See file 2M/99 P71
Folio 8.

GOLDEN TRIANGLE RESOURCES NL

MAIN CREEK MAGNESITE PROJECT, TASMANIA

CONCEPTUAL MINE PLANNING STUDY

BFP

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TABLE OF CONTENTS

APPENDIX

LIST OF FIGURES

EXECUTIVE SUMMARY

BFP Consultants Pty Ltd (BFP) have undertaken a conceptual study of mining possibilities and options for the Main Creek Magnesite prospect in NW Tasmania, for Golden Triangle Resources NL (GTR). Geological information, as provided in a report for GTR by Mr L. Newnham dated June 1998, has been used, along with general data with respect to general project objectives. Other details were outlined in the study briefing by Mr CA Laughlin, General Manager of GTR, prior to a site visit.

As a project base for the scale of project envisaged and assuming the material is suitable for or can be economically beneficiated to a suitable smelter feed, the Inferred Resource of 47.4Mt's @ 43.36 MgO represents a more than adequate starting point. Depending on the specifications required for a smelter feed, a substantial proportion of this resource, may, when better defined prove suitable for direct shipping of ore to a smelter, rather than requiring an intermediate beneficiation stage (to reduce SiO_2 content).

The site area is relatively difficult to access, making the prospects of an open pit mining operation problematical because of both the effects of geological and geographical features. These pose problems for any potential pit sites, and for any waste dump sites for overburden deriving therefrom in the course of mining. Only one site, based on Drill Sections MC1, 30 34, ref. the northern (Main Creek) lenses is considered to have potential as an open pit site, and then only for short term (4-5 years). Possible locations of waste dump sites and stream relocations would need to be verified before advancing this option.

It is likely that for a small scale, short term open pit operation in the environment of NW Tasmania, open pit mining costs would be of the order of $10/tonne of ore produced, assuming a strip ratio of approximately 2:1. This high cost is predicated by requirements for dump site access, rehabilitation of dump sites and creek diversion, etc.

For underground mining, the technically possible options are more varied. The most attractive proposition is that of C Lens in the Southern (Bowry Creek) Group of lenses. This lens could be accessed, at quite shallow depth in fresh rock, from a decline having a portal site adjacent to the existing access track thus ensuring lowest cost initial exploration/development costs.

A particular attraction of the sector identified is the major thickness (+65m) of low SiO_2 (1.26%) Magnesite rock along the hangingwall section of C Lens, in this vicinity. Such a sequence might be expected to contain horizons of even lower SiO_2 (<1.0%) content which if present over an extend strike length and/or depth, might allow for an ((initial) supply of direct feed grade ore to the smelter.

Several underground mining methods have been examined. The preferred option being an initial along strike oriented room and pillar exploratory phase on 1 or 2 upper levels with pillars cut through only where sampling confirms grade requirements. This would be followed by cut and fill mining or a conversion to up-hole benching.

The cut and fill approach would allow a high degree of selectively and flexibility particularly in the early stages of mining when the orebody is not well defined and understood. It would avoid the non-selectivity/dilution aspects likely with large scale conventional open stoping, and the risk of possible future subsidence in the absence of backfill for worked out stopes.

Backfill will be required for cut and fill mining and may be required to allow extraction of benched stoping blocks adjacent to the first-mined stopes. A backfill source could be provided from a separate backfill-rock quarry or, if such a site were practical and environmentally acceptable, from supplementary stopes in low grade magnesite reasonably close to the underground mining.

The latter option would entail extra costs, but because of the bulking factor of broken rock, could be logically considered in the context of this operation. The extra cost to mining might only be of the order of $5.00/tonne of ore.

For a small scale underground operation of the type envisaged in this report, it is anticipated that, depending on the method of stoping finally adopted, and the sourcing of backfill, if required, an operating cost of between $18-24/tonne of ore produced is likely.

G DAVISON
Manager, Mining

CE JAMES
Mining Consultant

1.0 INTRODUCTION

BFP Consultants Pty Ltd (BFP) have undertaken this study of mining options of the Main Creek Magnesite Prospect, near Savage River in Western Tasmania, at the request of Golden Triangle Resources NL (GTR) who hold an Option to Purchase Agreement with respect to the tenements covering the prospect. (refer Fig. 1)

This study follows completion by Mr Newnham (also acting on behalf of GTR) of a Resource Estimate for both the Main Creek and Bowry Creek deposits following an exploration programme of nine cored drill holes, for a total of 3,381 metres in the first half of Calendar 1998.

The total inferred resource is:

47.4 Mtonnes @ 43.36% MgO, 2.20% CaO, 2.66 SiO_2 and 1.95%, F_2O_3

This tonnage has been computed on the basis of five high grade lenses which have been identified within the host carbonate sequence rocks, which range from 180m to 400m in true thickness.

A breakdown of the 47.4M tonne Inferred Resource is given in Table 1.1, below.

TABLE 1.1
INFERRED RESOURCE, BY LENS*

Lens	Tonnes (Mt)	Grade (%)			
		MgO	CaO	SiO_2	Fe_2O_3
Bowry Creek					
Lens A	6.4	42.10	2.27	3.25	2.64
Lens C	16.7	42.86	2.23	2.81	2.20
Bowry Creek Sub-total	23.1	42.64	2.24	2.93	2.32
Main Creek					
Lens D	7.1	43.10	2.00	3.14	1.84
Lens E	7.3	43.92	2.23	2.43	1.42
Lens F	9.9	44.80	2.25	1.88	0.63
Main Creek Sub-total	24.3	44.03	2.17	2.41	1.22
Total Resource Estimate	**47.3**	**43.36**	**2.20**	**2.66**	**1.75**

(* As per report by GTR, 10 June 1998)

Prior to the execution of this study, BFP's Manager, Mining, Mr Gary Davison visited the site, in company with Messrs Newnham and Jones, the latter being a Principal of NSR Environmental Consultants, and also held discussions with GTR in their Melbourne offices concerning GTR's general expectations for the project.

Key elements are:
- A project scale of some 60-90,000 tonnes per annum of magnesium metal produced, is envisaged.
- This equates to a mining requirements of some 250,000-400,00t/a, approx.

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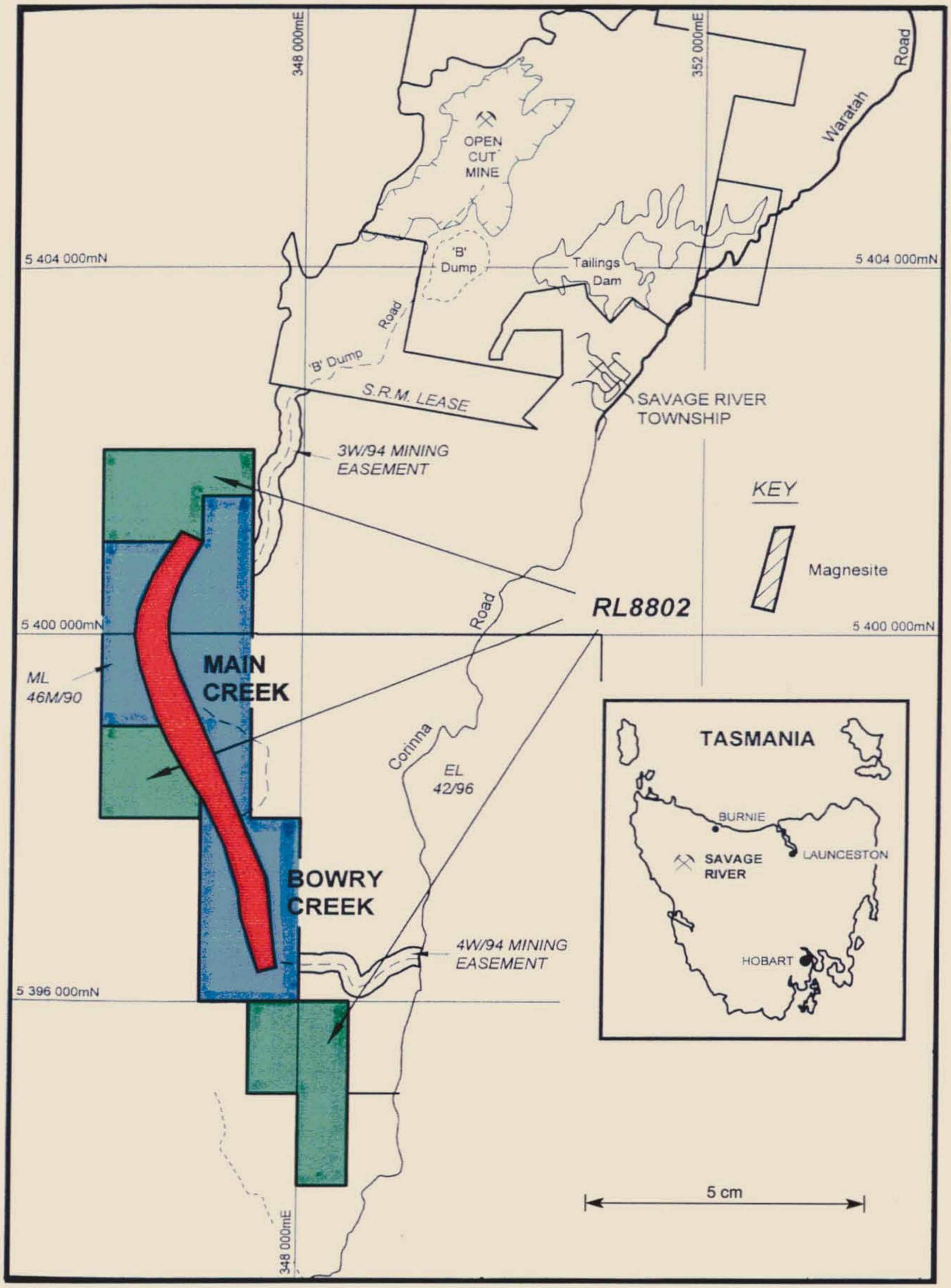


Courtesy Newnham Exploration and Mining Services

CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Prospect*	Location Plan		
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007 FILE Fig1.CDR	SCALE N.T.S. DATE 30/7/98	FIG. 1

- The area is considered "environmentally sensitive, having already been subject to river degradation from mining effluent. Further, the calcareous sequence hosting the Magnesite lenses abuts pyrite-bearing schists on both footwall and hangingwall. The footwall schists are considered to be extremely reactive and liable to acid generation, if disturbed. UB

- Access to the project area is difficult with undulating to precipitous topography, and dense tree or scrub cover. Rainfall at the adjacent Savage River Fe Mine is high.

- It has been suggested that a Smelter feed head grade of :
  - >40%MgO
  - < 3% CoO
  - $\leq$1% SiO_2 is required.

Selection of ore from areas having CaO <3% is assumed, and if SiO_2 levels can be kept to "near 1%", or less, then preliminary beneficiation of Magnesite ore, ahead of smelting could be avoided, at least in the initial years of mining.

2.0 REVIEW OF OPTIONS

2.1 The Resource Base

The resource is presently categorized as "Inferred", some 13 holes - 4 from an earlier program, and 9 from the recently completed programme having been completed in the ore horizons.

As such, interpretation of individual lens limits and dimensions/attitudes has been done using this limited information and any changes to this information may affect any present interpretations.

The tonnage identified is well in excess of that needed to provide a resource base for the project, with, say 15-25 years at 400,000 t/a, or 6Mt to 10Mt total, in mineable reserves being necessary.

The key questions therefore are:

- Can a sufficiently high proportion of the identified tonnage be mined at an acceptable cost?.
- Should the resource be mined by open cut methods? or,
- Should the resource be mined by underground methods?

It is noted that the resource base is of "Inferred" status, only, and that before any detailed study and/or project planning work is undertaken it will be necessary to upgrade a significant proportion of the preferred resource blocks, at least, to "Measured" and/or "Indicated" status. UB

2.2 Mining Concept, Scale Required

At 400,000 t/a the required scale of mining is quite modest, either by open pit, or by underground methods.

It is understood that for the "Main Creek" Magnesite prospect, whilst the absolute cost of mining will be a high ranking consideration in the selection of a preferred mining option, in the overall project context the key considerations will be reliability and consistency of the grade of ore to the "mill".

2.3 Technical Constraints

It is evident, from inspection of retained drill core, that the ore can only be differentiated from sub-ore and waste by assay control. Visually, it will be difficult or impossible to determine what is direct shipping ore, beneficiable ore, or sub-ore/waste. Hence, precise mining to survey plan and strict allocation of blasted rounds, according to assay results will be a prime determinant of the selective mining process.

If it occurs that a beneficiation plant is established on site, only minor quantities of material would be available as a backfill for any underground mining option.
The hangingwall pyritic schists pose acid generating problems and the footwall schists have given evidence of the same as well as regions of high pressure water. NB.

2.4 Environmental Constraints

It is understood that the project area is presently classified as "Multiple Use Forest", and managed by the Forestry Commission of Tasmania. As such it is available for the resources of the area, including mineral resources for commercial exploration.

As noted in the Newnham report to GTR June 1998, the southern part of the project area of predominantly eucalyptus forest has been selectively logged (±20 years, ago) and more recently subject to bush fire (early 1980's). Scrubby re-growth prevails. The northern sector is predominantly myrtle rainforest with Sassafras and celery top pines intermixed.

As noted earlier, the terrain is precipitous, if generally low-lying (mainly less than 200m in elevation), with numerous tributary streams to the two mains watercourses, Bowry Creek and Main Creek. Main Creek runs through the existing Savage River Mine workings and shows evidence of surface runoff/degradation at that site. Bowry Creek has not been subject to such sedimentation from mining. The heavy (+2500mm) annual precipitation does mean that all exposed surfaces are subject to scouring and soil denudation on a continuous basis.

In the light of the foregoing, it will be necessary for any surface works established for the project to be carefully engineered and designed to mitigate surface erosion and deposition of particulates (soil, rock and/or vegetative matter) in creeks.

A further complication is the presence of (highly-reactive) pyrite in the footwall schists material which is likely to cause serious acid leach problems, if exposed to any extent by mining, dumps, or roadworks, etc.

For these reasons any open pit mining is likely to have significant environmental impact, and/or mitigation costs associated with its implementation.

Therefore, only relatively shallow, small scale pits with minor quantities of waste generated have been examined at this stage, as:

- Land areas suitable for permanent stockpiling of overburden are limited.
- The pyritic schist on the western (footwall) side of the carbonate sequence host blocks should be left undisturbed to avoid acid generation.

It may transpire that for underground mining, substantial (say ± 200,000 tpa) quantities of backfill material could be required from a surface quarry. Such a requirement would also be best sourced from within the carbonate sequence of rocks. If even such a minor quarry proves environmentally unacceptable then it would be necessary to mine additional underground stopes in low grade (sub-ore) areas and use this as backfill.

Whichever primary mining option is adopted - open pit or underground, it will be necessary to upgrade the access road system to allow delivery of mine supplies, man transport to and from site, and the dispatch of 400,000 tonnes of Magnesite ore/year.

3.0 OPEN PIT OPTION

3.1 Prospective Sites

As noted earlier, the terrain is precipitous, with numerous tributary streams to the two main watercourses, Bowry Creek and Main Creek, the latter including the existing Savage River Mine workings in its catchment and showing evidence of surface runoff/degradation at that site. Bowry Creek has not been subject to such degredation from mining.

The heavy (+2500mm) annual precipitation does mean that all exposed surfaces are subject to scouring and soil denudation on a continuous basis.

In the light of the foregoing, it would be necessary of any surface works established for the project to be carefully engineered and designed to mitigate surface erosion and deposition of particulates (soil, rock and/or vegetative matter) in creeks.

A further complication is the presence of (highly-reactive) pyrite in the footwall schists material which are likely to cause serious acid leach problems, if exposed to any extent by mining, dumps, or roadworks, etc.

For these reasons any open pit mining is likely to have significant environmental impact, and/or mitigation costs associated with its implementation.

Therefore, only relatively shallow, small scale pits with minor quantities of waste generated could possibly implemented.

A examination of the geological sections provided also shows that, if early access to high grade sections of ore (i.e. low SiO_8 content) is required, then only limited parts of two three of the five individual lenses could be considered for open pit mining.
Two possible blocks (in terms of high grade) have been identified viz;

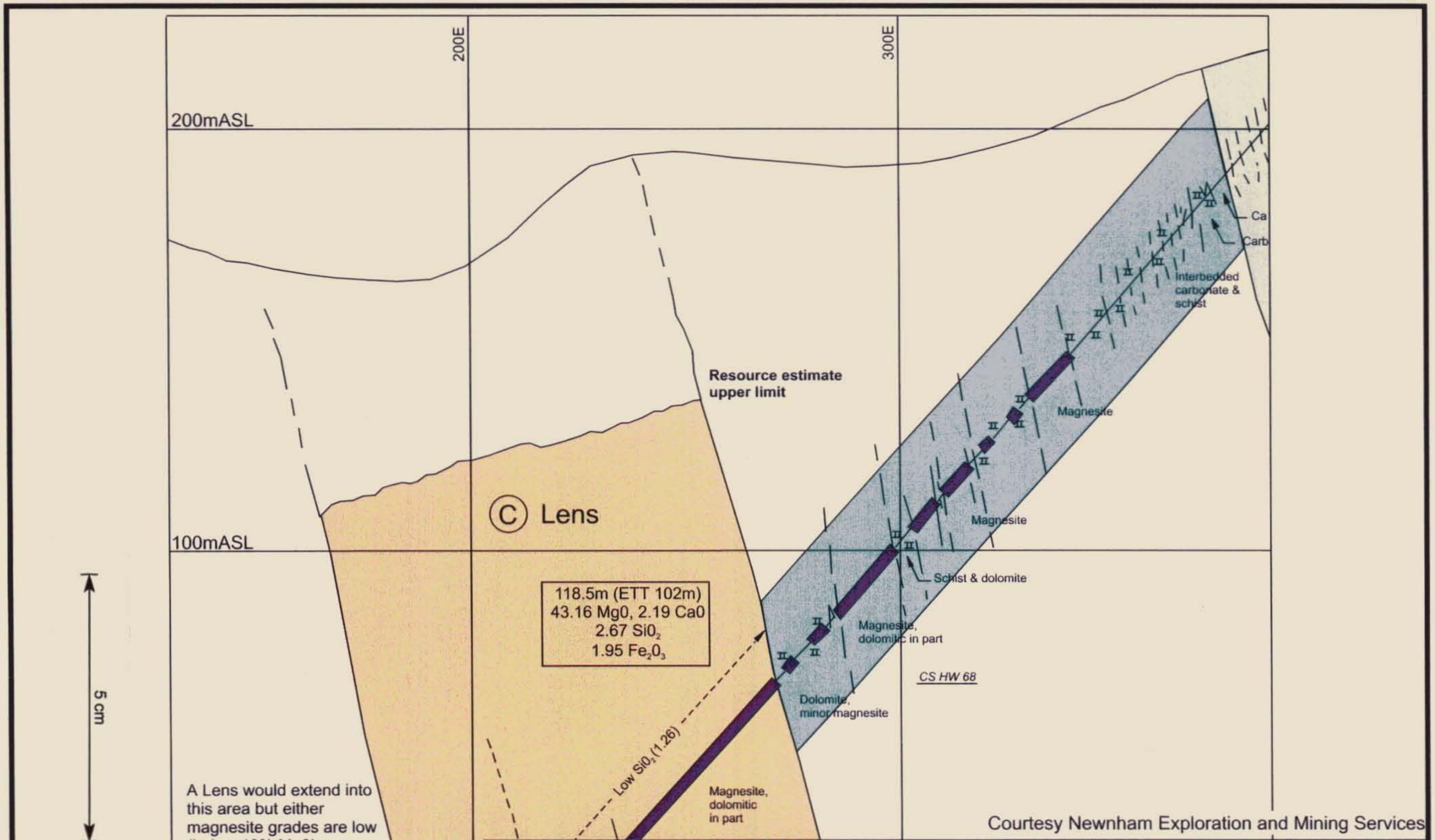
a) One area, based on C Lens (section MC32 - Refer Fig. 2) is estimated to contain up to 2M tonnes of ore of average 1.26% SiO_2 , at a strip ratio of approx. 6.1, over a strike length of 200 metres (100m ea. Side of intersection).

This area does, however, suffer from a lack of prospective overburden dump sites, nearby, and is adjacent to the as yet uncontaminated Bowry Creek.

(b) A second open pit area is that based on Lenses E, and F1 in the northern (Main Creek) section of the project (Refer Fig.3, Section MC1, 30 34 composite). This area similarly suffers from a lack of prospective overburden dump sites, however, it has been examined to put the cost and problems involved with an open cut into some perspective.

In this area the major unit (E lens) is estimated to have a block grade of 43.9% MgO, 2.48% CaO, 1.42% SiO_2, 1.50% Fe_2O_3. Over a true width of approx. 47m.

Additionally, a minor hangingwall Lens has a grade of 44.5% MgO, 2.34% CaO, 0.55% SiO_2 and 1.42% Fe_2O_3 over a true width of 17m, approx.
Plus



CLIENT *GOLDEN TRIANGLE RESOURCES NL*
Main Creek Magnesite Prospect

BFP Geotechnical, Mining & Geological Consultants

Geological Section Through C Lens, Bowry Creek Section (MC 32)

JOB# 13007	SCALE 1:1250		
FILE Fig-2.CDR	DATE 29/7/98		

FIG. 2

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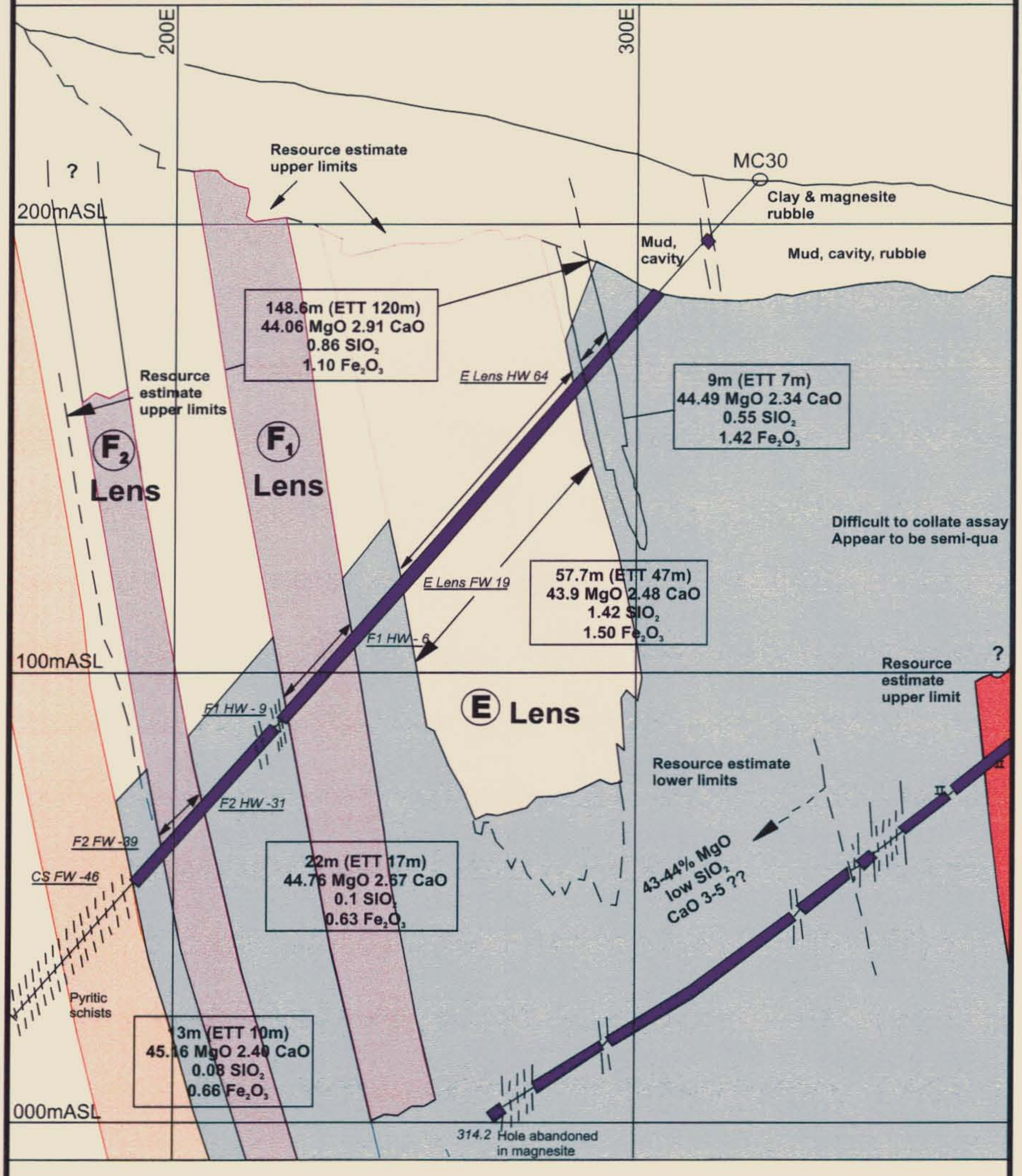

Courtesy Newnham Exploration and Mining Services

CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Prospect*	Geological Section Through E Lens (MC's 1,30,34)		
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007	SCALE 1:1250	FIG. 3
	FILE Fig-3.CDR	DATE 29/7/98	

F1, adjacent to , and on the footwall side of E Lens, grades 44.8% MgO 2.67% CaO 0.1% SiO_2 and 0.63% Fe_2O_3 over a true width of 17m, approx.

If taken to a depth of 40m ASL, or approx. 90m below surface, a block extending 100m North and 150m South towards sections on DDHs MC2, MC36 (i.e. 250m along strike) would yield approx. 1.7M tonnes, at a strip ratio of approx. 2.1, depending on the overall pit slopes adopted.

In terms of strip ratio, this second block is clearly more attractive, though much of the overlying weathered zone of clay and magnesite rubble will be difficult to handle and stockpile.

3.2 Possible Design Parameters

Based on site knowledge from a two day visit, inspections of core and core photos, and some minor geotechnical examination of a magnesite core section, it is considered that any open pit developed with in the carbonate sequence could be relatively steep walled.

Because of restriction from any significant excavation on the footwall side of the carbonate sequence (i.e. within the pyritic schists), open pit ramp access would have to be on the hangingwall side. Final footwall side benches (10m height) would be cut to 75° face angle, with 6m berm width, and an overall slope of approx. 48°. The top (weathered) section would be laid back to 30°, but keeping within the carbonate sequence and clear of the pyritic footwall schists.

On the hangingwall side to compensate for the extra width of pushback necessitated by the access ramp, benches could be merged to double height for final configuration, thus achieving a similar overall pits slope of ±48° (refer Fig. 4a).

Some waste may have to be stockpiled along the eastern boundary of the pit eventually extending further north and south of the pit, providing a buffer zone between the pit and Main Creek, though the majority would be transported to a relatively level area at approx 220m RL some distance from the pit (refer Fig. 4b).

A possible location for a major dump on the western side of the ridge to the west of the pit site would require a steep (1 in 10) haulroad of up to 1.2 km length, to access it. This would incur a major capital cost.

The following densities have been used in the mining calculations.

Bulk Density of ore ....3.0 tonnes/bm^3
Bulk Density of waste ....3.0 tonnes/bm^3
Bulk Density of weathered cover ....2.0 tonnes/bm^3

Total Ore	0.57M BCM
Total Weathered Waste	0.45M BCM
Total Unweathered Waste	0.28M BCM
Strip Ratio (by Volume)	approx. 2:1
Strip Ratio (by Tonnes)	approx. 2.5:1

3.3 Costs

It is assumed that any open pit operation would be of short term duration (4-5 years at full 400,000 t/a rating) and would need to be carried out in conjunction with or ahead of a longer term underground mining operation.

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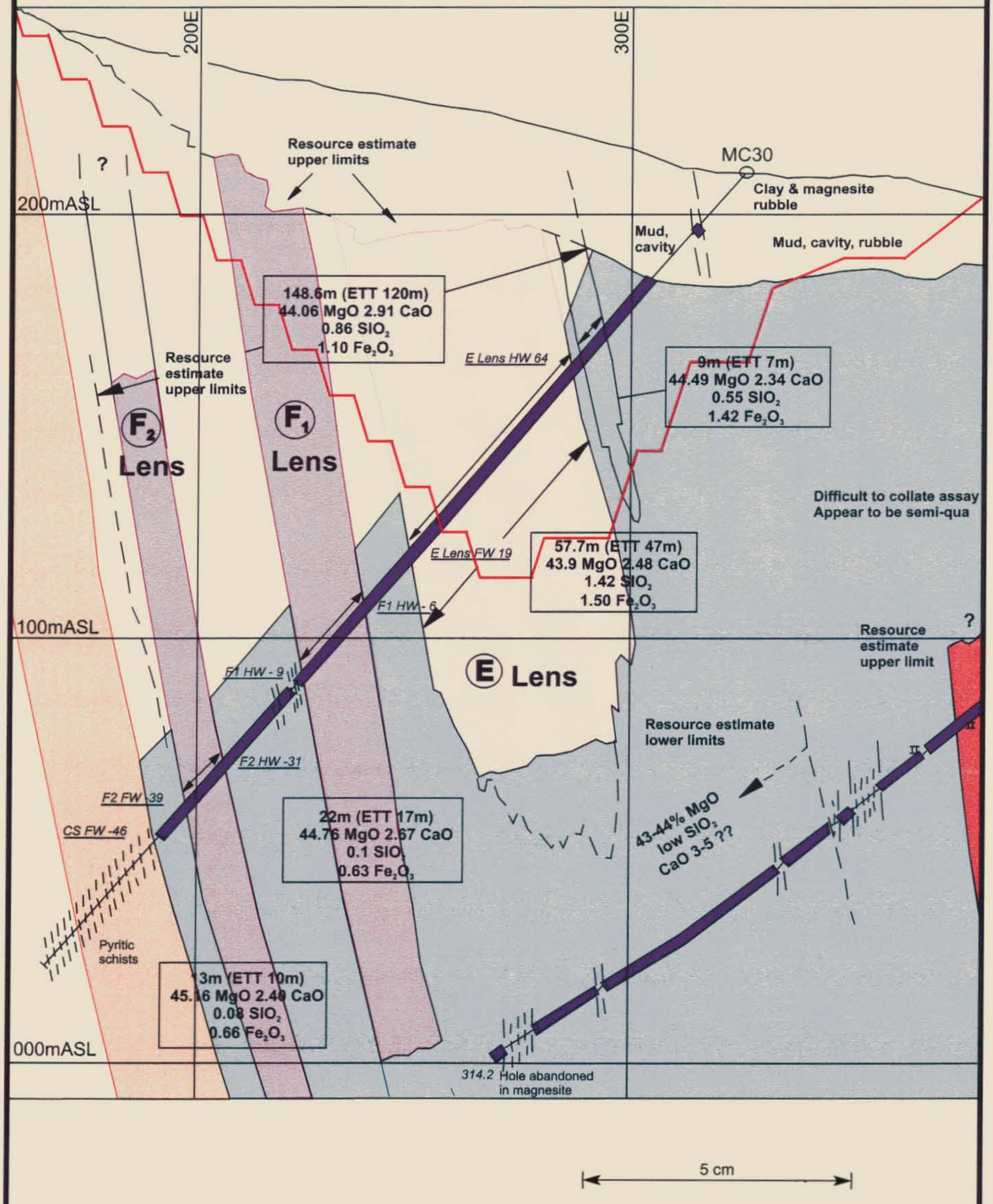

Courtesy Newnham Exploration and Mining Services

CLIENT *GOLDEN TRIANGLE RESOURCES* *Main Creek Magnasite Prospect*	Cross Section Through E Lens on MC 1,30,34 Showing Possible limits of an Open Pit to 15m ASL
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007 · SCALE 1:1250 · FILE Fig 4a.CDR · DATE 29/7/98 · FIG. 4a

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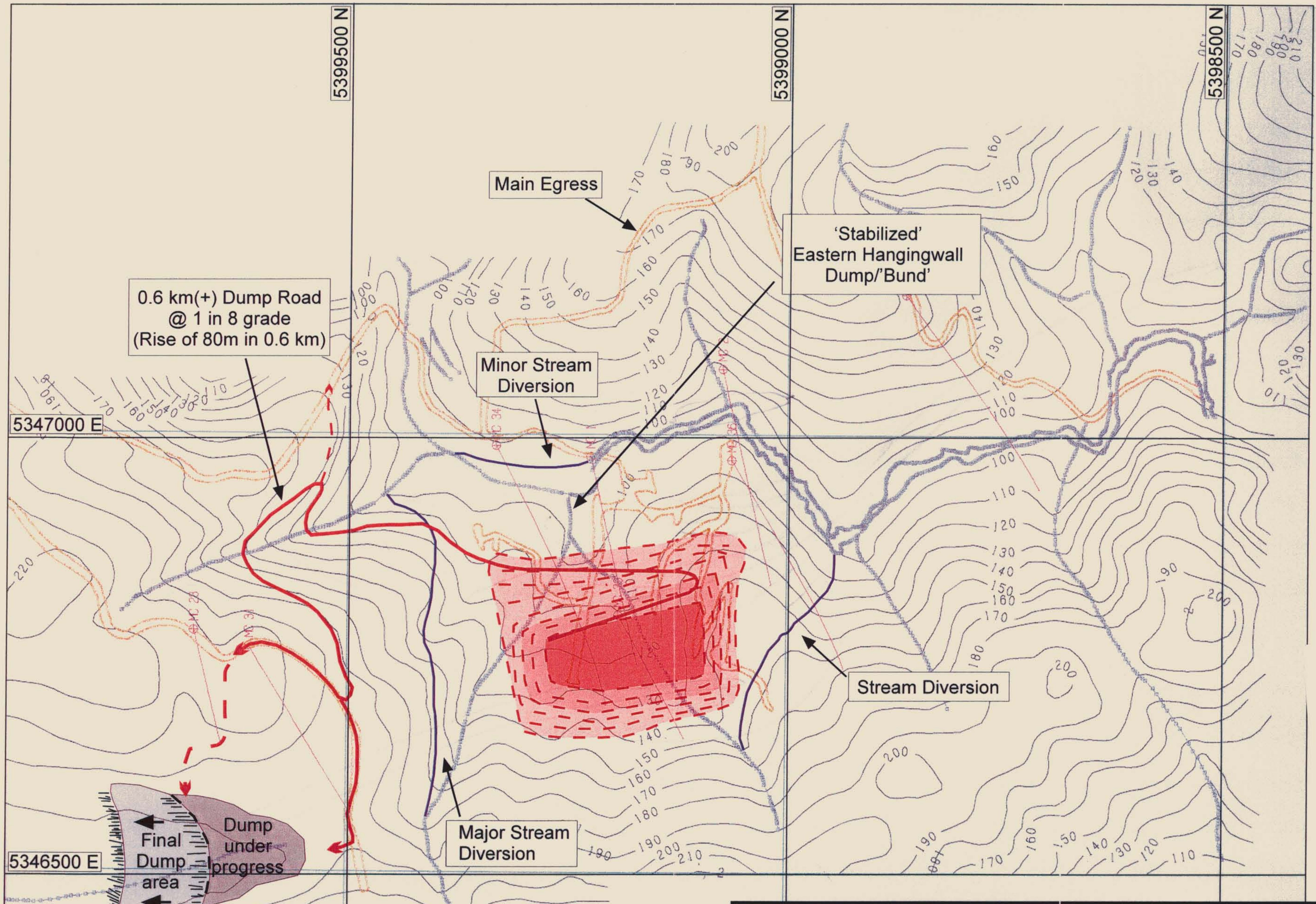


CLIENT GOLDEN TRIANGLE RESOURCES NL Main Creek Magnesite Prospect	Open Pit Option - Main Creek MC 1,30,34		
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007	SCALE NTS	FIG. 4b
	FILE Fig4b.CDR	DATE 30/7/98	

Due to the scale and short duration of the open pit option the use of a Mining Contractor, rather than a "Do-It-Yourself" approach is logical. Preliminary cost estimates are therefore for a Contract Mining basis.

(i)	Establishment			
	- Mobilization			$500,000
	- Perimeter Drainage Control			$150,000
	- Waste Dump-Site Preparation			$250,000
	- Ridge-Site Dump Access road (±1.2 km)			$1,200,000
			Sub-Total	**$2.10M**
(ii)	Mining			
	- Excavate & Stack Weathered O/B	1.15M BCM @ $2.0/m^3		= $ 1.3M
	- EXCESS Haulage Provision	1M BCM @ $2.0/m^3		= $ 2.9M
	- Drill & Blast Carbonate Waste	0.3 M BCM @ $2.50/m^3		= $ 0.7M
	- Drill & Blast Magnesite Ore	0.6 M BCM @ $3.00/m^3		= $ 1.7M
	- Load & Haul Carbonate Waste to Dump	0.3M BCM @ $1.50/m^3		= $ 0.4M
	- Load & Haul Magnesite Ore to stockpile	@ $2.00/m^3		= $ 1.1M
			Sub Total	**$ 9.1M**
(iii)	Dump Maintenance			
	- Securing of Eastern Wall	@ $1,000/m^2		= $ 0.4M
	Final Dump Rehabilitation 25 ha			= $0.5M
			Sub Total	**$ 0.9M**
	Contractors Other Costs (5% of Items (ii), (iii), above)			$ 0.6M
			Contractors Total	**$ 12.7M**
Owner's Costs – Contract Supervision				
	- Survey/Grade Control, Geology, etc.	$500,000/yr		$ 2.5M
Total Open Pit Mining Costs for 1.7Mt's of ore				**= $ 15.2M**
				or $8.94/t of Ore
			+10% Contingency	**$ 0.90/t, say**
			TOTAL	**$ 9.84/t**

3.4 Comments on Open Pit Mining Option

At around $10.00/t of ore produced, open pit mining would be potentially the lowest cost option for mining.

However certain aspects of an open pit option have inherent risk and are difficult to quantify in monetary terms, at this early stage.

Some are:-

- the practicality of the identified site for a waste (overburden) dump,
- the rehabilitation of the waste dump,
- the rehabilitation of the open pit,
- the cost and practicality of "minor" stream diversions,
- the difficulty of selective mining in an open pit operation of this type with no visual control basis,
- the possibility of rupturing the rock and allowing water to bleed into the footwall pyritic schsits and thereby allowing acid water to collect in the open pit.

Costs for site access clearance of pit and dump areas, and any post-mining rehabilitation work are difficult to assess at this stage.

Without recourse to mining the footwall schists as overburden, to allow extension of pit depth, the open pit option must be regarded as a short term option, with an underground mine required within 5 years of project start up.

An open pit operation of the type and extent described would require a surface area of approx. 6 hectares, plus a similar area for waste dumps. Together with ore stockpiles area and associated works area, stream diversions, etc, a total area of disturbance of 15 to 20ha could be anticipated.

4.0 UNDERGROUND

4.1 Prospective Areas

In contrast with the open pit options, the full strike lengths of the deposit(s) can be realistically considered as a basis for some form of underground mining.

However, because of grade considerations, selection reverts to the same two areas as the most attractive choices, both areas showing the desired substantial thickness of ore, and high grade (i.e. low SiO_2 content). For the underground mining alternative, though, the depth of cover (40-50 metres, to an estimated top of the ore zone for the southern (MC32) block, vs 20-30 metre cover for the northern (MC1, 30 34) block) is not so important.

Further, the apparent lack of a significant depth of weathered material to bedrock is a real bonus, as any portal /shaft collar would more quickly reach solid rock, with lower ground support costs. (Refer. Figs. 5a, 5b)

Additionally, the MC32 site, on C Lens offers the largest individual target, with a major thickness of low SiO_2 (<1.26% SiO_2) material in the 60m wide hangingwall portion of C Lens. Within that span it is possible that even lower SiO_2 sections will occur, making the production of direct shipping ore a possibility, even if smelter requirements are for as low as 1.0% SiO_2. Underground mining studies have therefore been based on this site.

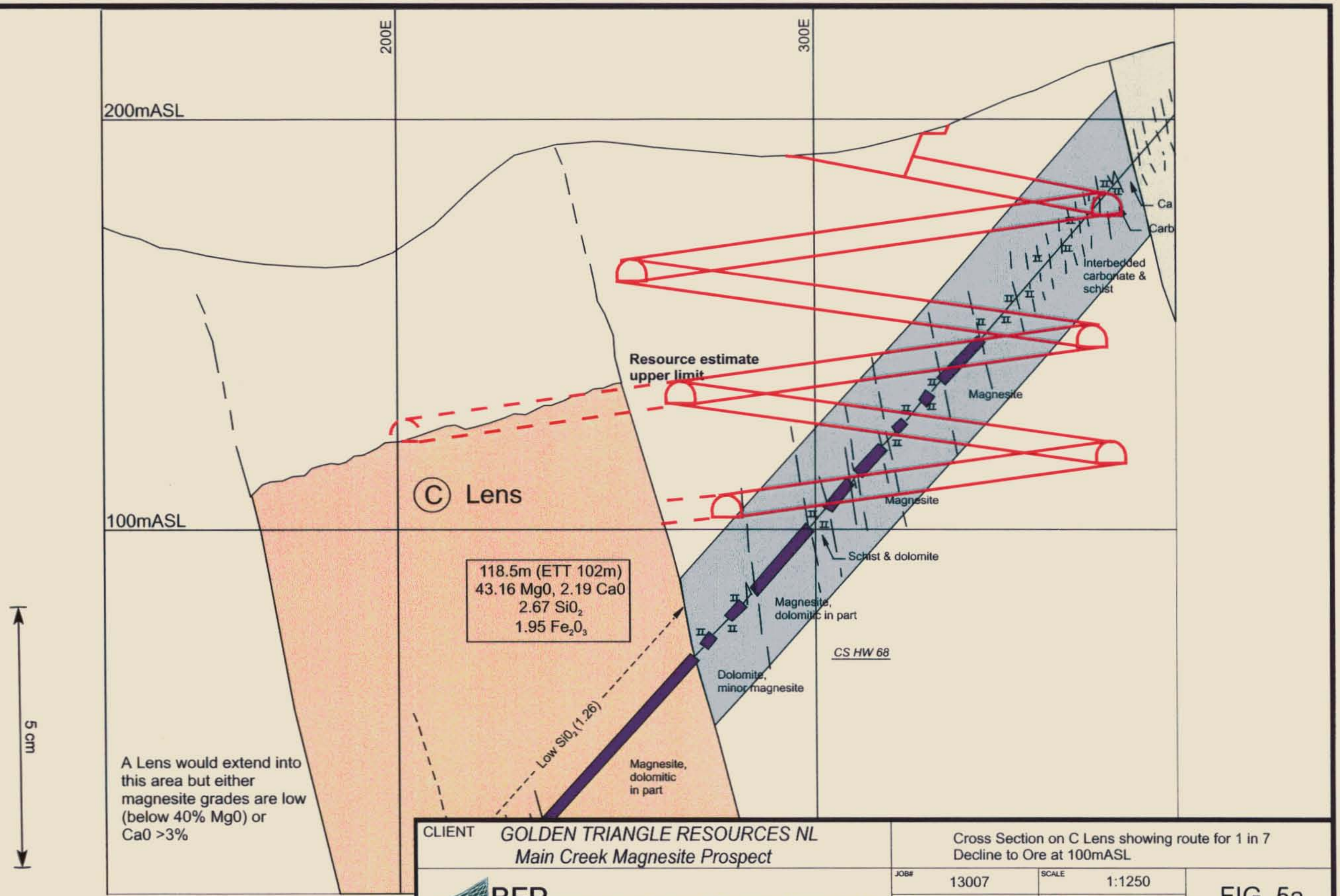
4.2 Underground Mining Options

The Main (and Bowry) Creek Magnesite deposits are (on the evidence so far available) both massive, competent and steep dipping. As such, a range of underground methods could be applied with varying costs, selectively, dilution, and (possible) long term surface stability and subsidence implications.

Given the apparent extent of the resource base (approx. 47Mt, inferred), relative to the intended project scale and duration, the percentage recovery or extraction factor is not expected to be a primary consideration.

The primary methods considered at this stage (refer Figs. 6 to 8c) are;

- Room-and-Pillar, Cut-and-Fill stoping (a hybrid of these two, Post-Pillar Cut and Fill, is also depicted).
- Open Stoping – conventional longhole drilling (Fig. 7).
- Open Stoping as longhole benching (Figs. 8a, 8b & 8c).



CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Prospect*	Cross Section on C Lens showing route for 1 in 7 Decline to Ore at 100mASL		
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007 / FILE Fig5a.CDR	SCALE 1:1250 / DATE 30/7/98	FIG. 5a

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5 cm

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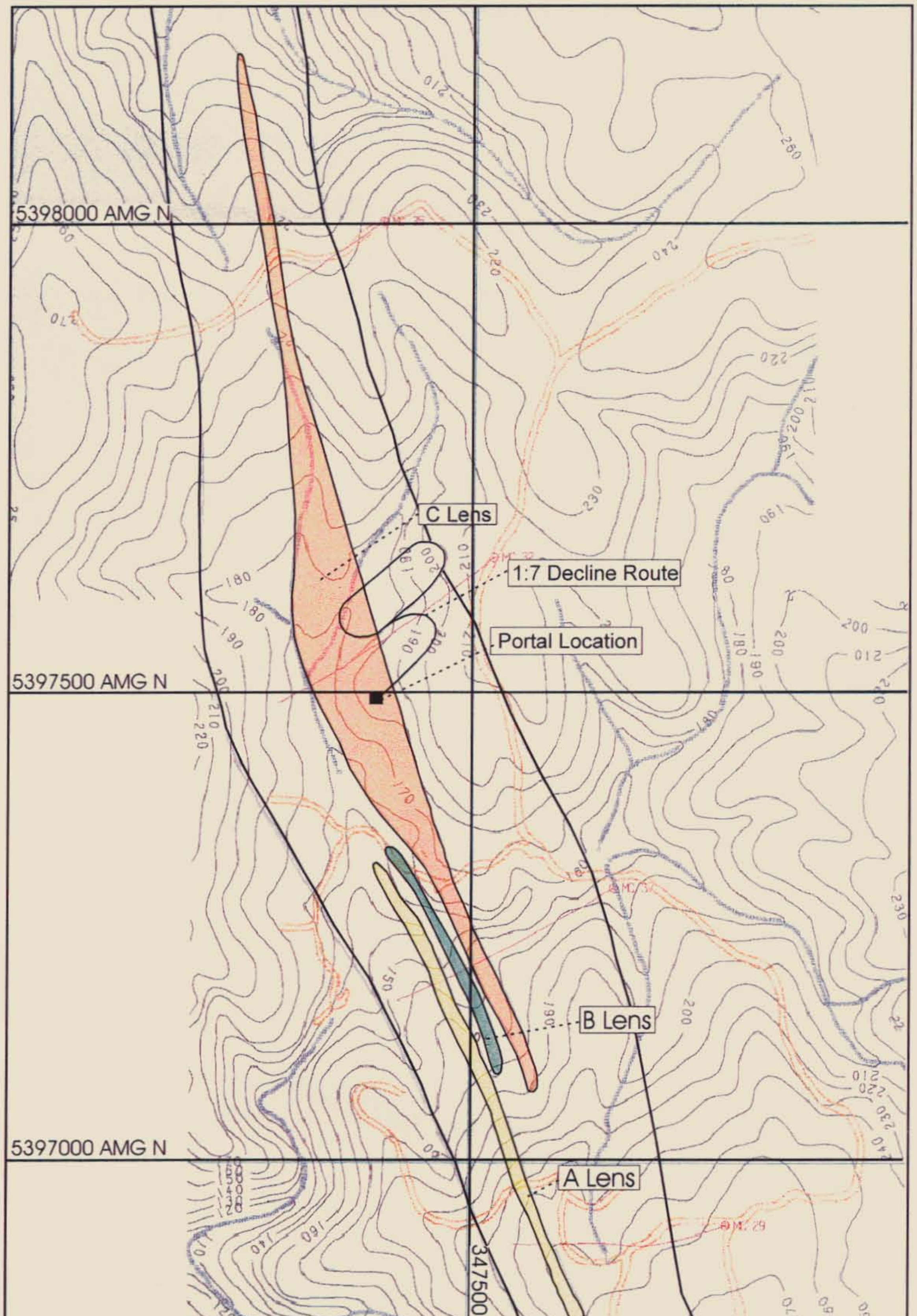
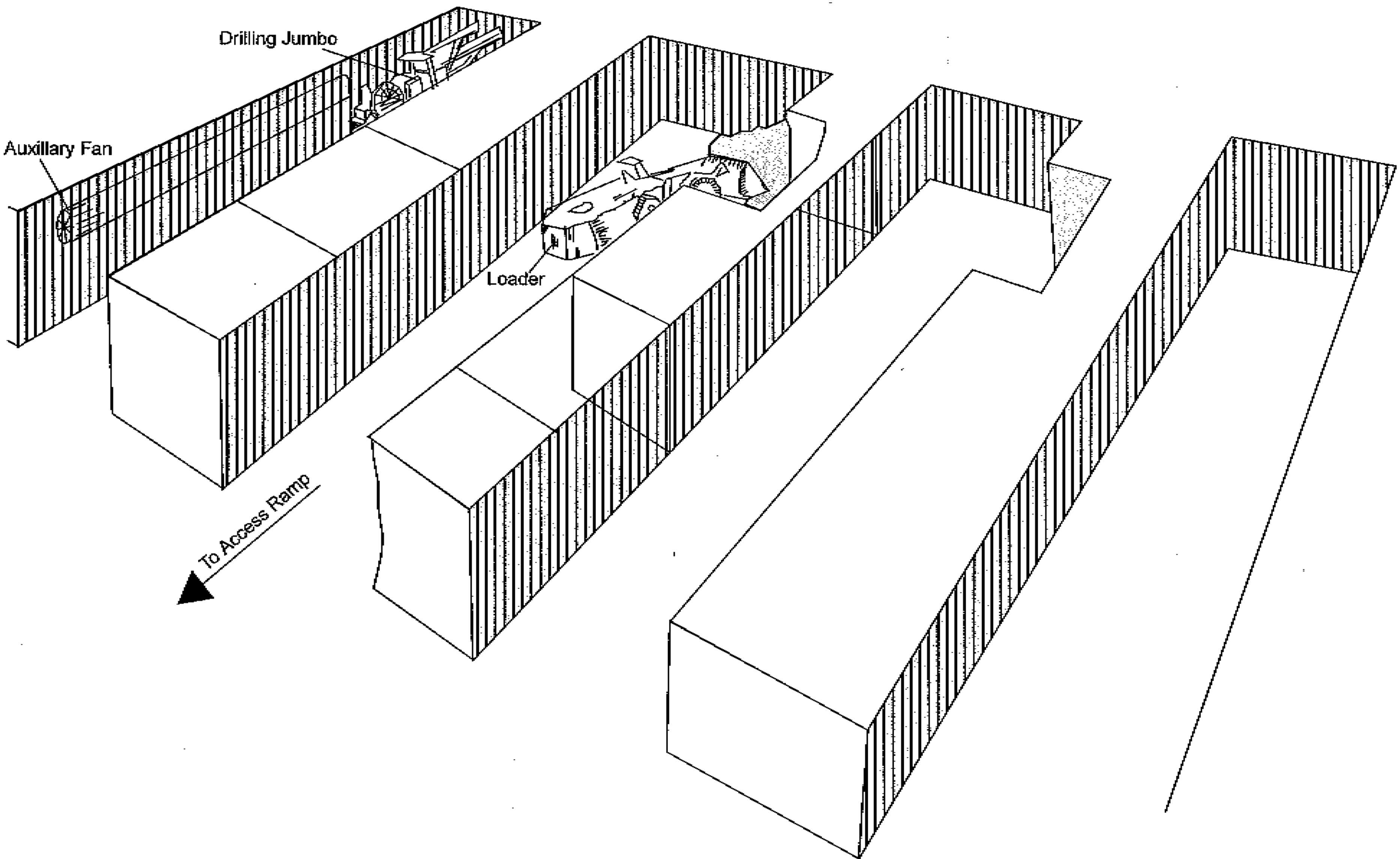


CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Deposit*	Portal Decline Location within Carbonate Sequence		
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007	SCALE 1:1250	FIG. 5b
	FILE Fig5.cdr	DATE 30/7/98	



CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Prospect*	Room and Pillar/Cut and Fill Stoping -Strike Drivage Stage		
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007	SCALE N.T.S.	FIG. 6
	FILE Fig-6.CDR	DATE 30/7/98	

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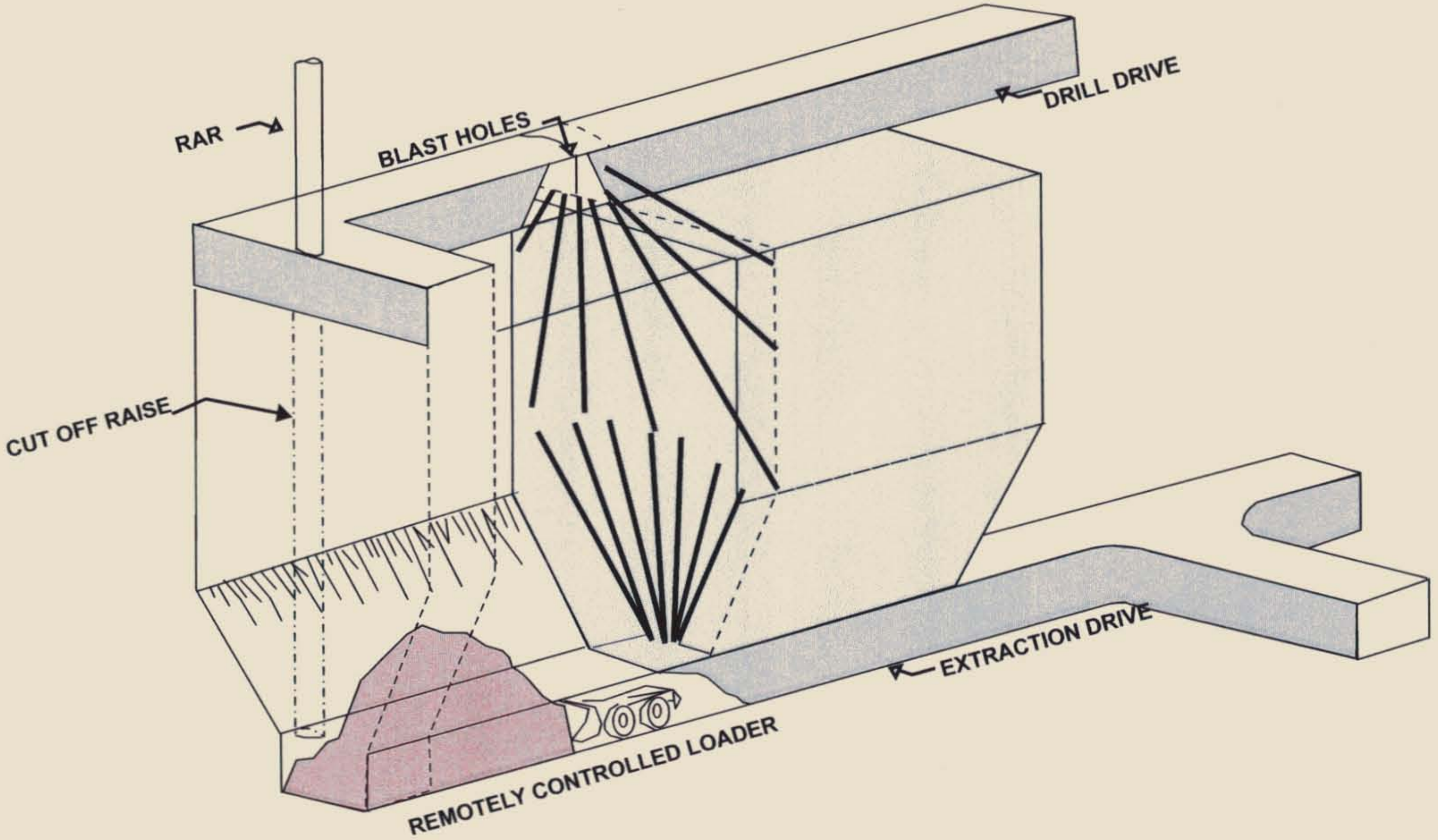

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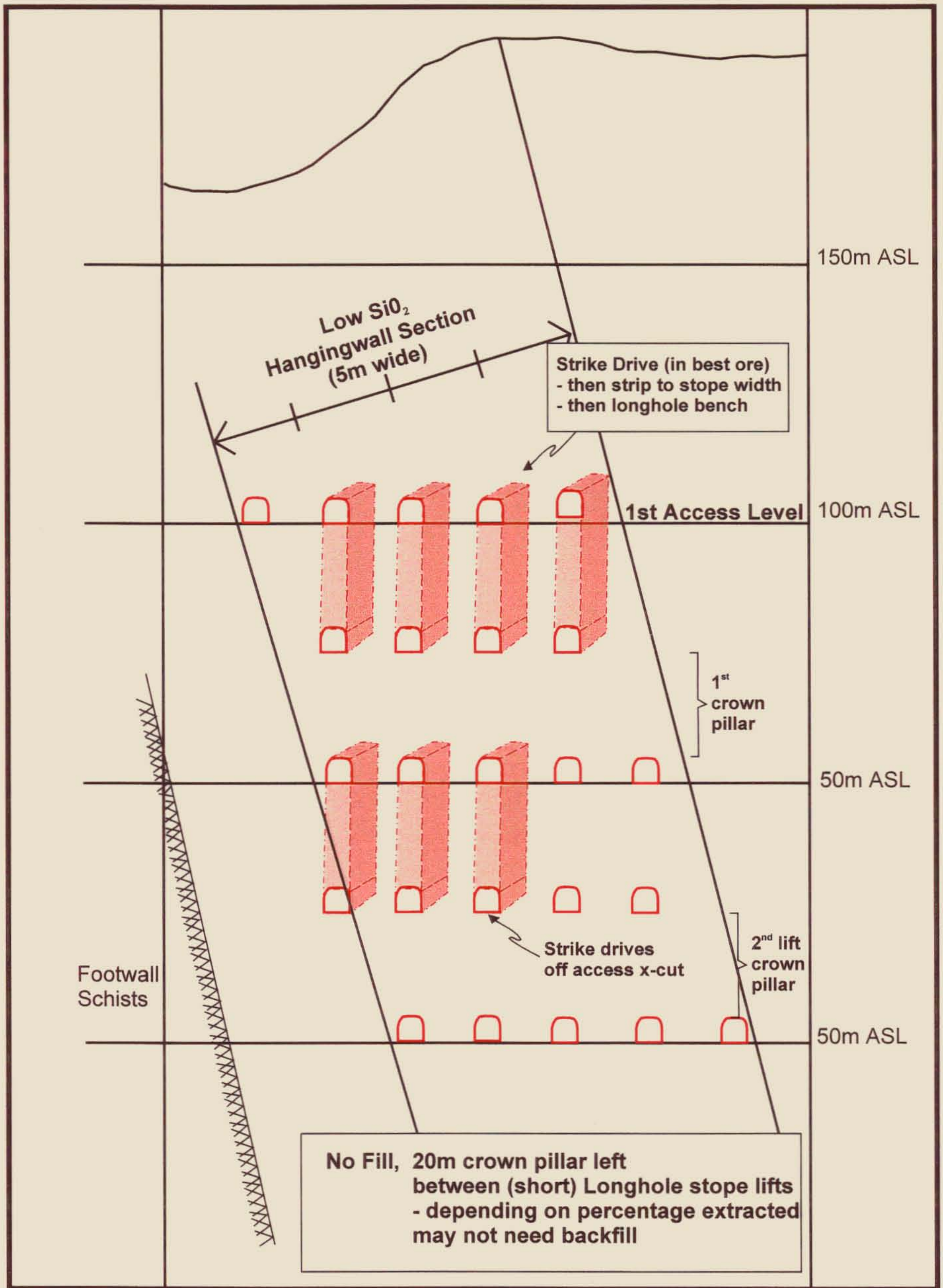
CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Prospect*	Long Hole Open Stoping Method Remote Controlled Loadout		
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007	SCALE N.T.S.	FIG.7
	FILE Fig7.CDR	DATE 30/7/98	



CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Prospect*	Uphole Benching Alternative benching/Open stoping configurations as applied to C Lens			
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007	SCALE 1:1000	FIG. 8a	
	FILE Fig8a.CDR	DATE 30/7/98		

5 cm

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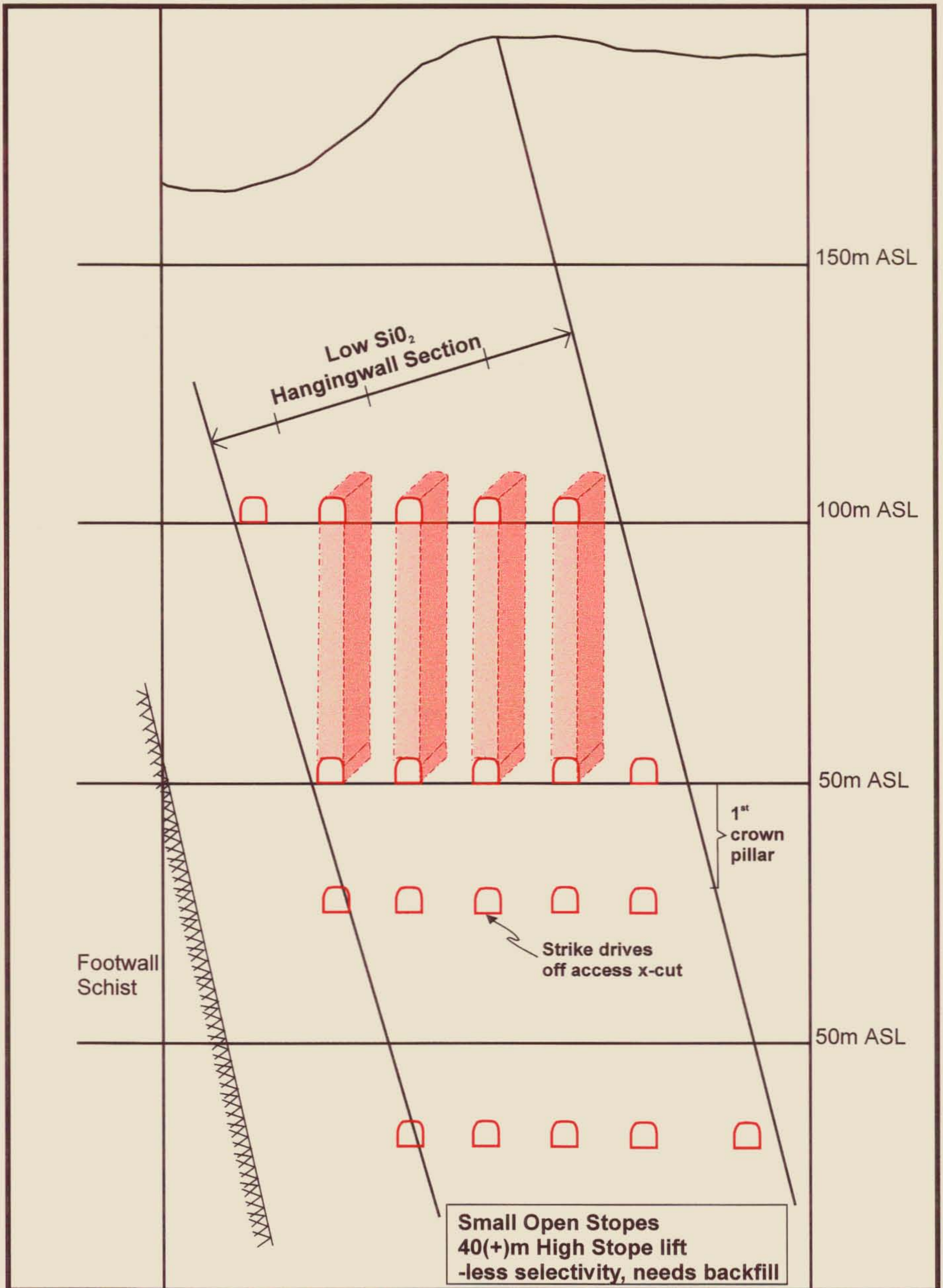
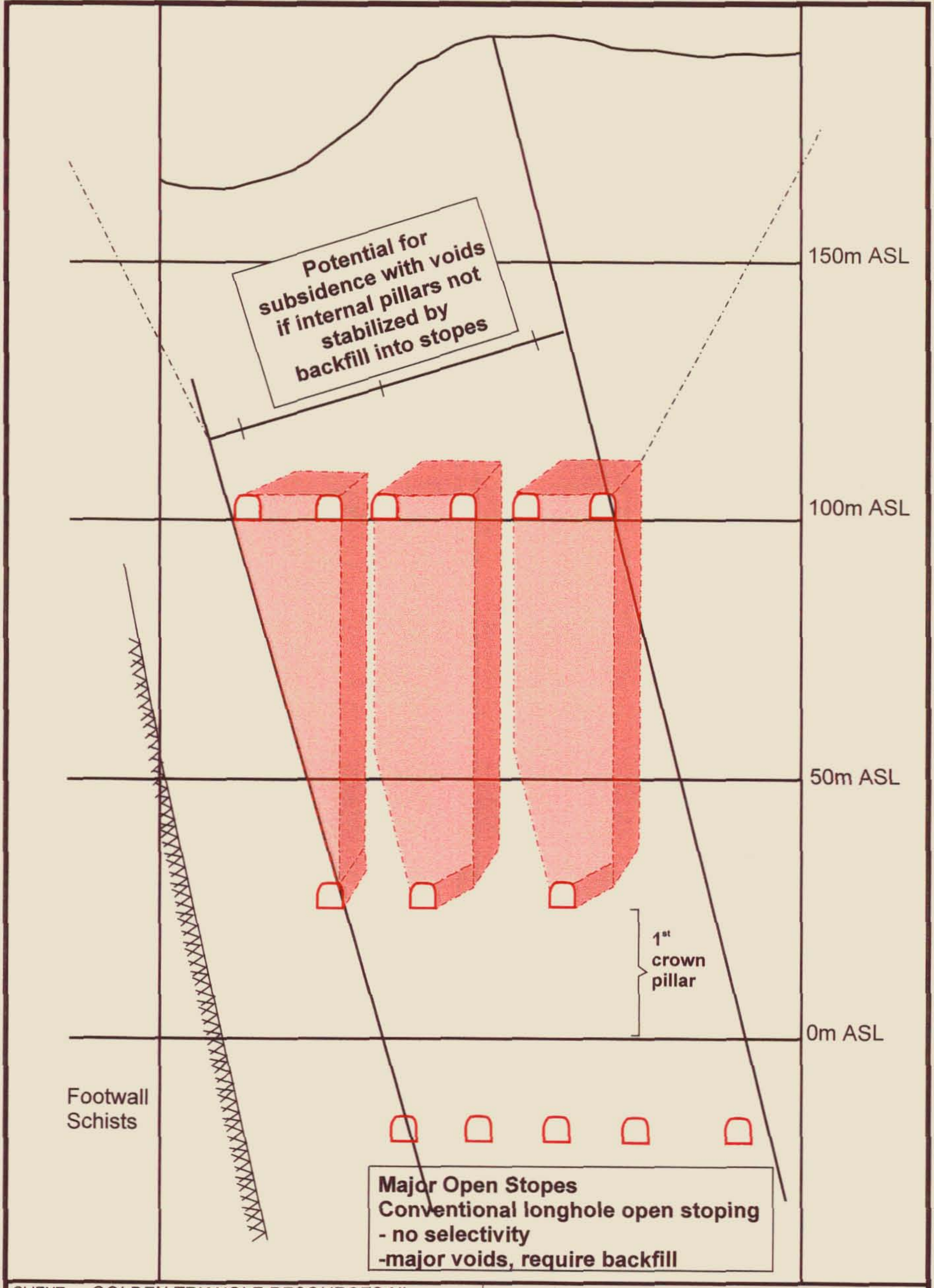


CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Prospect*		Narrow, Longhole/Open Stoping As applied to C Lens			
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007	SCALE 1:1000			FIG. 8b
	FILE Fig8b.CDR	DATE 30/7/98			

5 cm



CLIENT *GOLDEN TRIANGLE RESOURCES NL* *Main Creek Magnesite Prospect*	Conventional Longhole Open Stoping			
BFP Geotechnical, Mining & Geological Consultants	JOB# 13007	SCALE 1:1000		FIG. 8c
	FILE Fig8c.CDR	DATE 30/7/98		

This last noted alternative of conventional longhole stoping could be applied in a range of configurations, ranging from relatively small (narrow) blocks, through to extremely large blocks (say of up to 60m width x's 50m strike length xs 100m high).

For small scale cut and fill stopes, modest dilution and high selectivity could generally be achieved, whilst for bulk stopes, selectivity would be poor and dilution possibly high. There would also be increased long term risk of back failure for large open stopes and eventual surface subsidence, unless the open stopes were backfilled.

Costs for each method would vary-according to such aspects as:
- Drill and blast costs,
- Ground support needs,
- Stope block development requirements,
- Backfill needs.

As a general guide to their possible applications the various methods are presented in Table 4.1, below in terms of their ratings for various attributes (a scale of I to V is used, with I – representing best, through to V – representing worst.

TABLE 4.1
MINING METHODS- OPTIONS MATRIX

Parameter / Method	Room Pillar	Long Cut-&-Fill	Small Open Stoping Uphole Benching	Large Open Stopes
Selectivity of Method	I	II	II-III	V
Pre-Production Development Needs	I	II	II	IV
Mining costs	III	IV	II	I
Flexibility in Mining Plan	I	II	III	IV
Backfill Requirement	I	III	I	IV
Orebody Utilization	IV	III	III	II
Dilution Levels	I	III	II	IV

4.3 Design Constraints

General access to the project site, coupled with the difficult topography and leachate potential of the footwall schists, will dictate that underground mining access will be from the hangingwall side.

The projected scale of operation, at approximately 400,000 t/a, and shallow (long-strike) attitude and extent of the ore bodies within the carbonate sequence, suggests a decline based operation rather than shaft hoisting system. In any detailed feasibility study, such an option should be costed as a matter of record and due diligence.

On evidence of the site visit and examination/testing of carbonate sequence core, the orebodies, and near-hangingwall rocks are considered to be generally highly competent and suitable for substantial stope openings with minimal support.

There is, as yet, no definitive evidence available as to the distribution of the high grade (low SiO_2) ore within the lenses, however, the present interpretation is for continuity of such (high grade) along strike.

There is little experience in excavating in the type of material that comprises this deposit. Whilst the magnesite has few fractures, a high strength and is hard, the drilling and blasting of this material would be an activity that has little precedent. This is not to say that it will be difficult, however an important part of any future work would be an early examination of how well the magnesite responds to drilling and blasting.

CUT AND FILL STOPING, ALTERNATIVES

The implementation of this method, from start-up is shown in Fig. 6. Variations of open stoping is shown in Fig. 8(a) as the benching method and b) and c) as conventional open stoping methods which can be developed from the initial cut and fill layout, and are generally less expensive than cut and fill.

As a broad comparison, approximate mining costs for a production rate of 400,000 tonnes per annum follows:-

STOPING COSTS

• Cut & Fill	$20 /t
• Benching	$15 /t
• Open Stoping	$10 /t

Early development would cross cut the orebody (C Lens) at several levels to pick up where the preferred horizons occur, and then drive along strike, with regular assays, presumably with short hole (sludge) sampling at regular intervals.
Mining would retreat out from the parallel "in ore" development drive, selectively taking the panels of appropriate grade. All data would be collected for geostatistical analysis.

This approach is akin to a multi-level room-and-pillar mining technique, and could be developed or extended in the production phase to an overhead or sub-floor benching technique. (Refer Figs. 6 and 7 or post-pillar mining Fig. 8).

Backfill will be required-to provide a platform in the cut and fill scenario. Initially backfill will attainable from development however, if cut and fill mining is to continue a separate supply of backfill will be needed.

- One option would be the establishment of a small surface quarry as a supply source.
- Alternatively, if an open pit/surface quarry is excluded from consideration, additional "stoping" areas in the hangingwall, but away from the preferred mining zones could be opened up as needed to provide backfill.

Since broken rock expands to take up approx. 50% more volume than the "insitu" block of equal tonnage, for a mine out put of approx. 400,000 tpa of ore, only some 200,000 tpa of separately produced material would be necessary for backfill.

INITIAL FILL REQUIREMENT

Of the 200,00t initially required, 50,000t can be reclaimed from the initial decline development and the further 150, 000t would need to be derived from other sources.

ONGOING CUT AND FILL - FILL REQUIREMENT

Up to 30,000t per annum would come from ongoing decline development, the further 170,000t would need to be sourced elsewhere.

Provided such "backfill" stoping blocks were reasonably well spaced out, and were of limited individual size and away from the "ore mining" areas, they would not in turn require backfilling.

If the grade of the orebody proves consistent them it would be possible to convert the cut and fill stope to a bench stope or eventually to an open stope. As has been seen in the previous table these mining methods provide progressively cheaper mining at the cost of selectivity and set up time.

4.4 Underground Mining Costs for a Cut and Fill Mining Method

The Principal or a Contractor could undertake underground mining. Contractors generally set a fairly high profit margin to cover risks and capital. As a guide most costs for mining could be increased by 20% if a contractor was to perform the work.

The mining method examined is the cut and fill method which would be the most expensive and most selective.

4.4.1 Costs – Pre Production

a) If undertaking the work itself, GTR would have to buy or make alternative arrangement to acquire suitable mining equipment, possibly as follows;
(2nd Hand estimates)

1 grader & 1 dozer	$150,000
1 x drilling jumbo	$250,000
2 x 5cu m LHD's (load, haul, dumps)	$300,000
2 x 35t capacity underground trucks	$300,000
Other support gear (service vehicles, fans, pump sites)	$500,000
Sub-Total	**$1,500,000**

If new, such a listing could be of the order of $4-$5 million.

(i) Site Establishment

• Mobilisation to site (Equipment, manpower, etc.	$250,000
• General surface site preparation works (Workshop, Offices, Change house, etc)	$250,000
• Portal Preparation	Allow $100,000
• Other (Ventilation Shaft – R/B Collar) etc.	$ 50,000

(ii) Underground Decline Driveage, @ 1 in 7, 5m x 5m Section
Allow (700 +100m) xs $1800/M = $1,440,000

(iii) Other Underground Ventilation Rises/Egress

1 x 90m x 1.8m^2 dia. @ $1,500/M	
1 x 90m x 3.0m^2 dia. @ $2,000/M	$ 315,000
plus Ventilation Fans/Ladderways, etc.	$ 40,000
	$ 355,000

(iv) Initial Lateral Development (in ore)
- Allow 4 strike drives/level – 5 x 5 Section
(av.) 200m each = 800m xs $1,800 say = $1,440,000

(v) allow 10% "on" for extras
- ground support, roads, draws, pumping, Ventilation etc.= $ 323,000

$3,559,000

<u>Add</u> : GTR's Owner-Supervision Costs

Mine Staffing - say 3 staff @ $250,000/av xs 1.5yrs +50%
"On Cost" Supplies, etc. = $ 562,500

<u>Plus</u> : Geological Control/Sampling @ ½ of that $ 281,250

Project Total - Before First Production 1 = $4,402,250

Or (say) $5.0 Million

Production Costs - (CUT & FILL Stoping)

From there, on, <u>annual</u> (GTR) owner-operator mining costs would be of the order of:

- Underground Mining (i.e. Stoping) costs $10.00/t
- Backfill Placement

If ex. Open Pit or Quarry $1.67/t of ore
or if ex. Underground Source $6.70/t or ore
i.e. $11.67 to $16.70/t of ore

- Decline Drivage Extension 400m/yr @ $1,500/m = $1.50/t of ore

- Add Miscellaneous Mine Operating Costs @ 15% of prime costs (ground support, roads, vent. Pumping, etc.) = $2.00/t of ore

Total (Cheaper Backfill Operation) say = $15.17/tonne

- Supervision, Geo & Management costs $2.90/t

Total GTR Mining Costs = $18.07/tonne of ore
Or, $23.90/tonne
if underground sourced backfill has to be provided.

Note: these estimates do not make any specific provision for geological block control drilling.

The preceding are cost estimates for a conceptual study, only, and any further progression of the project would require additional analysis. As a preliminary estimate, it is suggested that an underground operation of the type and scale envisaged would require only a surface area disturbance of 1-2 ha, plus that for a small quarry for surface supply of backfill, (say 2ha) and up to 1ha (say) for on-site stockpiling) blenders of ore if needed i.e. a maximum of 5ha say.

APPENDIX 1

NOTIONAL PRODUCTION PLAN

Golden Triangle Resources N.L.

Planning Estimates for Main Creek Magnesite Project, NW Tasmania

	Year: -2	-1	1	2	3	4	5	6 - 10	11 - 15	16-20	Totals/Av.
1.0 Production Plan											
Total Required Ore Supply - 20 Years:		0	300,000	400,000	400,000	400,000	400,000	2,000,000	2,000,000	2,000,000	7,900,000
1.1 Open Pit Ore:											
Ore: [m. tonnes]	-	0	300,000	400,000	400,000	350,000	250,000				1,700,000
[b.cu.m]			*100,000*	*133,333*	*133,333*	*116,667*	*83,333*				*566,667*
O/Pit Waste:											
Weathered *[b.cu.m]*		*500,000*	*300,000*	*200,000*	*100,000*	*50,000*	*0*				*1,150,000*
Fresh *[b.cu.m]*			*100,000*	*50,000*	*50,000*	*50,000*	*33,000*				*283,000*
Total O/B: *[b.cu.m]*	*[say]*	*500,000*	*400,000*	*250,000*	*150,000*	*100,000*	*33,000*				*1,433,000*
1.2 *Underground Ore:*											
Development Ore [m. tonnes]						50,000	25,000				*75,000*
[lin. metres]						556	278				
ng Ore [tonnes]							125,000	2,000,000	2,000,000	2,000,000	6,125,000
Total Underground Ore						50,000	150,000	2,000,000	2,000,000	2,000,000	6,200,000
Development Waste:											
Lateral Devt. *[lin. metres]*					*400*	*400*	*400*	*1,000*	*1,000*		*3,200*
Vert. Devt. *[lin. metres]*						*100*	*100*	*200*	*200*		*600*
Total Waste: *[b.cu.m]*					*12,000*	*12,500*	*12,500*	*31,000*	*31,000*		*99,000*
Total Ore Supply - 20 Years:		0	300,000	400,000	400,000	400,000	400,000	2,000,000	2,000,000	2,000,000	7,900,000

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APPENDIX 2

OPEN PIT MINING COSTS

Golden Triangle Resources N.L.

Planning Estimates for Main Creek Magnesite Project, NW Tasmania

Year	-2	-1	1	2	3	4	5	6-10	11-15	16-20	Totals/Av.	
Mining Costs:	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	
2.0 Open Pit, [By Contractor]												
2.1 Establishment:												
Mobilization		500,000									500,000	
Perimeter Drainage Control		150,000									150,000	
Waste Dump Site Preparation		*250,000*									250,000	
Ridge-Site Dump Access Road	[+/- 1.2 km]	1,200,000									1,200,000	
2.2 Mining	$Rate/m3											
Excavate & Stack Weathered O/b	2.00	1,000,000	600,000	400,000	200,000	100,000	0				2,300,002	
Excess Haulage Provision for Waste	2.00	1,000,000	800,000	500,000	300,000	200,000	66,000				2,866,002	
Drill & Blast Carbonate Waste	2.50	0	250,000	125,000	125,000	125,000	82,500				707,503	
Drill & Blast Magnesite Ore	3.00	0	300,000	400,000	400,000	350,000	250,000				1,700,003	
Load & Haul Waste to Dump	1.50	0	150,000	75,000	75,000	75,000	49,500				424,502	
Load & Haul Ore to S/pile	2.00	0	200,000	266,667	266,667	233,333	166,667				1,133,335	
											9,131,346	
2.3 Dump Maintenance												
Securing Of Eastern Wall- @ $1000/m3		200,000	200,000								400,000	
Final Rehabilitation [25 ha.]						*250,000*	*250,000*				500,000	
2.4 Contractor's "On Costs"												
@ 5% of items 2.1, 2.2, 2.3		215,000	125,000	88,333	68,333	66,667	43,233				606,567	
2.5 Owner's Costs - Contract Supervision												
-Survey/Grade Control, Geology,etc		250,000	500,000	500,000	500,000	500,000	250,000				2,500,000	
Total Open Pit Mining Costs:		4,765,000	3,125,000	2,355,000	1,935,000	1,900,000	1,157,900	0	0	0	15,237,900	
Add Contingency,[basis:	10%											
Say:		**5,241,500**	**3,437,500**	**2,590,500**	**2,128,500**	**2,090,000**	**1,273,690**	**0**	**0**	**0**	**16,761,690**	
										or,	***$9.86***	***/tonne of o***

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APPENDIX 3

UNDERGROUND MINING COSTS

a) Pre-Production

b) Operating Costs

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Golden Triangle Resources N.L.

Planning Estimates for Main Creek Magnesite Project, NW Tasmania

Year:	-2	-1	1	2	3	4	5	6 -10	11- 15	16-20	Totals/Av.
3.0 Underground, As Owner-Operator	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]	$'S[A]
3.1 Initial Mining Equipment											
Drilling Jumbo						250,000					250,000
Load, Haul, Dump Units [5cu.m] -2, Off						300,000					300,000
U/G trucks [35 tonne cap.] -2, off						300,000					300,000
U/G Grader, dozer						150,000					150,000
Misc. Support Gear						500,000					500,000
Sub-total:						1,500,000					1,500,000
3.2 Site Establishment											
Mobilization To Site [Equipment, Manpower, etc.]					250,000						250,000
General Surface Site Preparation Works					250,000						250,000
Portal Preparation					100,000						100,000
Other (Vent Shaft - R/B Collar, etc)					50,000						50,000
Sub-total:					650,000						650,000
3.3 Underground Decline Drivage, @ 1 in 7 [5m x 5m Section]											
Rate: $1,800 /metre					720,000	720,000	720,000	1,800,000	1,800,000	0	5,760,000
3.4 Other U/G Devt. - Vent Raises/Egress											
Rate: $2,000 /metre						200,000					200,000
Rate: $1,500 /metre							150,000	300,000	300,000	0	750,000
[Plus: Ventilation Fans, Ladderways,etc]						40,000					40,000
3.5 Other Lateral Devt. - In Ore											
Rate: $1,800 /metre					0	1,000,000	500,000	0	0	0	1,500,000
3.6 Extras- allow 10% "On" for 10%											
-ground support, roads, drains, pumping, ventilation, etc					72,000	196,000	137,000	210,000	210,000	0	825,000
3.7 GTR's Owner Supervision *10%*					187,500	375,000	375,000				937,500
-say 3 staff,@ $250,000 for 1.5 yrs, +5(pluss Geo. Control					93,750	187,500	187,500				468,750
Total -U/G Pre-production Costs:					**1,723,250**	**2,718,500**	**959,750**				**5,401,500**
4.0 U/G Operating Costs:											
4.1 U/G Mining (i.e. Stoping) Costs $10.00 */tonne of ore stoped*							1,250,000	20,000,000	20,000,000	20,000,000	61,250,000
4.2 Backfill Placement $1.67 */total tonne of ore*							250,500	3,340,000	3,340,000	3,340,000	10,354,000
4.3 Add Misc. Mine Operating Cos 15%											
-ground support, *roads, drains, pumping, ventilation, etc*							225,075	3,501,000	3,501,000	3,501,000	10,740,600
4.4 Supervision, Geo & Mining, Mgement											
$2.90 */total tonne of ore*							435,000	5,800,000	5,800,000	5,800,000	17,980,000
Total -U/G Operating Costs:							2,160,575	32,641,000	32,641,000	32,641,000	100,324,600

or, ***$16.18*** */tonne of o*

[excluding Decline ExtensionCos

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