7	4.88	96	16.8	51.7	48.8	12.2	44.1	29.1	12.8
Rougher Cons 1 to 5		15.52	4.95	997.3	16.9	5.14	99	16.6	62.2	58.6	14.0	52.7	33.8	14.4

FLOTATION TEST DATA SHEET

TEST NO.	FT 4	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	1/9/00	RPM	60
SAMPLE :	A018A WC	MINUTES	35.5
WT (g)	1000	% SOLIDS	60

OBJECTIVE : As for test 3 with the crushed core composite
Float feed and tails were samples for magnetic separation tests

	REAGENTS		Eh	pH	COND	FLOAT
			mV		(min)	(min)
Cond	SIBX	MIBC 10	86	10	1	
Prefloat	Good froth. Some talc and slimes floating. Barren at the end.					5
Cond	20		52	10.0	2	
Rougher Stage 1						3
Cond	20		42	10.0	2	
Rougher Stage 2						3
Cond	20		29	10.0	2	
Rougher Stage 3						3
Cond	20		48	9.9	2	
Rougher Stage 4						3
Cond	40				2	
Rougher Stage 5						3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Prefloat	50.0	5.27	1.29	295	12.7	1.31	140	5.1	4.7	4.1	5.2	9.5
Rougher Conc 1	32.7	3.45	10.9	2280	19.9	12.0	270	28.0	23.7	4.2	30.8	12.0
Rougher Conc 2	31.0	3.27	7.60	1610	18.4	8.46	180	18.5	15.9	3.7	20.7	7.6
Rougher Conc 3	22.6	2.38	5.55	1210	17.2	5.56	170	9.8	8.7	2.5	9.9	5.2
Rougher Conc 4	19.6	2.07	3.45	780	16.3	3.38	150	5.3	4.9	2.1	5.2	4.0
Rougher Conc 5	23.4	2.47	2.40	565	15.5	2.45	105	4.4	4.2	2.4	4.5	3.3
Tail	768.9	81.09	0.48	155	16.2	0.392	56	29.0	37.9	81.0	23.7	58.4
Calc head	948.2	100.0	1.34	331	16.2	1.339	78	100.0	100.0	100.0	100.0	100.0
Actual Head			1.35	310	16.0	1.25	-					
Rougher Conc 1		3.45	10.90	2280	19.9	11.95	270	28.0	23.7	4.2	30.8	12.0
Rougher Cons 1 + 2		6.72	9.29	1954	19.2	10.25	226	46.4	39.6	7.9	51.5	19.5
Rougher Cons 1 to 3		9.10	8.31	1759	18.7	9.02	211	56.3	48.3	10.5	61.4	24.8
Rougher Cons 1 to 4		11.17	7.41	1578	18.2	7.98	200	61.6	53.2	12.5	66.6	28.8
Rougher Cons 1 to 5		13.64	6.51	1395	17.7	6.98	183	66.0	57.4	14.9	71.1	32.1

661C84

FLOTATION TEST DATA SHEET

TEST NO.	FT 5	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	1/9/00	RPM	60
SAMPLE :	A018A WC	MINUTES	35.5
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Based on test 4, without a prefloat, and with CMC to assist in dispersion and talc depression

	REAGENTS										Eh	pH	COND	FLOAT
			SIBX		MIBC		CMC (BDA)		CuSO4		mV		(min)	(min)
											58	10		
Cond			30				100						2	
Rougher Stage 1							20							3
Cond			20										2	
Rougher Stage 2														3
Cond			20				20				4	9.9	2	
Rougher Stage 3														3
Cond			40										2	
Rougher Stage 4														3
Cond			40						200		55	9.6	5/2	
Rougher Stage 5														3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	49.5	4.94	7.05	1510	17.2	7.12	200	25.8	22.3	5.2	26.9	7.9
Rougher Conc 2	43.8	4.37	8.00	1710	18.4	8.33	165	25.9	22.3	4.9	27.9	5.8
Rougher Conc 3	28.9	2.88	5.55	1200	16.7	5.86	145	11.8	10.3	3.0	12.9	3.4
Rougher Conc 4	20.8	2.07	3.60	825	16	3.67	135	5.5	5.1	2.0	5.8	2.3
Rougher Conc 5	14.1	1.41	5.85	2000	18.1	5.51	240	6.1	8.4	1.6	5.9	2.7
Tail	845.6	84.33	0.398	125	16.1	0.316	115	24.9	31.5	83.3	20.4	77.9
Calc head	1003	100.0	1.35	334	16.3	1.304	124	100.0	100.0	100.0	100.0	100.0
Actual Head			1.35	310	16.0	1.25	-					
Rougher Conc 1		4.94	7.05	1510	17.2	7.12	200	25.8	22.3	5.2	26.9	7.9
Rougher Cons 1 + 2		9.30	7.50	1604	17.8	7.69	184	51.7	44.6	10.1	54.8	13.7
Rougher Cons 1 to 3		12.19	7.04	1508	17.5	7.26	174	63.5	55.0	13.1	67.8	17.1
Rougher Cons 1 to 4		14.26	6.54	1409	17.3	6.73	169	69.0	60.1	15.1	73.6	19.3
Rougher Cons 1 to 5		15.67	6.47	1462	17.4	6.62	175	75.1	68.5	16.7	79.6	22.1

661085

FLOTATION TEST DATA SHEET

TEST NO.	FT 6	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	1/9/00	RPM	60
SAMPLE :	A019A WC	MINUTES	21
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Based on test 2.
Float feed and tails were samples for magnetic separation tests

	REAGENTS			Eh	pH	COND	FLOAT
		SIBX	MIBC	mV		(min)	(min)
Cond			10	19	10.0	2	
Prefloat	Talc and slimes at start. Barren after 2 min						5
Cond		20		32	9.7	2	
Rougher Stage 1	Lots of pentlandite.		10				3
Cond		10		36	9.7	2	
Rougher Stage 2	Good froth						3
Cond		10		43	9.7	2	
Rougher Stage 3							3
Cond		10	10	47	9.7	2	
Rougher Stage 4							3
Cond		40	10	45	9.6	2	
Rougher Stage 5							3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Prefloat	13.1	1.31	3.00	695	8.35	3.59	2160	1.1	1.1	0.7	1.1	9.0
Rougher Conc 1	43.1	4.30	28.8	6170	28.8	30.3	1300	35.9	31.0	8.1	29.5	17.8
Rougher Conc 2	31.5	3.14	24.5	5510	27.8	29.6	710	22.3	20.2	5.7	21.0	7.1
Rougher Conc 3	18.8	1.87	16.6	3870	23.0	20.7	655	9.0	8.5	2.8	8.8	3.9
Rougher Conc 4	12.5	1.25	12.7	3090	19.9	16.4	625	4.6	4.5	1.6	4.6	2.5
Rougher Conc 5	20.2	2.01	14.1	4100	24.6	21.2	770	8.2	9.6	3.3	9.7	4.9
Tail	863.9	86.12	0.755	250	13.7	1.30	200	18.8	25.1	77.7	25.3	54.8
Calc head	1003	100.0	3.45	856	15.2	4.417	314	100.0	100.0	100.0	100.0	100.0
Actual Head			3.48	840	15.1	4.19	-					
Rougher Conc 1		4.30	28.8	6170	28.8	30.3	1300	35.9	31.0	8.1	29.5	17.8
Rougher Cons 1 + 2		7.44	27.0	5891	28.4	30.0	1051	58.2	51.2	13.9	50.5	24.9
Rougher Cons 1 to 3		9.31	24.9	5484	27.3	28.1	971	67.2	59.6	16.7	59.3	28.8
Rougher Cons 1 to 4		10.56	23.5	5202	26.4	26.8	930	71.8	64.1	18.4	63.9	31.3
Rougher Cons 1 to 5		12.57	22.0	5025	26.1	25.9	905	80.0	73.8	21.6	73.6	36.2

661C8C

FLOTATION TEST DATA SHEET

TEST NO.	FT 7	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	4/9/00	RPM	60
SAMPLE :	A019A WC	MINUTES	21
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Based on test 6, without the prefloat, and with CMC to aid dispersion of slimes and for talc depression

	REAGENTS										Eh	pH	COND	FLOAT
			SIBX		MIBC		CMC (BDA)		CuSO4		mV		(min)	(min)
Cond			20				100				44	10.1	2	
Rougher Stage 1					10						22	10.2		3
Cond			10								25	10.1	2	
Rougher Stage 2														3
Cond			10								26	10.1	2	
Rougher Stage 3														3
Cond			10		10						25	10.1	2	
Rougher Stage 4														3
Cond			20		10				200		42	9.9	5/2	
Rougher Stage 5														3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	74.8	7.20	26.1	5720	27.7	29.6	1360	53.6	47.5	13.1	46.5	28.3
Rougher Conc 2	32.9	3.17	19.5	4610	26	24.9	805	17.6	16.8	5.4	17.2	7.4
Rougher Conc 3	18.0	1.73	14.5	3740	25.0	21.2	835	7.2	7.5	2.8	8.0	4.2
Rougher Conc 4	9.7	0.93	10.8	2910	20.5	15.2	775	2.9	3.1	1.3	3.1	2.1
Rougher Conc 5	34.3	3.30	11.7	4040	30.5	23.9	1020	11.0	15.4	6.6	17.2	9.7
Tail	869.5	83.67	0.321	100	12.9	0.442	200	7.7	9.7	70.8	8.1	48.4
Calc head	1039	100.0	3.50	867	15.2	4.59	346	100.0	100.0	100.0	100.0	100.0
Actual Head			3.48	840	15.1	4.19	-					
Rougher Conc 1		7.20	26.1	5720	27.7	29.6	1360	53.6	47.5	13.1	46.5	28.3
Rougher Cons 1 + 2		10.36	24.1	5381	27.2	28.2	1190	71.3	64.3	18.5	63.7	35.6
Rougher Cons 1 to 3		12.10	22.7	5146	26.9	27.2	1140	78.4	71.8	21.3	71.7	39.8
Rougher Cons 1 to 4		13.03	21.9	4986	26.4	26.3	1113	81.3	75.0	22.6	74.8	41.9
Rougher Cons 1 to 5		16.33	19.8	4795	27.2	25.8	1095	92.3	90.3	29.2	91.9	51.6

FLOTATION TEST DATA SHEET

TEST NO.	FT 8	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	14/9/00	RPM	60
SAMPLE :	18/19WC composite	MINUTES	25
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Based on test 6, with extra CuSO4 in the scavengers.

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond				65	10.0	2	
Prefloat	Barren after 2 to 3 minutes						5
Cond	20	10		36	9.9	2	
Rougher Stage 1							3
Cond	10					2	
Rougher Stage 2							3
Cond	10	10				2	
Rougher Stage 3							3
Cond			200	72	9.6	5	
Rougher Stage 4	10	10				2	3
Cond	10					2	
Rougher Stage 5							3
Cond			200	90	9.3	5	
Rougher Stage 6	20					2	3
Cond	10	10		78	9.4	2	
Rougher Stage 7							3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Prefloat	22.2	2.22	2.50	590	9.1	3.27	850	2.1	2.0	1.3	2.3	5.5
Rougher Conc 1	43.3	4.33	23.4	5020	25.3	24.5	1050	38.7	33.5	6.9	33.5	13.2
Rougher Conc 2	24.1	2.41	17.7	3940	23.4	19.5	630	16.3	14.7	3.5	14.8	4.4
Rougher Conc 3	17.9	1.79	13.5	3130	20.6	16.2	575	9.2	8.6	2.3	9.1	3.0
Rougher Conc 4	34.9	3.49	16.3	4790	30.6	26.3	885	21.7	25.8	6.7	29.0	9.0
Rougher Conc 5	11.4	1.14	5.6	1570	20.9	8.57	770	2.4	2.8	1.5	3.1	2.5
Rougher Conc 6	12.7	1.27	4.0	1230	16.5	5.27	805	1.9	2.4	1.3	2.1	3.0
Rougher Conc 7	10.0	1.00	2.3	655	15.2	3.13	705	0.9	1.0	1.0	1.0	2.0
Tail	823.5	82.35	0.213	72	14.6	0.20	240	6.7	9.2	75.5	5.1	57.4
Calc head	1000	100.0	2.62	648	15.9	3.17	344	100.0	100.0	100.0	100.0	100.0
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		4.33	23.4	5020	25.3	24.5	1050	38.7	33.5	6.9	33.5	13.2
Rougher Cons 1 + 2		6.74	21.4	4634	24.6	22.7	900	55.0	48.2	10.4	48.3	17.6
Rougher Cons 1 to 3		8.53	19.7	4318	23.8	21.4	832	64.2	56.8	12.7	57.4	20.6
Rougher Cons 1 to 4		12.02	18.7	4455	25.8	22.8	847	85.9	82.6	19.4	86.4	29.6
Rougher Cons 1 to 5		13.16	17.6	4205	25.3	21.6	840	88.4	85.4	20.9	89.5	32.1
Rougher Cons 1 to 6		14.43	16.4	3943	24.6	20.1	837	90.3	87.8	22.3	91.6	35.1
Rougher Cons 1 to 7		15.43	15.5	3730	24.0	19.0	829	91.2	88.8	23.2	92.6	37.1

FLOTATION TEST DATA SHEET

TEST NO.	FT 9	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	14/9/00	RPM	60
SAMPLE :	18/19WC composite	MINUTES	25
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Based on test 8, with CMC instead of a prefloat.

	REAGENTS					CMC	Eh	pH	COND	FLOAT
		SIBX		MIBC	CuSO4	(BDA)	mV		(min)	(min)
Cond				10		260		9.9	2	
Cond		20					25	9.8	2	
Rougher Stage 1										3
Cond		10		10			35	9.9	2	
Rougher Stage 2										3
Cond		10					33	9.8	2	
Rougher Stage 3										3
Cond					200		71	9.6	5	
Rougher Stage 4		10		10					2	3
Cond		10					53	9.6	2	
Rougher Stage 5										3
Cond					200		74	9.4	5	
Rougher Stage 6		10							2	3
Cond		10		10					2	
Rougher Stage 7										3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	67.8	6.78	21.7	4670	24.5	26.7	965	55.4	48.5	10.6	53.7	20.2
Rougher Conc 2	22.1	2.21	15.4	3550	21.3	18.6	640	12.8	12.0	3.0	12.2	4.4
Rougher Conc 3	10.9	1.09	13.5	3320	21.8	16.8	685	5.5	5.5	1.5	5.4	2.3
Rougher Conc 4	29.3	2.93	13.2	4290	29.9	22.0	945	14.6	19.3	5.6	19.1	8.6
Rougher Conc 5	8.9	0.89	5.4	1490	22.1	8.35	855	1.8	2.0	1.3	2.2	2.4
Rougher Conc 6	11.1	1.11	4.9	1560	17.7	5.34	925	2.0	2.7	1.3	1.8	3.2
Rougher Conc 7	8.0	0.80	2.1	565	14.1	3.17	870	0.6	0.7	0.7	0.8	2.2
Tail	836.5	83.65	0.228	72	14.3	0.19	220	7.2	9.2	76.1	4.8	56.9
Calc head	1000	99.5	2.66	652	15.7	3.365	324	100.0	100.0	100.0	100.0	100.0
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		6.78	21.7	4670	24.5	26.7	965	55.4	48.5	10.6	53.7	20.2
Rougher Cons 1 + 2		8.99	20.2	4395	23.7	24.7	885	68.2	60.6	13.6	65.9	24.6
Rougher Cons 1 to 3		10.08	19.4	4278	23.5	23.8	863	73.8	66.1	15.1	71.3	26.9
Rougher Cons 1 to 4		13.01	18.0	4281	24.9	23.4	882	88.3	85.4	20.7	90.5	35.5
Rougher Cons 1 to 5		13.90	17.2	4102	24.8	22.4	880	90.1	87.4	21.9	92.7	37.8
Rougher Cons 1 to 6		15.01	16.3	3914	24.2	21.2	883	92.2	90.1	23.2	94.4	41.0
Rougher Cons 1 to 7		15.81	15.6	3745	23.7	20.3	883	92.8	90.8	23.9	95.2	43.1

FLOTATION TEST DATA SHEET

TEST NO.	FT 10	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	14/9/00	RPM	60
SAMPLE :	18/19WC composite	MINUTES	25
WT (g)	1000	% SOLIDS	60

OBJECTIVE : No prefloat, and no CMC.
CuSO4 added from the start.

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	22	9.9	5	
Cond	20	10				2	
Rougher Stage 1	Good float, lots of mineralisation / sulphides. Good froth.						3
Cond	10	10		52	9.8	2	
Rougher Stage 2							3
Cond			200	51	9.6	5	
Rougher Stage 3	20	10				2	3
Cond	10	10		67	9.5	2	
Rougher Stage 4	Seemed very barren						3
Cond	10	10		74	9.4	5	
Rougher Stage 5						2	3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	82.1	8.21	21.5	5030	26.6	27.8	1040	66.9	63.4	13.5	66.3	25.5
Rougher Conc 2	33.4	3.34	12.1	3070	22.7	17.9	755	15.3	15.7	4.7	17.4	7.5
Rougher Conc 3	21.2	2.12	9.4	2510	23.5	13.7	920	7.6	8.2	3.1	8.5	5.8
Rougher Conc 4	16.2	1.62	2.6	675	13.7	3.88	730	1.6	1.7	1.4	1.8	3.5
Rougher Conc 5	13.1	1.31	1.9	485	13.4	2.65	700	0.9	1.0	1.1	1.0	2.7
Tail	836.2	83.62	0.243	78	14.7	0.21	220	7.7	10.0	76.2	5.0	54.9
Calc head	1000	100.2	2.64	651	16.1	3.436	335	100.0	100.0	100.0	100.0	100.0
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		8.21	21.5	5030	26.6	27.8	1040	66.9	63.4	13.5	66.3	25.5
Rougher Cons 1 + 2		11.55	18.8	4463	25.5	24.9	958	82.2	79.2	18.2	83.7	33.0
Rougher Cons 1 to 3		13.67	17.3	4160	25.2	23.2	952	89.8	87.3	21.3	92.2	38.8
Rougher Cons 1 to 4		15.29	15.8	3791	24.0	21.1	928	91.4	89.0	22.7	94.0	42.4
Rougher Cons 1 to 5		16.60	14.7	3530	23.1	19.7	910	92.3	90.0	23.8	95.0	45.1

FLOTATION TEST DATA SHEET

TEST NO.	FT 11	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 45µm
DATE :	14/9/00	RPM	60
SAMPLE :	18/19WC composite	MINUTES	36
WT (g)	1000	% SOLIDS	60

OBJECTIVE : No prefloat, and no CMC. Repeat of test 10 with a 45µm grind
CuSO4 added from the start.

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	80	9.8	5	
Cond	20	10		64	9.9	2	
Rougher Stage 1	Good float, lots of mineralisation / sulphides. Good froth.						3
Cond	10	10		60	9.8	2	
Rougher Stage 2							3
Cond			200	52	9.7	5	
Rougher Stage 3	20	10				2	3
Cond	10	10		64	9.6	2	
Rougher Stage 4	Seemed very barren						3
Cond	10	10		70	9.6	5	
Rougher Stage 5						2	3
Cond	10	10		66	9.6	2	
Rougher Stage 6							3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	87.8	8.75	20.0	4610	25	24.3	955	68.3	64.0	13.8	65.9	26.5
Rougher Conc 2	42.3	4.22	9.8	2610	20.6	14.4	630	16.1	17.5	5.5	18.8	8.4
Rougher Conc 3	22.4	2.23	7.7	2140	21.2	11.6	855	6.7	7.6	3.0	8.0	6.1
Rougher Conc 4	24.4	2.43	1.9	510	14.2	2.87	575	1.8	2.0	2.2	2.2	4.4
Rougher Conc 5	15.3	1.52	1.4	370	13.1	1.96	610	0.8	0.9	1.3	0.9	2.9
Rougher Conc 6	13.7	1.37	0.9	250	11.9	1.25	580	0.5	0.5	1.0	0.5	2.5
Tail	797.4	79.48	0.185	60	14.6	0.15	195	5.7	7.6	73.3	3.6	49.1
Calc head	1003	100.0	2.56	630	15.8	3.23	315	100.0	100.0	100.0	100.0	100.0
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		8.75	20.0	4610	25	24.3	955	68.3	64.0	13.8	65.9	26.5
Rougher Cons 1 + 2		12.97	16.7	3960	23.6	21.1	849	84.5	81.5	19.3	84.7	34.9
Rougher Cons 1 to 3		15.20	15.4	3692	23.2	19.7	850	91.1	89.0	22.3	92.7	41.0
Rougher Cons 1 to 4		17.63	13.5	3253	22.0	17.4	812	92.9	91.0	24.5	94.9	45.4
Rougher Cons 1 to 5		19.16	12.5	3024	21.3	16.1	796	93.8	91.9	25.7	95.8	48.4
Rougher Cons 1 to 6		20.52	11.8	2839	20.6	15.2	782	94.3	92.4	26.7	96.4	50.9

FLOTATION TEST DATA SHEET

TEST NO.	FT 12	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	15/9/00	RPM	60
SAMPLE :	18/19WC composite	MINUTES	25
WT (g)	500	% SOLIDS	60

OBJECTIVE : 1kg grind, product divided into two for 2 x 500g scouting flotation tests to examine gangue depressants.(tests 12 and 13). Test 12as a standard test without any depressants.

	REAGENTS										Eh	pH	COND	FLOAT
			SIBX		MIBC		CuSO4				mV		(min)	(min)
Cond							200				67	9.5	5	
Cond			20		20						56	9.3	2	
Rougher float stage 1														1
Stage 2														1
Stage 3														1

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	35.1	7.19	13.1	3040	19.6	15.0	715	35.7	33.9	8.8	34.1	19.9
Rougher Conc 2	22.6	4.63	13.8	3180	20.5	17.5	735	24.2	22.9	5.9	25.6	13.2
Rougher Conc 3	13.2	2.70	11.6	2710	20.1	14.2	695	11.9	11.4	3.4	12.1	7.3
Tail	417.2	85.47	0.867	240	15.3	1.04	180	28.1	31.8	81.8	28.1	59.6
Calc head	488	100.0	2.64	644	16.0	3.16	258	100	100	100	100	100
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		7.19	13.1	3040	19.6	15.0	715	35.7	33.9	8.8	34.1	19.9
Rougher Cons 1 + 2		11.82	13.4	3095	20.0	16.0	723	60.0	56.8	14.8	59.8	33.1
Rougher Cons 1 to 3		14.53	13.0	3023	20.0	15.6	718	71.9	68.2	18.2	71.9	40.4

661092

FLOTATION TEST DATA SHEET

TEST NO.	FT 13	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	15/9/00	RPM	60
SAMPLE :	18/19WC composite	MINUTES	25
WT (g)	500	% SOLIDS	60

OBJECTIVE : 1kg grind, product divided into two for 2 x 500g scouting flotation tests to examine gangue depressants.(tests 12 and 13). Test 13 with dextrin.

	REAGENTS								Eh	pH	COND	FLOAT
			SIBX		MIBC		CuSO4	Dextrin	mV		(min)	(min)
Cond							200		67	9.5	5	
Cond			20		20			80	56	9.3	2	
Rougher float stage 1			Froth started to become soupy / creamy with the dextrin.									1
Stage 2												1
Stage 3												1

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	32.2	6.82	9.9	2310	17.0	11.4	735	25.4	24.1	7.2	25.0	18.5
Rougher Conc 2	12.1	2.56	11.6	2740	17.2	13.8	860	11.2	10.7	2.7	11.4	8.1
Rougher Conc 3	11.5	2.43	10.6	2440	17.2	11.9	715	9.8	9.1	2.6	9.3	6.4
Tail	416.6	88.19	1.61	415	15.9	1.91	205	53.7	56.0	87.4	54.3	66.9
Calc head	472	100.0	2.65	653	16.0	3.10	270	100	100	100	100	100
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		6.82	9.9	2310	17.0	11.4	735	25.4	24.1	7.2	25.0	18.5
Rougher Cons 1 + 2		9.38	10.3	2427	17.1	12.0	769	36.6	34.9	10.0	36.4	26.7
Rougher Cons 1 to 3		11.81	10.4	2430	17.1	12.0	758	46.3	44.0	12.6	45.7	33.1

FLOTATION TEST DATA SHEET

TEST NO.	FT 14	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	19/9/00	RPM	60
SAMPLE :	18/19WC composite	MINUTES	25
WT (g)	500	% SOLIDS	60

OBJECTIVE : 1kg grind, product divided into two for 2 x 500g scouting flotation tests to examine gangue depressants (tests 14 and 15). Test 14 with Na2SiO3.

	REAGENTS						Eh	pH	COND	FLOAT
	SIBX		MIBC		CuSO4	Na2SiO3	mV		(min)	(min)
Cond					200		82	9.8	5	
Cond	20		20			600	75	9.7	2	
Rougher float stage 1	Strong froth.									1
Stage 2										1
Stage 3										1

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	58.0	11.66	10.4	2440	19.0	12.4	665	47.7	45.8	14.4	46.3	28.6
Rougher Conc 2	23.2	4.66	10.9	2550	18.4	13.0	685	20.0	19.1	5.6	19.5	11.8
Rougher Conc 3	14.0	2.81	8.5	2010	16.9	10.4	585	9.4	9.1	3.1	9.3	6.1
Tail	402.2	80.86	0.718	200	14.6	0.96	180	22.8	26.0	76.9	24.9	53.6
Calc head	497	100.0	2.54	622	15.4	3.12	272	100.0	100.0	100.0	100.0	100.0
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		11.66	10.4	2440	19	12.4	665	47.7	45.8	14.4	46.3	28.6
Rougher Cons 1 + 2		16.32	10.5	2471	18.8	12.6	671	67.7	64.9	20.0	65.7	40.3
Rougher Cons 1 to 3		19.14	10.2	2404	18.5	12.3	658	77.2	74.0	23.1	75.1	46.4

661094

FLOTATION TEST DATA SHEET

TEST NO.	FT 15	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	19/9/00	RPM	60
SAMPLE :	18/19WC composite	MINUTES	25
WT (g)	500	% SOLIDS	60

OBJECTIVE : 1kg grind, product divided into two for 2 x 500g scouting flotation tests to examine gangue depressants (tests 14 and 15). Test 15 with Na2SiO3 (reduced amount)

	REAGENTS					Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	Na2SiO3		mV		(min)	(min)
Cond			200			60	9.8	5	
Cond	20	20		200		50	9.6	2	
Rougher float stage 1	More normal froth.								1
Stage 2									1
Stage 3									1

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	45.9	9.13	11.7	2720	19.4	13.4	705	41.0	39.2	11.4	39.9	23.7
Rougher Conc 2	20.4	4.06	11.7	2740	18.9	13.3	735	18.2	17.5	4.9	17.6	11.0
Rougher Conc 3	11.4	2.27	11.9	2790	19.2	14.3	765	10.4	10.0	2.8	10.5	6.4
Tail	425.0	84.54	0.935	250	14.9	1.16	190	30.4	33.3	80.9	32.0	59.0
Calc head	503	100.0	2.60	634	15.6	3.065	272	100.0	100.0	100.0	100.0	100.0
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		9.13	11.7	2720	19.4	13.4	705	41.0	39.2	11.4	39.9	23.7
Rougher Cons 1 + 2		13.19	11.7	2726	19.2	13.4	714	59.3	56.7	16.3	57.5	34.6
Rougher Cons 1 to 3		15.46	11.7	2736	19.2	13.5	722	69.6	66.7	19.1	68.0	41.0

FLOTATION TEST DATA SHEET

TEST NO.	FT 16	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	5/10/00	RPM	60
SAMPLE :	A014A WC Comp	MINUTES	20.5
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Repeat of test 10 with this composite

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	43	9.6	5	
Cond	20	10 + 10				2	
Rougher Stage 1	Slightly brittle and effervescing at start : excess SIBX ??						3
Cond	10	10		77	9.7	2	
Rougher Stage 2	Good froth						3
Cond			200	58	9.4	5	
Rougher Stage 3	20	10				2	3
	Fairly barren, but with a good froth						
Cond	10	10		60	9.5	2	
Rougher Stage 4	barren						3
Cond	10	10		82	9.4	2	
Rougher Stage 5	barren						3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	71.6	7.16	27.3	5200	36.0	25.9	2580	74.1	71.7	26.0	53.5	46.6
Rougher Conc 2	32.3	3.23	12.0	2490	33.6	22.4	1080	14.7	15.5	11.0	20.9	8.8
Rougher Conc 3	18.8	1.88	4.90	990	25.4	13.3	820	3.5	3.6	4.8	7.2	3.9
Rougher Conc 4	23.9	2.39	1.61	325	15.2	6.40	525	1.5	1.5	3.7	4.4	3.2
Rougher Conc 5	10.0	1.00	1.49	300	15.0	6.18	630	0.6	0.6	1.5	1.8	1.6
Tail	840.3	84.03	0.179	44	6.3	0.50	170	5.7	7.1	53.0	12.1	36.0
Calc head	1000	99.7	2.64	519	9.9	3.46	397	100.0	100.0	100.0	100.0	100.0
Actual Head			2.07	420	9.4		203					
Rougher Conc 1		7.16	27.3	5200	36.0	25.9	2580	74.1	71.7	26.0	53.5	46.6
Rougher Cons 1 + 2		10.39	22.5	4358	35.3	24.8	2114	88.8	87.2	37.0	74.4	55.4
Rougher Cons 1 to 3		12.27	19.8	3842	33.7	23.0	1915	92.3	90.8	41.8	81.7	59.2
Rougher Cons 1 to 4		14.66	16.9	3268	30.7	20.3	1689	93.7	92.3	45.5	86.1	62.4
Rougher Cons 1 to 5		15.66	15.9	3079	29.7	19.4	1621	94.3	92.9	47.0	87.9	64.0

661696

FLOTATION TEST DATA SHEET

TEST NO.	FT 17	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	5/10/00	RPM	60
SAMPLE :	A019A WC Comp	MINUTES	21
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Repeat of test 10 with this composite

	REAGENTS										Eh	pH	COND	FLOAT
			SIBX		MIBC		CuSO4				mV		(min)	(min)
Cond							200				48	9.9	5	
Cond			20		10								2	
Rougher Stage 1	Good froth, lots of sulphides													3
Cond			10		10						58	9.9	2	
Rougher Stage 2	Good froth													3
Cond							200				57	9.6	5	
Rougher Stage 3			20		10								2	3
Cond			10		10						53	9.7	2	
Rougher Stage 4														3
Cond			10		10						63	9.6	2	
Rougher Stage 5														3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	111.6	11.16	25.8	6090	30.7	30.6	1380	77.3	75.0	21.5	73.8	34.4
Rougher Conc 2	28.1	2.81	15.2	3900	28.9	22.7	1180	11.5	12.1	5.1	13.8	7.4
Rougher Conc 3	19.7	1.97	8.35	2010	24.0	12.6	1270	4.4	4.4	3.0	5.4	5.6
Rougher Conc 4	10.3	1.03	3.30	760	15.1	4.91	1200	0.9	0.9	1.0	1.1	2.8
Rougher Conc 5	7.8	0.78	2.70	615	14.4	4.37	1230	0.6	0.5	0.7	0.7	2.1
Tail	855.9	85.59	0.235	76	12.8	0.28	250	5.4	7.2	68.8	5.2	47.8
Calc head	1000	103.3	3.73	907	15.9	4.63	448	100.0	100.0	100.0	100.0	100.0
Actual Head			3.48	840	15.1	4.19	-					
Rougher Conc 1		11.16	25.8	6090	30.7	30.6	1380	77.3	75.0	21.5	73.8	34.4
Rougher Cons 1 + 2		13.97	23.7	5649	30.3	29.0	1340	88.7	87.1	26.6	87.6	41.8
Rougher Cons 1 to 3		15.94	21.8	5200	29.6	27.0	1331	93.1	91.4	29.6	93.0	47.4
Rougher Cons 1 to 4		16.97	20.7	4930	28.7	25.7	1323	94.0	92.3	30.5	94.1	50.1
Rougher Cons 1 to 5		17.75	19.9	4741	28.0	24.7	1319	94.6	92.8	31.2	94.8	52.2

661097

FLOTATION TEST DATA SHEET

TEST NO.	FT 18	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	75µm
DATE :	5/10/00	RPM	60
SAMPLE :	A018AWC	MINUTES	35.5
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Repeat of test 10 with this composite

	REAGENTS					Eh	pH	COND	FLOAT
		SIBX	MIBC	CuSO4		mV		(min)	(min)
Cond				200		61	9.8	5	
Cond		20	10					2	
Rougher Stage 1	Good froth, lots of sulphides								3
Cond		10	10			64	9.8	2	
Rougher Stage 2	Strong froth, almost excessive								3
Cond				200		66	9.6	5	
Rougher Stage 3		20						2	3
	Still a good froth, without frother to this stage								
Cond		10	10			65	9.6	2	
Rougher Stage 4									3
Cond		10	10			57	9.6	2	
Rougher Stage 5									3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	70.3	7.03	13.1	3030	22.3	12.1	315	65.6	62.7	9.3	66.3	11.8
Rougher Conc 2	54.9	5.49	4.1	1000	17.7	3.91	235	15.8	16.2	5.8	16.7	6.9
Rougher Conc 3	38.7	3.87	2.4	735	16.9	2.15	325	6.5	8.4	3.9	6.5	6.7
Rougher Conc 4	33.6	3.36	0.8	215	14.9	0.76	335	1.8	2.1	3.0	2.0	6.0
Rougher Conc 5	21.2	2.12	0.5	130	14	0.56	395	0.8	0.8	1.8	0.9	4.5
Tail	798.7	79.87	0.165	42	16.1	0.12	150	9.4	9.9	76.3	7.7	64.0
Calc head	1000	101.7	1.40	340	16.8	1.29	187	100.0	100.0	100.0	100.0	100.0
Actual Head			1.35	310	16.0	1.25	-					
Rougher Conc 1		7.03	13.1	3030	22.3	12.1	315	65.6	62.7	9.3	66.3	11.8
Rougher Cons 1 + 2		12.52	9.1	2140	20.3	8.5	280	81.5	78.8	15.1	82.9	18.7
Rougher Cons 1 to 3		16.39	7.5	1808	19.5	7.0	291	88.0	87.2	19.0	89.4	25.5
Rougher Cons 1 to 4		19.75	6.4	1537	18.7	6.0	298	89.8	89.3	21.9	91.4	31.5
Rougher Cons 1 to 5		21.87	5.8	1401	18.2	5.4	308	90.6	90.1	23.7	92.3	36.0

661C98

FLOTATION TEST DATA SHEET

TEST NO.	FT 19	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	106 µm
DATE :	5/10/00	RPM	60
SAMPLE :	18/19 WC Comp	MINUTES	20.5
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Repeat test 10 with a coarser grind (same composite as for test 10)

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	64	9.9	5	
Cond	20	10				2	
Rougher Stage 1							3
Cond	10	10		62	9.8	2	
Rougher Stage 2							3
Cond			200	59	9.7	5	
Rougher Stage 3	20	10				2	3
Cond	10	10		60	9.7	2	
Rougher Stage 4							3
Cond	10	10		57	9.7	2	
Rougher Stage 5							3

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Rougher Conc 1	78.2	7.82	22.9	5310	28.1	26.9	1100	67.1	63.9	14.1	65.1	25.7
Rougher Conc 2	30.6	3.06	12.9	3220	23.5	17.9	845	14.8	15.2	4.6	17.0	7.7
Rougher Conc 3	18.2	1.82	10.3	2660	24.3	13.8	980	7.0	7.4	2.8	7.8	5.3
Rougher Conc 4	16.2	1.62	3.35	835	15.7	4.68	865	2.0	2.1	1.6	2.4	4.2
Rougher Conc 5	13.7	1.37	2.15	545	14.2	2.88	825	1.1	1.1	1.2	1.2	3.4
Tail	834.6	83.46	0.256	80	14.1	0.25	215	8.0	10.3	75.6	6.5	53.6
Calc head	1000	99.2	2.67	650	15.6	3.22	334	100.0	100.0	100.0	100.0	100.0
Actual Head			2.63	615	15.2	3.18	-					
Rougher Conc 1		7.82	22.9	5310	28.1	26.9	1100	67.1	63.9	14.1	65.1	25.7
Rougher Cons 1 + 2		10.88	20.1	4722	26.8	24.3	1028	81.8	79.0	18.7	82.1	33.4
Rougher Cons 1 to 3		12.70	18.7	4427	26.4	22.8	1021	88.9	86.5	21.6	89.9	38.8
Rougher Cons 1 to 4		14.32	17.0	4020	25.2	20.8	1004	90.9	88.6	23.2	92.2	43.0
Rougher Cons 1 to 5		15.69	15.7	3717	24.3	19.2	988	92.0	89.7	24.4	93.5	46.4

FLOTATION TEST DATA SHEET

TEST NO.	FT 20	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	75 µm
DATE :	5/10/00	RPM	60
SAMPLE :	18/19 WC Comp	MINUTES	25
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Roughing as per test 10 then two stages of cleaning

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	48	9.9	5	
Cond	20	10				2	
Rougher Stage 1							3
Cond	10	10		54	9.9	2	
Rougher Stage 2							3
Cond			200	68	9.6	5	
Rougher Stage 3	20	10				2	3
Cond	10	10		61	9.6	2	
Rougher Stage 4							3
Cond	10	10		56	9.7	2	
Rougher Stage 5							3
1st Cleaner					9.7		5
Stage	10			76	9.5	2	4.5
2nd cleaner stage 1				51	9.5		1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3							6

Product	Wt	Wt	Assays					Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	54.2	5.42	28.3	6810	34.2	29.6	1090	55.8	55.3	11.5	53.9	17.0
2nd Cleaner Satge 2	30.1	3.01	24.6	5650	34.2	27.7	1200	26.9	25.5	6.4	28.1	10.4
2nd Cleaner Stage 3	7.9	0.79	19.1	4300	31.2	24.8	1730	5.5	5.1	1.5	6.6	3.9
2nd Cleaner Tail	10.0	1.00	7.10	1930	19.9	7.57	1440	2.6	2.9	1.2	2.5	4.1
1st Cleaner Tail	37.5	3.75	1.45	345	10.9	2.04	705	2.0	1.9	2.5	2.6	7.6
Tail	861.8	86.18	0.23	72	14.3	0.218	230	7.2	9.3	76.7	6.3	57.0
Calc head	1000	100.2	2.75	667	16.1	2.97	348	100.0	100.0	100.0	100.0	100.0
Actual Head			2.63	615	15.2	3.18	-					
2nd Clnr conc 1		5.42	28.3	6810	34.2	29.6	1090	55.8	55.3	11.5	53.9	17.0
2nd Clnr Cons 1 to 2		8.43	27.0	6396	34.2	28.9	1129	82.7	80.8	18.0	82.0	27.4
2nd Clnr Conc 1 to 3		9.22	26.3	6216	33.9	28.6	1181	88.2	85.9	19.5	88.6	31.3
1st Clnr Conc		10.22	24.4	5797	32.6	26.5	1206	90.8	88.8	20.7	91.1	35.4
Rougher Conc		13.97	18.3	4333	26.8	19.9	1072	92.8	90.7	23.3	93.7	43.0

FLOTATION TEST DATA SHEET

TEST NO.	FT 21	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 106µm
DATE :	12/10/00	RPM	60
SAMPLE :	14/18/19WC comp	MINUTES	19
WT (g)	1000	% SOLIDS	60

OBJECTIVE : As for FT 10, with a coarser grind.
This is the first test with this composite.

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
After grind				115	10		
Cond			200	64	9.8	5	
Cond	20	10		56	9.8	2	
Rougher Stage 1							3
Cond	10	10		68	9.9	2	
Rougher Stage 2							3
Cond			200	60	9.6	5	
Rougher Stage 3	20	10		64	9.7	2	3
Cond	10	10		66	9.6	2	
Rougher Stage 4							3
Cond	10	10		60	9.6	2	
Rougher Stage 5							3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
Rougher Conc 1	56.3	5.63	23.6	5170	28.5	24.9	1510	46	56.3	53.2	12.5	46.7	25.6
Rougher Conc 2	38.8	3.88	15.0	3550	27.7	20.7	980	42	24.7	25.2	8.4	26.7	11.4
Rougher Conc 3	19.6	1.96	9.20	2230	26.2	16.5	990	41	7.6	8.0	4.0	10.8	5.8
Rougher Conc 4	11.0	1.10	4.25	1000	20.8	8.47	945	43	2.0	2.0	1.8	3.1	3.1
Rougher Conc 5	7.6	0.76	2.60	610	16.4	6.41	880	43	0.8	0.8	1.0	1.6	2.0
Tail	864.1	86.41	0.235	68	10.7	0.38	200	35	8.6	10.7	72.3	11.1	52.0
Calc head	1000	99.7	2.36	547	12.8	3.00	332	43	100.0	100.0	100.0	100.0	100.0
Actual Head			2.60	598	13.3	3.25	220	43					
Rougher Conc 1		5.63	23.6	5170	28.5	24.9	1510	46	56.3	53.2	12.5	46.7	25.6
Rougher Cons 1 + 2		9.51	20.1	4509	28.2	23.2	1294	45	80.9	78.4	20.9	73.5	37.0
Rougher Cons 1 to 3		11.47	18.2	4120	27.8	22.0	1242	44	88.6	86.4	25.0	84.2	42.9
Rougher Cons 1 to 4		12.57	17.0	3847	27.2	20.8	1216	44	90.6	88.4	26.7	87.3	46.0
Rougher Cons 1 to 5		13.33	16.2	3662	26.6	20.0	1197	44	91.4	89.3	27.7	88.9	48.0

FLOTATION TEST DATA SHEET

TEST NO.	FT 22	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 106µm
DATE :	12/10/00	RPM	60
SAMPLE :	14/18/19WC comp	MINUTES	19
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Cleaner test, with roughing as for FT21, and cleaner stages as per test 20.
The flotation time of the "2nd cleaner stage 3" was reduced (6 min to 3 min) as the froth was very barren.

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	55	9.8	5	
Cond	20	10				2	
Rougher Stage 1							3
Cond	10	10				2	
Rougher Stage 2							3
Cond			200			5	
Rougher Stage 3	20	10		63	9.5	2	3
Cond	10	10				2	
Rougher Stage 4							3
Cond	10	10				2	
Rougher Stage 5							3
1st Cleaner				67	9.7		5
Stage	10			61	9.5	2	4.5
2nd cleaner stage 1							1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3							3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	45.8	4.58	26.1	5840	32.9	32.6	1340	45	49.7	48.9	11.8	48.4	18.5
2nd Cleaner Satge 2	36.6	3.66	21.1	4690	33.0	24.8	1350	45	32.1	31.4	9.4	29.4	14.9
2nd Cleaner Stage 3	6.5	0.65	14.9	3320	33.2	20.7	1510	45	4.0	3.9	1.7	4.4	3.0
2nd Cleaner Tail	10.1	1.01	6.25	1480	20.3	8.54	1080	42	2.6	2.7	1.6	2.8	3.3
1st Cleaner Tail	24.5	2.45	1.86	430	11.8	3.17	680	43	1.9	1.9	2.3	2.5	5.0
Tail	875.3	87.53	0.263	70	10.7	0.44	210	38	9.6	11.2	73.2	12.5	55.4
Calc head	1000	99.9	2.40	547	12.8	3.086	332	44	100.0	100.0	100.0	100.0	100.0
Actual Head			2.60	598	13.3	3.25	220	43					
2nd Clnr conc 1		4.58	26.1	5840	32.9	32.6	1340	45	49.7	48.9	11.8	48.4	18.5
2nd Clnr Cons 1 to 2		8.24	23.9	5329	32.9	29.1	1344	45	81.9	80.2	21.2	77.8	33.4
2nd Clnr Conc 1 to 3		8.89	23.2	5182	33.0	28.5	1357	45	85.9	84.2	22.9	82.2	36.3
1st Clnr Conc		9.90	21.5	4805	31.7	26.5	1328	45	88.5	86.9	24.5	85.0	39.6
Rougher Conc		12.35	17.6	3937	27.7	21.9	1200	45	90.4	88.8	26.8	87.5	44.6

FLOTATION TEST DATA SHEET

TEST NO.	FT 23	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 106µm
DATE :	12/10/00	RPM	60
SAMPLE :	A018A WC	MINUTES	29
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Cleaner test, as for test 22.

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	60	9.9	5	
Cond	20	10				2	
Rougher Stage 1							3
Cond	10	10		53	9.9	2	
Rougher Stage 2							3
Cond			200			5	
Rougher Stage 3	20	10		64	9.7	2	3
Cond	10	10		63	9.7	2	
Rougher Stage 4							3
Cond	10	10		59	9.7	2	
Rougher Stage 5							3
1st Cleaner				59	9.6		5
Stage	10	10 @ 3 min				2	4.5
2nd cleaner stage 1	Light froth.			68	9.5		1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3							3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	12.6	1.26	27.2	5870	29.4	29.2	470	46	27.2	24.1	2.3	29.6	3.2
2nd Cleaner Satge 2	18.1	1.81	24.3	5370	29.5	25.9	545	45	34.9	31.7	3.4	37.6	5.3
2nd Cleaner Stage 3	7.3	0.73	18.3	4400	30.3	18.6	500	42	10.6	10.5	1.4	10.9	2.0
2nd Cleaner Tail	11.1	1.11	7.10	2430	17.6	5.94	470	29	6.3	8.8	1.2	5.3	2.8
1st Cleaner Tail	84.8	8.48	0.86	260	13.3	0.81	245	33	5.8	7.2	7.1	5.5	11.2
Tail	846.6	84.66	0.227	64	15.8	0.16	165	35	15.3	17.7	84.5	11.1	75.5
Calc head	1000	98.1	1.26	306	15.8	1.245	185	41	100.0	100.0	100.0	100.0	100.0
Actual Head			1.35	310	16.0	1.25	-	44					
2nd Clnr conc 1		1.26	27.2	5870	29.4	29.2	470	46	27.2	24.1	2.3	29.6	3.2
2nd Clnr Cons 1 to 2		3.07	25.5	5575	29.5	27.3	514	46	62.1	55.8	5.7	67.2	8.5
2nd Clnr Conc 1 to 3		3.80	24.1	5349	29.6	25.6	511	45	72.7	66.3	7.1	78.1	10.5
1st Clnr Conc		4.91	20.3	4689	26.9	21.2	502	43	79.0	75.1	8.3	83.4	13.3
Rougher Conc		13.39	8.0	1884	18.3	8.3	339	42	84.7	82.3	15.5	88.9	24.5

FLOTATION TEST DATA SHEET

TEST NO.	FT 24
PROJECT NO.	900
DATE :	12/10/00
SAMPLE :	A019A WC
WT (g)	1000

GRINDING MILL :	Stainless Steel Rod
Product fineness	80% passing 106µm
RPM	60
MINUTES	16
% SOLIDS	60

OBJECTIVE : Cleaner test, as for test 22.

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	76	9.8	5	
Cond	20	10		61	9.9	2	
Rougher Stage 1							3
Cond	10	10		74	9.9	2	
Rougher Stage 2							3
Cond			200	63	9.5	5	
Rougher Stage 3	20	10		65	9.6	2	3
Cond	10	10		70	9.7	2	
Rougher Stage 4							3
Cond	10	10		66	9.7	2	
Rougher Stage 5							3
1st Cleaner				65	9.7		5
Stage	10					2	4.5
2nd cleaner stage 1				70	9.5		1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3							3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	79.8	7.98	25.9	6360	31.8	32.6	1320	41	60.4	60.3	17.5	59.9	24.7
2nd Cleaner Satge 2	33.2	3.32	24.6	5770	32.9	31.1	1330	43	23.9	22.8	7.5	23.7	10.4
2nd Cleaner Stage 3	8.1	0.81	18.1	4180	30.0	25.7	1450	43	4.3	4.0	1.7	4.8	2.8
2nd Cleaner Tail	9.8	0.98	6.60	1520	17.4	9.01	1440	43	1.9	1.8	1.2	2.0	3.3
1st Cleaner Tail	23.1	2.31	1.73	405	9.6	2.47	970	43	1.2	1.1	1.5	1.3	5.3
Tail	846.7	84.67	0.337	100	12.1	0.42	270	34	8.3	10.1	70.6	8.2	53.6
Calc head	1000	100.1	3.42	842	14.5	4.345	426	41	100.0	100.0	100.0	100.0	100.0
Actual Head			3.48	840	15.1	4.19	-	41					
2nd Clnr conc 1		7.98	25.9	6360	31.8	32.6	1320	41	60.4	60.3	17.5	59.9	24.7
2nd Clnr Cons 1 to 2		11.30	25.5	6187	32.1	32.2	1323	41	84.3	83.0	25.0	83.7	35.1
2nd Clnr Conc 1 to 3		12.11	25.0	6052	32.0	31.7	1331	41	88.6	87.1	26.7	88.5	37.8
1st Clnr Conc		13.09	23.6	5713	30.9	30.0	1340	41	90.5	88.8	27.9	90.5	41.1
Rougher Conc		15.40	20.4	4917	27.7	25.9	1284	41	91.7	89.9	29.4	91.8	46.4

FLOTATION TEST DATA SHEET

TEST NO.	FT 25	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 106µm
DATE :	13/10/00	RPM	60
SAMPLE :	A014AWC	MINUTES	17
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Cleaner test, as for test 22.

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	50	9.7	5	
Cond	20	20				2	
Rougher Stage 1							3
Cond	10	10		85	9.6	2	
Rougher Stage 2							3
Cond			200	57	9.4	5	
Rougher Stage 3	20	10				2	3
Cond	10	10		85	9.4	2	
Rougher Stage 4							3
Cond	10	10		92	9.4	2	
Rougher Stage 5							3
1st Cleaner				80	9.5		5
Stage	10			96	9.3	2	4.5
2nd cleaner stage 1				75	9.3		1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3							3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	58.0	5.80	23.1	4540	33.2	29.4	1760	51	65.2	64.1	22.0	53.7	29.5
2nd Cleaner Satge 2	20.9	2.09	18.1	3510	32.9	26.0	1500	52	18.4	17.8	7.9	17.1	9.0
2nd Cleaner Stage 3	5.5	0.55	10.0	1970	31.3	20.3	1100	51	2.7	2.6	2.0	3.5	1.7
2nd Cleaner Tail	9.5	0.95	3.15	590	14.4	6.06	760	53	1.5	1.4	1.6	1.8	2.1
1st Cleaner Tail	39.1	3.91	1.21	245	9.15	3.00	480	49	2.3	2.3	4.1	3.7	5.4
Tail	862.2	86.22	0.237	56	6.35	0.74	210	42	9.9	11.7	62.5	20.2	52.3
Calc head	1000	99.5	2.05	411	8.8	3.172	347	50	100.0	100.0	100.0	100.0	100.0
Actual Head			2.04	413	8.9	3.38	203	49					
2nd Clnr conc 1		5.80	23.1	4540	33.2	29.4	1760	51	65.2	64.1	22.0	53.7	29.5
2nd Clnr Cons 1 to 2		7.89	21.8	4267	33.1	28.5	1691	51	83.6	81.9	29.8	70.8	38.5
2nd Clnr Conc 1 to 3		8.44	21.0	4117	33.0	27.9	1653	51	86.3	84.6	31.8	74.3	40.3
1st Clnr Conc		9.39	19.2	3761	31.1	25.7	1562	51	87.8	85.9	33.4	76.1	42.3
Rougher Conc		13.30	13.9	2727	24.7	19.0	1244	51	90.1	88.3	37.5	79.8	47.7

FLOTATION TEST DATA SHEET

TEST NO.	FT 26	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 106µm
DATE :	23/10/00	RPM	60
SAMPLE :	14/18/19WC comp	MINUTES	18.5
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Cleaner test as for test 22, with a regrind on the rougher conc before cleaning

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	105	10.1	5	
Cond	20	10		96	9.9	2	
Rougher Stage 1				90	10.0		3
Cond	10	10		104	10.0	2	
Rougher Stage 2							3
Cond			200			5	
Rougher Stage 3	20	10		80	9.7	2	3
Cond	10	10				2	
Rougher Stage 4							3
Cond	10	10		85	7.1	2	
Rougher Stage 5							3
Regrind	12 min regrind in stainless rod mil. Laser size o product was P80 of 22µm						
Cond			50			3	
1st Cleaner	5			125	9.0	2	5
Stage	10						5
2nd cleaner stage 1				132	9.0		1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3	5						3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	47.7	4.77	26.5	6760	31.4	31.8	2050	39	54.3	54.5	11.4	48.6	26.3
2nd Cleaner Satge 2	20.2	2.02	23.5	5780	31.7	30.1	1710	41	20.4	19.7	4.9	19.5	9.3
2nd Cleaner Stage 3	7.8	0.78	18.3	4800	32.5	27.6	1360	38	6.1	6.3	1.9	6.9	2.9
2nd Cleaner Tail	11.9	1.19	6.45	1570	18.1	11.91	1130	41	3.3	3.2	1.6	4.5	3.6
1st Cleaner Tail	57.3	5.73	2.70	635	18.4	6.29	925	43	6.6	6.2	8.0	11.6	14.3
Tail	853.1	85.31	0.251	70	11.1	0.33	190	36	9.2	10.1	72.1	8.9	43.6
Calc head	1000	99.8	2.33	591	13.1	3.12	371	39	100.0	100.0	100.0	100.0	100.0
Actual Head			2.60	598	13.3	3.25	220	43					
2nd Clnr conc 1		4.77	26.5	6760	31.4	31.8	2050	39	54.3	54.5	11.4	48.6	26.3
2nd Clnr Cons 1 to 2		6.79	25.6	6468	31.5	31.3	1949	40	74.7	74.3	16.3	68.1	35.6
2nd Clnr Conc 1 to 3		7.57	24.9	6297	31.6	30.9	1888	39	80.9	80.6	18.2	75.0	38.5
1st Clnr Conc		8.76	22.4	5654	29.8	28.3	1785	40	84.2	83.8	19.9	79.5	42.1
Rougher Conc		14.49	14.6	3670	25.3	19.6	1445	40	90.8	89.9	27.9	91.1	56.4

FLOTATION TEST DATA SHEET

TEST NO.	FT 27
PROJECT NO.	900
DATE :	23/10/00
SAMPLE :	A018A WC
WT (g)	1000

GRINDING MILL :	Stainless Steel Rod
Product fineness	80% passing 106µm
RPM	60
MINUTES	29
% SOLIDS	60

OBJECTIVE : Cleaner test as for test 23, with a regrind on the rougher conc before cleaning

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	83	10.0	5	
Cond	20	10		86	10.0	2	
Rougher Stage 1							3
Cond	10	10		69	9.9	2	
Rougher Stage 2							3
Cond			200			5	
Rougher Stage 3	20	10		69	9.8	2	3
Cond	10	10		71	9.8	2	
Rougher Stage 4							3
Cond	10	10		67	9.7	2	
Rougher Stage 5							3
Regrind	16 min regrind in stainless rod mil. Laser size o product was P80 of 37µm						
Cond			50			3	
1st Cleaner	5			96	9.6	2	5
Stage	10	10		98	9.5		5
2nd cleaner stage 1				93	9.4		1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3	5						3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	22.9	2.29	21.1	5350	23.3	22.4	575	39	37.6	35.8	3.0	40.8	4.3
2nd Cleaner Satge 2	14.0	1.40	21.3	5500	24.4	22.4	805	39	23.2	22.5	1.9	25.0	3.7
2nd Cleaner Stage 3	7.0	0.70	18.0	4960	23.0	18.8	750	36	9.8	10.2	0.9	10.5	1.7
2nd Cleaner Tail	36.8	3.68	2.60	740	15.1	2.85	410	35	7.4	8.0	3.2	8.3	4.9
1st Cleaner Tail	118.3	11.83	0.70	185	16.0	0.74	925	38	6.4	6.4	10.8	6.9	35.5
Tail	811.4	81.14	0.247	72	17.4	0.13	190	34	15.6	17.1	80.2	8.5	50.0
Calc head	1000	101.0	1.29	342	17.6	1.26	308	38	100.0	100.0	100.0	100.0	100.0
Actual Head			1.35	310	16.0	1.25	-	44					
2nd Clnr conc 1		2.29	21.1	5350	23.3	22.4	575	39	37.6	35.8	3.0	40.8	4.3
2nd Clnr Cons 1 to 2		3.69	21.2	5407	23.7	22.4	662	39	60.8	58.4	5.0	65.8	7.9
2nd Clnr Conc 1 to 3		4.39	20.7	5336	23.6	21.8	676	39	70.6	68.5	5.9	76.3	9.6
1st Clnr Conc		8.07	12.4	3240	19.7	13.2	555	38	78.0	76.5	9.0	84.6	14.5
Rougher Conc		19.90	5.5	1424	17.5	5.8	775	38	84.4	82.9	19.8	91.5	50.0

FLOTATION TEST DATA SHEET

TEST NO.	FT 28	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 106µm
DATE :	23/10/00	RPM	60
SAMPLE :	A019A WC	MINUTES	16
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Cleaner test as for test 24, with a regrind on the rougher conc before cleaning

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	67	10.1	5	
Cond	20	10		60	10.0	2	
Rougher Stage 1							3
Cond	10	10		48	10.0	2	
Rougher Stage 2							3
Cond			200	57	9.8	5	
Rougher Stage 3	20	10				2	3
Cond	10	10		56	9.9	2	
Rougher Stage 4							3
Cond	10	10		56	9.8	2	
Rougher Stage 5							3
Regrind	16 min regrind in stainless rod mil. Laser size of product was P80 of 28µm						
Cond			50			3	
1st Cleaner	5			109	8.9	2	5
Stage	10	10		109	9.0		5
2nd cleaner stage 1				114	9.0		1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3	5						3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	71.9	7.19	26.6	7000	30.2	31.5	1580	38	52.6	51.6	12.5	49.4	25.2
2nd Cleaner Satge 2	28.9	2.89	23.3	6540	28.1	31.0	1170	36	18.5	19.4	4.7	19.5	7.5
2nd Cleaner Stage 3	12.0	1.20	21.3	5840	30.7	30.2	1310	36	7.0	7.2	2.1	7.9	3.5
2nd Cleaner Tail	15.9	1.59	9.80	2750	19.5	15.27	1110	36	4.3	4.5	1.8	5.3	3.9
1st Cleaner Tail	42.1	4.21	5.85	1510	20.2	8.87	930	39	6.8	6.5	4.9	8.2	8.7
Tail	925.0	92.50	0.424	115	13.9	0.48	250	37	10.8	10.9	74.0	9.6	51.3
Calc head	1000	109.6	3.64	976	17.4	4.58	451	37	100.0	100.0	100.0	100.0	100.0
Actual Head			3.48	840	15.1	4.19	-	41					
2nd Clnr conc 1		7.19	26.6	7000	30.2	31.5	1580	38	52.6	51.6	12.5	49.4	25.2
2nd Clnr Cons 1 to 2		10.08	25.7	6868	29.6	31.3	1462	37	71.1	70.9	17.2	69.0	32.7
2nd Clnr Conc 1 to 3		11.28	25.2	6759	29.7	31.2	1446	37	78.2	78.1	19.3	76.9	36.2
1st Clnr Conc		12.87	23.3	6263	28.5	29.2	1405	37	82.4	82.6	21.1	82.2	40.1
Rougher Conc		17.08	19.0	5092	26.4	24.2	1288	37	89.2	89.1	26.0	90.4	48.7

FLOTATION TEST DATA SHEET

TEST NO.	FT 29	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 106µm
DATE :	23/10/00	RPM	60
SAMPLE :	A014A WC	MINUTES	17
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Cleaner test as for test 25, with a regrind on the rougher conc before cleaning

	REAGENTS			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	50	9.7	5	
Cond	20	10		40	9.7	2	
Rougher Stage 1							3
Cond	10	10		58	9.7	2	
Rougher Stage 2							3
Cond			200	58	9.6	5	
Rougher Stage 3	20	10				2	3
Cond	10	10		65	9.6	2	
Rougher Stage 4							3
Cond	10	10		63	9.6	2	
Rougher Stage 5							3
Regrind	16 min regrind in stainless rod mil. Laser size o product was P80 of 37µm						
Cond			50			3	
1st Cleaner	5			106	9.0	2	5
Stage	10	10		83	8.9		5
2nd cleaner stage 1				116	9.0		1.5
2nd cleaner stage 2		10					2
2nd cleaner stage 3	5						3

Product	Wt	Wt	Assays					Ratio	Distribution %				
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Ni/Co	Ni	Co	Fe	S	Cu
2nd Cleaner Stage 1	52.8	5.28	23.0	5350	32	30.5	2280	43	58.0	59.9	17.2	48.9	35
2nd Cleaner Satge 2	15.8	1.58	19.3	4400	33.9	28.9	1330	44	14.6	14.7	5.4	13.9	6
2nd Cleaner Stage 3	9.9	0.99	13.9	2840	34.9	26.3	955	49	6.6	6.0	3.5	7.9	3
2nd Cleaner Tail	17.9	1.79	4.25	895	16.4	9.24	555	47	3.6	3.4	3.0	5.0	3
1st Cleaner Tail	51.1	5.11	2.30	440	15.8	6.85	420	52	5.6	4.8	8.2	10.6	6
Tail	852.2	85.22	0.285	62	7.25	0.53	185	46	11.6	11.2	62.7	13.7	46
Calc head	1000	100.0	2.09	471	9.9	3.29	340	44	100.0	100.0	100.0	100.0	100.0
Actual Head			2.04	413	8.9	3.38	195	49					
2nd Clnr conc 1		5.28	23.0	5350	32	30.5	2280	43	58.0	59.9	17.2	48.9	35.4
2nd Clnr Cons 1 to 2		6.86	22.1	5131	32.4	30.1	2061	43	72.6	74.7	22.6	62.7	41.6
2nd Clnr Conc 1 to 3		7.85	21.1	4842	32.7	29.6	1922	44	79.1	80.6	26.1	70.7	44.4
1st Clnr Conc		9.64	18.0	4109	29.7	25.9	1668	44	82.8	84.0	29.1	75.7	47.3
Rougher Conc		14.75	12.5	2838	24.9	19.3	1236	44	88.4	88.8	37.3	86.3	53.6

FLOTATION TEST DATA SHEET

TEST NO.	FT 31	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	8/11/00	RPM	60
SAMPLE :	14/18/19WC comp	MINUTES	25
WT (g)	1000	% SOLIDS	60

OBJECTIVE : Cleaner test without the regrind and with CMC

	REAGENTS										Eh	pH	COND	FLOAT
			SIBX		MIBC		CuSO4	CMC			mV		(min)	(min)
Cond							200				112	9.8	5	
Cond			20		10						102	9.6	2	
Rougher Stage 1														3
Cond			10		10						95	9.6	2	
Rougher Stage 2														3
Cond							200				75	9.4	5	
Rougher Stage 3			20		10								2	3
Cond			10		10						75	9.4	2	
Rougher Stage 4														3
Cond			10		10								2	
Rougher Stage 5														3
Cond								60			104	9.4	2	
1st Cleaner														5
Stage			10					20					1	5
2nd cleaner					10			30			108	9.2	1	3.5
2nd cleaner stage					10									3

Product	Wt	Wt	Assays						Distribution %					
	g	%	Ni %	Co ppm	Fe %	S %	Cu ppm	Mg %	Ni	Co	Fe	S	Cu	Mg
2nd Cleaner Conc	97.4	9.74	22.6	5090	33.8	27.9	1360	2.60	88.6	87.1	25.1	87.0	43.4	1.6
2nd Cleaner Tail	18.5	1.85	4.75	1030	17.8	8.03	765	12.8	3.5	3.3	2.5	4.7	4.6	1.5
1st Cleaner Tail	61.6	6.16	0.59	140	9.00	1.04	385	17.0	1.5	1.5	4.2	2.0	7.8	6.6
Tail	819.2	81.92	0.194	56	10.9	0.24	165	17.5	6.4	8.1	68.1	6.2	44.2	90.3
Calc head	1000	99.7	2.48	569	13.1	3.13	306	15.9	100.0	100.0	100.0	100.0	100.0	100.0
Actual Head			2.60	598	13.3	3.25	220	-						
2nd Clnr conc		9.74	22.6	5090	33.8	27.9	1360	2.60	88.6	87.1	25.1	87.0	43.4	1.6
1st Clnr Conc		11.59	19.8	4442	31.2	24.8	1265	4.23	92.1	90.4	27.6	91.7	48.0	3.1
Rougher Conc		17.75	13.1	2949	23.5	16.5	960	8.66	93.6	91.9	31.9	93.8	55.8	9.7

Note : Mg assays were by AAS

APPENDIX 6

FLOTATION CYCLE TEST 30
DETAILS AND RESULTS

FLOTATION TEST DATA SHEET

TEST NO.	FT 30	GRINDING MILL :	Stainless Steel Rod
PROJECT NO.	900	Product fineness	80% passing 75µm
DATE :	30/10/00	RPM	60
SAMPLE :	14/18/19WC comp	MINUTES	25 per 1 kg cycle
WT (g)	1000 x 6 cycles	% SOLIDS	60

OBJECTIVE : Cycle test of 6 cycles with a regrind on the rougher conc before cleaning
pH and Eh data shown was recoded on Cycle 6

	REAGENTS (per cycle)			Eh	pH	COND	FLOAT
	SIBX	MIBC	CuSO4	mV		(min)	(min)
Cond			200	114	9.7	5	
Cond	20	10		109	9.7	2	
Rougher Stage 1							3
Cond	10	10		90	9.7	2	
Rougher Stage 2							3
Cond			200			5	
Rougher Stage 3	20	10		79	9.5	2	3
Cond	10	10		70	9.5	2	
Rougher Stage 4							3
Cond	10	10		75	9.5	2	
Rougher Stage 5							3
Regrind	Regrind in mini stainless rod mill. Laser size o products for each cycle are shown below.						
Cond			50			3	
1st Cleaner	5	10				2	5
Stage	10						5
2nd cleaner				106	9.3		3.5
Stage	5	10					3

Regrind Fineness (as P80 in µm) as measured by Malcvern Laser Sizer

	P80 µm	Regrind Time min
Cycle 1	22µm	10
Cycle 2	24µm	8
Cycle 3	26 µm	6
Cycle 4	27µm	5
Cycle 5	27µm	5
Cycle 6	27 µm	5

Cycle Test FT30 **14/18/19 WC Composite**

	Products	Wt g	Wt %	Assays					Distribution %				
				Ni %	Co ppm	Fe %	S %	Cu ppm	Ni	Co	Fe	S	Cu
Cycle 1	2nd Clnr Conc	88.2	9.9	23.8	5470	30.2	30.0	1530	-	-	-	-	-
	Tail	801.2	90.1	0.16	46	10.5	0.18	195	-	-	-	-	-
	Calc Total	889.4	100.0	2.51	583.9	12.5	3.14	327					
Cycle 2	2nd Clnr Conc	99.5	10.4	22.5	5150	30.2	27.9	1510	92.2	90.9	24.1	90.4	45
	Tail	852.9	89.6	0.22	60	11.1	0.35	215	7.77	9.08	75.9	9.58	55
	Calc Total	952.4	100.0	2.55	591.8	13.1	3.22	350	100	100	100	100	100
Cycle 3	2nd Clnr Conc	106.9	11.1	23.6	5390	32.5	28.3	1580	93.6	92.3	27.1	89.7	47.3
	Tail	857	88.9	0.20	56	10.9	0.41	220	6.42	7.69	72.9	10.3	52.7
	Calc Total	963.9	100.0	2.80	647.6	13.3	3.5	371	100	100	100	100	100
Cycle 4	2nd Clnr Conc	111	11.2	20.0	4560	28.8	27.8	1370	93	91.4	25.6	93.2	45.8
	Tail	877.4	88.8	0.19	54	10.6	0.26	205	7.02	8.56	74.4	6.79	54.2
	Calc Total	988.4	100.0	2.42	560	12.6	3.35	336	100	100	100	100	100
Cycle 5	2nd Clnr Conc	100	10.1	21.6	4980	28.9	27.8	1500	91.1	89.8	22.7	89	44.6
	Tail	886.4	89.9	0.24	64	11.1	0.39	210	8.9	10.2	77.3	11	55.4
	Calc Total	986.4	100.0	2.4	562.4	12.9	3.17	341	100	100	100	100	100
Cycle 6	2nd Clnr Conc	116.8	11.5	20.4	4670	29.8	27.5	1470	93.3	91.8	27.5	92.1	47.1
	Tail	898.1	88.5	0.19	54	10.2	0.31	215	6.65	8.17	72.5	7.93	52.9
	Calc Total	1015	100.0	2.51	585	12.5	3.44	359	100	100	100	100	100
	Circulating Load												
Cycle 6	2nd Clnr Tail	55.7	5.5	2.05	460	13.5	5.21	660	4.47	4.31	5.95	8.32	10.1
	1st Clnr Tail	141.7	14.0	0.43	175	10.2	1.84	425	2.36	4.17	11.4	7.47	16.5
	Calc Intermediate Concentrates												
Cycle 6	1st Cleaner Conc		17.0	14.5	3311	24.5	20.3	1208	97.8	96.1	33.5	100	57.1
	Rougher Conc		31.0	8.14	1896	18.1	12	855	100	100	44.9	108	73.7
	Overall Calc Total	5993	100	2.48	577	12.8	3.29	352	100	100	100	100	100
	Actual Head	6000		2.60	598	13.3	3.25	220					