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REPORT ON

FEASIBILITY STAGE 1
WINCHESTER PROSPECT
BATCHELOR MAGNESIUM PROJECT

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TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF APPENDICES

1.0 INTRODUCTION

This report presents the results of geotechnical, hydrogeological, hydrological, waste management and mining studies completed for the Stage 1 Feasibility study for the Batchelor Magnesium Project, Northern Territory.

1.1 Sources of Information

Information regarding the Magnesite resource to be used as the basis of this study came from cross-sections showing the defined recoverable resource blocks provided by Mt Grace Resources NL and Korab Resources Ltd. Information relating to mine operating performance and costs has been provided by appropriate suppliers and contractors and from Golder Associates internal databases for use in the study. The report has been updated during 2013, 2014 and 2015 using information provided by Bateman Tenova, Korab Resources, appropriate suppliers and contractors. The resource estimate has used in the original report dated 2001 has not been updated on the basis that the information used to prepare the resource statement has not materially changed since the estimate was last reported and that all the material assumptions and technical parameters underpinning this resource estimates continue to apply and have not materially changed. Consequently, no changes have been made to the overall design of the mine, including the pit, plant/crusher, ROM pad, and ancillary infrastructure. 2001 report assumed that the output from the mine would be processed in magnesium smelter to be built on site to produce magnesium metal, which would result in production of slag and other waste materials. This report assumes that no smelter will be built and that mine output will be crushed using mobile crushers to 25mm and sold as direct shipping magnesite ore to third parties to be processed off-site. Consequently the waste storage areas have been reduced in size to reflect modified space and waste storage requirements, and the plant/crusher site preparation has been revised to reflect modified foundation requirements. Costing of each item within the mine and mine infrastructure as well as the operating costs of the mine have been estimated on the basis of the information provided by Mt Grace, Golder Associates, Bateman Tenova, URS, Devmin Consultants and appropriate suppliers and contractors.

1.2 Accuracy of Study

This study has been carried out to a pre-feasibility level that is typically considered to have an accuracy of ±30%.

2.0 BACKGROUND INFORMATION

The Batchelor Magnesium Project is located approximately 85 km south of Darwin, approximately 4 km east of the township of Batchelor. The project is in close proximity to a variety of infrastructure including mains power, the sealed highway to Darwin and a gas pipeline (Figure 1).

It is intended to develop the Batchelor deposit to produce a direct shipping crushed magnesite rock (DSO) with an ultimate capacity of to supply 800,000 tonnes per year of coarse saleable magnesite rock from 1,000,000 tonnes per year ROM output. The initial first stage of the project is likely to be a mine with a plant consisting of crusher/screener and conveyors or wheel loaders with a ROM capacity of 250,000 tonnes per year, expanding over 3 years to ROM capacity of 1,000,000 tonnes per year - although the optimum size of the first stage mine and the speed of the capacity expansion will depend on the ability to secure offtake agreements and/or additional markets for DSO magnesite rock. Mining will be by conventional open pit methods.

3.0 SCOPE OF THE STUDIES

The scope of the proposed studies has been tailored to the Stage 1 feasibility study requirements and has largely been based on the technical studies already completed at the site. No significant field studies were undertaken at this stage other than a walkover survey and site familiarisation visit by a hydrogeological and a geotechnical specialist.

The completed scopes of work for the various components of the study are outlined below:

3.1 Geotechnical and Rock Mechanics Studies

The methodology was based on an approach that has been successfully applied to a number of similar studies that have been successfully conducted at many other sites and comprised a series of phases as follows:

- initial review of available reports and aerial photographs to provide a broad scale characterisation of the area and to identify major geomorphological, geological and geotechnical constraints
- an initial visit by a geotechnical engineer to confirm the desk top review studies, and assess the site conditions; the site visit focused on general ground conditions at the site, the exposures of material in the nearby Sundance open pit, and walkover surveys of alternative plant/crusher and waste disposal sites
- assessment of the field information and existing data and preparation of this report.

3.2 Mining and Geology Studies

The resource estimates that were used as the basis for this study were provided to Golder by Mt Grace as a series of hard copy cross-sections showing the defined recoverable resource blocks. These cross-sections were used as the basis for mine planning and scheduling for the study. Golder study was updated by incorporation of block models generated by Devmin Consultants Pty Ltd.

No site visit by a mining engineer was carried out as Golder personnel have some familiarity with the project area from previous studies carried out in the region.

The scope of the mine planning studies was as follows:

- identify the likely limits of open pit mining for the scale and extent of the project as currently proposed
- identify an appropriate part of the resource for the start of mining operations taking into account issues such as resource confidence and classification, grade variability of both Mg and relevant impurities, overburden thickness, ground water management, overburden disposal and ore stockpiling
- develop life of mine plan and pit sequencing including appropriate stage design
- select appropriate mining method and mining equipment including requirements for in-pit quality control and blending if necessary
- evaluate stockpiling and overburden disposal requirements
- liaise with groundwater specialists to develop ground water management and open pit drainage plan
- assess the open pit drainage requirements
- develop mine capital and operating cost estimates to ±30% based upon contractor mining including owner manning and equipment costs and an assessment of campaign mining
- assess likely impacts upon production schedule and mining costs of varying process inputs
- assess strategies available for selective mining and/or developing alternative mining areas to be able to react to changes in ore impurities if required
- develop closure and plant/crusher waste disposal strategies in conjunction with ground water and environmental specialists.

3.3 Hydrogeological Studies

The scope of the hydrogeological studies was as follows:

- an initial desk-top review of the available geological, climatic and hydrogeological information
- an initial site visit by a hydrogeological specialist to inspect site conditions
- evaluation of the available data and development of pit dewatering strategies to a standard commensurate with the requirements of the Stage 1 feasibility study.
- The information collected during the desk-top study and site visit were used to:
- provide preliminary design recommendations on the most appropriate methods of dewatering and depressurisation
- provide preliminary design recommendations on mine water management

- develop cost estimates for installation of dewatering facilities and estimates of likely power requirements for dewatering operations to an accuracy of ±30%.

3.4 Hydrological Studies

The scope of the hydrological studies was to:

- collect and collate available data, including stream flows, catchment areas and run-off characteristics, meteorological information and proposed design layouts
- prepare estimates of catchment yields for design storm events
- prepare preliminary designs of flood protection/control measures and river diversion - this will be integrated with the proposed mining schedule to allow for staged construction and deferment of capital expenditure
- prepare cost estimates to an accuracy of ±30%,

3.5 Waste Storage Studies

The scope of the waste design studies was to:

- identify options for the site for the waste storage facilities
- interpret results of chemical and physical analyses and assess the need for special measures, such as liners and underdrains
- prepare layouts for the various components of the facilities, including capacity calculations, extent and geometry of earthworks, drainage measures etc.
- consider options for a staged approach that could be adopted to minimise capital expenditure, including identification of the timing for moving to in-pit waste storage
- prepare cost estimates for the capital and operating costs to an estimated accuracy of ±30%.

4.0 GEOTECHNICAL STUDIES

Geomorphological and geotechnical conditions have been assessed using no sub-surface methods. This assessment involved a desk study comprising aerial photograph interpretation, study of the geological plan prepared by Mt Grace Resources as well as a walkover survey by a geotechnical engineer.

Geotechnical aerial photograph interpretation was undertaken by an Engineering Geologist Geotechnical Engineer using 1:20,000 scale colour aerial photography stereo-pair contact prints. Topographical, textural, colour and vegetation contrasts were used to map a total of three landform units and these are indicated on Figure 2. These landform units are denoted as follows:

- (SR) Rock present at the surface, conjectured to be present at shallow depth, or areas of coarse scree, pronounced relief.
- (RS) Red Soils including slope wash, gravelly scree and sediment deposits (conjectured fine to coarse grained soils), subdued relief.
- (SS) Black Soils principally associated with overbank flow from the watercourse (conjectured plastic fine grained soils), very subdued relief.

From studying the aerial photographs a number of geological contacts can be seen trending in a southwest to northeast direction and the topography appears to be controlled by alternate harder and weaker strata.

The geological map prepared by Mt Grace indicates that the higher topography to the south of the main valley on which the Magnesite Prospect is located principally comprises a sequence of dipping metasediments. These are Proterozoic in age and comprise shales, greywackes, siltstones with ironstone bands, sandstones, basic volcanics, quartzite and mafic extrusive igneous rocks.

4.1 Open Pit Stability

The assessment of likely open pit stability conditions was based on the available information provided by Mt Grace, comprising:

- surface geological maps
- boreholes logs from the exploration of the deposit conducted to date
- photographs of the recovered core from the diamond drill holes drilled as part of the exploration of the deposit, along with photographs of the trial pit excavation
- observations made of the general geological structure, rock strength and degree of fracturing observed in the recovered core and the trial pit excavation observed by Mr B Uren of Mt Grace
- a published paper on the Geology and Mining of the adjacent Sundance open pit (Simpson, 1994) provided to Golder by Mr Uren.

The Batchelor magnesium deposit occurs within the Coomalie Dolomite and it is understood that the pit will be developed entirely within this unit. The general characteristics of the dolomite, based on the information provided by Mt Grace are that it is generally massive, competent and below the upper weathered material and overlying deposits, of uniformly high to very high strength. Whilst the dolomite is veined on the micro-scale, the trial pit observations suggest that on the macro-scale (for example the scale of one 20m batter height) there is no apparent preferred orientation to the vein patterns. Mt Grace advised that there are no prominent joint, fault, shear or other dominant geological structures that were mappable in the trial pit. The core photographs and the photographs of the trial pit generally confirmed the Mt Grace observations.

The likely controls on the stability of the walls of the proposed open pit are as follows:

- the risk of rotational or circular failures in the upper soils and clays and weathered materials that overly the dolomite
- the influence of geological structure in the unweathered dolomite that could lead to a potential for structurally controlled pit wall instability
- the presence of cavities within the dolomite
- the influence of groundwater pressures behind the pit walls arising from the high regional groundwater table; any high groundwater pressures behind the pit walls could exacerbate any potential for structurally controlled and non-structurally controlled instability
- the influence of excavation technique – given the experience in the trial pit, poorly implemented blasting of the final pit walls could also significantly impact on the potential for wall instability.

4.2 Infrastructure Foundation Conditions

Geotechnical considerations in selecting a site for the plant/crusher include the following:

- avoiding land susceptible to frequent flooding
- avoiding areas of deep soft fine-grained surficial soils
- avoiding steep relief and topography
- selecting well drained sites with gentle relief
- selecting sites underlain by relatively shallow rock
- selecting sites where surficial soils are medium or coarse grained in preference to fine-grained.

On this basis, the (RS) red soil land form unit or (SR) shallow rock land form unit, (refer Figure 2) avoiding unfavourable topography, were assessed as the more favourable landform units on which to site the plant/crusher from a geotechnical perspective.

Possible alternative sites for the mine infrastructure were evaluated on the basis of:

- the inferred geomorphological and geotechnical conditions described above
- proximity to the electricity transmission line and gas pipeline
- hydrological conditions
- topographic constraints
- waste storage requirements
- road access needs
- the anticipated sizes of the plant/crusher and support facilities,

A footprint of 350 by 220 m has been allowed for the crusher and additional facilities such as office, ablution block, warehouse, etc.) with the ROM pad (500 by 220 m) directly abutting the crusher. The proposed infrastructure layout is shown on Figures 3 and 4, with the crusher located towards the eastern boundary of the tenement. The natural ground surface elevation across the crusher and ROM pad locations is between approximately RL80 and RL85. Formation of the surface level at approximately RL83 across the crusher site would lead to a balance of cut and fill in this area. This elevation is expected to be sufficiently above the anticipated peak flood levels.

It is inferred that the mafic extrusives, quartzite and siltstone make up the scree that flanks the northern edge of the hill range in the proposed location of the plant/crusher site.

North of the range in the main valley, fewer outcrops have been mapped due to the presence of surficial sediments covering the bedrock geology. The map indicates a number of areas of scree and some outcrops of Coomalie Dolomite and Whites Formation. The former is inferred to lie north of an inferred contact below the creek bed itself and as such is not believed to underlie the proposed plant/crusher site location. The Whites Formation possibly together with a siltstone member of the Wildman Formation may underlie the surficial scree, slope wash and pediment soil at and in the vicinity of the proposed plant/crusher site, The Whites Formation has two mapped units described as dark grey to black, carbonaceous(?), dolomitic(?) siltstone and tine grained sandstone, partially calcareous with dolomite bands.

With reference to Figure 2, foundation conditions across the proposed crusher site are inferred to comprise fine to coarse grained surficial soils including some gravel sized scree, underlain by relatively shallow rock which may be sandstone or siltstone and could be dolomitic in places. Minor gossanous cavities in a quartz vein are noted on the geological map in the vicinity. Cavities can also be present in dolomite and attention should be given during the next stage of the feasibility study to locating the presence of any possible cavities in the rock underlying the plant/crusher site. The size of cavities, if present, is likely to be relatively minor and at this stage they are not considered to be a sufficient reason to merit consideration of an alternative location.

The ground conditions are considered to be generally suitable for founding the likely plant/crusher infrastructure on compacted soil/gravel or on normal concrete pads.

Waste disposal facilities are also shown on Figures 3 and 4 and these are described more fully in a later section. Foundation conditions over the proposed fines dump area are likely to comprise (BS) black soil landform unit and (RS) red soil land form unit. At this stage the black soil unit expected to be present beneath parts of the fines dump is not anticipated to present significant stability problems as it is expected to be relatively shallow.

The location or a proposed access road to the crusher is shown on Figures 3 and 4, running off Crater Lake Road, which in turns runs off Batchelor Road. The proposed access road is not located within the tenement boundary, but has been sited so as to mainly traverse relatively high ground to avoid flood waters and the black soils present in the flood plain and so as to not require large amounts of cut and fill. An alternative access road alignment located within the tenement boundary and intersecting Batchelor Road is also shown as a contingency. However, the alternative alignment crosses the flood plain and is likely to encounter black soils over a significant length.

5.0 MINING STUDIES

5.1 Resource Model

Mt Grace provided a sectional, polygonal resource model for use as the basis for the mine planning studies (Uren, 2000). The model comprised a series of sections in PDF format showing block outlines interpreted from drill hole logging and assaying. A spreadsheet was also provided giving details of the volumes, tonnages and mineral qualities for each of the blocks shown on the sections. The ore blocks as supplied are not regular and are trapezoidal shapes with a general inclination of approximately 20°.

Mt Grace provided details of the definition of the ore blocks and resource estimation procedures as background for this study (Uren, 2000).

The Mineral Resources at a cut-off grade of 40% MgO are summarised in Table 1. The estimate was based on the cut-off grade and allowance for a minimum mining width of 4 -5m. The detailed estimate is included as Appendix A.

Table 1 Mineral Resources at a Cut-Off Grade of 40% MgO

Parameter	Units	Indicated	Inferred	Total
Quantity	Mt	12.20	4.40	16.60
SiO_2	%	5.70	5.50	5.70
Al_2O_3	%	0.70	0.84	0.74
TiO_2	%	0.03	0.04	0.03
Fe_2O_3	%	1.28	1.67	1.39
MnO	%	0.21	0.19	0.21
MgO	%	43.09	43.56	43.22
CaO	%	2.17	1.78	2.07
K_2O	%	0.01	0.00	0.01
Na_2O	%	0.09	0.11	0.10
SO_3	%	0.02	0.03	0.02
P_2O_5	%	0.04	0.05	0.04
LOI	%	46.41	46.25	46.37
$CaCO_3$	%	33.90	3.20	3.70
$MgCO_3$	%	85.80	85.90	85.80
MgO in Carbonate	%	41.00	41.10	41.00
Talc	%	6.50	7.80	6.90
Quartz	%	1.70	0.80	1.40

5.2 Mine Planning Model

In order to facilitate mine planning for the project, a model was developed by digitising the block outlines provided on the sections. Solid triangulations were then created by extending the digitised trapezoids by half the distance between sections. The triangulations were then filled with regular, rectangular blocks to approximate the ore blocks. Block qualities were then allocated from the spreadsheet. The model was validated against the original sectional resource estimates to ensure that a reasonable approximation of the resource tonnage was achieved.

It is important to note that this mine planning model is an approximate representation of the resource estimates provided by Mt Grace and is not suitable for the purposes of statutory reporting of ore reserves. The model is designed solely for use in this study for the development of mine plans and production schedules as the basis for cost estimation to an order of accuracy suitable for pre-feasibility study.

The parameters for the mine planning block model developed by Golder Associates are given in Table 2.

Table 2 Mine Planning Model Parameters

	X(Easting)	Y (Northing)	Z(RL)
Parent Block size (m)	10	10	5
Model Origin	723,000	8,556,400	-60
Model Limit	723,600	8,557,000	140

5.3 Open Pit Design

An ultimate open pit design was developed to include the majority of the ore in the resource model. It was considered unnecessary to carry out open pit optimisation studies at this stage of project development. The parts of the magnesite resource that will be mined will be defined on the basis of quality in terms of magnesium content and the amount of impurities present. The variable costs of mining and the waste:ore ratio will have little impact upon the size and shape of the open pit excavation over the current 18 year mine life examined in this study. The design criteria used are summarised in Table 3.

Table 3 Open Pit Design Criteria

Bench Height (m)	5
Distance between berms (m)	20
Berm Width (m)	5
Road Width (m)	20
Maximum Road Gradient (%)	12.5

The selection of open pit slope design parameters is detailed in Section 9.2 of this report based upon an initial assessment carried out by Golder Associates. The slope design parameters used in this study are summarised in Table 4.

Table 4 Final Pit Slope Design Parameters

Unit	Face Angle	Face Height
Overburden	40°	Irregular
Undisturbed rock	70°	20 m

The open pit contour for the Winchester deposit as at end of year 3 and at end of life is shown in Figure 5. The cross section of the open pit under staged development options is shown in Figure 8.

5.4 Open Pit Quantities

The open pit quantities are summarised on a bench by bench basis in Table 5.

Table 5 Open Pit Quantities

Ore						Waste rock		Overburden		Total		Waste/Ore Ratio	
Bench	bcm	tonnes	Mgo	CaO	SiO2	bcm	Tonnes	bcm	Tonnes	bcm	tonnes	BCM	tonnes
			%	%	%								
70	171,998	514,275	44.7	0.46	5.44	196,131	585,936	441,746	1,104,365	809,875	2,204,576	3.7	3.3
65	354,545	1,059,820	44.4	0.57	5.67	260,491	772,899	143,113	357,783	758,149	2,190,502	1.1	1.1
60	413,427	1,235,967	44.4	0.56	5.97	233,745	692,938	36,427	91,069	683,599	2,019,974	0.7	0.6
55	408,378	1,220,746	44.3	0.58	6.01	234,511	694,328	22,494	56,236	665,383	1,971,310	0.6	0.6
50	395,199	1,181,339	44.3	0.58	6.03	236,028	698,960	15,933	39,833	647,160	1,920,132	0.6	0.6
45	380,079	1,136,357	44.4	0.55	5.78	235,442	697,215	13,314	33,284	628,835	1,866,856	0.7	0.6
40	356,375	1,065,382	44.5	0.57	5.65	186,970	553,141	5,740	14,350	549,085	1,632,873	0.5	0.5
35	340,207	1,017,040	44.5	0.56	5.7	190,225	562,238	1,824	4,559	532,256	1,583,837	0.6	0.6
30	333,716	997,207	44.4	0.69	5.62	181,187	535,647	608	1,520	515,511	1,534,374	0.5	0.5
25	318,429	951,722	44.4	0.65	5.67	180,458	532,673			498,887	1,484,395	0.6	0.6
20	266,627	796,836	44.5	0.67	5.32	133,817	394,198			400,444	1,191,034	0.5	0.5
15	274,058	818,652	44.4	0.8	5.52	112,306	330,991			386,364	1,149,643	0.4	0.4
10	269,386	804,659	44.3	0.81	5.73	103,228	304,125			372,614	1,108,784	0.4	0.4
5	255,111	762,156	44.6	0.72	5.27	104,132	306,405			359,243	1,068,561	0.4	0.4
0	248,883	743,762	44.6	0.65	5.31	73,144	214,694			322,027	958,456	0.3	0.3
-5	250,694	749,375	44.6	0.57	5.41	58,750	172,493			309,444	921,868	0.2	0.2
-10	244,617	731,206	44.7	0.56	5.41	52,302	153,615			296,919	884,821	0.2	0.2
-15	230,258	688,269	44.7	0.53	5.44	54,268	159,387			284,526	847,656	0.2	0.2
Total	5,511,987	16,474,770	44.5	0.61	5.65	2,827,135	8,361,883	681,199	1,702,999	9,020,321	26,539,652	0.6	0.6

5.5 Production Rate

Winchester mine will produce and sell direct shipping crushed magnesite ore and the mine output is therefore not limited by the processing capacity. The key limiting factor is the ability to sell the magnesite to trading houses or end users (ea. quantities, timing and scheduling of sales under offtake agreements).

Consequently, the ramp-up schedule is very flexible and can be modified by accelerating or deferring the expansion of the mining capacity. It is assumed that the initial mine production will be 200,000 tonnes of coarse crushed magnesite ore per year in year 1 with an allowance to increase production capacity to 400,000 tonnes per year in year 2, and to 800,000 tonnes per year in year 3.

The ramp-up production schedule for magnesite ore is given in Table 6. The tonnage of saleable coarse DSO magnesite ore does not include allowance for 20% of fines that will be produced as by-product during preparation of saleable ore. The tonnage of ROM ore includes the allowance of 20% of fines.

Table 6 ROM Magnesite Production Schedule by Mineral Resources Classification

year	annual production	cumulative production	resource classification
1	250	250	indicated
2	500	750	indicated
3	1000	1750	indicated
4	1000	2750	indicated
5	1000	3750	indicated
6	1000	4750	indicated
7	1000	5750	indicated
8	1000	6750	indicated
9	1000	7750	indicated
10	1000	8750	indicated
11	1000	9750	indicated
12	1000	10750	indicated
13	1000	11750	indicated
14	450	12200	indicated
14	550	12750	inferred
15	1000	13750	inferred
16	1000	14750	inferred
17	1000	15750	inferred
18	725	16475	inferred

5.6 Mine Dumps

Overburden and waste rock storage sites were selected so that the outer toe will act as a drainage bund for the Coomalie Creek. The creek is a semi-permanent drainage across the proposed open pit site which will require diversion to allow continuity of the mining operations as discussed in Sections 9.5 and 9.6 of this report.

A dump height of 20m with a final slope of 20° degrees was adopted for the purposes of this study. These design criteria are appropriate for this stage of the study. Mine waste volumes are shown in Table 7.

Table 7 Mine Waste Volumes

Material	In situ volume (Mcm)	Loose Volume (Mcm)
Overburden	0.7	1.0
Waste Rock	2.8	3.9
Total	3.5	4.9

5.7 Open Pit and Mine Operations

A generally thin layer (up to 5m) of unconsolidated soil and unconsolidated alluvium overlies the massive magnesite at the Winchester deposit. This overburden will require progressive removal to expose the hard, consolidated magnesite. The overburden is saturated and will require draining as mining proceeds.

Conventional open pit mining methods using rubber-tyred trucks and either a hydraulic excavator or rubber-tyred, front end loader are proposed for excavation of the open pit materials. It is expected that the unconsolidated, overburden could be removed by tractor scraper if more convenient.

The magnesite will require blasting prior to excavation and mine benches are expected to be suitable for running rubber tyred mining equipment with minimal preparation.

The second alternative is to use continuous miners with wheel loaders and/or conveyors. This option would reduce the mining costs by removing the need for blasting and possibly crushing of ore as well. Either a TAKRAF, WIRTGEN, or VERMEER could be utilised to crush the rock in-situ. These continuous miners are available with conveyor loaders. Additional testing is recommended to assess this option. Preliminary assessment based on quotations from Bateman Tenova, Vermeer and Global Surface Mining suggests a 20%-30% reduction in mining/crushing costs per tonne over the life of mine.

Because of the low volumes of material to be excavated it is proposed that mining operations are carried out on a yearly campaign by employing a specialist mining contractor. The overall waste:ore ratio is for the proposed 18 year mine life is 0.6:1 t/t.

5.7.1 Magnesite Mining

The ore zones are in general characterised by a higher grade MgO and lower grade SiO_2 content.

Two options for mining method are being considered. Under Option 1, the magnesite ore will be drilled and blasted using conventional vertical blast holes prior to excavation.

Under Option 2, continuous miner will be used to crush ore in situ without blasting.

Five metre high mine benches are proposed as mining selectivity is not considered to be important.

Under Option 1, ore mining will be carried out using a Komatsu PC 1000 hydraulic backhoe or similar capacity front end loader.

Under Option 2, ore mining will be carried out using Vermeer T1255, Vermeer T1655, TAKRAF, or Wirtgen 4200 continuous miner with wheel loader and mobile conveyor.

The magnesite ore will be hauled to the stockpile using conventional 85t capacity rear dumps such as the Caterpillar 777.

The main access from the open pit to the ROM stockpile is 20 m wide with maximum grades of approximately 10% constructed from mine waste. The haul distance for the ore is about 1 km.

5.7.2 Waste Materials

Waste materials will be mined using the same equipment as for the magnesite ore. The unconsolidated overburden is likely to be amenable to removal by tractor scraper, but the choice of approach will largely be subject to equipment availability.

5.7.3 Mining Method

Drill and Blasting

Under Option 1, the magnesite will be blasted in 5m benches. Drilling and blasting will be carried out in advance of mining using 102 mm diameter holes and ANFO for dry holes and emulsion for wet holes. The typical pattern will be 4.0 x 3.5 m with 1 m sub-drill and a powder factor of 0.3 kg/t.

Pre-split blasting is proposed for the final walls of the open pit to ensure long term stability and minimise the risk of contamination from wall failures. Pre-splitting will be carried out using 89 mm diameter holes.

Continuous Mining and In-Situ Crushing

Under option 2, the magnesite will be crushed in-situ to a depth of 25cm using Vermeer T1255, Vermeer T1655, TAKRAF, or Wirtgen 4200 continuous miner or similar. The rock will be moved either by wheel loader or mobile conveyor onto trucks. If continuous miners are sued, there will be no requirement for pre-split blasting. This option needs to be assessed in more detail in the bankable feasibility study before final decision is made on the selection of mining method.

Method Selection

The selection of the method will depend also on the sales volumes. If the initial sales are low, it is possible that initially drill and blast method will be used and at a later date as the sales volumes (off-take agreements) increase the method will switch to using continuous miner and wheel loader.

5.7.4 Grade Control

If drill and blast method is used, grade control will be carried out by taking samples from the blast holes. Blast holes will be sampled at 2.5m intervals and assayed for MgO and major impurities. If continuous miner is used, samples will be taken from the in-situ crushed ore and assayed for MgO and major impurities.

5.7.5 Ancillary Equipment

The ancillary equipment fleet will provide services to ensure continued production and perform any general pit and dump work not directly involved in the mining operations. These will include:

- Road construction and maintenance.
- Dust suppression.
- Lighting of work areas.
- In pit refuelling and lubrication.
- Personnel transport.
- Mobile maintenance.
- Pit/slope dewatering.

- General construction and drainage earth works.

5.7.6 Manpower

The staff requirements for supervision and grade control are shown in Table 8. As it is proposed to carry out the mining operations on a campaign basis the supervision and grade control staff will only be required during the period when mining is in progress. Thus it is appropriate that contract and casual employees are used.

Table 8 Mining Staff

Production Manager (Casual)	1
Mine Surveyor (Contractor)	1
Mine Geologist/Quality Control (full time)	1
Crusher operator/mechanic	2
Samplers (Casual)	2
Total	7

5.7.7 Contractor Mining

It is proposed that a specialist contractor be used to carry out the open pit mining operations. The owner will be responsible for the mine planning, geology, survey and control of the operations (see above).

In 2001 indicative contractor rates were obtained from Henry Walker Eltin Ltd (HWE) a major contracting company with a large establishment in the Northern Territory. In 2014-2015, indicative contractor rates were obtained from several contractors including BGC Contracting and Global Surface Mining. A mining cost estimate has been developed based upon unit rates for the mining operations supplied by the contractors for use in this study as shown in Table 9.

For the purposes of this study it is also proposed to use a front end loader hired from the contractor to feed the crusher on a continuous basis. It may be appropriate for AusMag to consider purchase and operation of the front end loader. A mining cost estimate has been developed based upon unit rates for the continuous mining operations supplied by the contractors as shown in Table 9.

Table 9 Contractor Mining Costs – Drill and Blast

Overburden	Waste Rock	Magnesite Ore
$/bcm	$/bcm	$/bcm
3.0	9.0	12.0

Table 10 Contractor Mining Costs – Continuous Miner

Overburden	Waste Rock	Magnesite Ore
$/bcm	$/bcm	$/bcm
3.0	5.0	9.0

5.7.8 Summary of Operating Costs

Table 11 gives indicative mine operating costs for first 5 years of operation. Table 12 provides estimates project operating costs for Winchester magnesite DSO over the initial 5 year period when the project is ramped-up to 1million tonnes per year ROM capacity.

The cost for year 1 are estimated approximately $29/ty. Cost per tonne declines in year 2 to $25/t and declines further to approximately $20/t in subsequent years when project reaches its ultimate output capacity. The decline in operating cost per tonne is mainly due to lower waste rock to ore ratio in subsequent years and due to the fixed costs being spread over greater production volume.

Table 11 Summary of Mine Operating Costs over Initial 5 Years (Drill and Blast)

MINE OPERATING COSTS - (SHOVEL, DRILL AND BLAST DEVELOPMENT VARIANT)						
Year		1	2	3	4	5
Bench		70	70-65	65-60	60-55	55
Number of Excavators		2	2	2	2	2
Coarse Magnesite/Fines		80%	80%	80%	80%	80%
Coarse Magnesite		200,000	400,000	800,000	800,000	800,000
Fines		50,000	100,000	200,000	200,000	200,000
ROM Ore	T/yr	250,000	500,000	1,000,000	1,000,000	1,000,000
Overburden	T/yr	220,873	357,783	91,069	56,236	39,833
Waste Rock	T/yr	292,968	524,838	748,911	693,355	486,030
Total	T/yr	763,841	1,382,621	1,839,980	1,749,591	1,525,863
Staff	$/yr	317,500	508,333	930,000	930,000	930,000
Grade Control	$/yr	80,000	160,000	320,000	320,000	320,000
Dayworks	$/yr	40,000	40,000	40,000	40,000	40,000
ROM Loader	$/yr	150,000	150,000	300,000	300,000	300,000
Mobilisation	$/yr	75,000	75,000	75,000	75,000	75,000
Contract Mining	$/yr	2,143,952	4,003,853	6,356,015	6,147,548	5,505,888
Total	**$/yr**	**2,806,452**	**4,937,186**	**8,021,015**	**7,812,548**	**7,170,888**

Table 12 Winchester Project Operating Costs over Initial 5 Years (Drill and Blast)

WINCHESTER MINE OPERATING COST ESTIMATE OVER 5 YEARS (DRILL AND BLAST)					
Description	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5
WATER MANAGEMENT ($/YR.)	440,000	440,000	440,000	440,000	440,000
WASTE DUMPS ($/YR.)	180,000	180,000	180,000	180,000	180,000
MINE ($/YR.)	2,806,452	4,937,186	8,021,015	7,812,548	7,170,888
CRUSHING ($/YR.)	1,100,000	2,200,000	4,400,000	4,400,000	4,400,000
SUBTOTAL ($/YR.)	4,526,452	7,757,186	13,041,015	12,832,548	12,190,888
CONTINGENCY (30%)	1,357,935	2,327,156	3,912,304	3,849,764	3,657,267
TOTAL ESTIMATE	5,884,387	10,084,342	16,953,319	16,682,313	15,848,155
CAPACITY OUTPUT ROM MAGNESITE (T/YR.)	250,000	500,000	1,000,000	1,000,000	1,000,000
SALEABLE COARSE MAGNESITE COST ($/T)	29.42	25.21	21.19	20.85	19.81
COARSE MAGNESITE/FINES	80%	80%	80%	80%	80%
CAPACITY OUTPUT COARSE SALEABLE MAGNESITE (T/YR.)	200,000	400,000	800,000	800,000	800,000
CAPACITY OUTPUT FINES (T/YR.)	50,000	100,000	200,000	200,000	200,000

5.7.9 Capital Costs

The mining capital costs are summarised in Table 13.

Table 13 Summary of Project Capital Costs (Bench-by-Bench)

WATER MANAGEMENT	626,810
SITE INFRASTRUCTURE	1,079,310
WASTE DUMPS	108,925
MINE	1,293,290
SUBTOTAL	3,108,335
CONTINGENCY	932,501
TOTAL ESTIMATE	4,040,836

It is assumed that surveying of the operations for measurement and payment of the mining contractor will be carried out on a contract basis. Thus there is only a minimal requirement for technical facilities to support the mining operations. A more complete breakdown of costs is provided in Appendix B.

5.7.10 Contract Mining Operations

The proposed mining operations are very small scale and it is appropriate to consider campaign mining to reduce the costs of mining. Contractors advise that there is always unutilised mining equipment available in the Northern Territory during the wet season. This may mean that favourable mobilisation rates are available if the mining operations can be carried out during the wet season and a major component of the mining costs will be the mobilisation. In theory this is possible, as the magnesite should provide a good all weather, trafficable surface. It will be necessary to review the open pit design to ensure that an adequate sump and drains are established for appropriate management of in-pit water. Any proposals for wet season mining need to be considered in the context of the site water management plan and discharge licences.

6.0 HYDROGEOLOGICAL STUDIES

6.1 General

Although no dedicated hydrogeological investigations have been completed at the Winchester Prospect, hydrogeological data have been collected during earlier drilling programs and during excavation of the trial pit. These data include:

- groundwater levels
- ground water quality data
- ground water recovery times in drill holes after completion
- dewatering pumping rates during excavation of the trial pit
- ground water level drawdowns and recoveries associated with dewatering activities for the trial pit.

No hydrogeological data are available for the Sundance Gold Prospect, however, it is understood from discussions with Mr Tony McGill (Director of Mines, Department of Mines and Energy, Northern Territory) that;

- these pits were relatively shallow (approximately 20 m deep)
- caves intersected within the Coomalie Dolomite unit initially yielded high flows, but the long-term flows were manageable with "small sump pumps".

6.2 Hydrogeological Units

The major hydrogeological units at the site comprise:

- near-surface alluvial material (typically 2-15 m thick)
- Coomalie Dolomite

The near-surface materials comprise mixed alluvials deposited by the Coomalie Creek (right branch). These deposits include mostly clay and silt, with occasional lenticular sand and gravel bodies, presumably associated with prior stream channels.

The Coomalie Dolomite unit comprises both dolomite and magnesite in the area of the prospect. Where the ore occurs, there is commonly a thin veneer of weathered dolomite immediately below the contact, overlying fresh crystalline magnesite.

Drillhole logs for the Winchester Prospect commonly record the loss of circulation and/or sample return and possibly cavity formation, for several metres, at the interface of the alluvium and the upper weathered zone of the Coomalie Dolomite. This may indicate a higher transmissive zone at the base of these sediments and in the upper weathered zone of the Coomalie Dolomite.

Caves have been identified in the Coomalie Dolomite unit during exploration and resource drilling. The caves are commonly sediment-filled (generally clay and silt) and up to 12 m in thickness. In addition, minor cavities and vughs (less than 0.2 m in diameter) have also been observed in diamond drill core and samples of ore collected during development of the trial pit. It is understood that the caves are typically restricted to materials that are not classified as ore-grade. However, mining may intersect cavernous materials not intersected during earlier drilling programmes.

6.2.1 Aquifer Zones

The main aquifer zones are:

- the lenticular sand and gravel bodies within the alluvial deposits
- the weathered and cavernous zone at the top of the Coomalie Dolomite
- individual caves and cavernous zones within the Coomalie Dolomite.

The degree of hydraulic connection between individual cave systems and between these cave systems and the near surface alluvial deposits is not known and is expected to vary from place to place. Observations at the Sundance Gold Prospect noted above suggest poor interconnection between individual caves in the Coomalie Dolomite at that location. It is likely that caves that extend to the top of the dolomite receive ground water recharge from the overlying alluvial deposits.

Weathered and crystalline magnesite and clay zones within the alluvial deposits are all likely to have low permeabilities. Observations made during the dewatering of the trial pit indicate that the minor cavities and vughs that occur within the fresh magnesite are not major producers of groundwater. This suggests that these smaller openings are not interconnected, but rather blind cavities within the crystalline magnesite.

6.2.2 Water Levels and Groundwater Flows

The trial pit dewatering, documented in Uren (2001), provides a useful insight into various aspects of dewatering at the site.

Groundwater levels at the Winchester Prospect vary seasonally between 1 to 2 m below ground level (m bgl) during the dry season to at or above ground level during the wet season.

At the time of the site visit, a number of holes in the vicinity of the trial pit were slightly artesian. The aquifer(s) associated with these artesian flows were not identified, but are likely to comprise sand or gravel lenses within the alluvial deposits, confined by overlying low permeability clay. This interpretation is supported by the drilling results, which did not identify any groundwater intersections within the dolomite.

A number of holes drilled as part of the exploration and resource drilling programs were used to monitor groundwater level recoveries after completion of drilling. Low recovery rates were observed in four of these drill holes which were cased down through the overlying alluvial deposits and. therefore, monitored groundwater levels within the Coomalie Dolomite. The recovery rates varied from 1 to 2 m/d after one month, reducing to about 0.5 m/d after four months. Full recovery was not achieved until the wet season when the drill holes were inundated during flooding. Although this data is not sufficiently rigorous to estimate aquifer parameters, it does indicate that the magnesite/dolomite has low primary permeability.

More rapid water level recoveries were observed in other drill holes monitored. However, it is not known whether this response was due to ground water inflows within the dolomite or whether the surface casing was ineffective in sealing off the overlying alluvial deposits.

Sixty drill holes were used to monitor groundwater levels during dewatering operations at the trial pit, carried out between 27 September and 7 November 2000. The hydrographs for these drill holes show the following;

- pre-dewatering water levels were generally 1 m ±0.2 metres below ground level
- drawdown of water levels around the trial pit was about 2 metres after 3 weeks of mining from 25 September to 13 October 2000
- water levels stabilised at about 3 metres below ground level, from 13 October 2000 to end of dewatering on 7 November 2000 at a discharge rate of 1,300 to 1,730 ml/day (15 to 20 L/s)
- at completion of the trial pit the water levels in the drill holes rose above pre-dewatering levels to ground level or near ground level as a consequence of the onset of the wet season.

Discussions with Mr Bruce Uren indicate that the main inflows into the trial pit occurred from sand and gravel lenses within the alluvial deposits that were exposed in the pit wall during mining. This observation is broadly consistent with the observed small changes in groundwater level in comparison with the depth of the pit yet the moderately high rate of groundwater inflow. It is presumed that ground water in the alluvium drained laterally into the pit through a seepage face at the wall, thus limiting the amount of drawdown that could be developed.

Groundwater levels in some bores were measured before trial mining commenced. The distribution of water levels has been examined and, although a general flow gradient to the north east is indicated, the patterns of groundwater levels and changes in groundwater level are not clear from the data and are not presented in this report,

6.2.3 Groundwater Quality

Groundwater samples have been collected from two drill holes (MRC-I77 and MRC-214) and submitted for analysis of salinity, major ions, apart from chloride and sulphate) and heavy metals. Concentrations of total dissolved solids, major ions and heavy metals are low and within guidelines outlined for protection of aquatic ecosystems (ANZECC, 1992).

Total suspended solids (TSS) from pit discharge water during the dry season is a concern. During the dry season, water pumped from pit sumps would probably need to be routed through settling ponds to conform to guidelines of < 10% change in seasonal mean concentration (ANZECC. 1992)

6.3 Mine Dewatering

Dewatering will be required to maintain dry conditions during mining. This may not be practicable during parts of the wet season, when high rainfall rates and greater rates of groundwater inflow may flood the lowest part of the pit.

Dewatering requirements, methodologies and the predicted dewatering rates are discussed in the following sections.

6.3.1 Prediction of Dewatering Rates

There is a reasonable set of data with which to estimate the likely dewatering rates for the proposed open pit mine, as a result of monitoring the trial pit. There is not sufficient information to warrant establishing a numerical model, hut an analytical approach is appropriate. For the purpose of the prefeasibility study, the dry season dewatering rate (for the alluvial component) of the mine has been estimated by applying the Thiem equation for unconfined steady state radial flow to dewatering data collected during excavation of the trial pit.

A dewatering rate of 1,380 m^3/day (16 L/*s)* was used in the calculation. This rate was observed for the period after 6 weeks of trial pit mining, from 25 September to 7 November 2000.

The Thiem equation can be expressed as follows:

$$Q = \frac{\pi k(H^2-h^2)}{\ln(R/r)}$$

where:

Q= discharge rate from Trial Pit (1,380 m^3/day)

k = hydraulic conductivity (m/day)

H = estimated thickness of aquifer (15 m)

h =Saturated aquifer thickness at seepage face along the pit wall (3 m)

R = distance of zero water table drawdown from centre of pit (estimated as 500 m)

r = effective radius of Trial Pit (49 m, derived as 125 m length +30 m width)/π)

Re-arranged, this equation yields an estimated hydraulic conductivity of 14 m/day for the trial pit data. This estimate of hydraulic conductivity is judged appropriate for the more permeable parts of the alluvial material.

The Thiem analysis was checked using monitor bore data. Analysis using Jacob's straight line method yielded transmissivity values in the approximate range 320 to 430 m^3/day, which, for an aquifer thickness of 15 m, is equivalent to a hydraulic conductivity range of 21 to 29 m/day. These values are similar to those obtained by the Thiem steady state analysis.

The Thiem equation can then be re-applied to estimate the dewatering rate (Q) for the mine when it has penetrated the alluvial cover. In this approach, the term "In(R/r)" is adjusted for the larger size of the mine to estimate inflows for the same depth of the trial pit.

The Thiem equation can be expressed as follows:

$$Q = \frac{\pi k(H^2-h^2)}{\text{In}(R/r)}$$

In this case:

k = 14 m/day

R= distance of zero watertable drawdown from centre of pit (estimated as 500 m)

r = effective radius of mine (255 m, derived as 500 m length + 300 m width)/π.)

Hence Q = 5,300 m^3/day (60 L/*s)* for the mine at the same depth as the trial pit (i.e. fully penetrating the alluvium). This is about four times the rate of inflow from the alluvium towards the end of trial mining and is regarded as an estimate of the steady state dewatering rate, i.e. the rate that would apply towards the end of the dry season after the alluvium has been fully penetrated by the mine.

6.4 Water Supply Considerations

Dust suppression water will be required during the dry season, mainly for the haul roads. The quantity required is likely to be in the approximate range 500 to 1,000 m^3/day and this would be available from the dewatering discharge. "Dirty" water pumped from the pit sumps could be used for dust suppression.

6.5 Disposal of Surplus Mine Water

Figure 6 provides detail of dewatering infrastructure along the lines of shallow bore and inpit sump pumping. Alternative dewatering infrastructure which assumes staged development is shown in Figure 7.

Clean water from the shallow bores (or from options of interception trench/ underground drain) will then be pumped to holding ponds northeast of the pit adjacent to the diversion channel. Whilst a proportion of the water in the ponds will be used for processing and dust suppression, most will discharge into the diversion channel northeast of the pit. A second discharge point from the shallow bore ring may be extended the short distance to Coomalie Creek, north of the waste dump.

Dirty water pumped from the pit floor sumps during the dry season will be pumped into unlined settling pond located adjacent to the clean water holding ponds, northeast of the pit adjacent to the diversion channel. A discharge rate of 4,800 m_3/day into a settling pond of dimensions 20 x 6 m x 1.5 m depth will take approximately one hour for water to flow through and allow particles, consisting of medium to coarse grained silt fraction and sand greater than 20 μm in size, to settle out. The flow through water will then be discharged into Coomalie Creek. As the mining progresses into fresh rock, the quantity of suspended solids in the water may decrease sufficiently for the water to be discharged directly into Coomalie Creek during the dry season.

In the storms expected during the wet season it is anticipated that the larger quantities being pumped from pit sumps can be discharged directly into Coomalie Creek, which will then have a high level of suspended solids. However, it is recognised that the settling ponds will probably be required during drier periods, even in the wet season.

6.6 Estimated Costs for Dewatering Operations

A cost estimate for dewatering infrastructure is presented in Appendix B. Table 14 provides a summary of the cost estimate for bore, pumps and piping infrastructure for 10 shallow dewatering bores, in-pit sump pump, plant/crusher diversion and ponds. A contingency cost for 5 deep dewatering bores which may need to be installed at a later stage during the development of the pit has been included in cost estimates for dewatering infrastructure. It is likely that the deep bores will only be required if caves or other high transmissive features are identified during mining. It is therefore, not possible to estimate the timing for installation of these bores.

Piping costs are for the following lengths:

- pipeline linking up 10 shallow bores
- from shallow bore piping ring to discharge point in Coomalie Creek.
- from shallow bore piping ring to holding ponds,
- from holding ponds to plant/crusher
- from in-pit sump to settling pond.

Ten pumps will be required for the 10 shallow bores, a pump located in the in-pit sump and a pump at the holding ponds for directing water to the process plant.

Shallow bores preferably should be located in deeper zones of the cover sediments. Bore depth would be extended to within the weathered Coomalie Dolomite to a nominal depth between 10 and 25 metres depending on the depth to the bottom of the cover sediments. Pumps for the shallow bores should be capable of pumping efficiently up to 12 L/s.

Table 14 Summary of Estimated Dewatering Infrastructure Costs (Bench-by-Bench)

BORE CONSTRUCTION	225,000
DEWATERING PUMPS	301,700
PONDS	3,261
PIPELINE	96,849
WATER MANAGEMENT TOTAL	626,810

Pond dimensions of 25 x 6 x 1.5m (settling pond) and 2 ponds of 10 x 5 x 1.5m depth (holding ponds).

The main uncertainty in the dewatering cost estimate is associated with the allowance for deeper dewatering bores to intercept ground water potentially flowing through cave systems to the pit. As discussed, the rationale for allowing for external bores is that they should allow more simple discharge of clean water to the environment than water pumped from a sump in the mine floor.

The need for and practicability of installing these deeper bores may be assessed as part of the future feasibility studies.

An annual operating cost of about $95,000 has been estimated, and is calculated as 15% of the initial capital cost. The operating cost would cover expenses such as maintenance and replacement of the pipeline, bore and sump pumps.

6.7 Power Requirements

The power requirements have been estimated assuming that the bore pumps and holding pond pump will operate 100% of the time, with the sump pump operating for half of the year (i.e. 50%). It is estimated that about 450,000 kWh of power will be required per annum. This value also assumes the sump pump is pumping 100 m head of water to the settling pond at the surface. For a head of about 60 m, the power requirement is estimated to be about 410,000 kWh.

7.0 HYDROLOGICAL STUDIES

7.1 Background

The mine tenement is located in a catchment approximately 22 km^2 in area. Based on the digital contour information supplied by Mt Grace Resources, the land slopes towards the east. The proposed mine catchment consists of relatively flat terrain with some ridges towards the north of the catchment and towards the south eastern boundary. The creeks and streams within the mine catchment are mainly ephemeral, that is they flow periodically, after significant rainfall events. Defined creeks function as waterways during small events with larger events covering a wider area of floodplain. Interpretation of the topography and creeklines are based on maps provided by Mt Grace Resources and a 1:50,000 scale topographic map sheet for the Batchelor region. The contour interval in the data is 2 m.

The proposed minesite is situated on the Coomalie Creek. This creek is approximately 1.5 m wide and about 0.5 m deep at the proposed mine. During a site visit by Golder Associates in March 2001, it was noted that the creek was flowing and the surrounding floodplains had recently been covered by flood waters. It was also noticed that the floodwaters recede relatively quickly after a flood, in the order of hours.

7.2 Climate

Batchelor is located in an area described in Australian Rainfall and Runoff (Institution of Engineers, 1998) (ARR), as *humid zone* of the Northern Territory. The Bureau of Meteorology records closest to Batchelor are for Batchelor Post Office station (rainfall, based on 37 years of data) and Darwin (evaporation, based on 43 years of data). These provide a reasonable indication of likely climatic conditions at the site. (Table 15)

Table 15 Climatic Data

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Average Annual
Mean Monthly Rainfall (mm)	300	313	261	57	17	4	2	3	21	53	125	203	1346
Mean Monthly Daily Evaporation (mm/day)	6.6	6.0	6.1	6.9	7.2	7.1	7.3	7.6	8.2	8.4	7.9	7.1	7.3

(Data supplied by Bureau of Meteorology)

7.3 Rainfall Runoff Analysis

Australian Rainfall and Runoff (ARR) has been used to assess runoff from the catchment (I.E. Aust,1998). Peak flow will be generated for a storm event that coincides with the time of concentration (t_e) for the catchment. This is the time required for the runoff from the remotest part of the catchment to reach the point of interest. The t_e for the catchment contributing to the diversion channel is approximately 3 hours.

The intensity frequency distribution (IFD) curves for this locality have been generated by the Bureau of Meteorology from ARR. Appropriate data from these curves is summarised in Table 16.

Table 16 Summary of Rainfall Intensity

Storm Duration (hours)	Rainfall Intensity (mm/hr)					
	2yr ARI	5yr ARI	10yr ARI	20yr ARI	50yr ARI	100yr ARI
1	56.00	67.90	75.00	85.20	99.20	110.00
2	34.80	42.00	46.40	52.60	61.10	67.70
3	25.70	30.90	34.10	38.60	44.70	49.50
12	9.03	10.90	12.10	13.70	16.00	17.70
24	5.65	7.06	7.93	9.14	10.80	12.10
72	2.58	3.45	4.05	4.84	5.96	6.90

7.4 Selection of Design Storm Event

For the diversion channel and bund design, a 1 in 50 year (ARI) event, 3 hour duration has been chosen. There is a probability of approximately 60% that the design preliminary flows determined will be exceeded during the assumed 18 year life.

8.0 WASTE STORAGE

8.1 Waste Streams

In the 2001 feasibility study it was assumed that the processing of magnesite into magnesia and then into magnesium metal will result in production of waste in the form of feed fines, crushed slag, treated sludge and kiln dust.

The current feasibility study assumes production of direct shippable crushed magnesite ore. Therefore only the fines will be produced as by-product and no crushed slag, treated sludge and kiln dust will be produced thus removing the need to deal with the storage or treatment of these wastes.

8.1.1 Fines

Fines comprise minus 6 mm size magnesite that will be dry screened from the saleable ore after crushing. The material has an estimated 10% finer than 80 μm and a water content of about 5 to 10%. The fines will be deposited in a stockpile by conveyor and have been assumed to be transportable by a conventional loader or dozer. Approximately 50,000 tonnes per year of feed fines are anticipated in the first year of operation, increasing to 200,000 tonnes per year in year 3. For the purposes of this study, the bulk unit mass of the material was assumed as 1.4 t/m^3.

8.2 Storage Options

Options that were examined for storage of the waste streams included the following:

- in-pit storage using the two old Sundance pits located to the west of the deposit, about 2 km from the proposed plant/crusher site. The pits have an estimated total volume of about $60,000m_3$ and at the time of the site visit were full or partly full of water
- in-pit storage using the old quarry located to the east of the tenement, about I km from the proposed plant/crusher site. The quarry has a volume that is estimated to be about half that available in the Sundance pits and at the time of the site visit was dry
- above ground storage in individual waste dumps
- co-disposal of wastes either in-pit or above ground

2015 feasibility update has shown that fines can be sold to agricultural and feedstock users, and so in general it is planned that the fines should be stored in easily accessible location. Therefore in-pit storage of the fines has been discounted. However, it is anticipated that the fines will be relatively easy to handle and so storage in relatively close proximity to the crusher site is likely to be the most cost effective solution.

9.0 CONCLUSIONS AND RECOMMENDATIONS

9.1 Crusher Site and Infrastructure

Foundation conditions across the proposed crusher site (Figure 2) are inferred to comprise fine to coarse grained surficial soils including some gravel sized scree, underlain by relatively shallow rock which may be sandstone or siltstone and could be dolomitic in places. Minor gossanous cavities in a quartz vein are noted on the geological map in the vicinity. Cavities can also be present in dolomite and attention should be given during the feasibility stage to locating the presence of any possible cavities in the rock underlying the crusher site. The size of cavities, if present, is likely to be relatively minor and at this stage they are not considered to be a sufficient reason to merit consideration of an alternative location.

The ground conditions are considered to be generally suitable for founding the crusher and ancillary infrastructure on normal concrete pads.

Waste disposal facilities are also shown on Figures 3 and 4. Foundation conditions over the proposed fines dump area are likely to comprise (BS) black soil landform unit and (RS) red soil land form unit. At this stage the black soil unit expected to be present beneath parts of the fines dump is not anticipated to present significant stability problems as it is expected to be relatively shallow.

The location of a proposed access road to the crusher is shown on Figures 3 and 4, running off Crater Lake Road, which in turns runs off Batchelor Road. The proposed access road is not located within the tenement boundary, but has been sited so as to mainly traverse relatively high ground to avoid flood waters and the black soils present in the flood plain and so as to not require large amounts of cut and fill. An alternative access road alignment located within the tenement boundary and intersecting Batchelor Road is also shown as a contingency. However, the alternative alignment crosses the flood plain and is likely to encounter black soils over a significant length.

9.2 Open Pit Stability

Based on the Mt Grace interpretations of the likely geology of the proposed open pit and assuming that the final walls will be developed in competent dolomite of high to very high strength, with a lack of continuous veins, faults, shear and/or joints we recommend the following open pit slope design parameters:

- Face Height
    - Initial (10m below surface to -1): 10m face heights
    - below 10m: 20m face heights
- Face Angle
    - Initial (0 – 10m): 40 degrees
    - Below 10m: 70 degrees
- Berm Width: 5 m

These preliminary slope design recommendations are based solely on the limited geological information that is available at this time. Given the limited data available at this time, it has not been possible to conduct and quantitative analysis of likely slope stability. These criteria will need to be confirmed by further field investigations and analysis during the next phase of the feasibility study. The recommendations appear to be consistent with the mined final wall slopes at the nearby Sundance open pit, based on the limited information given in Simpson (1994).

These preliminary slope design recommendations are based on the following assumptions:

- That the final pit walls are effectively dewatered and depressurised
- That controlled final wall blasting techniques are employed during the excavation of the final wall slopes. Trial blasting will be required to develop the optimum blasting technique to limit blasting induced damage to the final walls.

Open pit and slope design is shown in Figure 8.

9.3 Mining Studies

The results of this study confirm the strong likelihood that the Winchester deposit can support a long life operation of approximately 18 years including more than 14 years of magnesite ore sales at a target rate of 800,000 tonnes of saleable coarse crushed magnesite ore per year with an additional revenue stream potential from sales of 200,000 tonnes per year of fines (sub-6mm magnesite).

Capital costs for establishing the project with 1 million tonnes/year ROM capacity (800,000 tonnes/year coarse fraction output) apart from the crusher/screener which are to be leased or operated by the contractor are estimated to be approximately $4 million (including 30% contingency).

Long run project operating costs (after year 4) for open pit mine, crusher, screener at 800,000 tonnes/year coarse fraction output (1million tonnes/year ROM capacity) are estimated as $15.9 million per year including 30% contingency. This translates into operating cost of approximately $20/tonne of saleable coarse fraction DSO magnesite ore, or approximately $16/tonne of ROM ore. Initial project operating costs (Year 3) at 800,000 tonnes/year coarse fraction output 1 million tonnes/year ROM capacity) are estimated as $16.9 million per year ($21/tonne of saleable rock) due to higher waste rock ratio.

Project operating costs per tonne of saleable magnesite for Year 1 of operation will be higher than the typical long run costs because of higher waste to ore ratio in the initial years and the lower output having to absorb fixed operating costs of the project.

The proposed mining operations use conventional, well tried methods and it is considered that there are no significant technical obstacles to establishing an open pit mining operation to provide magnesite ore from the Winchester deposit.

9.3.1 Resource Estimates

The resource estimates as currently presented are difficult to use as the basis for both short and long term mine planning. It is recommended that a digital block model be developed to facilitate mine planning for the project feasibility study. The resource block model can then be constructed to include all the important properties required for mine planning. It may be appropriate to further evaluate short term grade variability to ensure that the deposit is fully characterised by the model.

Additional work is required to achieve the JORC Standard for the statutory reporting of ore reserves.

9.3.2 Production Schedule

The production schedule developed for this study assumes that the open pit will be developed sequentially on a bench by bench basis. A staged development approach is also valid especially if wet season mining is considered. There will be minimal change in operating costs between the two scenarios however capital costs will be slightly lower with the staged approach. The design of the pit for staged development is shown in Figure 7.

9.4 Hydrogeological Studies

It is concluded that ground water inflows to the mine will be about four limes those to the trial pit, with other, probably short term larger flows possible when (and if cavities are intersected below the water table.

Hydrogeological work for the pre-feasibility study has of necessity been limited to evaluation of existing information and observations made by others of dewatering in shallow pits in the Coomalie Dolomite. Whilst this has been adequate for a pre-feasibility study, further field investigations would be necessary for a bankable feasibility study.

9.4.1 Dewatering Requirements

Dewatering requirements are mostly those of ensuring accessibility of the pit floor for advancing the mining. Neither trafficability nor pit wall stability are interpreted as being much affected by ground water, provided the floor is not flooded. Whilst wet blasting conditions may be encountered, it is unlikely that strongly flowing blastholes would be encountered.

For work in the dry season, dewatering would provide a source of dust suppression water and the dewatering rates expected greatly exceed these requirements, making provision of a separate water supply unnecessary.

9.4.2 Predicted Dewatering Rates

The rates of inflow to the trial pit varied with time and similar behaviour would be expected when the mine is being developed. Higher rates of groundwater inflow, particularly in the short term, could occur us a result of intersection of cavernous material outside the ore zone as well as being a consequence of the variable nature of the alluvial and weathered zone materials.

For the early stages of mining through the alluvium, ,which we understand will be carried out in the dry season, the anticipated rates of groundwater inflow are predicted to be about four times those encountered during the trial pit, i.e. to range approximately between an early 140 L/s and a late 65 L/s. This range of flow rates is probably representative of dry season to wet season fluctuations after mining through the alluvium.

At greater depths, i.e. while mining through the magnesite ore and dolomite wall rocks, experience reported to us from other pits in this material suggests that some larger, short term inflows may occur. The mining method will accommodate periods of temporary inundation of the pit floor and we anticipate that sump pumps of a size designed for the wet season should suffice for dewatering.

The estimated inflow rates are expected to be much the same in total, regardless of the relative proportion of external bores and internal sump pumps.

9.4.3 Predicted Groundwater Level Drawdowns

The trial pit only caused drawdown of approximately 1-2 m for most monitored drillholes, most of which was developed within 4-6 weeks. Predicted rates of inflow to the mine are about four times those for the trial pit, which has a much larger area.

The distribution of drawdown around the mine can reasonably be expected to be similar to that around the trial pit, with probable complete or near-complete recovery of ground water levels during each wet season. That is, drawdowns are unlikely to exceed 5 m, except close to the pit, within the near-surface alluvial material.

9.4.4 Proposed Approach to Mine Dewatering

The logical dewatering strategy will be to intercept as much ground water (i.e. the dry season inflow) as feasible by pumping from bores outside the pit, in order to minimise the need to treat this water to remove suspended solids before releasing it to the surface environment.

In the wet season, runoff within the pit and greater rates of ground water inflow are inevitable. Sumping will be the method of removing this water, which would be discharged after passing through settling ponds. During the wet season, natural surface flows tend to be turbid and the requirement for reducing the suspended solid concentrations may be less than in the dry season.

Shallow production bores (10 to 20 m) would be used from the beginning of the project to intercept a large proportion of inflow from the shallow alluvium, allowing it to be discharged as clean water without the need to be treated to remove suspended solids. As project planning is developed beyond the pre-feasibility stage, there will be opportunities to optimise the relativity between pumping from bores and sumps.

Deeper production bores would target known zones of cavernous development, some of which may be identified from existing information. Other bores would be drilled at sites identified from obvious sites of inflow or cave development exposed in the pit walls. Further investigation will be required to establish the feasibility of (and need for) such bores.

The dewatering methodology will be strongly influenced by the mining methodology. For example, if a wet season mining strategy is adopted, the need for external dewatering bores should be reviewed (as part of the feasibility study).

It is inevitable that sumping will be required, both to remove wet season runoff from the pit walls and to collect some groundwater that will enter the pit despite the external bores. Sumps will produce water with a variable load of suspended solids, which would be treated in settling ponds before release, as discussed in Section 6.5.

9.5 Hydrological Studies

9.5.1 Water Diversion Systems

Diversion and Flood Protection Bunds

The requirement to divert the large design flow through the tenement is best achieved by a dedicated floodway. Within the floodway, an engineered channel will convey low flows.

It is proposed to use the southern wall of the rock waste dump as the northern bank of the diversion channel. Channel construction is proposed for the southern bank in the form of a flood protection bund. Based on the time of concentration and flow runoff generation, it is recommended that the diversion channel dimensions replicate the existing floodplain geometry, that is to have wide and shallow dimensions as opposed to narrow and deep. It is envisaged that the channel divert the 3 hour storm of 50 year recurrence and have the following dimensions:

Channel Width: 200 m

Minimum depth of Channel: 0.8 m

Estimated Maximum flow Velocity: 1.44 m/s

It is envisaged that the channel be constructed on the floodplain with minimal disturbance to vegetation. Vegetation can act as an erosion control measure and minimise surface scouring. The material used for the bund walls should withstand the 1.44 m/s flow velocity. It is anticipated that the material be sourced during the initial pre-mining stage of the pit.

Diversion bunding will be required on the western side of the waste dump to divert the creek into the new diversion channel. A second diversion bund may be necessary at the confluence downstream of the minesite to minimise the possibility of water flooding the pit. It is envisaged that this bund be a vertical extension of the existing western creek bank. These bunds would be in the order of 2 to 5 m high.

Diversion Channels

Within the flat 200m floodway is a narrow channel approximately 1.5 m wide, 1 m deep will be excavated to carry low flow volumes around the pit.

It is anticipated that a second diversion channel be constructed upstream of the crusher and other mine infrastructure (office, fuel storage, generators, ablution blocks, warehouse, etc.) for protection against surface runoff flow towards infrastructure. This channel is designed based on a 1 in 100 Year, 1 hour intensity event. It is envisaged that the diversion channel will have dimensions in the order of:

Channel Width: 5 - 10 m

Minimum depth of Channel: 1.3 m

Estimated Maximum flow Velocity: 1.60 m/s

Apart from these excavated channels, no disturbance to vegetation within the flood way is envisaged.

9.5.2 Runoff Routing

To properly assess the flood characteristics of the catchment at the minesite, a more detailed runoff (outing study would be required. It is recommended that surface hydrological modelling be conducted to determine the design specifications mentioned in this report. While the specifications provided are only a guide, modelling of the surface flows using a computer package can provide more information on flood levels and the impact of these.

9.6 Waste Storage

The proposed waste storage options for each waste stream are described below and are illustrated in Figure 3 and 4 at the end of 3 years and at the end of mine life respectively. Typical cross sections are diagrammatically shown in Figure 9.

9.6.1 Fines

The fines will be deposited in a stockpile by a conveyor leading from the crushing and screening plant. It is proposed that the material will then be transported and spread with conventional earthmoving equipment in a dump area located immediately to the north of the crusher site and ROM pad. The ultimate footprint of the proposed dump is about 270 by 570 m, with a final top surface elevation at about RL108. The existing ground surface elevations in this area are between about RL75 and RL80.

The side slopes of the fines dump will initially be formed at the angle of repose of the material as it is dumped, likely to be approximately 1V:1.5H. Parts of the fines dump over the lower elevation ground may become saturated following high rainfall events. To minimise the likelihood of stability problems arising from water flow within the dump, and for rehabilitation, the side slopes should be formed at no greater than 1V:3H. The placement of waste rock armouring around the toe of the dump may be necessary for erosion control. Topsoil from the dump footprint area should be stockpiled and used for rehabilitation purposes. Progressive rehabilitation of the fines dump may be feasible as mining proceeds although sales of the fines to agricultural and feedstock sectors are likely to reduce the amount of material accumulated in the fines dump.

10.0 IMPORTANT INFORMATION

Your attention is drawn to the document - "Important Information About Your Geotechnical Engineering Report". This document has been prepared by the ASFE *(Professional Firms Practicing in the Geosciences),* of which Golder Associates is a member. The statements presented in this document are intended to advise you of what your realistic expectations of this report should be, and to present you with recommendations on how to minimise the risks associated with the ground works for this project. The document is not intended to reduce the level of responsibility accepted by Golder Associates, but rather to ensure that all parties who may rely on this report are aware of the responsibilities each assumes in so doing.

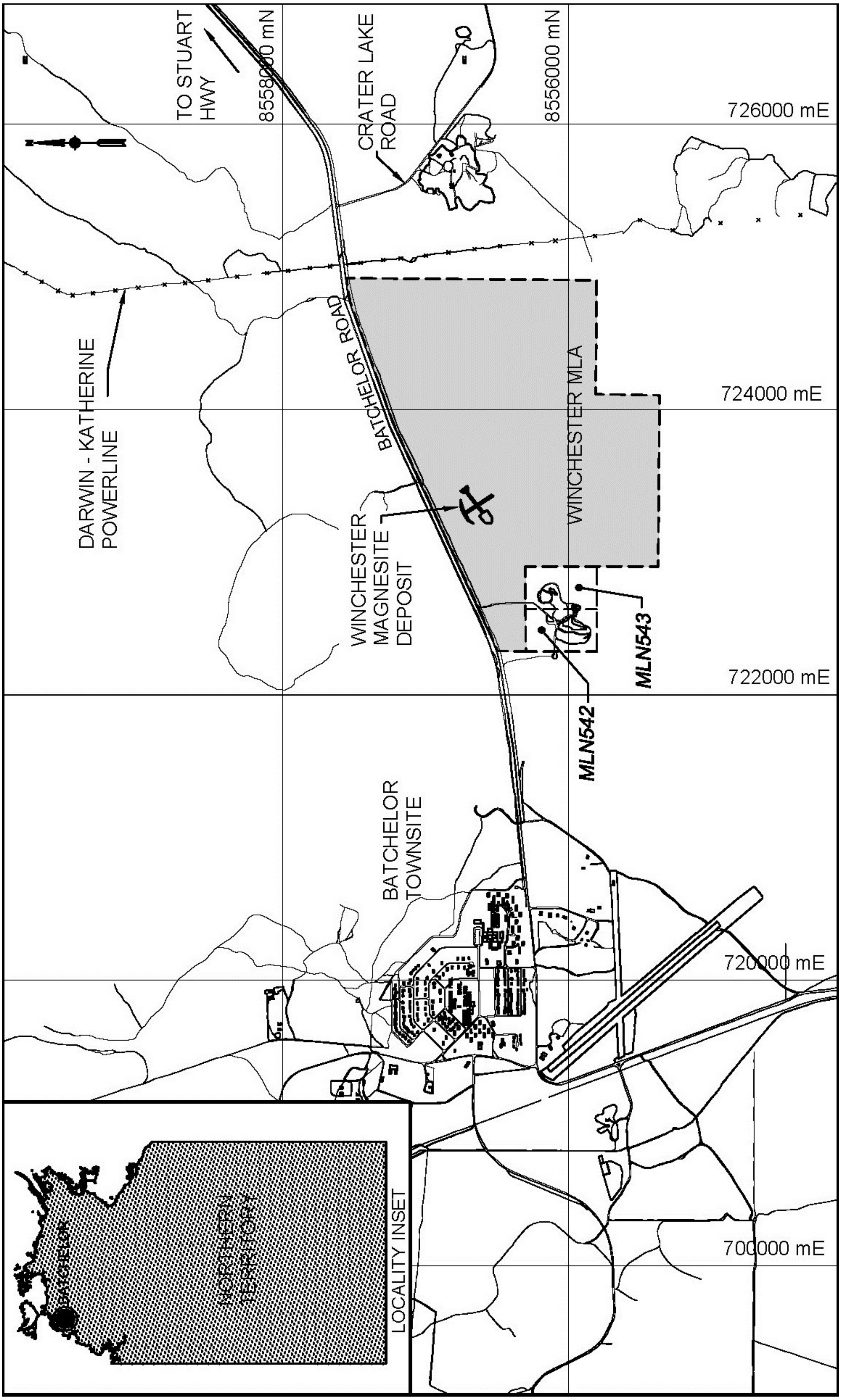

Figure 1 Site Locality Plan

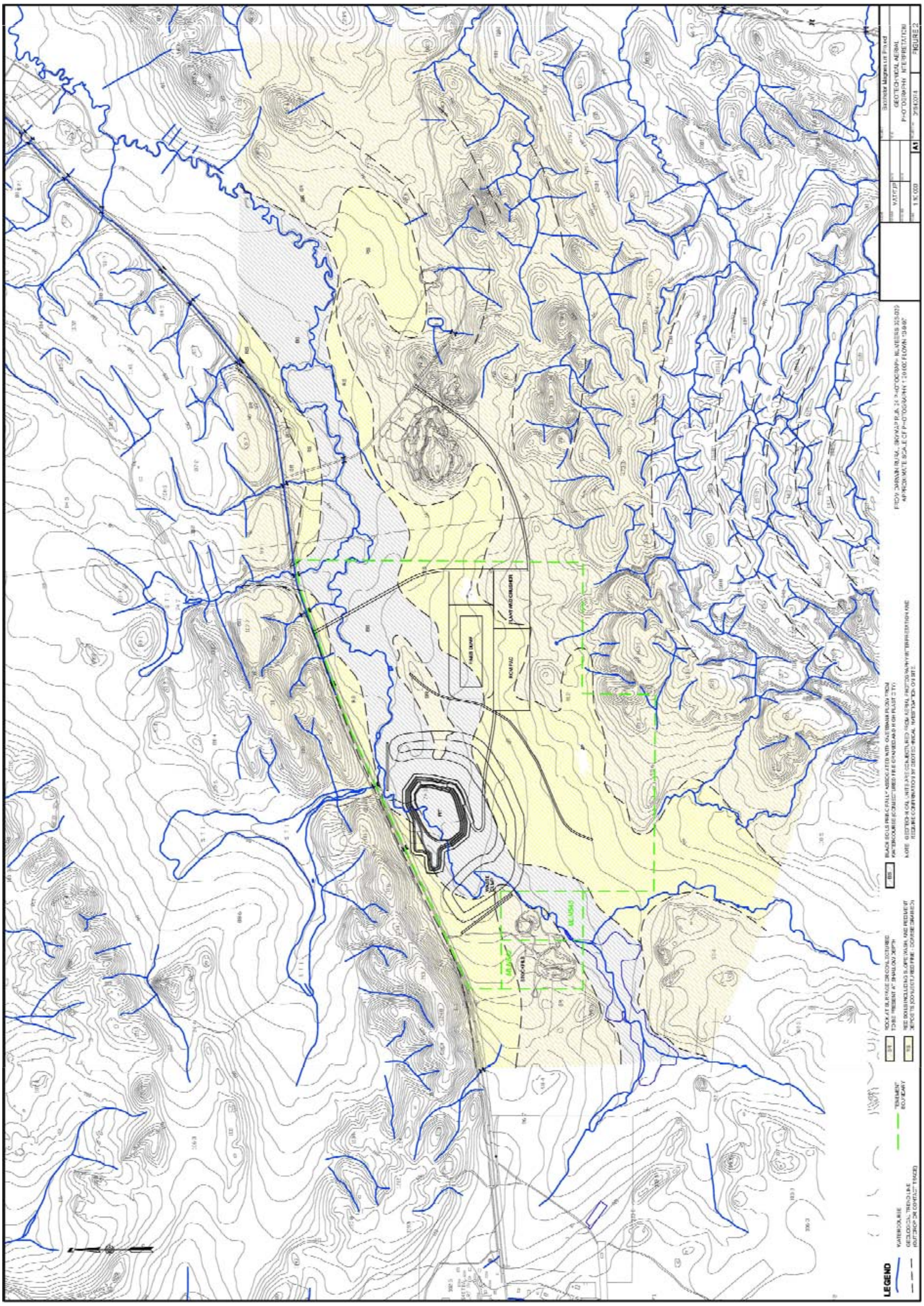

Figure 2 Geotechnical Aerial Photography Interpretation

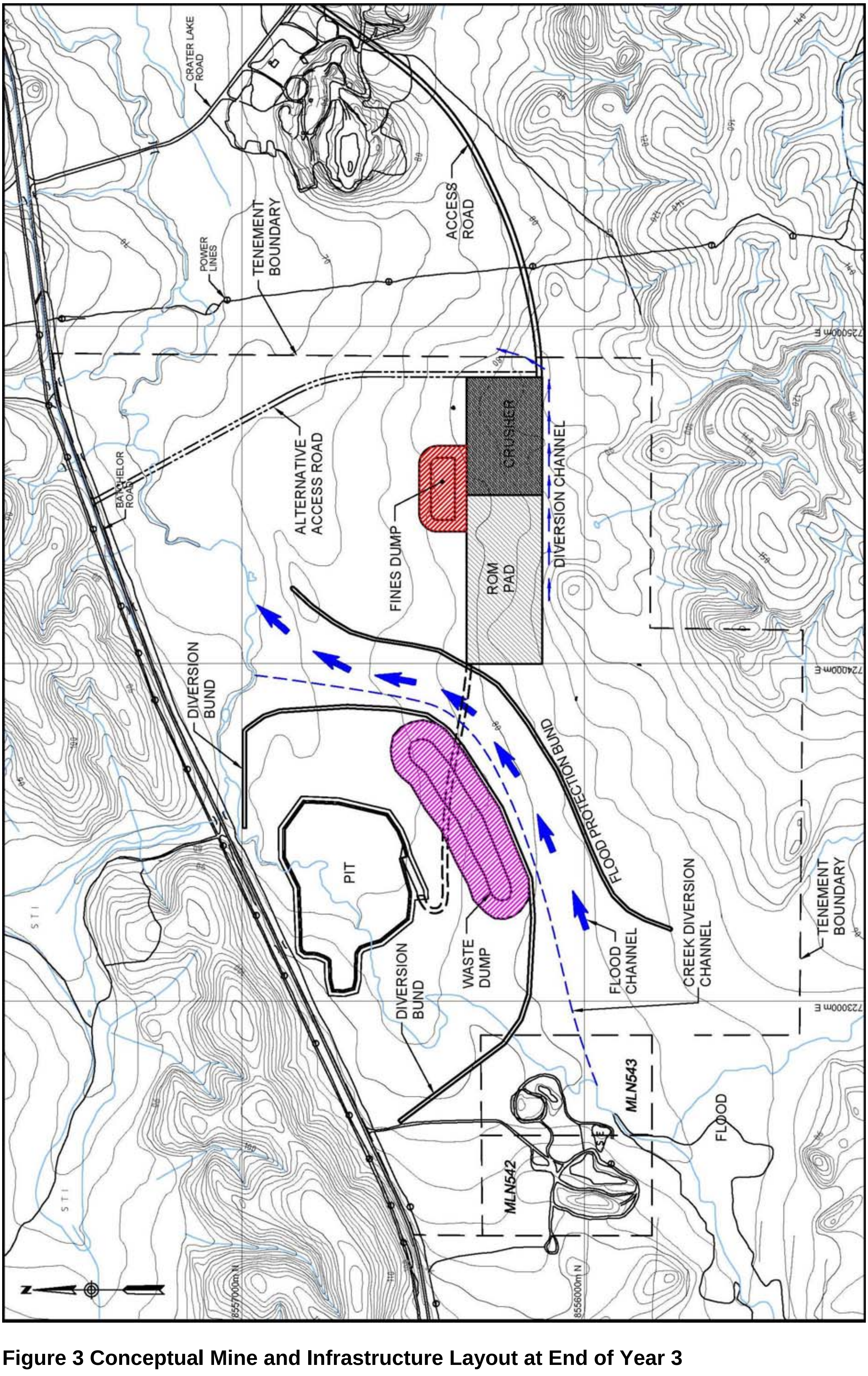

Figure 3 Conceptual Mine and Infrastructure Layout at End of Year 3

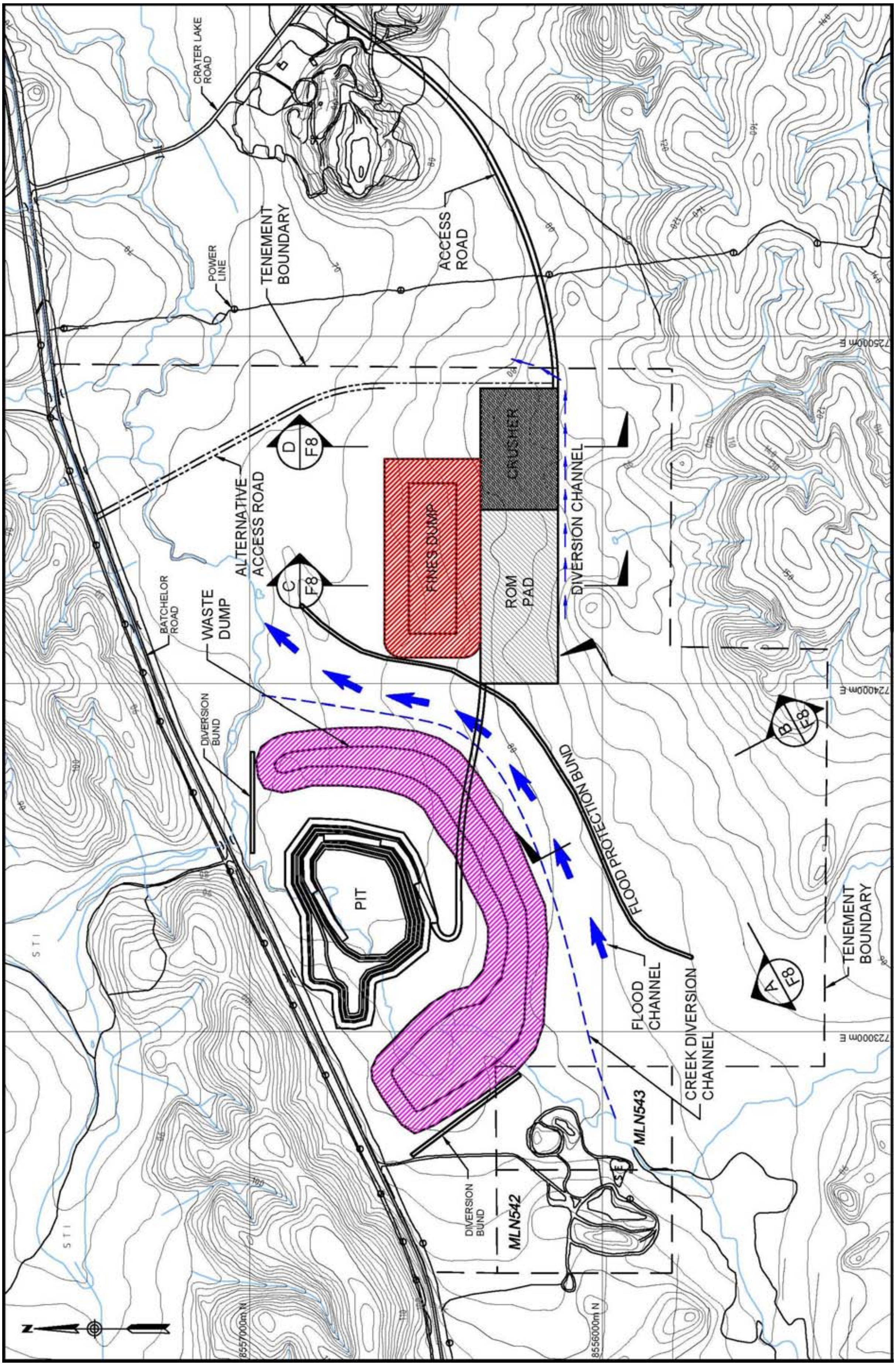

Figure 4 Conceptual Mine and Infrastructure Layout at End of Life

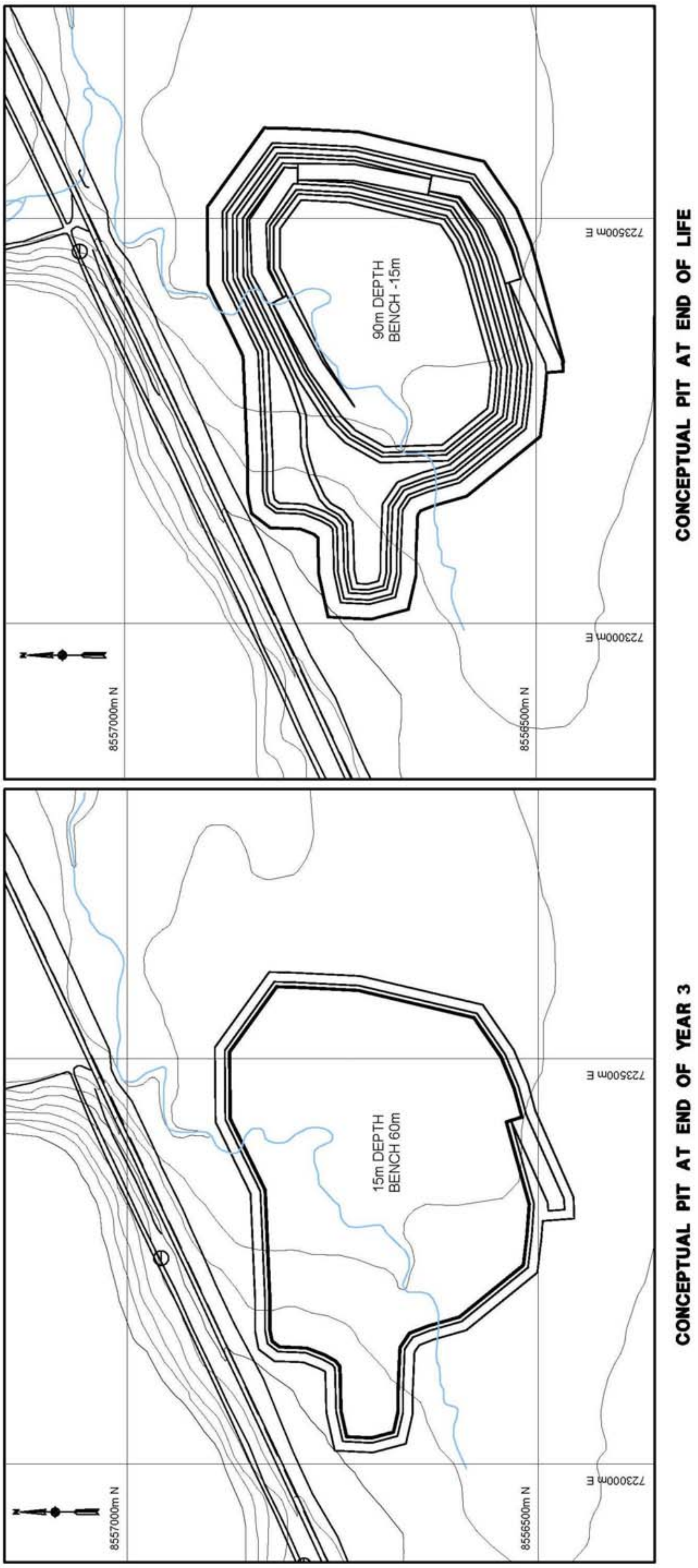

Figure 5 Pit Outlines at End of Year 3 and End of Life

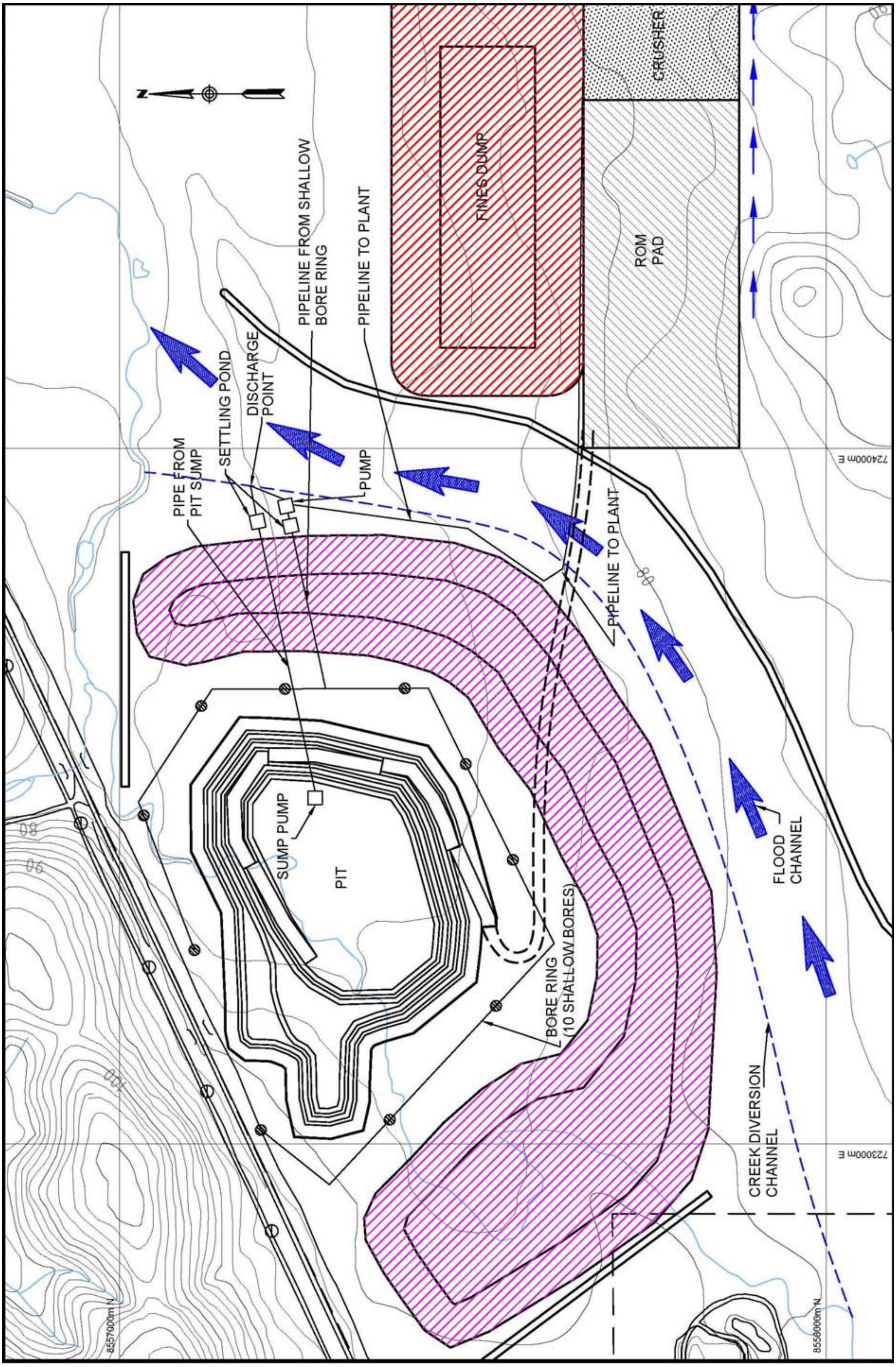

Figure 6 Dewatering Infrastructure - Bench by Bench Development

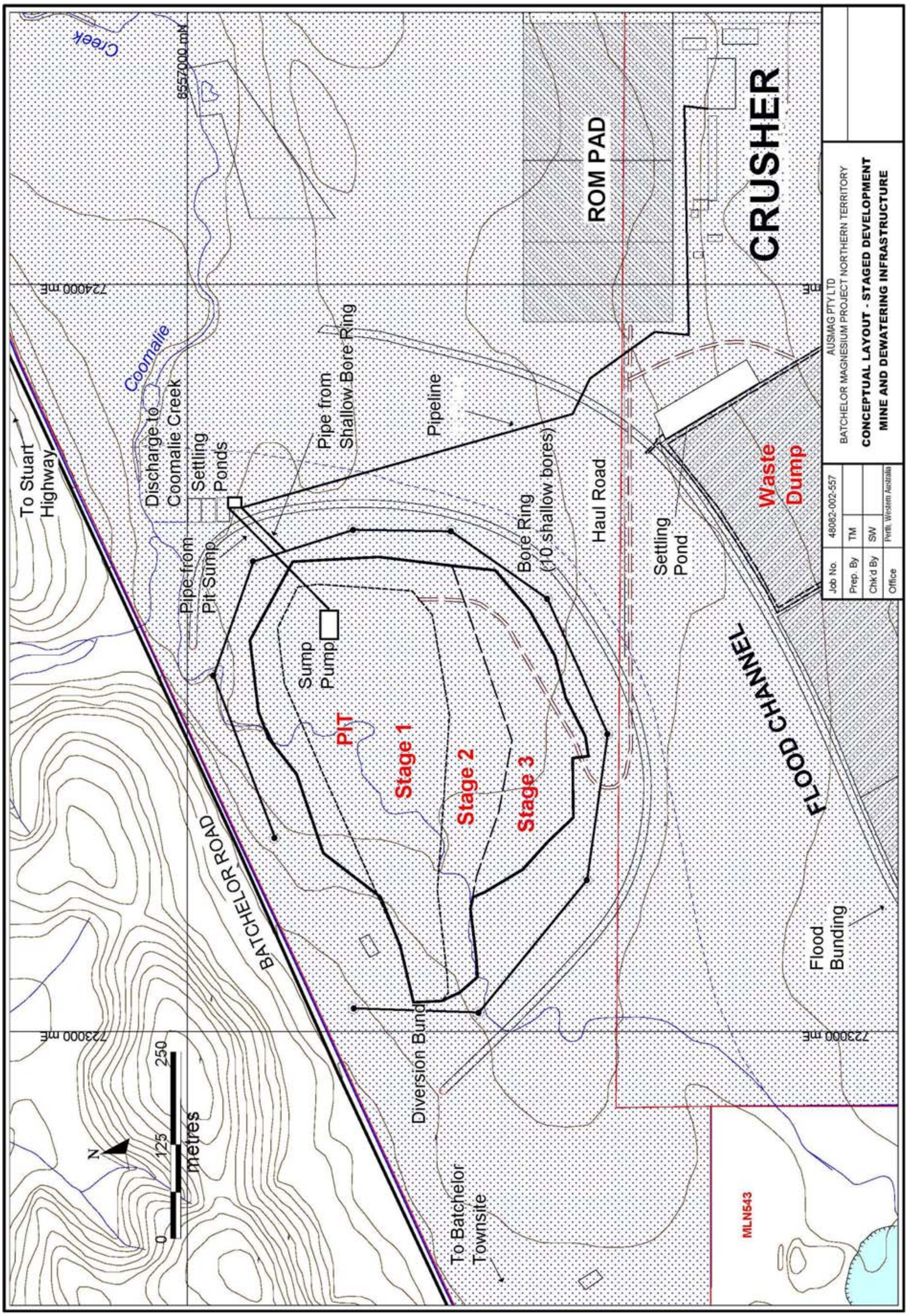

Figure 7 Alternative Dewatering Infrastructure and Pit Outline - Staged Development

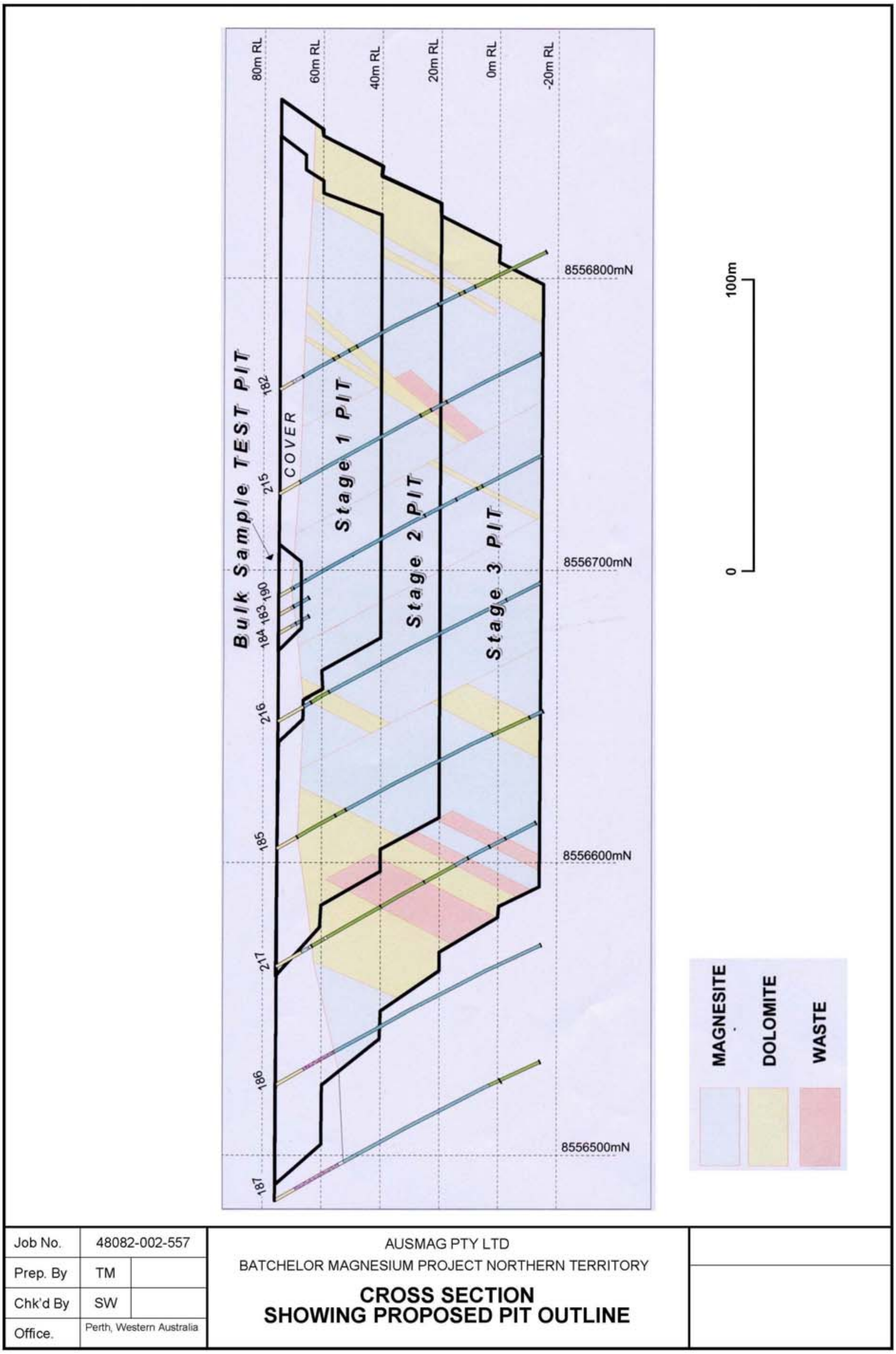

Figure 8 Open Pit Cross Section - Staged Development

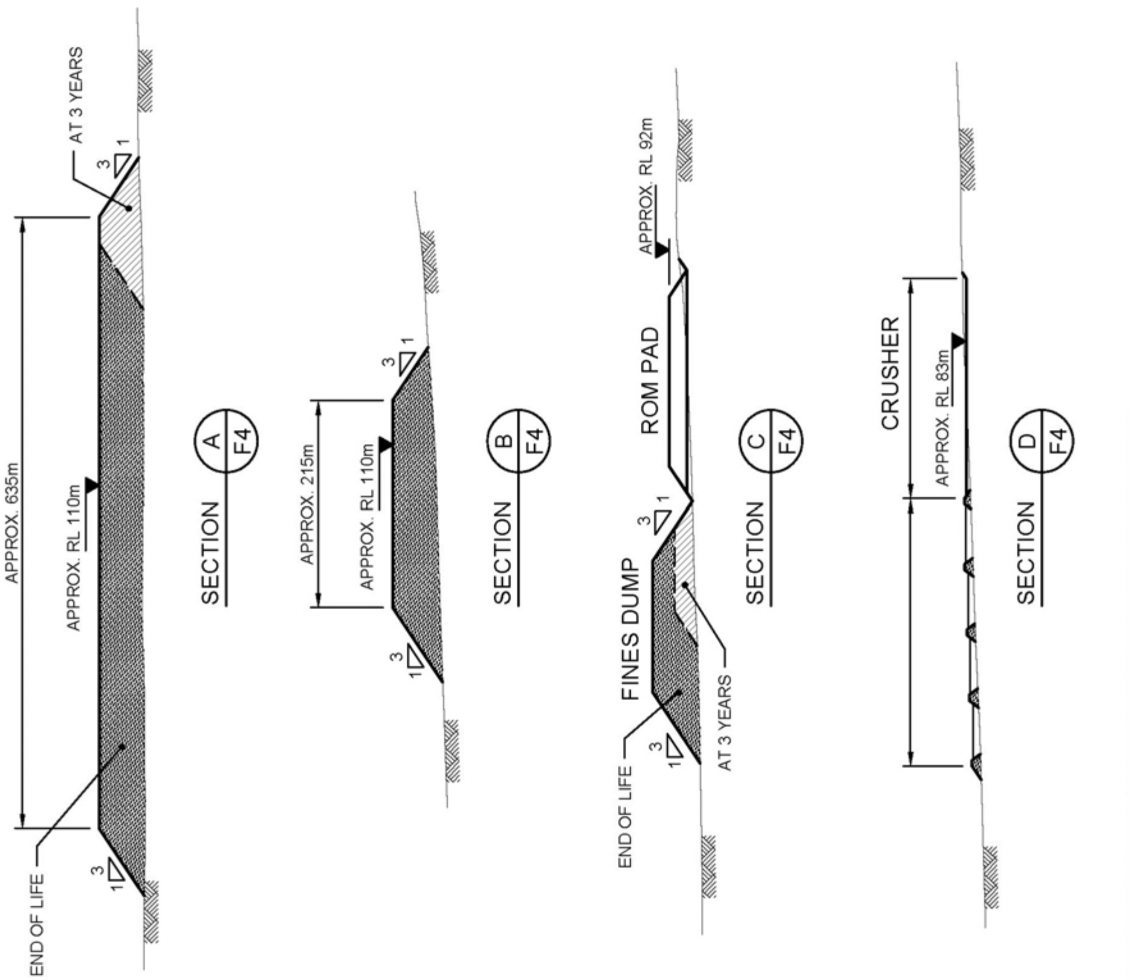

Figure 9 Typical Cross Sections of Infrastructure

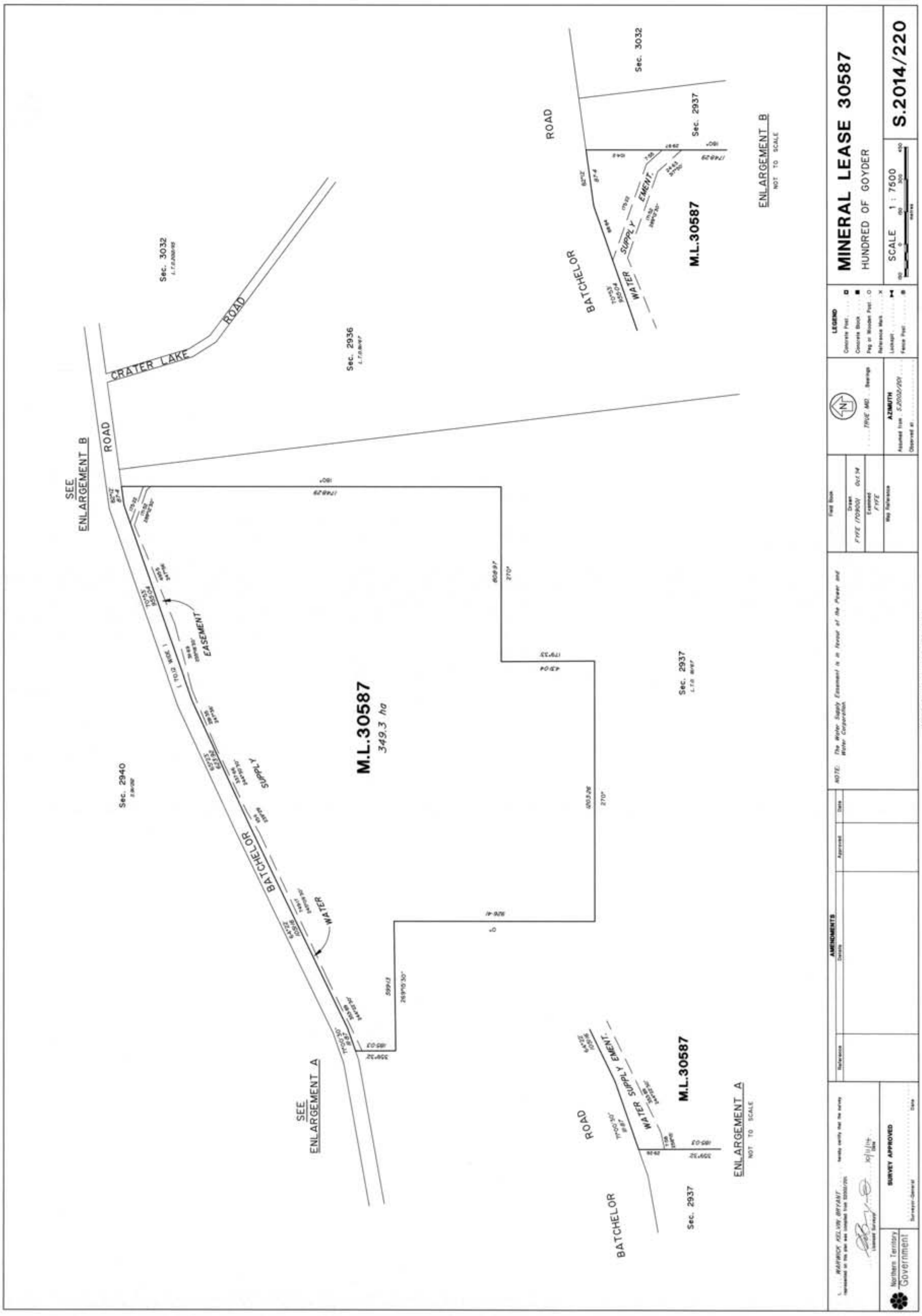

Figure 10 Survey Plan of Winchester Mineral Lease

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APPENDIX A - MINERAL RESOURCES ESTIMATES

April 2001

- A1 -
BATCHELOR MAGNESITE
PROJECT
WINCHESTER PROSPECT JUNE 2000 INDICATED INFERRED (MINUS HIGH QUARTZ) RESOURCE

01640074

SECTION	HOLE	FROM	TO	Block Number	Rock Type	Resource Category	Quality Code	Area (Sq m)	Strike Extent (m)	Volume (Cubic m)	SG	Tonnage	TOTAL TONNAGE on SECTION	SiO_2 % LW(ave)	Al_2O_3 % LW(ave)	TiO_2 % LW(ave)	Fe_2O_3 % LW(ave)	MnO % LW(ave)	MgO % LW(ave)	CaO % LW(ave)	K_2O % LW(ave)	Na_2O % LW(ave)	SO_3 % LW(ave)	P_2O_5 % LW(ave)	LOI % LW(ave)	$CaCO_3$ % LW(ave)	$MgCO_3$ % LW(ave)	MgO in carb % LW(ave)	Talc % LW(ave)	Quartz % LW(ave)
723100mE	MRC115	6	27	1	Mag	Inferred	LTLQ	1097	75	32,309	2.99	246,104		2.18	0.40	0.01	1.63	0.28	45.93	0.40	0.00	0.10	0.02	0.00	49.33	0.71	93.89	44.88	3.30	0.07
723100mE	MRC115	27	37	2	Mag	Inferred	HTLQ	520	75	39,037	2.99	116,719		11.05	0.56	0.01	1.34	0.21	43.68	0.45	0.01	0.13	0.02	0.03	42.71	0.61	91.14	38.79	15.35	1.32
723100mE	MRC115	37	42	3	Mag	Inferred	LTLQ	189	75	14,171	2.99	42,373		1.70	0.70	0.02	1.02	0.31	48.17	0.57	0.00	0.10	0.03	0.02	49.65	1.02	94.28	45.65	3.50	0.00
							TOTAL FOR SECTION 723 100mE						405,196	4.7%	0.47%	0.01%	1.48%	0.26%	45.31%	0.43%	0.00%	0.10%	0.02%	0.01%	47.46%	0.8%	90.3%	43.1%	6.8%	0.4%
723200mE	MRC095	3	30	1	Mag	Inferred	MTLQ	1936	75	145,182	2.99	434,125		4.19	1.14	0.06	2.28	0.19	44.78	0.47	0.00	0.10	0.02	0.07	46.78	0.64	96.87	42.48	7.21	0.00
723200mE	MRC095	30	37	2	Mag	Inferred	MTHQ	436	75	32,720	2.99	0		12.83	0.57	0.03	1.43	0.18	41.69	0.39	0.00	0.08	0.02	0.04	42.45	0.69	90.73	38.59	10.34	6.27
723200mE	MRC095	37	52	3	Mag	Inferred	MTLQ	1053	75	78,969	2.99	236,177		3.53	1.32	0.08	1.87	0.22	45.18	0.58	0.00	0.10	0.02	0.12	47.58	1.00	90.28	43.14	6.31	0.00
723200mE	MRC095	52	72	4	Dol "B"	Inferred	MCMTLQ	1360	75	102,011	2.90	295,831		5.56	1.82	0.10	2.24	0.18	36.24	10.09	0.01	0.10	0.03	0.08	43.72	18.02	38.58	32.77	10.45	0.00
723200mE	MRC095	72	102	5	Mag	Inferred	MTHQ	1427	75	107,012	2.99	0		15.90	1.49	0.06	0.76	0.08	39.44	0.54	0.02	0.09	0.18	0.02	40.42	1.14	76.47	36.55	9.04	11.17
723200mE	MRC094	3	22	6	Mag	Inferred	LTLQ	1500	75	112,530	2.99	336,464		2.16	0.51	0.02	1.27	0.19	45.83	0.44	0.00	0.10	0.03	0.01	49.61	0.79	94.38	45.11	2.55	0.56
							TOTAL FOR SECTION 723 200mE						1,302,597	3.9%	1.16%	0.06%	1.84%	0.19%	43.20%	2.03%	0.00%	0.10%	0.03%	0.06%	46.85%	4.6%	85.9%	41.1%	6.7%	0.1%
723250mE	MRC135	10	27	1	Mag	Inferred	MTLQ	1189	50	59,449	2.99	177,753		6.79	1.23	0.05	1.25	0.15	44.55	0.38	0.01	0.12	0.06	0.07	45.53	0.70	36.64	41.41	9.64	0.55
723250mE	MRC135	27	32	2	Mag	Inferred	LTHQ	273	50	13,630	2.99	0		9.66	0.50	0.02	1.07	0.06	42.53	0.22	0.00	0.11	0.25	0.00	45.87	0.38	57.54	41.84	2.13	8.31
723250mE	MRC135	32	52	3	Mag	Inferred	HTLQ	1293	50	64,639	2.99	193,271		10.34	0.50	0.02	2.06	0.10	43.52	0.31	0.00	0.12	0.08	0.03	43.13	0.54	92.17	39.28	13.29	1.91
723250mE	MRC135	52	101	4	Mag	Inferred	LTLQ	3410	50	170,493	2.99	506,773		4.57	0.70	0.02	2.46	0.20	44.27	0.50	0.00	0.11	0.03	0.04	47.12	0.00	89.52	42.79	4.66	1.62
723250mE	MRC134	4	32	5	Mag	Inferred	HTLQ	1917	50	95,862	2.99	286,626		11.70	0.78	0.04	2.09	0.15	43.19	0.34	0.00	0.11	0.02	0.05	41.59	0.80	79.17	37.84	16.78	1.07
723250mE	MRC134	32	62	6	Mag	Inferred	LTLQ	1992	50	99,578	2.99	297,740		2.64	0.57	0.02	2.70	0.23	46.20	0.40	0.00	0.11	0.02	0.04	48.30	0.71	91.93	43.94	3.95	0.13
723250mE	MRC134	62	72	7	Dol "B"	Inferred	LCLTHQ	632	50	31,589	2.90	0		13.74	0.70	0.03	1.00	0.13	34.85	7.57	0.02	0.10	0.04	0.03	42.14	13.52	68.33	33.14	5.36	10.34
723250mE	MRC134	72	77	8	Mag	Inferred	MTHQ	297	50	14,863	2.99	0		14.73	0.33	0.01	0.95	0.07	40.61	0.52	0.00	0.14	0.07	0.01	42.41	0.63	80.48	38.46	6.74	10.46
723250mE	MRC134	77	97	9	Mag	Inferred	MTLQ	914	50	45,690	2.99	136,600		5.51	1.80	0.08	0.78	0.19	44.75	0.70	0.01	0.11	0.09	0.07	48.19	1.24	87.35	41.74	8.36	0.00
723250mE	MRC134	97	101	10	Dol "B"	Inferred	LCMTHQ	621	50	31,038	2.90	0		17.51	2.71	0.12	0.91	0.04	33.84	8.62	0.14	0.11	0.59	0.09	38.19	11.62	63.19	30.20	11.40	10.38
723250mE	MRC133	13	37	11	Mag	Inferred	LTLQ	1827	50	91,385	2.99	273,182		5.30	0.56	0.01	0.84	0.13	44.28	0.91	0.00	0.12	0.11	0.01	47.61	1.63	89.83	42.94	4.21	2.63
							TOTAL FOR SECTION 723 250mE						1,874,944	6.3%	0.78%	0.03%	1.92%	0.17%	44.24%	0.50%	0.00%	0.11%	0.05%	0.04%	45.90%	0.9%	87.2%	41.7%	8.1%	1.3%
723300mE	MRC221	6	25	1	Mag	Indicated	LTMQ	617	50	30,846	2.99	92,234		4.09	0.57	0.03	1.57	0.21	44.38	0.58	0.02	0.08	0.01	0.04	48.31	1.00	91.73	43.83	1.71	3.01
723300mE	MRC221	25	31	2	Mag	Indicated	LTLQ	192	50	9,604	2.99	28,717		1.85	0.33	0.05	1.70	0.25	45.50	0.46	0.01	0.09	0.01	0.12	49.05	0.86	93.30	44.60	2.63	0.08
723300mE	MRC221	31	36	3	Mag	Indicated	HTLQ	181	50	9,065	2.99	27,106		9.30	1.28	0.07	1.29	0.21	43.72	0.48	0.01	0.10	0.01	0.14	43.19	0.85	82.02	39.20	14.16	0.32
723300mE	MRC221	36	66	4	Mag	Indicated	LTLQ	1041	50	52,054	2.99	155,643		2.65	0.31	0.02	1.06	0.13	45.85	0.31	0.01	0.09	0.01	0.02	49.51	0.55	94.38	45.11	2.31	1.19
723300mE	MRC221	66	79	5	Mag	Indicated	MTLQ	455	50	22,737	2.99	67,984		6.67	0.71	0.03	1.00	0.11	44.64	0.33	0.02	0.10	0.02	0.04	46.09	0.59	87.60	41.97	9.03	0.96
723300mE	MRC221	79	86	6	Mag	Indicated	LTLQ	223	50	11,142	2.99	33,316		0.82	0.15	0.01	1.25	0.14	48.28	0.46	0.01	0.11	0.01	0.02	50.98	0.80	96.74	48.24	0.08	0.76
723300mE	MRC221	86	89	7	Dol "B"	Indicated	LCLTLQ	89	50	4,450	2.90	12,904		2.72	0.06	0.01	0.80	0.13	41.10	6.38	0.01	0.08	0.01	0.01	49.08	11.39	84.38	40.33	2.40	1.20
723300mE	MRC221	89	101	8	Mag	Indicated	HTLQ	495	50	24,731	2.99	73,947		10.96	1.58	0.07	0.80	0.09	43.37	0.35	0.02	0.10	0.03	0.06	42.30	0.63	80.51	38.48	15.32	1.24
723300mE	MRC113	13	33	9	Mag	Indicated	HTHQ	748	50	37,417	2.99	0		15.96	1.50	0.07	1.63	0.13	40.82	0.33	0.02	0.08	0.01	0.05	39.42	0.59	75.02	35.86	15.55	6.10
723300mE	MRC113	33	38	10	Mag	Indicated	LTLQ	249	50	12,443	2.99	37,203		4.24	0.45	0.02	0.81	0.13	44.78	0.90	0.03	0.10	0.01	0.04	46.76	1.61	92.05	44.00	2.44	2.69
723300mE	MRC113	38	43	11	Dol "B"	Indicated	LCLTLQ	168	50	8,407	2.90	24,379		3.79	0.21	0.01	0.55	0.16	40.17	6.74	0.01	0.08	0.02	0.03	48.35	12.04	82.45	39.43	2.33	2.31
723300mE	MRC113	43	69	12	Mag	Indicated	LTLQ	1030	50	51,478	2.99	153,920		3.89	0.58	0.03	0.83	0.14	44.97	0.74	0.06	0.09	0.01	0.03	48.73	1.32	92.24	44.09	2.76	2.14
723300mE	MRC113	69	91	13	Dol "B"	Indicated	LCLTLQ	760	50	37,979	2.95	112,037		1.98	0.41	0.02	0.43	0.16	42.40	5.08	0.02	0.09	0.02	0.02	49.38	9.07	88.95	41.58	2.62	0.32
723300mE	MRC113	91	101	14	Mag	Indicated	LTLQ	578	50	28,881	2.99	86,355		2.44	0.80	0.04	0.87	0.16	45.91	0.48	0.04	0.08	0.01	0.09	49.08	0.86	93.30	44.60	4.11	0.09
723300mE	MRC197	5	10	15	Mag	Indicated	LTMQ	137	50	6,831	2.99	20,425		6.91	0.89	0.05	1.27	0.16	42.89	1.09	0.04	0.07	0.01	0.04	46.42	1.95	87.29	41.72	3.66	4.59
723300mE	MRC197	10	12	16	Dol "B"	Indicated	LCLTLQ	56	50	2,728	2.90	7,910		2.30	0.81	0.03	0.56	0.17	40.31	7.04	0.01	0.08	0.01	0.03	48.58	12.57	82.47	39.42	2.79	0.53
723300mE	MRC198	2	9	17	Mag	Indicated	LTMQ	201	50	10,064	2.99	30,091		3.72	0.60	0.05	1.59	0.22	44.49	0.44	0.03	0.08	0.01	0.04	48.68	0.79	92.59	44.26	0.72	3.26
723300mE	MRC198	9	29	18	Mag	Indicated	LTLQ	617	50	30,871	2.99	92,305		2.87	0.43	0.01	1.69	0.16	45.39	0.36	0.01	0.08	0.01	0.03	49.12	0.88	93.53	44.71	2.14	1.26
723300mE	MRC198	29	71	19	Mag	Indicated	MTLQ	1468	50	73,287	2.99	219,127		7.67	1.14	0.05	1.36	0.19	43.62	1.04	0.02	0.08	0.01	0.07	44.46	1.86	83.61	39.96	11.46	0.40
723300mE	MRC198	71	102	20	Mag	Indicated	LTLQ	948	50	47,380	2.99	141,667		4.70	0.52	0.02	1.46	0.18	44.81	0.34	0.01	0.08	0.02	0.02	47.67	0.61	90.81	43.41	4.40	1.81
723300mE	MRC112	7	37	21	Mag	Indicated	MTLQ	901	50	45,050	2.99	134,700		6.05	0.49	0.02	1.43	0.23	44.63	0.44	0.01	0.10	0.01	0.04	46.37	0.79	88.17	42.14	6.42	0.74
723300mE	MRC112	37	60	22	Dol "B"	Indicated	MCMTLQ	740	50	36,987	2.90	107,262		8.63	1.02	0.05	1.44	0.33	35.90	9.70	0.01	0.08	0.02	0.05	42.84	17.32	67.47	32.25	11.44	1.38
723300mE	MRC112	60	84	23	Mag	Indicated	LTMQ	766	50	38,278	2.99	114,452		5.32	0.65	0.02	1.81	0.21	44.28	0.45	0.01	0.10	0.01	0.03	47.67	0.80	90.64	43.33	2.98	3.43
723300mE	MRC112	84	101	24	Mag	Indicated	LTLQ	607	50	30,358	2.99	90,769		3.00	0.47	0.02	1.66	0.22	45.46	0.40	0.01	0.10	0.01	0.02	48.81	0.71	92.90	44.41	3.30	0.91
723300mE	MRC196	6	28	25	Mag	Indicated	LTLQ	752	50	37,594	2.99	112,406		4.66	0.54	0.02	2.30	0.24	44.01	0.51	0.01	0.09	0.01	0.04	47.35	0.91	90.03	43.02	3.11	2.69
723300mE	MRC196	28	36	26	Dol "B"	Indicated	LCMTLQ	270	50	13,477	2.90	39,083		8.81	0.83	0.04	1.85	0.26	37.28	7.11	0.01	0.10	0.01	0.05	43.02	12.70	71.71	34.28	9.44	2.92
723300mE	MRC196	36	45	27	Mag	Indicated	LTLQ	307	50	15,351	2.99	45,900		2.26	0.37	0.02	1.81	0.21	45.09	0.49	0.01	0.10	0.01	0.01	49.50	0.86	94.09	44.96	0.36	2.06
723300mE	MRC196	45	46	28	Mag	Indicated	HTHQ	39	50	1,963	2.95	0		38.42	1.41	0.11	1.55	0.07	31.53	0.37	0.01	0.11	0.01	0.06	26.56	0.66	50.32	24.06	23.43	23.58
723300mE	MRC196	46	60	29	Mag	Indicated	LTLQ	478	50	23,914	2.99	71,503		1.89	0.35	0.02	1.47	0.20	45.67	0.41	0.01	0.10	0.01	0.02	49.74	0.73	94.67	45.25	1.31	1.06
723300mE	MRC196	60	72	30	Mag	Indicated	HTLQ	403	50	20,143	2.99	60,228		8.61	0.82	0.03	1.30	0.10	43.86	0.36	0.01	0.11	0.06	0.05	43.61	0.64	83.00	39.68	13.18	1.25
723300mE	MRC196	72	85	31	Mag	Indicated	LTLQ	433	50	21,645	2.99	64,720		2.56	0.43	0.02	1.19	0.20	45.75	0.37	0.01	0.10	0.01	0.03	49.14	0.68	93.58	44.73	3.19	0.54
723300mE	MRC196	85	98	32	Mag	Indicated	MTLQ	336	50	16,791	2.99	50,204		5.29	1.20	0.07	2.29	0.24	44.11	0.70	0.01	0.09	0.01	0.05	45.71	1.25	86.51	41.35	8.64	0.00
723300mE	MRC196	98	102	33	Mag	Indicated	MTHQ	244	50	12,210	2.99	0		11.78	0.81	0.02	1.80	0.13	41.74	0.48	0.01	0.08	0.04	0.02	43.01	0.88	81.87	39.04	6.47	8.41
723300mE	MRC111	2	7	34	Mag	Indicated	LTMQ	239	50	11,962	2.99	35,767		4.38	0.56	0.03	2.12	0.29	44.00	0.39	0.01	0.09	0.01	0.03	47.99	0.70	91.35	43.68	1.05	3.71
723300mE	MRC111	7	49	35	Mag	Indicated	LTLQ	1344	50	67,200	2.99	200,927		3.77	0.48	0.02	1.86	0.20	45.08	0.37	0.01	0.10	0.02	0.03	48.03	0.68	91.45	43.72	4.28	1.06
723300mE	MRC111	49	67	36	Mag	Indicated	MTLQ	629	50	31,428	2.99	93,970		5.74	0.68	0.03	1.02	0.17	45.04	0.40	0.01	0.10	0.03	0.03	46.78	0.71	89.01	42.55	7.81	0.79
723300mE	MRC111	67	97	37	Dol "B"	Indicated	LCMTLQ	378	50	18,877	2.90	54,742		6.72	1.35	0.05	0.72	0.19	38.45	6.13	0.02	0.09	0.02	0.02	45.14	10.95	77.25	36.92	7.92	1.70
723300mE	MRC195	6	24	38	Mag	Indicated	MTLQ	627	50	31,349	2.99	93,731		8.02	0.83	0.03	1.04	0.16	43.02	0.97	0.01	0.00	0.02	0.05	44.58	1.55	84.99	40.20	10.73	1.82
723300mE	MRC195	24	27	39	Dol "B"	Indicated	LCHTLQ	137	50	6,825	2.95	20,135		10.44	0.92	0.04	0.81	0.11	40.26	4.52	0.01	0.08	0.08	0.01	42.70	8.07	75.00	35.85	13.83	1.67

Golder Associates

April 2001

- A3 -

BATCHELOR MAGNESITE PROJECT

WINCHESTER PROSPECT JUNE 2000 INDICATED INFERRED (MINUS HIGH QUARTZ) RESOURCE

SECTION	HOLE	FROM	TO	Block Number	Rock Type	Resource Category	Quality Code	Area (Sq m)	Strike Extent (m)	Volume (Cubic m)	SG	Tonnage	TOTAL TONNAGE on SECTION	SiO_2 % LW(ave)	Al_2O_3 % LW(ave)	TiO_2 % LW(ave)	Fe_2O_3 % LW(ave)	MnO % LW(ave)	MgO % LW(ave)	CaO % LW(ave)	K_2O % LW(ave)	Na_2O % LW(ave)	SO_3 % LW(ave)	P_2O_5 % LW(ave)	LOI % LW(ave)	$CaCO_3$ % LW(ave)
723385mE	MRC182	21	23	32	Dol "B"	Indicated	LCHTLQ	61	35	2,121	2.85	6,257		9.79	1.03	0.09	0.81	0.12	40.94	3.96	0.01	0.09	0.01	0.14	42.00	7.07
723385mE	MRC182	23	27	33	Mag	Indicated	LTLQ	112	35	3,919	2.89	11,717		2.92	0.73	0.04	1.41	0.28	45.46	0.84	0.01	0.10	0.01	0.10	48.33	1.14
723385mE	MRC182	27	30	34	Dol "B"	Indicated	LCLTLQ	107	35	3,750	2.80	10,076		2.85	1.20	0.07	2.11	0.28	36.25	7.24	0.01	0.07	0.01	0.11	46.58	12.93
723385mE	MRC182	30	61	35	Mag	Indicated	LTLQ	897	35	31,400	2.99	93,886		2.05	0.38	0.02	1.00	0.19	46.10	0.41	0.01	0.10	0.01	0.03	49.67	0.73
							TOTAL FOR SECTION 723 385mE						1,511,988	5.3%	0.72%	0.03%	1.43%	0.22%	43.30%	2.05%	0.02%	0.09%	0.01%	0.04%	45.73%	3.7%
723403mE	MRC208	12	20	1	Mag	Indicated	MTMQ	367	17	6,245	2.88	18,573		8.16	0.95	0.05	1.16	0.21	43.57	0.35	0.01	0.09	0.01	0.03	45.27	0.63
723403mE	MRC208	20	24	2	Dol "B"	Indicated	LCLTHQ	116	17	1,987	2.85	0		10.61	0.53	0.01	1.86	0.10	30.46	3.17	0.01	0.07	0.01	0.04	44.20	5.68
723403mE	MRC208	24	72	3	Mag	Indicated	LTMQ	1736	17	29,505	2.99	88,220		4.52	0.64	0.02	1.21	0.19	44.33	0.36	0.08	0.09	0.01	0.03	46.43	0.64
723403mE	MRC208	72	82	4	Mag	Indicated	MTMQ	354	17	6,024	2.99	18,011		9.67	0.56	0.02	2.15	0.20	42.17	0.50	0.05	0.09	0.01	0.04	44.24	0.89
723403mE	MRC208	82	89	5	Mag	Indicated	LTLQ	208	17	3,537	2.99	10,576		3.28	0.53	0.02	3.04	0.24	44.22	0.42	0.01	0.09	0.01	0.06	48.08	0.75
723403mE	MRC208	89	96	6	Mag	Indicated	MTMQ	164	17	2,790	2.99	8,341		9.44	0.52	0.02	2.07	0.20	43.09	0.34	0.01	0.09	0.01	0.03	44.03	0.61
723403mE	MRC208	96	101	7	Mag	Indicated	LTMQ	294	17	4,990	2.99	14,921		5.09	0.74	0.03	2.00	0.25	43.98	0.40	0.01	0.08	0.01	0.04	46.85	0.71
							TOTAL FOR SECTION 723 403mE						158,742	5.8%	0.66%	0.02%	1.56%	0.20%	43.63%	0.39%	0.05%	0.09%	0.01%	0.03%	47.18%	0.7%
723420mE	MRC053	9	33	1	Mag	Indicated	LTHQ	780	43	33,672	2.89	0		7.24	0.42	0.02	1.62	0.23	42.96	0.44	0.02	0.06	0.01	0.09	48.81	0.79
723420mE	MRC053	33	54	2	Mag	Indicated	LTMQ	1106	43	47,552	2.89	142,161		4.20	0.38	0.01	2.40	0.23	44.12	0.40	0.01	0.10	0.01	0.07	48.01	0.71
723420mE	MRC053	64	80	3	Mag	Indicated	MTLQ	567	43	24,388	2.89	72,920		6.28	0.07	0.06	1.86	0.13	43.61	0.53	0.01	0.10	0.01	0.07	44.30	0.06
723420mE	MRC053	80	83	4	Mag	Indicated	LTHQ	103	43	4,430	2.89	0		11.98	0.45	0.02	1.34	0.07	41.14	0.45	0.01	0.06	0.01	0.06	44.14	0.80
723420mE	MRC053	83	93	5	Mag	Indicated	MTLQ	288	43	12,364	2.99	36,969		8.49	0.66	0.03	1.67	0.08	43.65	0.54	0.01	0.11	0.01	0.08	44.20	0.96
723420mE	MRC053	93	102	6	Mag	Indicated	MTHQ	377	43	16,202	2.89	0		17.33	0.86	0.04	1.50	0.07	38.20	0.51	0.01	0.09	0.01	0.13	40.15	0.91
723420mE	MRC175	5	35	7	Mag	Indicated	LTLQ	967	43	41,583	2.99	124,333		4.38	0.54	0.03	1.87	0.23	44.55	0.42	0.01	0.09	0.01	0.05	47.79	0.75
723420mE	MRC175	35	37	8	Mag	Indicated	Waste	70	43	3,023	2.89	0		12.09	4.79	0.26	1.32	0.13	41.64	0.35	0.01	0.11	0.01	0.06	38.06	0.63
723420mE	MRC175	37	51	9	Mag	Indicated	LTLQ	498	43	21,403	2.99	63,998		5.05	0.81	0.04	2.22	0.26	44.28	0.38	0.01	0.09	0.01	0.05	48.70	0.84
723420mE	MRC175	51	80	10	Dol "B"	Indicated	HCMTMQ	1025	43	44,073	2.85	125,609		10.53	0.74	0.03	1.20	0.50	24.86	20.81	0.01	0.06	0.02	0.07	40.87	37.18
723420mE	MRC175	80	85	11	Mag	Indicated	HTMQ	172	43	7,388	2.99	22,084		14.12	1.79	0.06	3.98	0.13	41.14	0.72	0.04	0.11	0.01	0.04	40.53	1.29
723420mE	MRC175	85	89	12	Mag	Indicated	LTLQ	128	43	5,498	2.99	16,439		3.43	0.83	0.03	3.86	0.17	44.77	0.55	0.13	0.09	0.01	0.04	48.86	0.98
723420mE	MRC175	89	102	13	Mag	Indicated	LTLQ	543	43	23,365	2.99	69,831		1.12	0.31	0.01	1.11	0.21	48.11	0.49	0.03	0.10	0.01	0.02	50.52	0.88
723420mE	MRC052	14	56	14	Mag	Indicated	LTHQ	1669	34	58,004	2.90	0		8.44	0.84	0.04	1.51	0.22	42.37	0.46	0.13	0.09	0.01	0.04	45.89	0.80
723420mE	MRC052	56	135	15	Mag	Indicated	LTLQ	2809	34	95,523	2.99	285,613		5.10	0.56	0.03	1.39	0.20	44.34	0.54	0.02	0.09	0.02	0.04	47.64	0.98
723420mE	MRC052	135	139	16	Mag	Indicated	HTMQ	110	34	3,748	2.99	11,206		15.91	0.58	0.04	0.79	0.05	42.47	0.36	0.01	0.12	0.18	0.03	39.58	0.64
723420mE	MRC052	139	146	17	Dol "B"	Indicated	MCLTLQ	150	34	5,097	2.90	14,760		5.29	0.60	0.04	1.56	0.30	37.89	8.45	0.01	0.06	0.02	0.02	45.85	15.09
723420mE	MRC052	146	150	18	Mag	Indicated	MTLQ	246	34	8,379	2.99	25,052		5.19	1.39	0.07	1.88	0.25	44.21	0.88	0.01	0.09	0.02	0.08	48.30	1.18
723420mE	MD2	13	21.9	19	Dol "B"	Indicated	HCMTMQ	352	43	15,132	2.85	43,125		9.57	0.98	0.04	0.88	0.55	24.97	21.74	0.01	0.05	0.02	0.05	40.97	38.62
723420mE	MD2	21.9	26.9	20	Dol "B"	Indicated	LCMTMQ	160	43	6,876	2.95	20,285		10.28	1.71	0.08	0.81	0.25	41.35	2.89	0.04	0.08	0.04	0.08	43.27	3.79
723420mE	MD2	26.9	45.2	21	Mag	Indicated	LTMQ	644	43	27,695	2.99	82,809		7.88	0.63	0.03	0.76	0.13	43.47	0.43	0.02	0.07	0.01	0.03	48.46	0.77
723420mE	MD2	45.2	59.2	22	Dol "B"	Indicated	HCMTLQ	489	43	21,008	2.85	59,887		8.08	0.90	0.04	0.84	0.42	26.18	20.72	0.01	0.05	0.02	0.07	42.57	37.00
723420mE	MD2	59.2	79.2	23	Mag	Indicated	LTLQ	703	43	30,210	2.99	90,328		3.15	0.50	0.02	1.67	0.23	45.19	0.52	0.01	0.09	0.01	0.05	48.81	0.59
723420mE	MD2	79.2	83.85	24	Mag	Indicated	HTLQ	156	43	6,719	2.99	20,091		10.85	0.34	0.01	1.45	0.14	43.61	0.27	0.01	0.11	0.16	0.01	43.06	0.48
723420mE	MD2	83.85	87	25	Mag	Indicated	LTLQ	111	43	4,763	2.99	14,212		1.53	0.27	0.01	1.70	0.20	45.89	0.40	0.01	0.06	0.01	0.04	50.10	0.71
723420mE	MD2	87	89	26	Dol "B"	Indicated	MCMTLQ	92	43	3,953	2.90	11,492		5.24	0.82	0.04	2.85	0.31	37.34	8.45	0.01	0.06	0.01	0.08	44.50	15.09
723420mE	MD2	89	105.2	27	Mag	Indicated	= LTLQ	544	43	23,408	2.99	69,988		3.06	0.56	0.02	2.47	0.25	44.76	0.45	0.01	0.06	0.01	0.02	48.25	0.80
723420mE	MD2	105.2	109.2	28	Dol "B"	Indicated	LCLTLQ	136	43	5,860	2.95	17,288		2.56	0.23	0.01	0.87	0.15	44.37	2.29	0.01	0.06	0.03	0.01	49.43	4.09
723420mE	MD2	109.2	120.7	29	Mag	Indicated	LTLQ	388	43	16,678	2.99	49,866		3.02	0.33	0.01	1.93	0.19	45.21	0.36	0.01	0.06	0.01	0.02	48.97	0.64
723420mE	MD2	120.7	126.2	30	Mag	Indicated	HTLQ	183	43	7,869	2.99	23,528		10.51	0.73	0.03	0.61	0.11	43.87	0.31	0.01	0.06	0.11	0.02	43.30	0.55
723420mE	MD2	126.2	140.8	31	Mag	Indicated	LTLQ	425	43	18,296	2.99	54,704		1.81	0.51	0.02	0.82	0.26	48.12	0.39	0.01	0.07	0.01	0.02	50.36	0.70
723420mE	MRC051	7	25	32	Mag	Indicated	LTMQ	586	43	25,194	2.99	75,330		7.42	0.99	0.05	1.14	0.21	43.01	0.80	0.01	0.10	0.02	0.07	46.34	1.07
723420mE	MRC051	25	81	33	Mag	Indicated	LTLQ	1903	43	81,843	2.99	244,710		2.89	0.54	0.02	1.47	0.23	45.24	0.62	0.01	0.11	0.02	0.03	49.03	1.11
723420mE	MRC051	81	90	34	Mag	Indicated	HTLQ	293	43	12,607	2.99	37,695		11.33	0.74	0.04	0.92	0.06	43.33	0.34	0.02	0.14	0.24	0.02	42.96	0.61
723420mE	MRC051	90	86	35	Mag	Indicated	LTLQ	242	43	10,417	2.99	31,147		2.57	0.48	0.02	0.62	0.19	45.94	0.44	0.01	0.11	0.02	0.01	49.75	0.79
723420mE	MRC051	86	114	36	Mag	Indicated	MTLQ	527	43	22,643	2.99	67,704		6.84	0.88	0.06	0.81	0.14	43.08	0.55	0.01	0.11	0.05	0.03	44.71	0.56
723420mE	MRC173	10	14	37	Mag	Indicated	LTLQ	261	43	11,211	2.99	33,521		4.08	0.73	0.04	1.03	0.26	45.03	0.47	0.01	0.06	0.01	0.01	48.99	0.75
723420mE	MRC173	14	19	38	Mag	Indicated	MTHQ	118	43	5,056	2.99	0		10.79	0.72	0.03	1.12	0.18	42.27	0.48	0.03	0.06	0.02	0.01	43.99	0.86
723420mE	MRC173	19	20	39	Cave	Indicated	Cave	26	43	1,116	0.00	0														
723420mE	MRC173	20	26	40	Mag	Indicated	MTMQ	185	43	7,970	2.99	23,831		11.29	0.94	0.05	1.10	0.19	42.41	0.81	0.02	0.08	0.02	0.02	42.93	1.09
723420mE	MRC173	26	60	41	Mag	Indicated	LTLQ	1186	43	51,072	2.99	152,706		4.55	0.52	0.02	1.00	0.23	45.14	0.45	0.01	0.10	0.02	0.02	47.59	0.50
723420mE	MRC173	60	64	42	Mag	Indicated	LTLQ	140	43	6,024	2.99	18,012		8.27	0.54	0.01	1.04	0.21	44.33	0.52	0.01	0.09	0.01	0.05	47.32	0.93
723420mE	MRC173	64	102	43	Mag	Indicated	MTLQ	1214	43	52,205	2.99	156,092		4.67	0.67	0.03	0.85	0.20	45.15	0.64	0.01	0.10	0.01	0.04	47.42	1.14
723420mE	MRC050	4	69	44	Mag	Indicated	LTLQ	2240	43	96,303	2.99	287,945		4.82	0.47	0.02	1.12	0.18	45.25	0.46	0.01	0.11	0.05	0.03	47.78	0.82
723420mE	MRC199	7	35	45	Mag	Indicated	MTLQ	626	43	26,933	2.99	80,529		6.55	0.46	0.02	1.05	0.20	44.71	0.64	0.01	0.10	0.02	0.02	46.21	1.14
							TOTAL FOR SECTION 723 420mE						2,777,787	5.5%	0.62%	0.03%	1.35%	0.23%	42.85%	2.28%	0.01%	0.09%	0.03%	0.04%	46.87%	4.1%
723470mE	MRC123	12	31	1	Mag	Indicated	MTHQ	615	50	30,739	2.99	0		13.49	0.43	0.02	1.01	0.18	41.64	0.56	0.01	0.08	0.02	0.12	42.46	1.00
723470mE	MRC123	31	37	2	Mag	Indicated	LTLQ	199	50	9,949	2.99	29,748		2.17	0.73	0.03	1.35	0.17	44.66	1.93	0.01	0.10	0.02	0.13	48.59	3.45
723470mE	MRC123	37	47	3	Mag	Indicated	Waste	340	50	17,017	2.99	0		8.09	0.68	0.04	8.16	0.18	40.54	0.46	0.01	0.08	0.02	0.11	41.57	0.82
723470mE	MRC123	47	61	4	Mag	Indicated	HTLQ	470	50	23,511	2.99	70,299		8.75	1.61	0.05	1.00	0.10	43.42	0.30	0.01	0.10	0.02	0.05	42.97	0.69
723470mE	MRC123	61	65	5	Dol "B"	Indicated	HCHTLQ	138	50	6,766	2.86	19,370		9.99	1.15	0.07	0.63	0.52	25.72	22.29	0.01	0.07	0.03	0.08	39.57	39.80
723470mE	MRC123	65	71	6	Dol "B"	Indicated	HCMTHQ	194	50	9,692	2.86	0		15.86	1.04	0.06	0.72	0.69	20.31	23.47	0.01	0.07	0.04	0.15	37.31	41.61
723470mE	MRC123	71	74	7	Dol "B"	Indicated	HCHTLQ	108	50	5,389	2.85	15,359		10.88	0.81	0.05	0.74	0.81	26.20	21.04	0.01	0.06	0.02	0.13	39.42	37.57
723470mE	MRC123	74	77	8	Dol "B"	Indicated	HCMTHQ	251	50	12,574	2.85	0		12.40	1.01	0.06	0.81	0.65	22.94	21.99	0.01	0.06	0.02	0.12	39.93	39.27

Dolcer Associates

April 2001 - AS - 01640074

BATCHELOR MAGNESITE PROJECT

WINCHESTER PROSPECT JUNE 2000 INDICATED INFERRED (MINUS HIGH QUARTZ) RESOURCE

SECTION	HOLE	FROM	TO	Block Number	Rock Type	Resource Category	Quality Code	Area (Sq m)	Strike Extent (m)	Volume (Cubic m)	SG	Tonnage
							TOTAL FOR SECTION 723 520mE					

	TOTAL TONNAGE or SECTION	SiO_2 % LW(ave)	Al_2O_3 % LW(ave)	TiO_2 % LW(ave)	Fe_2O_3 % LW(ave)	MnO % LW(ave)	MgO % LW(ave)	CaO % LW(ave)	K_2O % LW(ave)	Na_2O % LW(ave)	SO_3 % LW(ave)	P_2O_5 % LW(ave)	LOI % LW(ave)	$CaCO_3$ % LW(ave)	$MgCO_3$ % LW(ave)	MgO in carb % LW(ave)	Talc % LW(ave)	Quartz % LW(ave)
TOTAL FOR SECTION 723 520mE	862,838	6.7%	0.87%	0.03%	0.97%	0.21%	41.82%	3.85%	0.01%	0.10%	0.01%	0.04%	45.39%	6.9%	81.2%	38.8%	9.5%	0.8%
	TOTAL TONNAGE	SiO_2%	Al_2O_3%	TiO_2	Fe_2O_3	MnO	MgO	CaO	K_2O	Na_2O	SO_3	P_2O_5	LOI	$CaCO_3$	$MgCO_3$	MgO in carb	Talc	Quartz
TOTAL INDICATED & INFERRED (MINUS HIGH QUARTZ) RESOURCE	16,649,441	5.7%	0.74%	0.03%	1.39%	0.21%	43.22%	2.07%	0.01%	0.10%	0.02%	0.04%	46.37%	3.7%	85.8%	41.0%	6.9%	1.4%
	TOTAL TONNAGE	SiO_2%	Al_2O_3%	TiO_2	Fe_2O_3	MnO	MgO	CaO	K_2O	Na_2O	SO_3	P_2O_5	LOI	$CaCO_3$	$MgCO_3$	MgO in carb	Talc	Quartz
RESOURCE (MINUS HIGH QUARTZ) STATEMENT	16.6Mt	5.7%	0.7%	0.03%	1.4%	0.2%	43.2%	2.1%	0.01%	0.10%	0.02%	0.04%	46.4%	4%	86%	41%	7%	1%
INDICATED (MINUS HIGH QUARTZ) RESOURCE STATEMENT	12.2Mt	5.8%	0.7%	0.03%	1.3%	0.2%	43.1%	2.2%	0.01%	0.09%	0.02%	0.04%	46.4%	4%	86%	41%	7%	2%
INFERRED (MINUS HIGH QUARTZ) RESOURCE STATEMENT	4.4Mt	5.5%	0.8%	0.04%	1.7%	0.2%	43.6%	1.8%	0.01%	0.11%	0.03%	0.05%	46.2%	3%	86%	41%	8%	1%

Strictly invalid mineralogical calculation but only just over the threshold

Estimated mineralogical values only - elevated aluminium or iron makes calculation invalid

WINCHESTER PROSPECT JUNE 2000 INDICATED & INFERRED (MINUS HIGH QUARTZ) RESOURCE FIGURE 49

File: 01888a

Golder Associates

APPENDIX B – CAPITAL COST ESTIMATES

	CAPITAL COST ESTIMATE - WINCHESTER MAGNESITE MINE AT 1,000,000 T/YR.. ROM						
	Description	**Table**	**Note**	**Unit**	**Rate**	**Amount**	**Total**
	SUMMARY						
	WATER MANAGEMENT	1					626,810
	SITE INFRASTRUCTURE	2					1,079,310
	WASTE DUMPS	3					108,925
	MINE	4					1,293,290
	SUBTOTAL						**3,108,335**
	CONTINGENCY				30%		932,501
	TOTAL ESTIMATE						**4,040,836**
	Description	**Table**	**Note**	**Unit**	**Rate**	**Amount**	**Total**
1	**WATER MANAGEMENT**						
1.1	**BORE CONSTRUCTION**						
1.1.1	Drill, construct and test shallow dewatering bores of 20 m depth			item	9,000	10	90,000
1.1.2	Drill, construct and test shalJow dewatering bores of 100 m depth			item	27,000	5	135,000
							225,000
1.2	**DEWATERING PUMPS**						
1.2.1	Supply pump with shroud, cable, rising main, headworks and control box for 10 shallow bores			Item	6,420	10	64,200
1.2.2	Supply pump with shroud, cable, rising main, headworks and control box for 5 deep bores			item	30,000	5	150,000
1.2.3	Supply pump, flotation module, 50 m cable for sump pump (no installation costs provided)			item	80,000	1	80,000
1.2.4	Supply in-pond pump and control box for pumping from holding ponds to process plant			item	7,500	1	7,500
							301,700
1.3	**PONDS**						
1.3.1	Excavate settling pond for dirty water pumped from in-pit			m3	6.00	225	1,350
1.3.2	Prepare foundation of settling pond			m2	1.80	150	270
1.3.3	Excavate two holding ponds for clean water pumped from bores			m3	6.00	150	900
1.3.4	Prepare foundation of holding ponds			m3	1.80	95	171
1.3.5	Supply and install liner for holding pond			m2	6.00	95	570
							3,261
1.4	**PIPELINE**						
1.4.1	Supply polypipe PN 6.3. PE 100, 125 mm			m	9.50	870	8,265
1.4.2	Installation, buttwelding and labour			m	3.50	870	3,045
1.4.3	Supply polypipe PN 6.3, PE 100. 200 mm			m	26.50	580	15,370
1.4.4	Installation, buttwelding and labour			m	3.90	580	2,262
1.4.5.	Supply polypipe PN 6.3, PE 100, 225 mm			m	31.00	260	8,060
1.4.6	Installation, buttwelding and labour			m	4.70	260	1,222
1.4.7	Supply polypipe PN 6.3, PE 100. 315 mm			m	48.00	230	11,040
1.4.8	Installation, buttwelding and labour			m	9.50	230	2,185
1.4.9	Supply polypipe PN 10. PE 100, 225 mm			m	45.00	425	19,125
1.4.10	Installation, buttwelding and labour			m	7.00	425	2,975
1.4.11	Supply polypipe PN 6.3. PE 100. 110 mm			m	9.50	1,200	11,400
1.4.12	Installation, buttwelding and labour			m	7.00	1,200	8,400
1.4.13	Fittings (atr valves. y-join, reducers, etc.)			allow	3,500	1	3,500
							96,849
	WATER MANAGEMENT TOTAL						**626,810**
	Description	**Table**	**Note**	**Unit**	**Rate**	**Amount**	**Total**
2	**SITE INFRASTRUCTURE**						
2.1	**CRUSHER SITE**						
2.1.1	Clear and grub footprint			m2	0.20	77,000	15,400
2.1.2	Clear and grub topsoil stockpile area		*1*	m2	0.20	8,250	1,650
2.1.3	Remove and stockpile topsoil 0.15 m thick		*1*	m3	2.50	12,000	30,000
2.1.4	Compact subgrade in fill area		*1*	m2	0.90	2,000	1,800
2.1.5	Excavate cut and grade surface		*1*	m3	12.00	1,000	12,000
2.1.6	Haul, place and compact engineered fill		*1*	m3	1.60	1,000	1,600
2.1.7	Excavate and grade stormwater diversion channel			m3	16.00	9,750	156,000
							218,450
2.2	**ROM PAD**						
2.2.1	Clear and grub footprint			m2	0.20	110,000	22,000
2.2.2	Clear and grub topsoil stockpile area			m2	0.20	8,250	1,650
2.2.3	Remove and stockpile topsoil 0.15 m thick			m3	6.00	16,500	99,000
2.2.4	Excavate cut and grade surface			m3	16.00	8,000	128,000
2.2.5	Cart ROM waste for fill			m3	0.40	250,000	100,000
							350,650
2.3	**ACCESS ROAD (unsealed)**						
2.3.1	Clear and grub footprint			m2	0.20	30,000	6,000
2.3.2	remove and stockpie topsoil 0.15 m thick			m3	2.50	4,500	11,250
2.3.3	Excavate cut and grade surface			m3	16.00	10,000	160,000
2.3.4	Compact subgrade			m2	1.80	30,000	54,000
2.3.5	Haul, place and compact engineered basecourse			m3	3.50	36,000	126,000
2.3.6	Excavate and grade stormwater trenches			m3	16.00	1,500	24,000
2.3.7	Supply and install culverts			allow	2.00	10,000	20,000
							401,250
2.4	**CREEK DIVERSION**						
2.4.1	Clear and grub low flow channel			m2	0.20	10,800	2,160
2.4.2	Excavate and grade low flow channel			m3	16.00	6,300	100,800
2.4.3	Construct bund waits from Rom waste			m3	0.20	30,000	6,000
							108,960
	SITE INFRASTRUCTURE TOTAL						**1,079,310**

CAPITAL COST ESTIMATE - WINCHESTER MAGNESITE MINE AT 1,000,000 T/YR.. ROM							
	Description	**Table**	**Note**	**Unit**	**Rate**	**Amount**	**Total**
3	**BYPRODUCT DUMPS (5 years storage)**						
3.1	**WASTE DUMP**						
3.1.	Clear and grub footprint			m2	0.20	58,800	11,760
3.1.2	Clear and grub topsoil stockpile area			m2	0.20	4,450	890
3.1.3	Remove and stockpile topsoil 0.15 m thick			m3	3.50	8,850	30,975
3.1.4	Excavate and grade stormwater diversion channel			m3	16.00	2,450	39,200
							82,825
3.2	**FINES DUMP**						
3.2.1	Clear and grub footprint			m2	0.20	36,000	7,200
3.2.2	Clear and grub topsoil stockpile area			m2	0.20	2,625	525
3.2.3	Remove and stockpile topsoil 0.15 m thick			m3	3.50	5,250	18,375
							26,100
	BYPRODUCT DUMPS TOTAL						**108,925**
	Description	**Table**	**Note**	**Unit**	**Rate**	**Amount**	**Total**
4	**MINE**						
4.1	**ESTABLISHMENT**						
4.1.1	Mobilisation						
4.1.2	Fuel storage			item			45,000
4.1.3	Crib room, ablution block			item			12,000
4.1.4	Workshop, store			item			38,000
4.1.5	Light vehicle			item			25,000
4.1.6	Computer			item			5,000
4.1.7	Explosive magazines			item			12,000
4.1.8	Construct protective bunds			m3	0.20	10,000	2,000
4.1.9	Construct haulroad to ROM pad			m3	0.50	40,000	20,000
							159,000
4.2	**PIT**						
4.2.1	Clear and grub footprint			m2	0.20	50,000	10,000
4.2.2	Clear and grub topsoil stockpile area			m2	0.20	4,000	800
4.2.3	Remove and stockpile topsoil 0.15 m thick			m3	3.50	7,500	26,250
4.2.4	Mine overburden			m3	3.00	353,397	1,060,190
							1,097,240
4.3	**WASTE DUMP**						
4.3.1	Clear and grub footprint			m2	0.20	50,000	10,000
4.3.2	Clear and grub topsoil stockpile area			m2	0.20	4,000	800
4.3.3	Remove and stockpile topsoil 0.15 m thick			m3	3.50	7,500	26,250
							37,050
	MINE TOTAL						**1,293,290**
	NOTES						
1	*Not required for DSO operation*						

APPENDIX C – MINE OPERATING COSTS ESTIMATES (DRILL AND BLAST)

MINE OPERATING COSTS - (SHOVEL, DRILL AND BLAST DEVELOPMENT VARIANT)						
Year		1	2	3	4	5
Bench		70	70-65	65-60	60-55	55
Number of Excavators		2	2	2	2	2
Coarse Magnesite/Fines		80%	80%	80%	80%	80%
Coarse Magnesite		200,000	400,000	800,000	800,000	800,000
Fines		50,000	100,000	200,000	200,000	200,000
ROM Ore	T/yr	250,000	500,000	1,000,000	1,000,000	1,000,000
Overburden	T/yr	220,873	357,783	91,069	56,236	39,833
Waste Rock	T/yr	292,968	524,838	748,911	693,355	486,030
Total	T/yr	763,841	1,382,621	1,839,980	1,749,591	1,525,863
Staff	$/yr	317,500	508,333	930,000	930,000	930,000
Grade Control	$/yr	80,000	160,000	320,000	320,000	320,000
Dayworks	$/yr	40,000	40,000	40,000	40,000	40,000
ROM Loader	$/yr	150,000	150,000	300,000	300,000	300,000
Mobilisation	$/yr	75,000	75,000	75,000	75,000	75,000
Contract Mining	$/yr	2,143,952	4,003,853	6,356,015	6,147,548	5,505,888
Total	**$/yr**	**2,806,452**	**4,937,186**	**8,021,015**	**7,812,548**	**7,170,888**

WINCHESTER MINE OPERATING COST ESTIMATE OVER 5 YEARS (DRILL AND BLAST)					
Description	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5
WATER MANAGEMENT ($/YR.)	440,000	440,000	440,000	440,000	440,000
WASTE DUMPS ($/YR.)	180,000	180,000	180,000	180,000	180,000
MINE ($/YR.)	2,806,452	4,937,186	8,021,015	7,812,548	7,170,888
CRUSHING ($/YR.)	1,100,000	2,200,000	4,400,000	4,400,000	4,400,000
SUBTOTAL ($/YR.)	4,526,452	7,757,186	13,041,015	12,832,548	12,190,888
CONTINGENCY (30%)	1,357,935	2,327,156	3,912,304	3,849,764	3,657,267
TOTAL ESTIMATE	5,884,387	10,084,342	16,953,319	16,682,313	15,848,155
CAPACITY OUTPUT ROM MAGNESITE (T/YR.)	250,000	500,000	1,000,000	1,000,000	1,000,000
SALEABLE COARSE MAGNESITE COST ($/T)	29.42	25.21	21.19	20.85	19.81
COARSE MAGNESITE/FINES	80%	80%	80%	80%	80%
CAPACITY OUTPUT COARSE SALEABLE MAGNESITE (T/YR.)	200,000	400,000	800,000	800,000	800,000
CAPACITY OUTPUT FINES (T/YR.)	50,000	100,000	200,000	200,000	200,000

APPENDIX D - MINE OPERATING COSTS ESTIMATES (CONTINUOUS MINER)

MINE OPERATING COSTS - (CONTINUOUS MINER AND LOADER DEVELOPMENT VARIANT)						
Year		1	2	3	4	5
Bench		70	70-65	65-60	60-55	55
Number of Excavators		2	2	2	2	2
Coarse Magnesite/Fines		80%	80%	80%	80%	80%
Coarse Magnesite		200,000	400,000	800,000	800,000	800,000
Fines		50,000	100,000	200,000	200,000	200,000
ROM Ore	T/yr	250,000	500,000	1,000,000	1,000,000	1,000,000
Overburden	T/yr	220,873	357,783	91,069	56,236	39,833
Waste Rock	T/yr	292,968	524,838	748,911	693,355	486,030
Total	T/yr	763,841	1,382,621	1,839,980	1,749,591	1,525,863
Staff	$/yr	317,500	508,333	930,000	930,000	930,000
Grade Control	$/yr	80,000	160,000	320,000	320,000	320,000
Dayworks	$/yr	40,000	40,000	40,000	40,000	40,000
Mobile Loader	$/yr	150,000	150,000	300,000	300,000	300,000
Mobilisation	$/yr	75,000	75,000	75,000	75,000	75,000
Contract Mining	$/yr	1,503,328	2,804,069	4,357,467	4,223,075	3,857,849
Total	**$/yr**	**2,165,828**	**3,737,402**	**6,022,467**	**5,888,075**	**5,522,849**

WINCHESTER MINE OPERATING COST ESTIMATE OVER 5 YEARS (CONTINUOUS MINER)					
Description	**YEAR 1**	**YEAR 2**	**YEAR 3**	**YEAR 4**	**YEAR 5**
WATER MANAGEMENT ($/YR.)	440,000	440,000	440,000	440,000	440,000
WASTE DUMPS ($/YR.)	180,000	180,000	180,000	180,000	180,000
MINE ($/YR.)	2,165,828	3,737,402	6,022,467	5,888,075	5,522,849
CRUSHING ($/YR.)	575,000	1,150,000	2,300,000	2,300,000	2,300,000
SUBTOTAL ($/YR.)	3,360,828	5,507,402	8,942,467	8,808,075	8,442,849
CONTINGENCY (30%)	1,008,248	1,652,221	2,682,740	2,642,422	2,532,855
TOTAL ESTIMATE	**4,369,076**	**7,159,623**	**11,625,207**	**11,450,497**	**10,975,704**
CAPACITY OUTPUT ROM MAGNESITE (T/YR.)	250,000	500,000	1,000,000	1,000,000	1,000,000
SALEABLE COARSE MAGNESITE COST ($/T)	**21.85**	**17.90**	**14.53**	**14.31**	**13.72**
COARSE MAGNESITE/FINES	80%	80%	80%	80%	80%
CAPACITY OUTPUT COARSE MAGNESITE (T/YR.)	200,000	400,000	800,000	800,000	800,000
CAPACITY OUTPUT FINES (T/YR.)	50,000	100,000	200,000	200,000	200,000