

4. GEOLOGY AND MINERAL RESOURCES

4.1 Regional Geology

The magnesite deposits of the Arthur and Lyons Rivers area are located within the Arthur River lineament, a north north-westerly striking belt of metamorphosed pre-Cambrian (Proterozoic) rocks, which extend from Wynyard in the north to Granville Harbour on the west coast, a distance of some 105 kilometres (refer Figure 1).

The regional geological setting of the area covering RL8717 and RL8718 comprises a steeply-dipping (65 - 80 degrees) sequence of folded quartz schist, quartzite and phyllite with minor dolomite, overlying a magnesite and dolomite sequence up to 400 metres thick which in turn overlies pyritic siltstone, mudstone and quartzite with minor carbonate and amphibolite. Intrusive into the Proterozoic sequence is mafic dolerite / gabbro dykes and/or plugs of both Proterozoic and Jurassic age.

These structures are overlain by younger Permian-aged siltstones, mudstones, fine-grained sandstones and carbonaceous phyllite.

In the Lyons River area there is extensive cover of Tertiary basalt, both as flows and possibly plugs. In the Arthur River tenement area, with the exception of minor exposures in the watercourses, almost all the resource zone is concealed beneath a cover of Quaternary-aged alluvium, scree and residual soils (Figure 3).

4.2 Detailed Geology

The geology of the deposits is almost totally concealed beneath a 10-15 metres deep cover of recent sand, gravel and boulder sediments, scree and residual soils together with a dense cover of regrowth forest vegetation.

Outcrop is negligible, being confined to in-situ magnesite in watercourses draining the area, together with scree material commonly exposed adjacent to the main, formed gravel forestry maintenance track which skirts the southern side of the resource zone (Figure 4).

In addition to mapping carried out by earlier workers, the bulk of the sub-surface geological information has been obtained from drill core recovered during the three exploration programs carried out by CRAE in 1983, TasMag in 1997 and Indcor in 1998/1999.

Figure 3 – Regional Geology

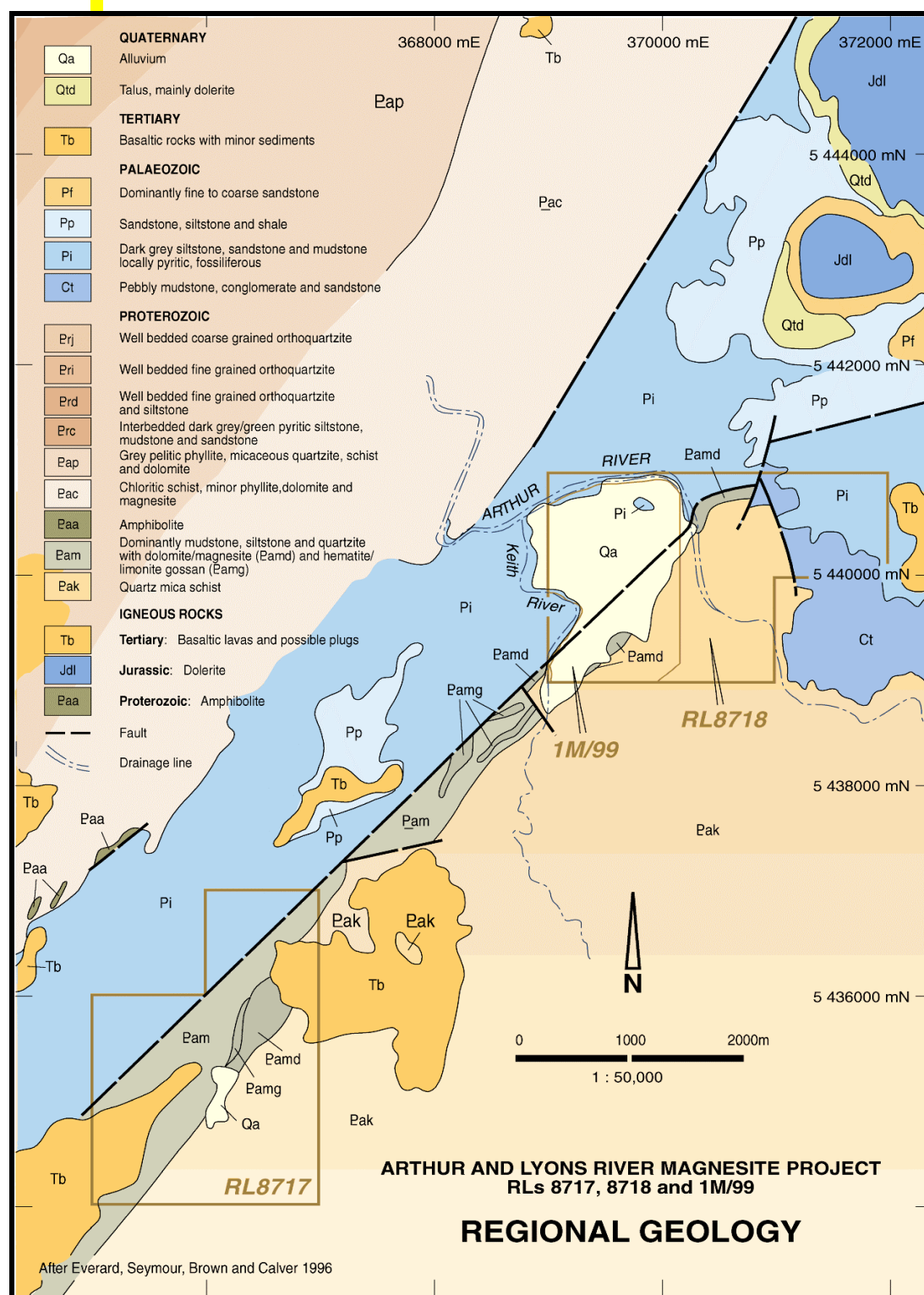
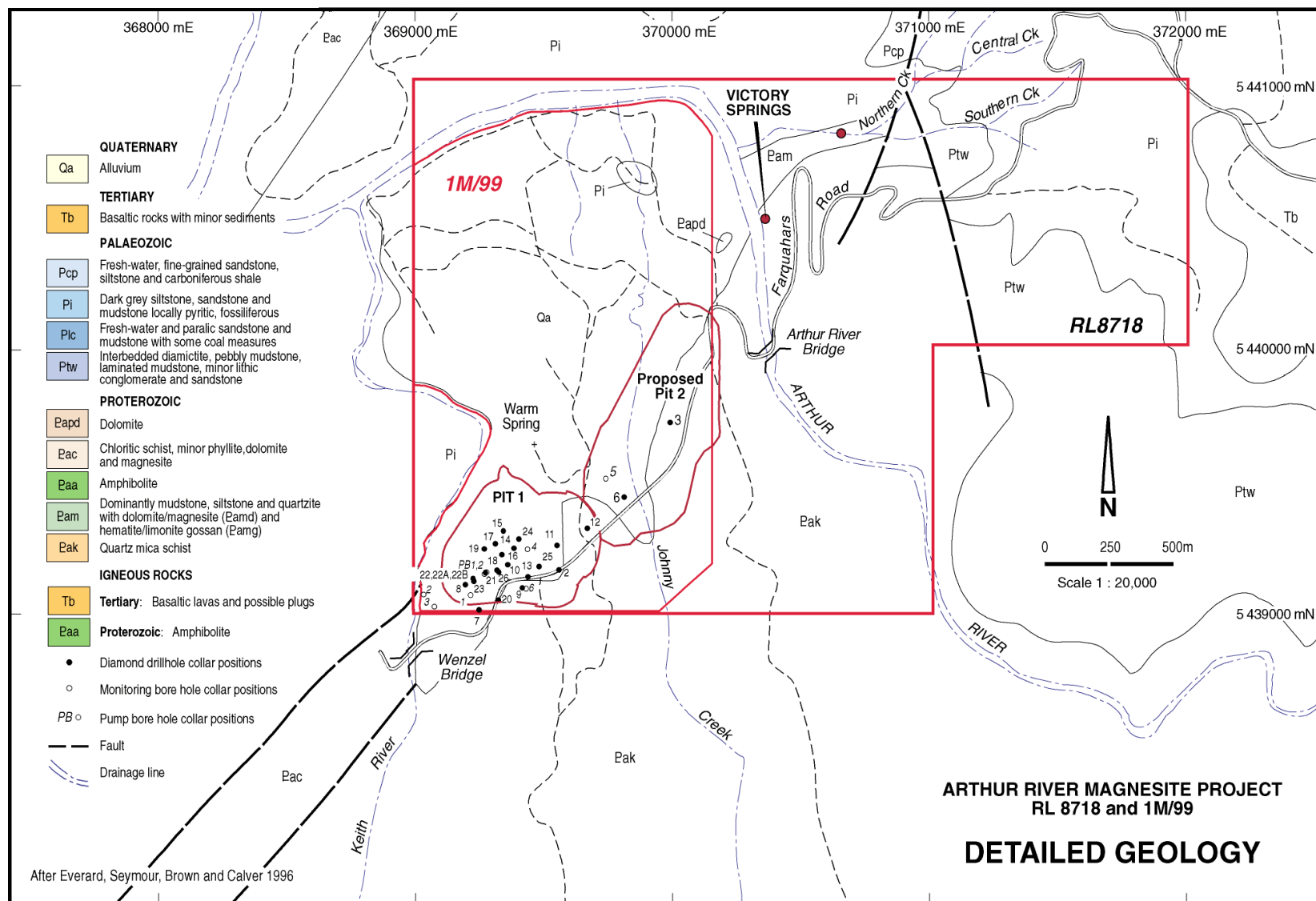


Figure 4 – TasMag Geology



The geological setting is summarised as follows:

- Recent/Quaternary
 - Grey sand/silt/boulder alluvium, red-brown limonitic clay at the north-eastern end of the resource zone (where dolerite scree indicates on the presence of underlying igneous intrusives), schistose scree and residual soils. The igneous intrusive(s) were encountered in several holes, but in the absence of sufficient drilling information, their geological setting is still unclear.
- Tertiary
 - Basaltic rocks with minor sediments.
- Paleozoic
 - Permian sandstones, siltstones and mudstones.
- Proterozoic
 - Hanging wall quartz schist.
 - Magnesite with minor dolomitic horizons.
 - Footwall pyritic schist, dolomite and siltstone which is commonly contorted and brecciated adjacent to the contact with the overlying magnesite.
- Proterozoic/Jurassic
 - Intrusive dolerite dykes or plugs.

4.3 Summary of Exploration Work

The Arthur River magnesite deposit was first discovered in 1925. In 1970, Mineral Holdings Australia Pty Ltd was granted a large exploration licence (EL43/70) over the area and during the next three years carried out exploration in association with a number of joint venture partners. Up to 1981, Mineral Holdings continued exploration, which resulted in the discovery of magnesite at Lyons River, about 4 kilometres along strike south of the Arthur River deposit.

Between 1982 and 1988 Mineral Holdings, in joint venture with CRAE, carried out exploration comprising geological mapping, geophysical gravity surveys, diamond drilling, metallurgical testing and feasibility and marketing studies with the view to assessing the deposit as a source of dead-burned magnesite, caustic calcined magnesite and direct shipping ore.

This work delineated a magnesite body at Arthur River occurring over a 3,500 metres strike length and a similar magnesite body at the Lyons River, some 2,000 metres long.

In 1997, TasMag entered into an option agreement to purchase the deposit from Mineral Holdings. Check and exploratory diamond drilling at Arthur River comprising seven holes totalling 1,254.3 metres confirmed the results of earlier workers. TasMag concluded that an Indicated Resource totalling some 29 million tonnes at an average grade of 42.8% MgO and 5.3% SiO₂ existed in the south-western corner of RL8718.

It has been estimated from past and recent drilling that the Arthur and Lyons Rivers magnesite deposits possibly contain as much as 180 million tonnes of high grade magnesite mineralisation to a vertical depth of 150 metres. Diamond drilling evidence indicated that the high grades encountered continue at depth to at least 400 metres and probably much deeper.

Whilst the footwall contact is fairly well understood, the dip of the hanging wall contact has still not been established because exploratory holes designed for this purpose had to be terminated when they reached the southern boundary of Indcor's Retention Licenses. It seems probable however, that the dip is somewhat flatter (at possibly 65-75°) than that assigned by earlier works. In each case Indcor drill holes AR16, AR23 and AR26, drilled in a southerly direction, failed to intersect hanging wall schists and were terminated in high-grade, crystalline magnesite.

4.3.1 CRAE Exploration

Between 1982 and 1984, CRAE put down 19 diamond drill holes at the Arthur and Lyons River deposits (now RL8718 and 8717) split as follows:

- Arthur River (now RL8718) 7 holes for 1,610 m
- Lyons River (now RL8717) 11 holes for 2,571m

Following these drill programs, CRAE concluded that the Arthur River (now RL8718) and Lyons River (now RL8717) each contained an estimated 30Mt of plus 40% magnesite.

Results of this drilling are summarised in Table 2 below.

Table 2(a): Summary of CRAE Drill Results – Arthur River

Hole Number	Intersection (m)	MgO (%)	CaO (%)	SiO ₂ (%)	Fe ₂ O ₃ (%)	LOI (%)
AR2	120	41.7	3.0	8.6	0.7	45.8
AR3	180	39.8	4.3	6.6	2.1	46.9
AR5	28	39.6	5.0	6.5	1.6	47.1
AR6	237	39.2	3.7	8.7	1.4	46.4
AR7	195	43.0	4.1	2.5	0.7	49.4

Table 2(b): Summary of CRAE Drill Results – Lyons River

Hole Number	Intersection (m)	MgO (%)	CaO (%)	SiO ₂ (%)	Fe ₂ O ₃ (%)	LOI (%)
LR1	224	42.8	2.8	4.1	1.7	48.5
LR2	284	41.1	3.6	7.1	0.7	47.6
LR3	15	43.0	4.1	2.5	0.7	49.4
LR5	218	39.1	3.8	10.4	0.7	45.4
LR6	102	40.1	2.9	9.9	1.5	45.4
LR7	131	40.5	2.1	10.8	0.5	45.9
LR9	92	39.3	2.9	10.7	1.5	45.4
LR10	31	38.5	5.0	9.3	2.3	44.6
LR11	9	39.3	2.9	10.7	1.5	45.4

Beneficiation testwork completed by CRAE determined that lower grade (+ 35% MgO) can be easily beneficiated by flotation to remove the dolomite (and most of the SiO₂) leaving high grade magnesite with low iron.

CRAE concluded in September 1984 that no further exploration work was required at these deposits as sufficient resource had been outlined and, as a result, future work would be of a development nature.

4.3.2 TasMag Exploration Prior to Indcor Involvement

In mid 1997, TasMag commenced a 1,250 metre diamond drill program on RL8718 which included twinning of CRAE's earlier holes AR2 and AR7, together with five additional exploratory holes to test the width, grade and continuity of the magnesite horizons. The check holes correlated very well and confirmed CRAE's previous work. Abbreviated results are summarised in Table 3 below.

Table 3: Summary of TasMag 1997 Drill Results

Hole Number	Intersection (m)	MgO (%)	SiO ₂ (%)
AR2 (check)	117	42.5	7.8
AR7 (check)	210	40.8	7.6
AR8	136	42.5	4.0
AR9	180	43.0	3.4
AR10	159	43.8	3.9

Based upon the results of this drilling, TasMag delineated an indicated resource (covering less than 400m of the total strike length) of 29Mt @ 42.8% MgO (refer section 4.5 for further details).

TasMag did not undertake any drilling within RL8717.

4.3.3 TasMag Exploration Subsequent to Indcor Involvement

Upon acquiring the Project, Indcor undertook a further drilling program within RL8718 designed to upgrade the Indicated Resource delineated by TasMag (in 1997) to Measured Resource status for inclusion in a bankable feasibility study.

From November 1998 to March 1999, a program of in-fill diamond drilling totalling 2,760 metres in fifteen holes was carried out by Indcor within RL8718.

The drilling was undertaken using non-coring Tricone bits to penetrate the overburden with coring commencing when hard rock was encountered. All holes were started in PQ size, using both 1.5 and 3.0 metre triple tube, split barrels. When poor ground was encountered, the holes were reduced to HQ and if absolutely necessary, to NQ. Only two holes were completed in NQ size, namely AR21 and AR23.

Skid-mounted Longyear 38, Longyear 44 and Mindrill 52 machines were used by the drilling contractors, Diamond Drilling Tasmania and Contract Diamond Drillers, both Tasmanian-based companies.

The drill core was collected in 1 metre long aluminium core trays and immediately photographed and then logged. Each day all core was removed from the site and delivered to Analabs Pty Ltd, Burnie. The core was then quartered using a diamond saw and one quarter prepared for analysis. Sample analyses were carried out by Analabs in Perth, Western Australia. Bulk samples comprising both drill core from 1983 (CRAE) and 1997 (TasMag) exploration and pulps from the 1998/1999 (Indcor) drilling were submitted for metallurgical test work.

The sample preparation and analytical methods used by Analabs are summarised below:

- Sample Preparation
 - Diamond drill core samples were cut using a diamond saw. The angle of cutting was clearly marked on the core prior to the samples being submitted to the laboratory.
 - The total sample is dried, to a core temperature of 110°C, jaw crushed, split and milled in a tungsten-carbide bowl to a nominal 90% passing 75µm. An analytical pulp of approximately 100g was achieved and the coarse residue retained for future reference.

- Analytical Methods

The sample was fused with 12/22 flux to form a glass fusion disc. This disc is presented to the XRF instrument for determination of the silicate rock elements. An LOI is performed at 1,000°C.

In the initial analyses, internal laboratory quality-control checks were made by performing repeats on 70 samples, as well as insertion of certified standards. No bias is evident from these results.

All holes were logged using standard Drill Log Sheets and nomenclature. Geological descriptions were simplified and coded, and entered into a geology table in the database. Cavities that had been recorded on the logs were also entered into the geology table.

The remaining core, sample duplicates and excess pulps are stored in a secure storage unit adjacent to the Analabs operation in Burnie, Tasmania.

Results of Indcor's drilling are summarised in Table 4 below.

Table 4: Summary of TasMag 1998/99 Drill Results

Hole Number	Intersection (m)	MgO (%)	CaO (%)	SiO ₂ (%)	Fe ₂ O ₃ (%)
R13	104	44.1	3.3	1.9	0.5
AR14	72	43.6	2.5	4.7	0.4
AR15	87	33.5	3.5	12.8	5.6
AR16	261	41.9	2.2	7.6	0.8
AR17	154	37.7	6.2	6.1	2.7
AR18	228	42.4	2.1	not reported	2.8
AR19	74	38.8	3.5	6.6	4.2
AR20	198	41.8	2.1	6.0	2.1
AR21	188	42.2	2.5	4.6	1.7
AR22	Abandoned				
AR22A/B	190	41.8	3.7	3.6	1.7
AR23	50	41.7	1.1	6.9	3.3
AR24	Abandoned				
AR25	Abandoned				
AR26	80	42.8	1.1	7.1	0.9

Indcor has not undertaken any drilling within RL8717.

Figure 5 – Drill Hole AR18: 228m @ 42.4% MgO



4.4 Mine Geological Model

4.4.1 Introduction

Drill hole data has been converted into digital format to allow resource modelling and estimation for the Arthur River Magnesite deposit (RL8718).

Drilling data was provided in the form of hand-written geological logs and digital text files for the drill core assays, whilst drill hole collar surveys were keyed in from data from contract surveyors.

Drill hole sections were established on a local grid for simpler computer processing. Resource estimates were completed using Surpac2000 software, and comprise both a classical sectional resource model and a block model.

For the block model, wireframes were constructed and used to constrain interpolation of block grades. The inverse distance weighting technique was used for grade interpolation.

4.4.2 Data Collection

4.4.2.1 Surveying

In 1999, both ground Theodolite and GPS surveys were carried out by Peacock, Darcey and Anderson Pty Ltd of Burnie in the Arthur River Project Area to establish the location of all completed drill holes, pump and monitoring water bore holes and access tracks.

A local grid was placed over the aero-magnetic grid (“AMG”) for resource modelling and estimation purposes. Local grid north is oriented 60° east north-east of AMG north. Any coordinates referred to in this report, or shown on drawings, relate to the AMG. Coordinates shown on sections are in local grid coordinates.

Almost all drill hole collars were surveyed by GPS and instrument surveys carried out by contract surveyors. Drill rigs were oriented on the grid using a compass or sighter pegs. Drill hole declination was set using a clinometer placed on the drilling rods, after the drilling rig had been levelled.

A study of drill hole surveys carried out by earlier workers revealed that there were no significant deviations during drilling. As a check, one downhole survey was carried out on AR23 during the 1997-1999 drilling programme. No dip deviation was recorded.

Holes drilled by CRAE were downhole surveyed using a single-shot Eastman camera and these reported surveys were used in the database.

4.4.2.2 Drilling

All holes drilled to date on RL8718 are within the proposed 1M/99.

Resource delineation drilling was by the diamond coring method. Results of some CRAE diamond core holes were included in the data base. The drill hole pattern was generally 40m fences with holes variably spaced along these lines.

4.4.2.3 Sampling Procedures

Sampling procedures are described in section 4.3.3.

4.4.2.4 Bulk Density Testwork

Initial bulk density assumptions were based on CRAE testwork. In April 1999, further testwork was carried out by Analabs in Burnie, with results supporting the assumptions used in the resource modelling. Both datasets resulted from testing using the water immersion method.

Densities used in the sectional resource modelling are detailed below:

Rock type	Density g/cm ³
Footwall/Hangingwall schist & siltstones	2.5
Overburden	2.0
Hard crystalline magnesite	2.7
Clayey magnesite	2.6
Broken hard magnesite	2.6
Cavernous, decomposed magnesite	2.6
Dolerite	2.7

4.4.3 Resource Estimation

Sectional ore zones were interpreted at cut off grades of 38.0% MgO, 4% CaO, 6% Fe₂O₃ and 12% SiO₂, with the MgO and CaO assays being the principal controls.

Metallurgical cut off grades were supplied by Hatch following completion of testwork on drill hole core samples.

4.4.3.1 Geological Interpretation and Procedures

A sectional interpretation at 40-50m sectional intervals was completed in April 1999. The sectional polygons followed a model of steeply east-dipping ($\approx 65-70^\circ$, local grid east) footwall and hangingwall rocks bounding the magnesite zone. To the north (local grid) of the main magnesite zone (as currently drilled), the magnesite is intersected by two dolerite dykes,

interpreted to be parallel to each other but cross-cutting the magnesite. Mineralised structures were modelled as being parallel to the footwall/hangingwall contacts. Magnesite was classified into clayey, cavernous, broken, crystalline and high CaO% zones.

4.4.3.2 Polygon Parameters, Procedures and Classification

Polygons were interpreted on sections using the following parameters:

- footwall and hangingwall contacts, with the contacts interpreted where no drill hole intercept data were available.
- high CaO% zones, parallel to footwall or hangingwall rocks, considered to be continuous down-dip on the footwall side and discontinuous down-dip on the hangingwall side, restricted to 40m away from the drillhole intercept on the hangingwall side, or halfway to the nearest hole, whichever was the lesser;
- no down-hole edge dilution;
- all polygons were classified as measured;
- magnesite classified as clayey was further subdivided into high grade and low grade zones. High grade clay resource was stockpiled for possible blending whilst low grade resource material was discarded to the waste stockpile;
- alluvium/overburden zones were modelled on each section.

4.4.3.3 Block Modelling

Wireframes were constructed for the main magnesite zone inclusive of broken, hard, iron stained, cavernous decomposed and high CaO% zones; for the high CaO% zones; for the high grade clayey and low grade clayey magnesite; and for the interpreted dolerite dykes.

Samples were composited to equal lengths of 1.5m. Grades for MgO, CaO, Fe₂O₃ and SiO₂ were assigned to block centroids by interpolation using IDW squared methods. Large search ellipsoids were used due to the large spacing between samples, particularly on some of the sections. Search ellipsoid radii and orientation are detailed in Table 5 below:

Table 5: Block Modelling Ellipsoids

Rocktype	Ellipsoid Radius (m)			Ellipsoid Orientation	
	Major Axis	Semi major Axis	Minor Axis	Bearing	Dip
Magnesite	100	100	25	0	-70
High CaO% zones	100	100	25	0	-70
High grade clayey magnesite					
- Steep dipping	100	100	25	0	-70
- Flat dipping	100	100	25	0	5
Low grade clayey magnesite					
- Steep dipping	100	100	25	0	-70
- Flat dipping	100	100	25	0	5

A maximum of 10 of the nearest samples was used to interpolate grades into each of the cells.

All cells were classified as Measured and were assigned a flag of 3.

The natural surface was mined off using a digital terrain model (DTM), constructed from the 1:25,000 topographic contours provided by Pitt & Sherry Holdings Pty Ltd in March 1999, along with surveyed points along tracks within the proposed 1M/99, from Peacock, Darcey & Anderson Pty Ltd.

The sectional resource estimate within the large pit, from surface is shown in Table 6 below:

Table 6: Measured Resource Estimate – Sectional Estimate

Rocktype	000T	MgO %	CaO %	Fe ₂ O ₃ %	SiO ₂ %
Hard crystalline magnesite	9,829	43.5	2.0	1.1	4.5
Iron stained hard magnesite	1,409	40.9	1.6	2.8	7.6
Broken hard magnesite	1,976	43.0	2.1	1.7	4.0
Cavernous decomposed magnesite	156	40.8	1.4	3.4	7.9
Subtotal	13,370	43.1	2.0	1.4	4.8
High grade clayey magnesite	1,696	41.2	0.7	3.3	7.7
Low grade clayey magnesite	676	28.7	9.2	6.5	13.0
Subtotal	2,372	37.6	3.1	4.2	9.2
Hard high CaO% magnesite	2,277	37.7	8.1	1.5	5.1

The block model resource estimate within the large pit, from surface is shown in Table 7 below:

Table 7: Measured Resource Estimate – Block Model Estimate

Rocktype	000T	MgO %	CaO %	Fe ₂ O ₃ %	SiO ₂ %
Magnesite	13,158	43.3	1.9	1.3	4.9
High grade clayey magnesite	1,553	40.9	0.7	3.3	8.3
Low grade clayey magnesite	619	28.4	9.5	5.8	14.7
Subtotal clayey magnesite	2,172	37.3	3.2	4.0	10.1
Hard high CaO% magnesite	2,160	39.8	6.2	1.3	3.8

Note: $\geq 38\%$ MgO and $\leq 4\%$ CaO tonnes reported as magnesite.

4.4.3.4 Further Work

Further work may be necessary to determine the orientation of the clayey and high CaO% magnesite zones which have largely been inferred from drilling data. The hangingwall has not been defined on some sections, so the approximate position has been interpreted. Further drilling could also possibly establish the orientation of the hangingwall and footwall and any structural controls, as well as provide information concerning the orientation and thickness of the dolerite dykes.

However, because the available tonnages of high grade magnesite resource are very large, further exploration can probably be deferred until mining commences, at which time some grade control measures may have to be implemented in these areas where high CaO% magnesite is known to occur.

Drilling along strike from the current proposed pit position, between drillholes AR003 and AR006, is recommended. This additional work may delineate better quality magnesite, in particular less clayey magnesite and magnesite uninterrupted by dolerite intrusions. It may also establish a magnesite resource without the restrictions imposed on the current resource, namely the proximity to the Keith River (and possible water ingress from the river into the pit), and the southern boundary of the mining lease the proposed 1M/99.

4.5 Mineral Resources

4.5.1 Indicated Resources

Following the 1997 TasMag drilling and the previous CRAE work, an indicated resource of **29 million tonnes at 42.8% MgO and 5.3% SiO₂** was determined to exist within RL8718.

4.5.2 Measured Resources

As a result of the in-fill drilling undertaken by Indcor in 1998/99, part of this indicated resource has been lifted into the measured category.

From preliminary metallurgical test work the cut-off grades used in this resource estimate were 38% MgO, 4% CaO, 6% Fe₂O₃ and 12% SiO₂.

Based upon the resource modelling undertaken (refer section 4.4) and applying cut off grades derived (38% MgO, 4% CaO, 6% Fe₂O₃ and 12% SiO₂), a Measured Resource was identified over a 250 metre strike length and to an average depth of 145 metres, of **13 million tonnes grading 43.4% MgO, 1.9% CaO, 1.3% Fe₂O₃ and 4.9% SiO₂** (Figure 6).

At a mining rate of approximately 410,000tpa which would be adequate to feed a 90,000tpa magnesium metal facility, this existing Measured Resource could provide a minimum 25-30 years feed to the metal facility.

4.5.3 Other Resources

The Measured Resource quoted in paragraph 4.5.2 above:

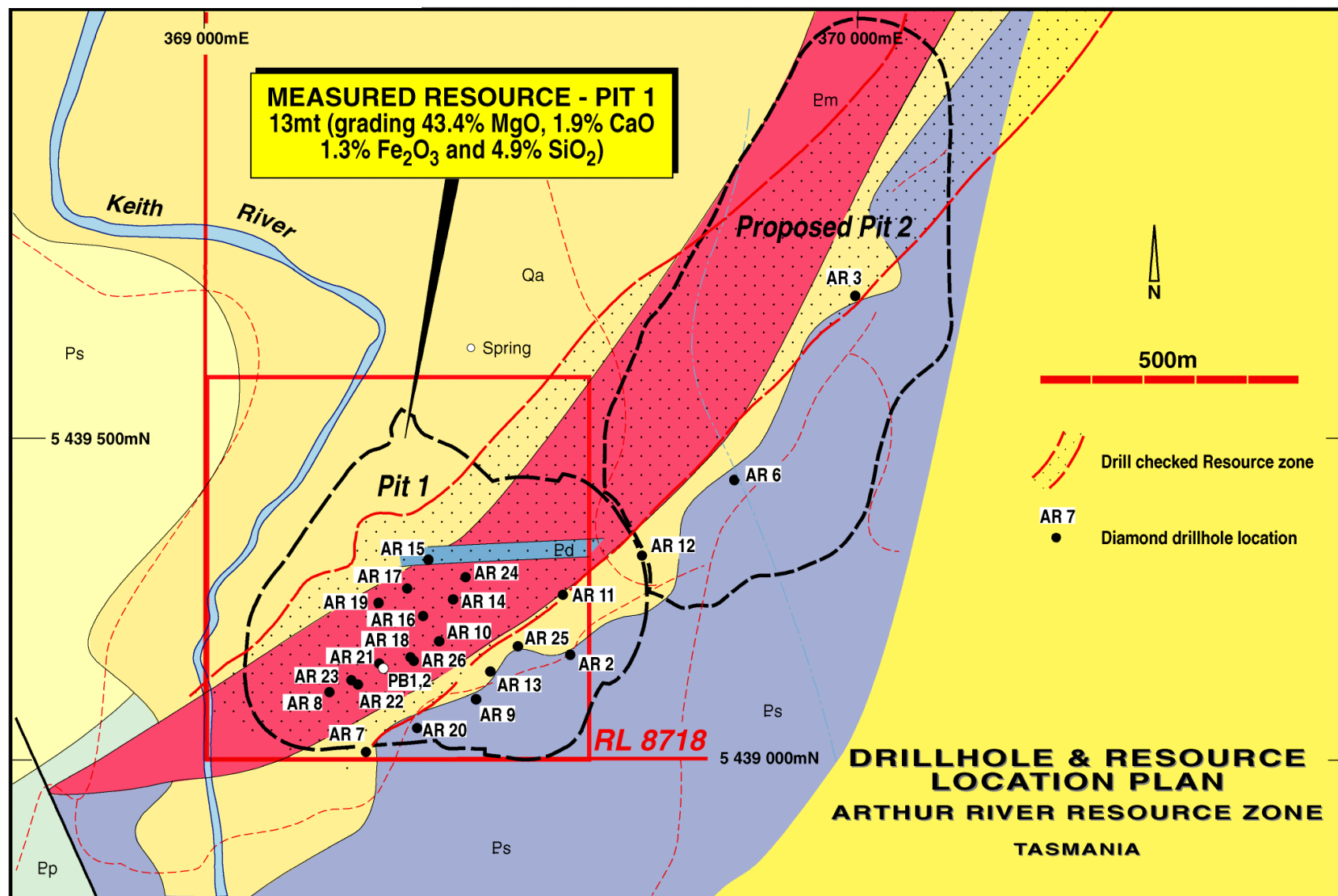
- Forms part of the 29 million tonnes Indicated Resource referred to in paragraph 4.5.1 above;
- Does not include 3.7 million tonnes of clayey and calcium oxide rich, high-grade magnesite which will be used for blending as feedstock for the proposed treatment plant; and
- Does not include any of the known mineralisation contained within RL8717.

The entire mineralised zone identified so far has a total strike length of about 5 kilometres, a maximum width in excess of 300 metres and a depth of at least 400 metres with no indication of weakening grade.

It has been recommended that further drilling be carried out in between AR3 and AR6 (CRAE 1983). The results from the above two holes indicated the presence of a high grade magnesite resource some 100 metres thick, having a strike length of at least 500 metres and extending to a depth in excess of 300 metres vertical. Application has been made and preliminary approvals were given, to carry out this in-fill drilling program, but this work has been indefinitely deferred.

It is anticipated that this drilling will identify a further 10.0 - 15.0 million tonnes of high-grade magnesite resource.

Figure 6 – Drill Hole Locations (RL8718)



5. MINING AND RESERVES

5.1 Proposed Mining Lease

The delineated measured and indicated resource occurs within RL8718 which has an area of 5km². In February 1999, a Mining Lease (1M/99) having an area of about 195 hectares was applied for within RL8718, covering about 1.5 kilometres strike length of high grade magnesite mineralisation. The proposed mining operation to extract some 15 million tonnes of this high grade resource is located in the south-western corner of the proposed mining lease and covers a resource zone some 250 metres long by 200 metres wide, and to a vertical depth of 145 metres. The area of the fully developed Pit No. 1 will be in the vicinity of 600 metres long by 500 metres wide and to a depth of about 140 metres.

The geology of the mining lease is almost totally concealed beneath a 10-15 metre deep cover of sand, gravel and boulder sediments. Outcrop is negligible being confined to in-situ magnesite in watercourses draining the area together with schistose scree material commonly present adjacent to the main formed gravel access track which skirts the southern edge of the resource zone.

5.2 Mine Plan and Pit Layout

A pit layout and mining schedule has been developed, the key features of which are set out below.

- Mining of magnesite will be by open pit methods. Sequencing of the open pit excavation will need to ensure that the yearly strip ratio does not exceed 5:1.
- The initial pit as developed for the 90,000tpa Mg metal case results in final open pit to mine the current Measured Resource totals 37.7 million tonnes comprising 13.3 million tonnes high grade magnesite, 2.2 million tonnes high calcium magnesite and 1.6 million tonnes high grade clayey magnesite and 20.6 million tonnes of waste. The clayey and lower grade resource categories will be stockpiled for possible future blending with high grade treatment plant feedstock.
- Mining will be by truck and excavator and the material will be broken by drilling, charging and blasting.
- All open pit faces will be excavated and dressed to reduce erosion. Faces will be supported by concrete grouted bird-caged bolts or cable bolts, as required.
- The top face has been designed to have a 40° batter slope which may be supported with fibrecrete and grouted birdcaged bolts. The top surround adjacent to the Keith River will have a 2m high bund wall, with sides sloping at 30° and a top width of at least 2 metres

- A windrow will be positioned around the pit circumference and starting at least 2 metres back from the high wall.
- The first wall face of 40° will be 5 metres high onto a 5 metre wide berm. This berm may be formed as a drain and possibly fibrecrete lined. The next face is proposed at 45° for a 20 metre high lift to a 5 metre wide berm, with the following face proposed at 55° also for a 20 metre high lift to a 5 metre wide berm. Subsequent faces are proposed at 65° face angle for each face of 20 metres height.
- All catch berms are 5 metres wide.
- Design controls for the final open pit layout are the Keith River where the pit edge is located at least 50 metres from the stream edge.
- Along the southern boundary of proposed 1M/99 the bund is located 30 metres from the boundary to allow a windrow and a lined spoon drain.
- The final floor level of the pit is planned for RL10 being controlled by the proximity of the Keith River.
- A 15 metre wide ramp at 1 in 8 gradient is proposed. This is wider than normal but allows for an adequate windrow and a drain which may be required in the wet months.

5.3 Overburden and Waste Rock

5.3.1 Overburden Characteristics

Information derived from drilling, drill-site preparation works and exploratory pitting, indicates that the overburden largely comprises grey sandy gravels containing some silt and pebble/boulder horizons. The gravels are composed of both well rounded and angular schist, dolerite, magnesite and quartz material, whilst the sand size fragments comprise both sub-rounded and well rounded fractions.

In addition to the gravel overburden, there is an extensive area of red-brown clay developed, on rising ground at the north-easterly end of the resource zone. These red brown clays are associated with two dolerite dykes which were encountered during drilling of diamond drill holes AR2, 11, 12, 13, 14, 24 and 25.

5.3.1.1 Test Pitting

In 1999, a short program of test pitting was undertaken by Pitt & Sherry, Consulting Engineers in Tasmania to check on the characteristics of the overburden in the vicinity of the proposed open pit.

A total of five excavation pits were located within the planned mining pit boundaries.

Excavation pits MM1 (located approximately 200m east of AR7 and the closest of the test pits to the Keith River) and MM2 (located adjacent to AR8) exposed silts overlaying river wash gravels of mixed origin. The gravels were relatively unconsolidated and became coarser with depth. Below the water table the gravels became semi fluid when disturbed. Excavation of MM1 beneath 4m was not possible due to hole collapse. In MM2 extremely weathered magnesite was recovered from approximately 4.8m which was beneath the surface of the fluid gravels.

Extremely weathered magnesite with an irregular surface was exposed below 2m in MM3 (located between AR18 and AR10), underlying a varied exposure of Quaternary sediments. MM3 was stable to 6.8m with no water inflows.

Excavation MM4 (located adjacent to AR15) contained relatively unconsolidated angular gravels of mixed metamorphic origin which became finer graded with depth. A distinct and horizontal boundary separated the gravels from the underlying grey silt. The silt contained some fine angular quartzite gravel which occurred in inter-bedded layers increasingly with depth. The strength of the silt was variable with pocket penetrometer values of 35 to 40kPa up to 200kPa measured for full depth. The silt and metamorphic gravel are likely to be of Quaternary age.

Excavation MM5 (located adjacent to AR2) at RL174 was located on hill slopes on the east side of the pit. The pit exposed angular metamorphic gravels inter-bedded with irregular silt layers. The sediments are likely to be of Quaternary age. Some hole collapse occurred below 4m where water seepage occurred.

5.3.2 Waste Rock Characteristics

The material designated waste rock comprises:

- Soil
 - possibly 350,000t.
 - requires special dump area for future rehabilitation works.
- Overburden
 - possibly 7.3 million tonnes.
 - includes some clays developed over doleritic intrusions at north-eastern end of resource zone and areas of clay developed over hangingwall schist which had no drill control on overburden thickness. No special stockpile requirements apart from possibly lower angle batter slopes.
- Waste
 - possibly 13.50 million tonnes.
 - Includes all hangingwall and footwall schist, low grade clay and dolerite and pyritic footwall schist (which possibly totals 900,000 tonnes).

In addition to waste rock stockpiles, the high and low grade clayey magnesite resource together with magnesite containing >4% CaO will also be selectively stockpiled for probable future blending with hard, high grade magnesite which will be used for plant start-up.

5.3.3 Overburden and Waste Rock Management

Following initial excavation, top soil will be stored progressively in banded areas away from the pit. This top soil will be used for the progressive rehabilitation of the overburden and waste rock dump revegetation.

Overburden suitable for fill material will be used as such. Material over and above this will be placed on the waste dump.

Footwall material containing disseminated pyrite has been intersected in the resource drilling. This material will be sealed in a clay area within the waste dump.

The waste dump will have a drain on each berm at the intersection of the batter. These drains will be graded to either inlet of the settling pond and discharge directly into it. Thus any erosion will only result from rain falling on to the batter face.

A drain will be constructed at the toe of the dump.

5.4 Mining Stages and Timing

Five mining stages are planned. Stage 1 is a small pit to 80 metres depth to ensure that if any flooding occurs it should fill this pit and allow production to be maintained from other stages.

Stage tonnages are as follows:

Stage	Ore Tonnes	Waste Tonnes	Strip Ratio	Cumulative Ore Tonnage and Timing
1	2,600,000	5,100,000	2.01:1	2,600,000 @ 0.41 M tpa 6.34 Years
2	1,500,000	6,800,000	4.56:1	4,100,000 @ 0.41 M tpa 10.00 Years
3	1,900,000	7,200,000	3.81:1	6,000,000 @ 0.41 M tpa 14.63 Years
4	2,600,000	3,000,000	1.20:1	8,600,000 @ 0.41 M tpa 20.98 Years
5	4,700,000	2,000,000	0.49:1	13,300,000 @ 0.41 M tpa 32.44 Years
	13,300,000	24,400,000	1.86:1	
Total	37,700,000 Tonnes			

5.5 Stockpiles

Run of mine ore will be stockpiled within the mining area for transport to the process plant facility.

Stockpiles will probably comprise:

- High grade massive magnesite - plus 40%
- Medium grade massive magnesite - plus 38%
- High grade clayey magnesite - for probable future blending
- High grade magnesite with Calcium - below 4%
- High grade magnesite with Iron - below 6%
- High grade magnesite with Silica - below 12%

All top soil removed from the open pit, waste dump areas, and during the construction of road areas will be stockpiled for use for the re-generation of ground cover. Top-soil stockpiles will be established at several locations.

A crushed material stockpile will be created at the end of the crushed material belt.

5.6 Pit Dewatering

5.6.1 Pit Pumping

Mine dewatering will be required since the pit will be excavated to at least 100m below the level of the water table. It is envisaged that dewatering will be achieved by sump pumping from the pit floor, allowing free inflow through the pit walls unless preventative measures such as perimeter grouting or pumping bores are installed. The dewatering rate will increase progressively from about 100-120 L/s in Year 2 to 150-180 L/s by Year 27. About 80% of this water is predicted to come from leakage through the bed of the Keith River.

Taking the five year recurrence interval storm of 24 hours duration as the design storm and allowing 24 hours to drain the pit floor, it is estimated that a maximum total pumping capacity of 470 L/s may be required for continuous mining operations.

An alternative approach would be to campaign mine over the dry months to avoid wet weather, water inflows and generally provide better working conditions. Owing to the small amount of material to be mined in a year, it is possible that sufficient material to feed a 90,000tpa metals facility may be mined and stockpiled in a 4 month period. Further investigation is required to fully assess and develop detailed costings for this option.

5.6.2 Treatment of Pit Water

Settlement ponds will be used to collect solids entrained in the water. A retention time of 6 hours has been adopted for design purposes allowing particles of 0.01 mm and above to be captured.

Given the five year maximum ingress flow-rate of 470 L/s from the pit, an area of 10,000m² is required with an active moving water depth of approximately 1m. The pond is envisaged to have a depth of 2m. As allowance has not been made for adverse effects an area of 20,000m² has been provisionally allowed to accommodate settlement ponds for the handling of the dewatering process.

5.7 Stormwater and Site Drainage

5.7.1 Loadout and Waste Dump Area Drainage

A 24 hour, 1 in 5 year rainfall event has been used resulting in a design volume figure of 41,500m³. Drainage trenches on the berms of the dump, at the toe and at the loadout area are graded to the settling pond situated at the north of the waste dump. A sump and pump is provided adjacent to the crusher plant at a natural low lying area. This would also pump to this settling pond.

Settling is again achieved in a 20,000m² area pond however surges are accommodated by a storage pond of 15,000m³.

5.7.2 Stormwater and Site Drainage

The southern edge of the pit will be protected from water ingress by a spoon drain. The incoming water will be from virgin ground to the south and therefore will not contain contaminants. The spoon drain will be lined and discharge into the Keith River.

A 2m high windrow will be constructed along the western side of the lease 40m from the eastern edge of the Keith River. This will ensure that the 100 year flood elevation of 145 RL will not overflow the lip of the pit.

A lined spoon drain will cut off water draining from the eastern slopes to the pit. This, and the other virgin eastern areas will drain down an existing gully to the Arthur River.

5.7.3 Grout Curtain Between the Keith River and the Proposed Open Pit

A concrete/fibrecrete grout curtain may be an option to reduce groundwater flow into the proposed open pit from the Keith River and pit perimeter. Exploratory and in-fill diamond drilling completed to date indicates that most cavities developed in the magnesite occur within the upper 70 metres and it is believed a grout curtain to this depth, or to a depth where solid rock is encountered will significantly reduce groundwater seepage from the Keith

River by way of cavities, fault zones, rock fractures and bedding planes developed within the magnesite and along rock contact zones.

Initially a curtain comprising holes at 25 metre centres to a vertical depth of 70 metres is proposed. Depending on the results of this work additional grout holes may need to be installed. Influx of water from the north-eastern end of the pit is not expected to be a problem because of the presence of a dolerite dyke and associated clay which are expected to be relatively impermeable.

Prior to installation of the curtain it is likely that further bores will be drilled between the pit and the Keith River and at other selected sites and pump tested both as a check on probable grout intake and the effect on groundwater drawdown generally.

It is possible that as an alternative to a grout curtain installed by way of holes located between the pit and the Keith River, it may be easier to install a grout barrier from inside the pit as excavation progresses. The worth of such a proposal will be evaluated from an examination of pump and monitoring bore tests which will be carried out when perimeter pump bores are being tested.

5.8 Rehabilitation Activities

The mine works for the TasMag Project will comprise an open pit which will be progressively deepened to a vertical depth of over 145 metres below current ground surface. In addition to the developing pit other site works will comprise soil, waste and ore stockpiles, perimeter and run-off drains and two settlement ponds. Together with general clearing for site buildings progressive rehabilitation will be restricted to the side slopes of the pit and the waste stockpile and for these structures it is planned that available slopes will be progressively covered with at least 200mm of soil and re-seeded with selected, shallow rooted nature species.

In 1982 the entire mine area, including the proposed open pit, was razed by a major bush fire. Coloured aerial photography from October 1984 clearly shows the cleared mine site area. This area was then aerial seeded with various local and native species. This aerial seeding has been very successful resulting in all traces of past exploration by previous mining companies comprising mine sites, test pits and access tracks being overgrown by re-growth of eucalypts and shallow rooted species comprising ferns, dogwood, ti-tree, myrtle, sassafras and others native to the area. Evidence of this successful re-growth is clearly shown in 1998 aerial photography of the mine site. It is intended that revegetation with a similar cover will be progressively carried out on the pit and waste dump slopes as they are developed.

Rehabilitation will be undertaken on all areas that have been disturbed as soon as is practicable. This will be primarily the side slopes of the waste pit. These slopes will be covered with top soil taken from the dumps and re-seeded with native plants similar to those removed at the onset of the works.

6. PROCESS AND PROCESSING FACILITY

6.1 Introduction

Extensive process Testwork, flowsheet development and basic engineering design has been undertaken by Indcor.

Process flow diagrams and process simulation using Metsim have been completed to establish equipment sizing, material, heat and reagent usage. This information is incorporated into the various feasibility study documentation held by Indcor.

As technology and scale up considerations are seen as key parameters for any review on magnesium metal production this Memorandum overviews testwork and process design work in substantial detail. Extensive process and basic engineering design will be made available in the Indcor data room.

6.2 Process Definition

6.2.1 Raw Material and Reagent Preparation

Magnesite ore is the primary raw material for the production of metallic magnesium metal. Magnesite may also be used to produce MgO by calcination for reagent usage in the leach process.

Run of mine magnesite ore from the open pit mining operation will be transported to the plant site and delivered to stockpiles to be used as feed to the leaching and calcination processes.

The magnesite ore will be crushed at the plant site in a closed circuit crushing and screening operation to an average particle size of less than 2 mm. Crushed ore will have the following average chemical composition (Table 8).

Table 8: Magnesite Ore Average Chemical Composition

Component	Weight Fraction
MgO	42.80
SiO ₂	7.07
FeO	0.89
CaO	2.50
LOI	47.02

Run of mine magnesite ore will also be calcined to produce magnesia (MgO). The magnesia product will be used in the process as a reagent leaching additive to precipitate out the calcium for brine purification or magnesia could be produced and prepared to commercial sale. Hot MgO is conveyed to leaching for gypsum precipitation, while part of it is slurried in a tank to

produce a 20% solids slurry for scrubbing (alkaline scrubber and SO₂ scrubber).

A recirculation cooling water circuit with a cooling tower will be used to supply cooling water. Cooling water make-up will be from process water, which will require prior treatment.

Other solid, liquid and gaseous reagents or utilities are purchased externally and delivered to the plant site in the form in which they are required.

6.2.2 Magnesite Leaching and Brine Purification

The hydro-metallurgical recovery begins with the leaching of magnesite ore (MgCO₃) with hydrochloric acid (HCl):



Magnesite ore is continuously leached with hydrochloric acid in a series of agitated leach reactors. Slurry overflows from one reactor to the next. Slurry from the last reactor is pumped to a leach solution clarifier and a filter to separate the unleached residue (SiO₂, etc) from the resulting MgCl₂ brine. The leach reactor off-gas is treated in the leach scrubber to recover a 4 to 7 wt% HCl solution which is recycled to the leach reactors.

The overflow from the leach solution clarifier and the primary filtrate from the leach filter is neutralised in a series of agitated neutralisation reactors, whereas the secondary filtrate is recycled to the leach reactors to recover its magnesium content. The brine is neutralised with MgO to adjust the pH and to purify the solution from metallic impurities. Precipitated impurities are removed by thickening and filtration before being disposed of to the tailing storage facility. The neutralised MgCl₂ brine is further purified by the removal of boron, nickel and manganese by means of ion-exchange.

The resultant brine is stored in a pure brine storage tank.

6.2.3 Spent Electrolyte Granulation

The purpose of this plant area is to produce a high purity MgCl₂.KCl brine from pure MgCl₂ brine, KCl make-up and from molten spent electrolyte.

Molten spent electrolyte from electrolysis is transferred to a holding furnace. It is then contacted with a portion of the pure brine, by means of a rotating device in a granulation tower.

The resulting spent electrolyte powder is transferred to dissolution reactors, in which most of the pure brine is injected, resulting in the dissolution of most soluble species and the production of a synthetic carnallite solution which is filtered and pumped to the carnallite crystallisation holding tank.

Air is injected in the granulation tower and in the dissolution reactors in order to dilute the off-gas, which may contain traces of hydrogen gas formed during the hydrolysis of molten magnesium contained in the spent electrolyte.

The off-gas from the granulation tower, which contains mainly inerts and water vapour, is directed to a cooler scrubber where fresh water is injected to capture solids and condense the water vapour. The scrubber underflow is recycled to the dissolution reactors.

6.2.4 Carnallite Crystallisation

The purpose of this plant area is to produce synthetic carnallite hexahydrate crystals ($\text{KCl} \cdot \text{MgCl}_2 \cdot 6\text{H}_2\text{O}$) from the synthetic carnallite solution and to remove calcium chloride from the process by means of gypsum precipitation.

The synthetic carnallite solution is first directed to a calcium sulphate crystalliser where calcium sulphate hemihydrate ($\text{CaSO}_4 \cdot 0.5\text{H}_2\text{O}$) crystals are formed. These crystals are then removed from the carnallite solution by means of a clarifier. The synthetic carnallite solution is then directed to a carnallite crystalliser, in which carnallite hexahydrate ($\text{MgCl}_2 \cdot \text{KCl} \cdot 6\text{H}_2\text{O}$) is precipitated. The slurry from the carnallite crystalliser is passed through a clarifier and a centrifuge. The resulting raw synthetic carnallite crystals pass through a screw feeder, in which sodium chloride (NaCl) and potassium chloride (KCl) make-up are added, and then are directed to a stockpile which makes up the feed to the pyrometallurgical area.

Mother liquor is bled from the crystallisation circuit and directed to a gypsum precipitation tank, where the addition of sulphuric acid causes the conversion of calcium chloride (CaCl_2) to calcium sulphate (CaSO_4). Calcium sulphate is almost entirely precipitated to gypsum ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$), and the resulting gypsum slurry is returned to the leach plant, where gypsum leaves the process with the leach residue. A small quantity of dissolved calcium sulphate reports to the leach brine and hence to the synthetic carnallite solution.

6.2.5 Carnallite Dehydration

The purpose of this plant area is to produce highly dehydrated carnallite from raw carnallite hexahydrate. This is achieved in a fluidised bed dehydrator with selective fluidising gas addition.

Electrolytic chlorine is burned with natural gas and steam to produce an HCl -containing hot combustion gas, which supplies the heat required in the fluid-bed dehydrator. The presence of HCl in the fluidised-bed dehydrator minimises the amount of MgCl_2 hydrolysis to $\text{Mg}(\text{OH})\text{Cl}$.

The resulting dry carnallite crystals are fed to a storage silo by dense-phase pneumatic conveying, and from there, to electrolysis through a separate dense-phase pneumatic conveying system.

The dust-laden dehydrator off-gas passes through a cyclone bank. The cyclone underflow is recycled to the dehydrator, whereas the off-gas, containing HCl, is treated by absorption for hydrochloric acid production.

6.2.6 Electrolysis

The electrolysis plant is intended for production of magnesium metal and chlorine gas by electrolytic reduction of MgCl_2 from dehydrated carnallite.

Electrolysis takes place in the electrolytic cells; 108 units are required to meet the overall plant capacity of approximately 90,000tpa. The electrolytic cells are combined into 4 flow line systems. All units of the flow line are connected to form an integrated hydrodynamic system and to allow for the continuous flow of molten electrolyte and magnesium metal from the electrolytic cells to the separating cell.

Highly dehydrated solid carnallite is transferred to the electrolysis plant by dense-phase pneumatic conveying and discharged into the feed bins located at each electrolytic cell. From the feed bin, carnallite is metered and continuously fed into the electrolytic cell. In the electrolytic cell, carnallite is fused with the heat provided by the molten electrolyte. Magnesium chloride from the carnallite feed is electrolytically reduced to Mg metal and Cl_2 gas.

The resulting magnesium metal is carried out from the electrolytic cells with the flow of electrolyte.

The magnesium metal is separated from the electrolyte and preliminarily refined from non-metallic impurities in the separating cell installed at the end of each of the flow lines.

Electrolytic chlorine gas is evacuated from the cells through a system of ducts and transferred to the bag-house area for removal of condensed sublimates from electrolyte vapours before being returned to the process in the carnallite dehydration stage.

6.2.7 Refining and Casting

The refining and casting plant area is intended for the refining of crude magnesium metal to produce pure metal in accordance with ASTM specifications. Magnesium alloy will also be produced.

For the production of refined pure Mg metal, Continuous Refining Furnace (CRF) technology and equipment will be utilised. Molten crude magnesium metal is transferred from the electrolysis area into the CRF. There are two layers of molten materials in the CRF - a layer of molten chloride salt electrolyte at the bottom of the furnace and a layer of molten Mg metal on top of the electrolyte. The furnace is heated by AC electrodes mounted through the side wall of the furnace and immersed into the electrolyte layer.

All non-metallic impurities (oxides and chlorides) are precipitated from the Mg metal and accumulated at the bottom of the furnace. Refined magnesium is then analysed to verify its compliance with specifications and cast into ingots on the in-line casting machine.

Magnesium alloys are produced batch-wise in the crucible furnaces, where the metal is superheated to a temperature of approximately 730 to 750°C. A calculated amount of pre-heated alloying elements (aluminium ingots or zinc ingots or both) are charged into the furnace and thoroughly mixed with the magnesium metal. A calculated amount of manganese chloride (MnCl_2) is added during mixing to adjust the Mn content of the melt.

Once mixing is complete, the alloy is analysed for major components (ie, Al, Zn and Mn) and transferred into the Alloy Continuous Refining Furnace (ACRF). In the ACRF, the alloy is refined from non-metallic impurities. Once the refining process is completed, an alloy sample is taken and analysed for compliance with specifications and the alloy is cast into ingots on the in-line casting machine.

6.2.8 Chlorine Handling

The purpose of this plant area is to remove entrained particulate matter from the electrolytic chlorine and re-compressed for its reuse in dehydration.

Chlorine produced at the electrolysis plant is treated in a bag-house for solids removal. The gas then passes through a wash tower equipped with a venturi, where water is used to collect HCl and solids. The resulting wet chlorine gas is compressed before being directed to carnallite dehydration.

The HCl solution from the chlorine wash tower is passed through a chlorinated hydrocarbons (“CHC”) removal system. It is anticipated that a few grams of CHC will be produced on an annual basis, and the portion of the CHC’s generated during electrolysis are captured using activated carbon. The solution is then directed to the HCl tank of the HCl absorption plant, where it is mixed with HCl produced by absorption of HCl from the dehydration off-gas.

Solids collected in the chlorine duct to the bag-house and from the bag-house are dissolved with water. The resulting solution is used for absorption and particle collection in the wash tower.

The activated carbon containing the CHC waste is collected and transported off site for long term storage.

6.2.9 HCl Absorption

The purpose of this plant area is to recover HCl from the dehydration off-gas to produce an HCl solution for use in leaching.

HCl-containing off-gas from carnallite dehydration is contacted with water in a HCl absorber in order to recover an 18 wt% HCl solution.

Before being fed to leaching, the HCl produced is directed to a storage tank in which pure 33% HCl is used as make-up. The storage tank is maintained under a slightly negative pressure to prevent leaks to the plant environment. Vent gas from this tank is treated in the leach scrubber.

The off-gas from the absorber passes through an alkaline scrubber, in which a magnesium hydroxide slurry is used to absorb HCl and potentially uncombusted chlorine to levels making the gas suitable for release to atmosphere.

6.3 Testwork Programme

The production of metallic magnesium from magnesite requires the following process steps using the patented UTI/VAMI technology with a high purity magnesite feed:

- Production of a purified 20% MgCl_2 solution by, dissolution of magnesite in hydrous hydrochloric acid, precipitation and separation of dissolved heavy metals and insoluble impurities.
- Production of a hot extract synthetic carnallite solution that contains KCl, NaCl and MgCl_2 from 20% MgCl_2 solution and spent-electrolyte.
- Production of pure and coarse grained carnallite crystallisate from an extract solution and a circulated carnallite mother liquor by evaporation and cooling crystallisation.
- Dehydration of the carnallite crystallisate.
- Fused-salt electrolysis of the dehydrated carnallite to produce metallic magnesium, chlorine and spent-electrolyte.
- Production of hydrochloric acid from electrolytic chlorine

Extensive pilot and laboratory scale testwork was completed under the direction of Multiplex and Hatch during the Optimisation program. This programme included:

Leaching

- Testwork by Process Research ORTECH of Mississauga, Canada confirmed that TasMag's magnesite resource is amenable to the production of a pure magnesium chloride brine suitable for the production of synthetic carnallite (Phase I test programme);

Carnallite Crystallisation

- Kali-Umwelttechnik (K-Utec) of Sondershausen, Germany confirmed that pure electrolytic grade carnallite crystallisate can be produced from MgCl_2 solution and spent electrolyte (Phase II and III test programmes);

Carnallite Dehydration

- Carnallite crystallisate of more than 50kg produced from Phases II and III formed the basis of the process which was undertaken by VAMI of St Petersburg, Russia. This confirmed the UTI/VAMI process for the dehydration step of the synthetic carnallite feed and resulting on-processing by electrolysis to Magnesium metal (Phase IV test programme);

Electrolysis

- Testwork was undertaken by UTI in Zaporoshe, Ukraine. Twelve magnesium metal ingots were produced by UTI from the carnallite crystallisate produced in the Phase IV test programme.

In addition, as part of optimisation program, Hatch also undertook a detailed review and report preparation of the following:

- the dehydration process proposed by UTI/VAMI;
- the chlorine combustion system proposed by UTI/VAMI; and
- the gas handling system throughout the process flowsheet, namely:
 - ⇒ HCl absorber and alkaline scrubbers;
 - ⇒ Chlorine bag-house and wash tower;
 - ⇒ Emergency chlorine scrubber;
 - ⇒ Leach scrubber;
 - ⇒ Cooler scrubber; and
 - ⇒ SO₂ scrubber.

Hatch concluded that the selected process represents an improvement over that at the Dead Sea magnesium plant (commissioned in 1997 using UTI/VAMI technology) and should provide significant benefits to the TasMag Project once operations commence.

Further details of the principal testwork undertaken and conclusions reached follows.

6.3.1 Leach Testwork Programme – ORTECH (Phase I)

6.3.1.1 Introduction

Whilst it is well known that most magnesite ores will leach to some extent in strong hydrochloric acid, experience has shown that material from different ore bodies may behave differently. For example, some ores are more refractory than others, or have differing levels of impurities. For such reasons Hatch recommended that a leaching testwork program be carried out in order to confirm the fundamental parameters for the leach circuit. This testwork program was separated into two stages:

- in the first stage, scoping leach tests have been carried out to confirm the preliminary leaching, oxidation and neutralisation conditions for the production of a purified 20% MgCl₂ brine from the TasMag ore; and
- in the second stage, it is proposed that continuous leach and neutralisation tests be carried out to optimise the parameters for the engineering design work to be used for the commercial plant.

The objectives of the stage I testwork program, which was undertaken by Process Research ORTECH of Mississauga, Ontario, Canada and supervised by Hatch were to confirm fundamental parameters and also to produce a process sample for ongoing testwork and evaluation.

The first objective was to confirm the fundamental leach and neutralisation/purification parameters necessary to achieve the production of a purified 20% magnesium chloride brine, suitable for sending to the carnallite crystallisation section of the plant. The test programme incorporated the following:

- preliminary bench scale batch leaching (scoping leach tests) to determine:
 - extent of magnesium extraction;
 - kinetics of magnesium extraction;
 - batch residence time;
 - temperature of leaching;
 - acid to ore ratio;
 - effect of particle size;
 - impurity leaching;
 - behaviour of different core samples; and
 - any unusual problems.
- based on the results from the preliminary leach tests, select the preferred conditions, and then continue with oxidation (with chlorine gas) and neutralisation (with reactive magnesia) tests to confirm:
 - preferred sequence of oxidation and neutralisation;
 - preferred oxidation potential to maximise removal of Ni, Mn and Fe;
 - reactive magnesia requirement and initial estimate of utilisation efficiency;
 - residence time for both oxidation and neutralisation;
 - impurity deportment / efficiency of removal;
 - effect of temperature; and
 - any unusual problems.

The second objective of the testwork program was to carry out, based on the results obtained from the above series of tests, a bulk leach / oxidation / neutralisation to produce 400 L of pure brine to be sent to K-Utec, Germany for subsequent carnallite crystallisation testwork.

6.3.1.2 Scoping Leach Tests

The initial scoping leach tests were carried out at the bench scale (approximately 1L), which is normal practice, by adding a fixed weight of ore to a set volume of 18% HCl and heating to the desired temperature. Thereafter, the slurry was maintained at this temperature for a period of four hours, with slurry samples being taken for chemical analysis after 15, 30, 60, 120, and 180 minutes of reaction. A similar procedure was used for testing of all of the core samples received. The amount of acid added (acid to ore ratio) was varied based on there being a calculated magnesium extraction efficiency of 85%, 90%, 95% and 100%.

6.3.1.3 Leach/Oxidation/Purification Tests

Based on the preferred conditions identified in the scoping leach tests (residence time, acid to ore ratio, temperature), further tests were carried out wherein the resultant leach solution was oxidised with elemental chlorine gas and neutralised with reactive magnesia, as follows:

- the unreacted solids were removed by filtration, followed by chlorine sparging to constant redox (reduction-oxidation) potential, followed by neutralisation with either additional magnesite and/or reactive magnesia to a final pH of 6.0.
- these tests were carried out at 70°C, since if vacuum filtration is used commercially, this temperature is about the maximum that is operable without imposing unacceptable duties on the vacuum system due to steam flashing, when operating at a vacuum of 20-24 inches.
- variations in the order of sparging, neutralisation and filtration were investigated. Thus, in some tests, the leach solids were not removed until the end of the test (since they are such a very small amount), and/or partial neutralisation to pH 3-4 was carried out prior to chlorine sparging (which was carried out for up to two hours to ensure a constant redox potential).

A commercially available reactive magnesia, nominally containing $\approx 97\%$ MgO was used for these tests, rather than calcining magnesite received from TasMag. This was considered to be the more practical approach at this stage of the testwork program, where it was really only necessary to demonstrate a workable flow sheet.

6.3.1.4 Preliminary Leach Tests

The following observations can be drawn from these preliminary tests (Table 9):

- between 83% and 97% of the mass of the feed material generally dissolved. This is advantageous from a mining standpoint, but did cause problems in the testwork in obtaining a good material balance, and hence obtaining consistent results, due to there being almost no solids left at the end of the test;

- magnesium extraction of > 90% was achieved after 60 minutes of leaching. Increasing the leach time to four hours did give slightly higher extractions, but the additional recovery is not considered to be economically attractive;
- virtually all of the calcium dissolved in all samples;
- about 90% of the iron dissolved during leaching for the high grade samples (lower net dissolution but higher final solution concentration for the lower grade samples), and will, therefore, require subsequent oxidation and precipitation to effect its removal;
- there was little difference in leach behaviour for particle size between 6 mesh and 20 mesh;
- leaching was most effective at 95°C, rather than 25°C or 60°C, although there was little difference between the two higher temperatures;
- there were some apparent small differences in magnesium extraction between the different samples, but this may be due to the difficulties in obtaining a good material balance; and
- based on these tests, acid addition can be estimated to be approximately 95% of that required for complete 100% Mg recovery.

Other observations, not reported in Table 9, but deemed significant are:

- the ore itself was very soft, and was very easily crushed. This indicates that the requirements for ore preparation should be relatively simple.
- frothing, due to the evolution of carbon dioxide, was not observed when the ore was added to acid at room temperature, and the mixture then heated to 95°C. However, when added at 60°C or 95°C, then significant frothing was observed. The problem was worse at the highest temperature, and dictated that the ore had to be added slowly over a period of 25 minutes rather than all at once. The implications for continuous operation are not clear at the present time, but will need to be investigated during the Phase II testwork. Since, in a continuous operation, the reactors operate their exit conditions, the bulk acid content will be lower, which helps to minimise any frothing problems. Other approaches to deal with this problem are to increase agitation speed, or to add a defoamer.

Table 9: Summary of Scoping Leach Test Conditions and Results

Test No.	Core/ Grade	Stoich. (%)	Particle size	Temp. (°C)	Time (hrs)	Mg extract. 1 hr (%)	Mg extract. 4 hr (%)	Ca extract. 4 hr (%)	Fe extract. 4 hr (%)
1	AR8/ LG	100	20 mesh	95	4	92.5	98.3	100	97.0
2	AR10/ HG	100	20 mesh	95	4	96.4	98.8	99.9	95.4
3	AR8/ LG	85	20 mesh	95	4	91.8	87.2	99.7	2.4
4	AR10/ HG	85	20 mesh	95	4	88.5	88.4	99.3	71.1
5	AR8/ LG	90	20 mesh	95	4	88.2	91.3	99.8	20.5
6	AR10/ HG	90	20 mesh	95	4	92.7	93.5	99.6	83.5
7	AR8/ LG	95	20 mesh	95	4	86.3	96.1	99.9	65.8
8	AR10/ HG	95	20 mesh	95	4	98.2	98.3	100	97.2
9	AR8/ LG	90	6 mesh	95	4	92.5	90.7	81.9	11.9
10	AR10/ HG	90	6 mesh	95	4	88.1	95.5	81.0	62.9
11	AR10/ HG	95	20 mesh	24	5	51.1	75.6	98.0	69.5
12	AR10/ HG	95	20 mesh	60	5	87.4	94.4	99.7	91.9
16	AR2/ HG	95	20 mesh	95	4	89.2	97.2	100	96.7
17	AR8/ HG	95	20 mesh	95	4	83.1	96.2	99.8	72.2
18	AR9/ HG	95	20 mesh	95	4	93.4	96.1	99.8	55.7
20	AR8/ LG	100	20 mesh	95 ¹ (hot acid)	4	95.5	97.5	100	90.2
21	AR10/ HG	95	20 mesh	95 ¹ (hot acid)	4	93.2		99.7	90.1

Note 1: Ore added directly at 95°C

One of the more important criteria to study is the deportment of potentially deleterious impurities. Table 10 below summarises the solution composition at the end of each scoping leach test.

Table 10: Summary of Leach Tests - Final Solution Analysis

Test No.	Assay	Mg (g/L)	Ca (g/L)	Fe (mg/L)	Si (mg/L)	Mn (mg/L)	B (mg/L)	Ni (mg/L)	Terminal pH
1	5321	68.2	8.28	2230	38	190	<5	<5	<0
2	5329	64.8	3.02	930	89	120	<5	<5	<0
3	5335	62.0	7.75	39	17	170	<5	<5	4.7
4	5355	67.4	3.14	690	60	100	<5	<5	1
5	5360	65.0	7.82	380	22	190	<5	<5	0.8
6	5381	68.5	2.83	820	67	110	<5	<5	2.8
7	5386	69.9	8.02	1310	39	190	<5	<5	2.7
8	5391	65.6	2.77	910	66	110	<5	<5	0.3
9	5398	62.7	7.33	290	23	170	<5	<5	3.7
10	5410	72.0	3.14	760	55	110	<5	<5	3.2
11	5447	47.2	2.88	660	110	72	<5	<1	<0
12	5454	59.9	2.77	810	110	9.7	<5	<1	<0
16	5527	71.1	2.02	890	45	1.2	<5	<1	<0
17	5533	74.2	3.90	690	31	110	<5	<1	<0
18	5541	66.0	3.33	270	26	72	<5	<1	0.3
20	5611	65.9	7.80	1730	33	190	<5	2.9	<0
21	5618	67.8	3.28	810	130	110	<5	<1	<0

From this data, the following observations can be drawn:

- there are significant levels of calcium in solution (2.0 - 8.3 g/L Ca, equivalent to 0.46-1.92% CaCl_2), indicating that it will be necessary to have a calcium removal step in the flow sheet;
- the iron and manganese levels are such that it will be necessary to include oxidation and precipitation to reduce them to acceptable levels; and
- nickel and boron levels are sufficiently low as not to be a problem. However, because of the various recycle streams, especially within the crystallisation area, ion exchange circuits will, nevertheless, be installed to remove these impurities. These circuits will be very simple due to the low levels, and will act as insurance against potential build-ups.

6.3.1.5 Testwork Conclusions - Leach

Based on the results of the scoping leach tests, the following was concluded:

- a magnesium extraction of 90% is achieved in a batch residence time of one hour.

- the preferred operating temperature is 95°C.
- the acid to ore ratio should be based on sufficient free acid being present to extract 95% of the contained magnesium. It should be noted that this will not be a design criterion, rather the acid/ore ratio will vary based on the terminal pH of the second leach reactor in a continuous train.
- particle size in the range 6 to 20 mesh has no effect on leach kinetics or extraction.
- iron, manganese and calcium will have to be removed from the leach solution.
- there was no significant difference in the leach behaviour of the four core samples tested.

6.3.1.6 Observations from Neutralisation and Purification Tests

The following observations may be made from the laboratory and bench scale neutralisation and purification pilot testwork completed by ORTECH and reviewed by Hatch:

- high magnesium extraction (generally > 90%) was achieved in the one-hour leach.
- using magnesite (MgCO_3) as the initial neutralising agent was not very effective, requiring a much higher addition on a Mg basis compared to magnesia (MgO), with a consequent low extraction of magnesium (5-56%). It is, therefore, proposed that only MgO be used for neutralisation.
- Mg extraction from the MgO was generally 100%.
- the MgO dosage rate was in the range 0.3-3.0% on a weight basis of the amount of magnesite added to the leach. These levels are extremely low. It should be noted that these values are for using high grade material. For lower grade ores, the requirement for MgO will increase proportionately to the increase in iron levels.
- no chemical advantage was seen whether or not the intermediate leach solids were removed.
- impurity removal was generally very effective, although some problems were experienced with erratic manganese removal (possibly due to over oxidation to permanganate). Nevertheless, the objective of demonstrating that these impurities could be reduced to low levels was adequately achieved.
- the solution seemed to have a preferred oxidation potential between 1,030 and 1,080 mV (versus the saturated calomel electrode), irrespective of sparging time. This will have to be confirmed during a continuous run.
- it was noticed after the production of the bulk samples, that the pH of the final filtered solution decreased with time, and there was a noticeable smell of chlorine. This is a direct result of carrying out the neutralisation

and oxidation at 70°C which is addressed in the review of testwork elsewhere.

6.3.1.7 Testwork Conclusions - Neutralisation and Oxidation

- The use of magnesite to effect initial neutralisation proved unattractive.
- The controlling redox potential for the oxidation of iron and manganese should be investigated more precisely in a continuous run.
- The reactive magnesia requirement was shown to be very low, in accordance with the generally low levels of iron present. Higher levels of iron will require higher addition rates of magnesia. The effective utilisation rate, which is very low, proved to be difficult to determine with any confidence in the small scale tests, will have to be determined in the continuous run.
- Residence time for oxidation was not rigorously investigated, with one hour established for oxidation and one hour established for neutralisation. This figure was chosen since a batch residence time of one hour was selected for the leach, and in a continuous circuit, each reactor will have the same residence time. It was, however, determined from these tests that both oxidation and neutralisation could be adequately effected within this residence time.
- The oxidation and neutralisation were conducted at 70°C. Whilst this gave acceptable results in terms of operating the tests on the small scale, the slow release of chlorine noticed after completion of the test means that the temperature will have to be raised to that of the leach, where chlorine has negligible solubility in the brine.
- The tests showed that all impurities, with the exception of calcium, could be effectively removed to their specification limits. Other calcium removal techniques need to be addressed.

6.3.1.8 Bulk Leach Production

The objective of this part of the program was to produce 400 L of 20 wt% MgCl_2 pure brine, required for the crystallisation tests at K-Utec, Germany. The ore used for the bulk leach was a mix of AR2, 8, 9 and 10 high grades. Additional scoping tests were undertaken to ensure that there were no significant differences in the behaviour of samples from each of the AR 2, 8, 9 and 10 drill core.

The following observations can be made from the bulk leach testwork undertaken by ORTECH and supervised by Hatch:

- some foaming occurred during addition of ore to the acid. At the larger scale, the mixing certainly was not as intense as in the bench scale tests, and may have contributed to this phenomenon. However, it confirms that the issue of foaming will have to be addressed during Stage II tests.

- the first batch was carried out with minus 20 mesh material. However, it was found that the discharge port of the leach vessel (centre bottom drainage - not a configuration that would be used commercially) was plugged with material that settled at the end of the test. Consequently, for the remaining two batches, minus 48 mesh material was used, which largely alleviated the problem. Bottom drainage will not be used in the continuous run, nor in the commercial plant, and hence the coarser feed will be used.
- Difficulties were observed with the initial filtration of the final slurries. The discharge solution from the filter press was not clear and contained a fine red material (ferric hydroxide). A second filtration was carried out with the press having a pre-coat of dicalite. This resulted in a final solution that was clear and colourless.
- Based on the analyses shown in Table 11 below, Hatch approved shipment of the solution to K-Utec, Germany.

Table 11: Summary of Results of Bulk Leach Tests

Test No.	% Mg extract from ore after 1 hr	Mg extract from MgO*	g MgO per g ore	Final Solution Analysis (mg/L)						
				Mg	Ca	Fe	Si	Mn	Ni	B
BL 1				63,300	2,970	<1	15	<1	<1	<5
BL 2				64,100	2,940	<1	20	<1	5	<5
BL3	-			65,100	2,840	1.2	14	<1	2.3	<5
BL 1, 2 and 3	88	77	0.018	64,167	2,917	1	16	<1	3	<5

* Average between pH adjustment and neutralisation

6.3.1.9 Calcium Removal

Significant levels of calcium in solution were observed during the test program which necessitated a calcium removal circuit in the flow sheet. Although part of the Stage II test program, Hatch requested ORTECH to carry out a scoping review program to test alternate removal processes. Two options were tested - carbonation and solvent extraction.

Although it is practised industrially, carbonation is a very difficult process to control, and incurs a significant loss of magnesia, which is a relatively expensive reagent. ORTECH's scoping test confirmed this.

Three reagents were tested for solvent extraction, of which D2EHPA (di 2-ethyl hexyl phosphoric acid, a very common reagent) showed promise. The scoping tests showed that extraction at a pH≈1 has the potential to remove most of the calcium with very little co-extraction of magnesium.

6.3.1.10 Check Analysis

Hatch requested that ORTECH send random samples to an alternate analytical laboratory for check analyses. Overall, given the instability of the solutions (the check analyses were carried out after the solutions had aged for at least a week), it was concluded by Hatch that the analyses generated by ORTECH are reliable at the level of acceptable accuracy.

6.3.1.11 Phase I Testwork - Conclusions

Phase I testwork clearly demonstrated that the TasMag ore is amenable to leaching to give a pure magnesium chloride brine suitable for proceeding to magnesium metal production. Specifically, a set of process design conditions can be extrapolated to be used as the basis for future continuous testing. Design parameters established are:

- leach temperature of 95°C, with a batch residence time of one hour;
- ore particle size minus six plus twenty mesh;
- acid to ore ratio initially based on sufficient acid being present to extract 95% of the contained magnesium. This ratio is controlled by the pH value in the second leach reactor to allow for variations in the head grade composition;
- oxidation with chlorine gas followed by neutralisation with reactive magnesia to pH 6.0. This to be at 95°C, with each a one hour residence time per stage; and
- as removal of the leach solids is not critical, it is proposed that the neutralisation solids (pH 6.0) be returned to the final leach reactor to use unreacted magnesia, resulting in step for the discharge of solids.

6.3.2 Carnallite Crystallisation Testwork Programme - Kali-Umwelttechnik (K-Utec) (Phase II)

6.3.2.1 Introduction

Approximately 400L of 20% MgCl_2 solution from the ORTECH Phase I bulk testwork programme was provided to K-Utec to produce a carnallite crystallisate.

A test program consisting of eight experimental cycles was carried out by K-Utec in Sondershausen, Germany.

6.3.2.2 Methodology and Solution/Spent Electrolyte Preparation

The MgCl_2 solution received from Canada was strongly acidic ($\text{pH} \approx 1$), containing chlorine and yellow in colour.

As a first step the free chlorine was destroyed by adding excess (double of the calculated amount) of hydrogen peroxide (1 ml 30% H_2O_2 per litre of MgCl_2

solution). Following the hydrogen peroxide additives, the MgO solution was agitated with pure $\text{Mg}(\text{OH})_2$ to adjust the pH to approximately 6.5.

During neutralisation a relatively stabile yellow foam was created and a yellow, brown sludge was built consisting of precipitated heavy metals, principally iron and excess $\text{Mg}(\text{OH})_2$. The precipitated sludge was filtered leaving a nearly clear and colourless MgCl_2 solution. No post precipitation was noted and the chlorine odour disappeared.

The spent-electrolyte was delivered in metal drums from UTI. The received spent-electrolyte lumps were crushed to a particle size of < 8 mm and the milled material was sampled and analysed.

6.3.2.3 Preliminary investigations for the production of extract solution from spent electrolyte and 20% MgCl_2 solution

The production of carnallite requires a raw material feed that is of the same molar ratio of MgCl_2 : KCl as is present in the carnallite crystal.

Spent-electrolyte was dissolved in a solution of 20% MgCl_2 , KCl, NaCl and H_2O (at near boiling point temperature) and fed into the crystallisation process.

During investigations it was determined that the operating temperature has to be at $95^\circ\text{C} \pm 5^\circ\text{C}$. At these temperatures, it will not be possible to carry out solids separation by sedimentation or to carry out vacuum filtration. However good results were obtained by pressure filtration with a filtration pressure of 2 bars. Testwork also found that a small amount of water is necessary for dilution of the MgCl_2 brine so as to avoid crystallisation of KCl in the filter cloth during the filtration process.

6.3.2.4 Evaporation of MgCl_2 solution for production of carnallite mother liquor

The production of carnallite requires approximately 3 to 3.5 litres of carnallite mother liquor for each kilogram of crystallisate. This mother liquor is not consumed in the process and is used again in the following cycle. For the first cycle the solution has to be synthesised by concentrating a part of the 20% MgCl_2 solution in an evaporation apparatus. In this way, 70 litres of concentrated MgCl_2 solution were produced. The evaporation process was not difficult and no foam or encrustations were produced. The condensate was colourless, nearly odourless and weakly acidic.

6.3.2.5 Production of extract solution

The amount of KCl-saturated hot extract solution required for the production of carnallite was produced in a separate experimental step. The necessary amount of extract solution for the 8 carnallite cycles was prepared in four batches by dissolution of ground spent-electrolyte in hot 20% MgCl_2 solution

and some water at a dissolution temperature of about 100°C in a 50L enamel agitator kettle.

After reaching the predetermined dissolution temperature the hot solution was discharged together with the small amounts of insoluble residues at short periods in portions of 5L each and immediately filtered by a PTFE pressure nutsche with a feed volume of 5.5L. The filter cloth was made of glass-fibre paper. At a filtration pressure of 2 bar, each filtration took less than 1 minute. The filtration paper was not removed at the filtration of a charge of extract solution. In this way, a clear and colourless hot solution and a dissolution residue resulted. The hot solution of one charge was divided into two halves by weight during the filtration and stored in two well lockable plastic tanks until it was used for the carnallite production.

The dissolution residue consisted of some crumbs of KCl/NaCl which were not fully dissolved and a grey dissolution residue of insoluble components of the spent-electrolyte along with a small quantity of metallic magnesium. Nevertheless, the metallic magnesium caused the development of hydrogen that restricted sedimentation of the finest particles and additionally, it was responsible for the creation of gaseous hydrogen in the gaseous atmosphere of the dissolution tank. The small development of hydrogen did not disturb either the hot dissolution process or the filtration process of the hot solution.

6.3.2.6 Production of carnallite in cycles

Carnallite was produced in eight cycles according to VAMI specifications. The production was carried out in a 50L enameled agitator kettle with a heating jacket, PTFE-agitator and bottom outlet.

Each cycle was carried out as follows:

- Step 1:* Filling with carnallite mother liquor from the preceding cycle and heating to about 60°C.
- Step 2:* Addition of the extract solution in the form of KCl dissolution mixture created by cooling.
- Step 3:* Heating to boiling temperature and evaporation of the calculated amount of water (measuring as condensate).
- Step 4:* Cooling from about 127°C to room temperature by slowly agitating during the night on the way to crystallisation of the carnallite.
- Step 5:* Discharging the crystallisate-solution mixture. Decantation of the liquid from the coarse crystallisate and filtration of the crystallisate on a vacuum nutsche.
Filtration of the mother liquor to separate possible fine particles before it is re-used.
- Step 6:* Dewatering of the carnallite crystallisate on a sieve cup centrifuge to < 5% of adhering solution.

Step 7: Sampling and analyses of the products and residues created.

6.3.2.7 Testwork Results - Chemical composition of raw materials, intermediate products, residues and final products

The following materials and material compositions were analysed by K-Utec:

- 20% MgCl_2 brine received from ORTECH;
- spent-electrolyte received from UTI;
- produced dechlorinated neutralised solution;
- produced neutralisation sludge;
- produced synthetic mother liquor for the start of the first cycle;
- produced extract solution of the spent electrolyte and the leaching residue; and
- carnallite crystallisates from all 8 cycles.

The composition of the extract solutions produced are shown in Table 12.

Table 12: Chemical composition of raw materials, intermediate products, residues and final products

	Extract Solution 1		Extract Solution 2		Extract Solution 3		Extract Solution 4	
	g/L	g/g H_2O	g/L	g/g H_2O	g/L	g/g H_2O	g/L	g/g H_2O
MgCl_2	211.4	255.5	209.8	253.4	212.2	255.0	210.6	254.3
KCl	166.6	201.4	169.2	204.4	161.0	193.5	163.9	197.9
NaCl	50.5	60.9	47.1	56.9	50.2	60.3	48.3	58.3
CaCl_2	8.9	10.8	8.9	10.8	8.0	9.6	8.6	10.4
CaSO_4	0.4		0.2		0.4		0.3	
H_2O	827.3		827.3		832.2		828.3	
Dichte (g/mL)	1,265		1,263		1,264		1,260	
Temp. ($^{\circ}\text{C}$)	95		95		95		95	

At $95 \pm 5^{\circ}\text{C}$ all KCl is dissolved in the MgCl_2 solution. The solution is almost saturated with KCl and the molar ratio KCl : MgCl_2 is at 1 : 1. NaCl saturation was not reached.

A mixed sample of all 4 extract solutions was analysed again for heavy metals and other by-products. The insoluble residue obtained with hot pressure filtration was also analysed. Additionally in the residual sludge of the spent-electrolyte the approximate content of Mg metal was determined by investigating the dissolution residue in a gas developing apparatus at a pH-value of ≈ 1 . From the release of H_2 gas (860mL of H_2 -gas per kg of

dissolution residue) the approximate content of Mg metal in the dissolution sludge of the spent-electrolyte was calculated as approximately 0.09%

6.3.2.8 Analysis of carnallite crystallisate from cycles 1 through 8

Eight cycles of approximately 12kg each of crystallisate were produced. The carnallite crystallisate analysis is summarised in Table 13.

Table 13: Carnallite Crystallisate Analysis

Cycle #	Carnallite	NaCl	Haftlosung	KCl
1	91.4%	4.4%	3.9%	-
2	90.8%	6.3%	3.3%	-
3	89.5%	6.6%	3.9%	-
4	89.0%	6.7%	3.0%	-
5	88.8%	6.3%	4.0%	-
6	86.6%	8.2%	4.3%	-
7	90.4%	6.6%	4.1%	-
8	90.1%	5.4%	3.6%	0.3%

The carnallite crystallisates were wet screened using their mother liquor. An average sample of all 8 carnallite crystallisates and carnallite from the 4th cycle was dry screened after an appropriate preparation with alcohol and acetone. The screen analyses showed an average particle diameter of $d = 0.9$ mm.

The single screen fractions of the average carnallite sample of carnallite are summarised in Table 14.

Table 14: Carnallite Screen Fractions

	1.0-2.0 mm	0.8-1.0 mm	0.5-0.8 mm	0.25-0.5 mm	< 0.25 mm
NaCl	2.6	2.7	4.6	23.0	76.2
KCl	26.3	26.1	25.8	20.8	6.3
MgCl ₂	33.8	3.6	33.1	26.7	7.8
Carnallite	98.2	97.5	96.3	77.7	23.2

It can be seen from Table 14 that the rock salt is crystallised mostly as a fine-grained salt. Approximately half the total quantity of NaCl can be found in the fraction of < 0.5 mm but this fraction is only about 10% of the total mass of the carnallite.

6.3.2.9 Analysis of carnallite mother liquor of the cycles 1 to 8

The filtrates obtained during separation of the carnallite crystallisate by filtration and centrifugation, were mixed and used again for the following cycle. As a result, the impurities contained in the solution will be slightly enriched.

The composition of the carnallite mother liquors 1 to 8 (CML 1 to 8) are shown in the Table 15.

Table 15: Composition of Carnallite Mother Liquor

	Dichte g/mL	Temp °C	MgCl₂ g/L	CaCl₂ g/L	KCl g/L	NaCl g/L	H₂O g/L	S g/L
CML0	1,280	25	375.0	12.0	0.1	0.3	892.5	0.2
CML1	1,302	25	395.0	15.0	7.0	11.6	873.0	0.3
CML2	1,302	25	388.3	20.8	6.7	12.0	874.0	0.2
CML3	1,300	26	384.3	26.5	6.5	10.7	872.0	0.3
CML4	1,303	25	380.4	30.0	6.5	11.4	874.3	0.3
CML5	1,302	25	372.2	36.1	6.9	11.9	874.4	0.3
CML6	1,306	25	373.7	39.7	6.7	10.4	875.1	0.3
CML7	1,306	25	369.8	40.8	7.4	10.4	870.0	0.3
CML8	1,305	25	364.3	46.3	7.8	12.0	874.3	0.2

6.3.2.10 Analysis of fine salt from the carnallite mother liquors 1 to 8

60 to 70 g of fine salt were separated by filtration as a residue during the purification of the carnallite mother liquor. An average sample of all 8 fine residues was analysed. From this analysis, the composition of the fine salt was estimated as:

- 65% carnallite
- 20% NaCl
- 1-2% CaSO₄ * 2H₂O
- 13 -14% adhering solution

6.3.2.11 Production of carnallite

Eight carnallite charges in total were produced. Dewatering of the carnallite was carried out by decanting, filtration on a vacuum nutsche and centrifugation.

The majority of the carnallite mother liquor for the usage in the following carnallite cycle was obtained in the first two steps. The carnallite mother liquor obtained in the long lasting centrifugation step was normally used not in the next but in the following cycle. Any shortfall in mother liquor was made-up by use of the starting mother liquor and appropriate amounts of KCl.

6.3.2.12 Evaluation of results

The following conclusions were reached by K-Utec.

- The testwork undertaken has proven the theoretical concept of the test program in all items. By using the chosen technology it is possible to produce a carnallite crystallisate, within the required specification, from a 20% MgCl_2 solution and spent electrolyte via an intermediate step of a KCl-MgCl_2 extract solution by evaporation and cooling crystallisation.
- It is possible to produce a pure neutralised MgO_2 solution from acidic and chlorine containing MgCl_2 solutions without the use of any additional substances. With this procedure iron and copper and partly zinc will be precipitated, but not manganese and nickel.
- The spent-electrolyte can be dissolved in hot MgCl_2 solution at a dissolving temperature of about 95 to 100°C. A small quantity of hydrogen gas is developed during this process. Hot filtration of the solution by a pressure filter at about 2 bar creates a high flow rate, without plugging the pores of the filter cloth by crystallisation,
- It is possible to settle (by gravity) the coarse particles of the insoluble leach residue before the pressure filtration. It is not possible to obtain a clear extract solution only by sedimentation.
- The evaporation of a required water quantity from the mixture of re-circulated carnallite mother liquor and fresh extract solution is possible up to temperatures of 127°C. By this procedure no crystallisate will be created. The condensate produced is colourless, nearly odourless and shows a pH-value between 6 and 7 and could be used as process water.
- The cooling crystallisation with precipitation of visible crystals commences at around 110°C and is completed at room temperature. Below about 50°C the rate of increase in the amount of crystallisate reduces. The crystallisate contains a mixture of coarse carnallite and some finer NaCl .
- The carnallite crystallisate produced contains all the characteristics specified by UTI/VAMI. The required KCl-MgCl_2 molar ratio in the carnallite crystallisate can only be adjusted indirectly. The best method is the addition of a calculated amount of potassium chloride as KCl 99 . The distribution of the particle size results in a medium particle size of about 0.7 to 0.9 mm and largely meets the VAMI requirements.
- The carnallite crystallisate obtained after the 8th cycle shows a similar particle size distribution to those of the other cycles.
- The enrichment of CaCl_2 in the carnallite mother liquor is nearly steady for each cycle. The maximum CaCl_2 content that can be tolerated in the

solution has to be determined by further special investigations at much higher levels of the CaCl_2 content in the circulating solution of the carnallite crystallisation.

- The intake of CaSO_4 together with the 20% MgCl_2 solution was relatively low. There were no signs of CaSO_4 crystallisation up to the 5th cycle. Analysis of the carnallite crystallisation of all 8 cycles did not show a significant increase of the CaSO_4 content in the crystallisate.

The fine particles separated from the carnallite mother liquor were analysed together after the 8th cycle. The analysis of this fine salt shows the following composition:

- 85 % carnallite + NaCl
- approx. 13 % adhering carnallite mother liquor
- 1 to 2 % $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$

Because CaSO_4 was not introduced with the raw material it is assumed to be created by crystallisation. There are no signs of CaSO_4 crystallisation in the hot part of the process and no encrustations caused by CaSO_4 were detected. The gypsum crystallisation which was detected appears obviously under the test conditions of a discontinuously cyclic course of the process at the cool parts of the apparatus. The precipitated gypsum is very fine and can easily be separated from the coarse grained carnallite-NaCl crystallisate by elutriation.

- The boron contained in the 20% MgCl_2 solution did not lead to an excess of the allowed boron concentration in the carnallite crystallisate in the course of 8 cycles. Nevertheless, in the course of repeated cycles an enrichment of boron appeared in the mother liquor.
- The bromide contained in the carnallite crystallisate was very low, with a maximum bromide concentration in the solution of about 70-mg of bromide per litre after the 8th cycle. It was not established whether the bromide limit will be exceeded in continued circulation of the mother liquor.
- The enrichment of soluble components (CaCl_2 , boron, bromide) in the carnallite mother liquor requires that either an appropriate discharge be provided or these components have to be removed from a side stream of carnallite mother liquor. The regenerated mother liquor can be taken back again into the process. This procedure has to be investigated in further experimental programs.

The possible methods that should be considered to remove the soluble components in the carnallite mother liquor include:

- Removal of CaCl_2 by adding MgSO_4 as $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ or $\text{CaSO}_4 \cdot 0.5\text{H}_2\text{O}$.

- Removal of boron by ion exchange with boron specific resins.
- Removal of bromide by, oxidation to elementary bromine and stripping.

The NaCl is readily separable from the coarser carnallite by classification and can be added to the carnallite at a later stage.

The testwork has clearly shown that NaCl crystallises simultaneously with the carnallite but in different particle sizes and particle shapes. Carnallite always appears coarser than the NaCl.

Carnallite forms crystals of rounded crystal shapes or good crystallised single crystals, whereas NaCl builds smaller cubic crystals or crystal agglomerates of several NaCl-cubes.

6.3.3 Metal Production Testwork - UTI / VAMI (Phase IV)

Carnallite crystallisate produced by K-Utec was dispatched to UTI/VAMI laboratories and research centre in the Ukraine and St Petersburg, Russia.

Utilising the patented UTI carnallite dehydration process coupled with the VAMI electrolysis process, twelve magnesium metal ingots were produced (Figures 7 and 8). Owing to the confidential nature of the patents and the rights to those patents held by Indcor, details of the testwork undertaken by UTI and VAMI are not disclosed in this Memorandum.

Recipients of the Memorandum who provide a Letter of Intent and are selected to review the data room will be provided with additional information relating to the UTI/VAMI process.

Figure 7 – TasMag Drill Core & Magnesium Metal Ingots



Figure 8 – Magnesium Metal Ingots



6.4 Technology

By an agreement between Indcor and UTI/VAMI, Indcor has secured an option over the exclusive rights throughout Australia and New Zealand to UTI/VAMI know how and technology in relation to the production of magnesium metal. The rights exist for the life of the TasMag Project and are exclusive for 20 years and thereafter are royalty-free non-exclusive rights.

Option fees totalling US\$1.35 million have been paid to date. The license fee exercise price of US\$20 million in aggregate is payable on the plant achieving performance criteria. The new management of Indcor are presently re-negotiating an extension to the option period, and potentially, the option exercise price.

UTI/VAMI technology is used as a basis for construction for process facilities that currently produce 140,000 tpa magnesium metal out of a total world market of 450,000 tpa. These facilities are based in:

- the former Soviet Union (all 5 plants);
- in China (1 plant);
- USA (1 plant); and
- Israel (Dead Sea plant).

In addition, some elements of this technology are being used in Noranda's Magnola plant in Canada which is currently being commissioned.

Table 16 below summarises the involvement of UTI / VAMI in existing magnesium metal plants.

Table 16: UTI / VAMI design input to Magnesium Plants

Project	UTI / VAMI Input	Raw Material	Start-up
Zaporozhie Ti/Mg complex, Ukraine	Complete UTI / VAMI technology	Dehydrated carnallite & recycled MgCl_2 from Ti plant	1935
Solikamsk Mg Plant, Russia	Complete UTI / VAMI technology	Synthetic carnallite	1936
Berezniki Ti/Mg complex, Russia	Complete UTI / VAMI technology	Synthetic carnallite & recycled MgCl_2 from Ti plant	1943
Fushun Mg Plant, China	UTI / VAMI Mg production technology	Magnesite	1957

Ust-Kamenogorsk Ti/Mg complex, Kazakstan	Complete UTI / VAMI technology	Synthetic or dehydrated carnallite & recycled MgCl_2 from Ti plant	1965
Oriana Mg Plant, Kalush, Ukraine	Complete UTI / VAMI technology	MgCl_2 solution converted to carnallite	1969
American Magnesium Company, Snider Texas, USA	UTI / VAMI MgCl_2 electrolysis technology		1972
Dead Sea Mg Plant, Israel	Complete UTI / VAMI technology	Carnallite	1997
Magnola Mg Plant, Canada	UTI / VAMI Mg refining equipment (continuous refining furnace)	Asbestos Tailing	2000

6.5 Process Flowsheet

Indcor in conjunction with BHPE, Hatch, UTI/VAMI and other process consultants and advisors have developed detailed process diagrams. The process has been modelled by Hatch using Metsim for material flows, water and heat balance. This information will be made available to recipients of the Memorandum who provide a Letter of Intent and are selected to review the data room.

In particular, Hatch has completed process flow diagrams (PFDs) based upon the ORTECH and K-Utec testwork, UTI/VAMI specifications and Metsim outputs. PFD's have been produced for the following areas:

- feed preparation;
- leaching;
- purification and neutralisation;
- spent electrolyte granulation;
- carnallite crystallisation;
- carnallite dehydration;
- HCl absorption;
- electrolysis - cells;
- electrolysis - chlorine handling;
- pure magnesium refining and casting;

- alloy preparation, refining and casting;
- liquid reagents and utilities;
- gaseous reagents and utilities; and
- solid reagents and utilities.

A simplified overall block model diagram of the Indcor process flowsheet is provided in Figure 9. The process flowsheet and process description is summarised in paragraph 6.2 of this Memorandum.

Hatch have completed on behalf of Indcor a Technology Patent Search and assembled a data base of Magnesium Process Patents worldwide.

6.6 Engineering Design

6.6.1 Introduction

The principal target of all feasibility testwork and design undertaken by Indcor is to:

- secure the best possible proven technology on a worldwide basis to incorporate into the TasMag Project;
- obtain representative samples and undertake adequate testwork and process review to establish process viability for the TasMag Project;
- complete, in conjunction with world recognised proven equipment suppliers and engineering consultants, adequate design of each of the major process blocks to establish utility, reagent and infrastructure requirements, generate sized equipment lists for process and mechanical equipment and model material, heat and water balances, complete layout, plans and elevation drawings to confidently estimate capital and operating cost for the project.

Process and Engineering design packages have been completed for all plant areas.

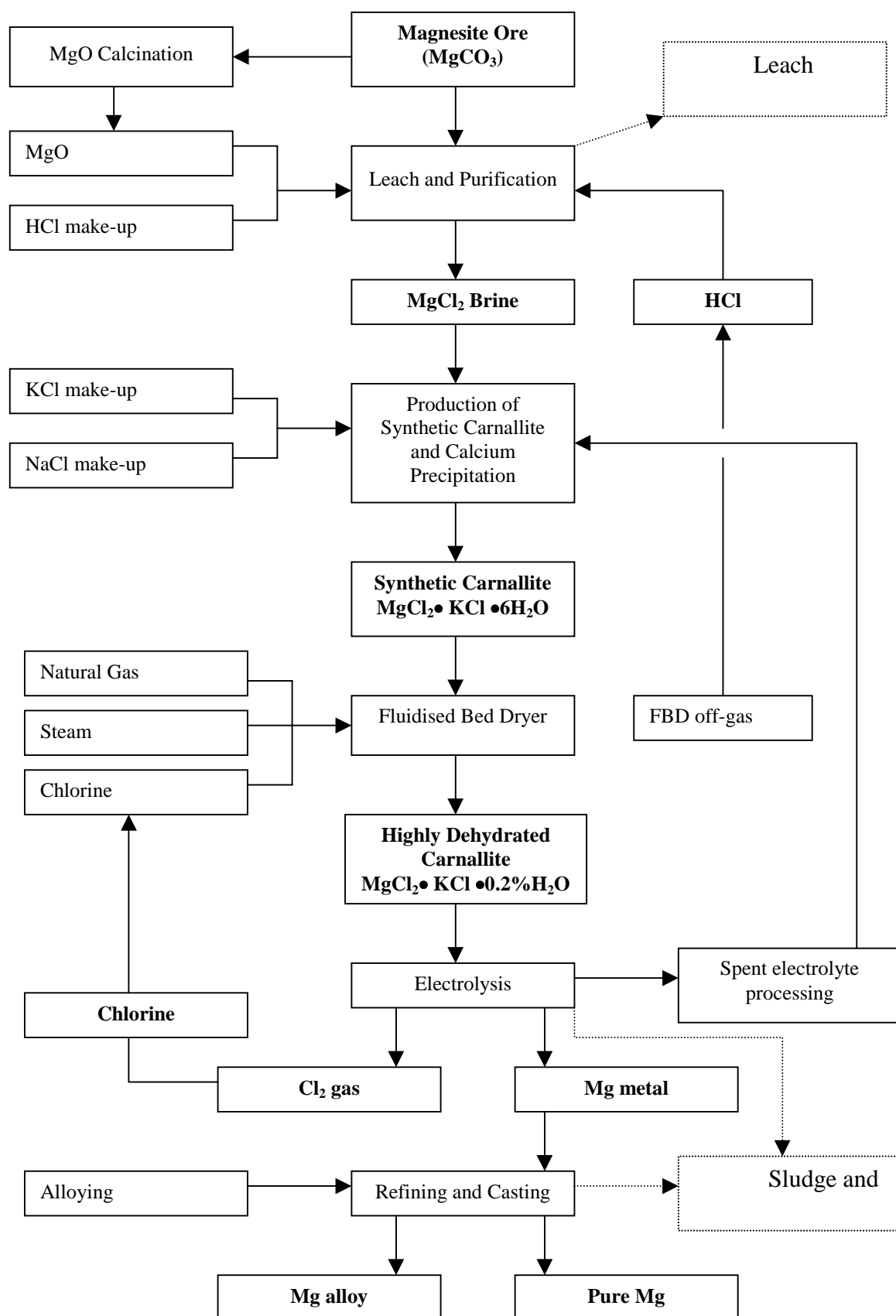
6.6.2 Mine Area

Detailed mine plans for a number of operating scenarios have been completed by Hatch. Scenarios relate to project size, initial resource area to be mined and various stages of development to meet plant feed requirements.

On-site plant infrastructure and facilities have also been designed incorporating road design, stockpile location and sizing, bridge and access requirements for transport of run of mine product together with support facilities and utilities at the minesite.

Design has incorporated environmental requirements recommended by the environmental consultants in liaison with political and community groups.

Figure 9 – Process Flowsheet



6.6.3 Ore Transport

The requirements for ore transport from the minesite have been identified.

Engineering data and infrastructure support facilities have been identified and preliminary designs finalised to enable capital and operating costings to be finalised.

The transport requirements remain flexible to enable differing locations of the final plant location to be assembled as a result of project feasibility data and sponsor preference.

6.6.4 Ore Receival / Preparation

Designs based on the receival of run of mine ore and other reagents have been completed by Hatch. The designs are not site specific and can incorporate rail or truck delivery. Designs for receival, storage and preparation of the ore, consider:

- ore feed crushing and screening of magnesite ore and fine ore storage; and
- calcination of up to 10% of the magnesite feed to produce a high quality MgO product for addition to the process for calcium removal.

6.6.5 Leaching, Purification and Neutralisation

Following the testwork undertaken by ORTECH in Canada, determination of the flowsheet and comprehensive process design data generated, Hatch have completed engineering design for:

- equipment, sizing and selection;
- plant layout;
- elevation drawings;
- material and utility requirements and flow; and
- basic civil and structural design.

Engineering work incorporates proven processes and equipment based on information, drawings and specifications supplied by proven suppliers and contractors.

Subject to minor outstanding testwork to confirm impurity removing process and size the leaching, purification and neutralisation section of the plant will be bid to suitably qualified process engineers and contractors for bankable pricing and process guarantee requirements.

6.6.6 Carnallite Crystallisation

K-Utec in conjunction with Messo, an equipment and process plant supplier based in Germany, and Hatch have completed a comprehensive and detailed design of the carnallite crystallisation and spent electrolyte granulation.

A comprehensive offer incorporating advanced engineering design and drawings has been negotiated with Messo and will be firmed up to a bankable offer with process and engineering guarantees. Some confirmatory testwork will need to be completed to achieve this position.

6.6.7 Carnallite Dehydration and HCl Absorption

UTI in conjunction with VAMI have provided detailed process and engineering data, equipment specifications, equipment sizing data and detailed process drawings and layouts.

Equipment specifications have been discussed with western fabricators and suppliers based upon the detailed engineering drawings provided by VAMI in order to establish viability of construction and costings.

Detailed plans and elevations together with material flow and utility requirements have been completed.

This area of the facility can be designed and formalised to a bankable level in a minimal period once location, size and sponsor requirements are finalised.

6.6.8 Electrolysis, Chlorine Handling, Refining and Casting

VAMI has provided detailed process and engineering data, equipment specifications, equipment sizing together with detailed process drawings and layouts.

Equipment specifications have been discussed with western fabricators and suppliers based upon the detailed engineering drawings provided in order to establish viability of constructions and costings.

Detailed plans and elevations together with material flow and utility requirements have been completed.

This area of the facility can be designed and formalised to a bankable level in a minimal period once location, size and sponsor requirements are finalised.

6.6.9 Reagents, Utilities and Infrastructure

Hatch have completed a final pass review of the specification, sizing and design of the reagent, utility and infrastructure requirements. Layouts, plans, elevations, equipment sizing and specifications have been completed.

A full plant Metsim model has been completed by Hatch.

A comprehensive industry review completed by Hatch of the following process areas has also been completed:

- Assessment of Fluidised Bed Dehydrators;
- Deacon Equilibrium Considerations for Dehydrator Burners; and
- Assessment of Industry Scrubbing Technology.

A preliminary Summary Activity Programme and Master Plan for project implementation has been completed by Hatch.