



**ICON RESOURCES LTD**

**ANNUAL REPORT**  
**Period ending 9 February 2009**

**HENTY ROAD – EL47/2004**

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## **1 ABSTRACT**

Henty Road EL47/2004 is held by South Eastern Resources Pty Ltd (SER), a wholly owned subsidiary of Icon Resources Ltd.

The 23 sq km licence straddles the Henty Road, south of Zeehan in Western Tasmania. The main focus of exploration on the lease has been on the Grieves Siding prospect where zinc mineralisation occurs within the Ordovician Gordon Limestone, mostly as a complex of zinc 'oxides' and in near-surface peat-hosted sphalerite.

Metallurgical test work of bulk peat samples, collected in 2007 and 2008, was continued in an effort to find a cost-effective process for zinc extraction. This work was conducted by Rogers Chemical Engineering and consultant Robert Groenward.

The various tests returned some encouraging results with respect to the percentage of zinc extracted, but in general the usage of re-agents and additives was too high to be economical.

## **2 INTRODUCTION**

This report details the work completed on EL47/2004 for the period ending 9<sup>th</sup> February 2009.

The main focus of work has been the Grieves prospect of carbonate-hosted zinc mineralisation and overlying peat-hosted zinc. The mineralisation lies under swampy button grass plains adjacent to Henty Road, about 12 km south of Zeehan.

## **3 TENEMENT STATUS**

Henty Road (EL47/2004) covers 23 sq km on either side of Henty Road, south of Zeehan in Western Tasmania (Figure 1).

Henty Road was granted to South Eastern Resources Limited (SER) on 10 February 2004 for a period of five years.

In September 2005 Zinifex Australia Limited (Zinifex) signed a one-year option agreement with SER. The option committed Zinifex to expenditure of \$120,000 on the EL and expired on 30<sup>th</sup> September 2006.

In January 2006 Icon Resources Ltd, (Icon), in a related-party transaction, purchased two-thirds of South Eastern Resources and the remaining (unrelated) one-third after Icon listed on the ASX in June 2006.

## **4 TOPOGRAPHY AND ACCESS**

The Henty Road, linking Zeehan and Strahan transects the license, providing ideal access. East of the road the license falls within the Dundas Regional Reserve and west of the road is within State Forest.

In the area of the zinc prospects the Henty Road traverses the Badger River valley. For several hundred metres either side of the road there are low-lying swampy button grass plains overlying weathered limestone. The plains are flanked west and east by escarpments of sandstone quartzite 70-90m high.

## **5 GEOLOGY**

### **5.1 Regional Geology**

The regional geology of western Tasmania was dominated by rifting during the Late Precambrian to Early Cambrian. This was followed in the Early to Mid Cambrian by arc-continent collision, subduction and ultramafic allochthon emplacement. The Dundas Trough developed containing siliciclastics and volcanic derived sediments. To the east and interfingering with the sediments the Mount Read Volcanics were being formed. The base of the Ordovician sequence is typically localized conglomerates and grades up to sandstones and carbonates. The Mid Ordovician carbonates of the Gordon Group are part of a widespread sedimentary basin with variable rates of subsidence. These are the host rocks for prospects within the license (Figure 2).

## **5.2 Local Geology**

The rocks in the license are a conformable Ordovician to Devonian sedimentary sequence overlying Cambrian basement. Cambrian basement rocks occur in the southern part of the license and are mainly interbedded siltstone and sandstone with some magnetic mafic igneous rocks.

The basal unit of the Ordovician sequence is the pink, silicified and coarse grained Owen Conglomerate. Within the lease it is overlain by siliceous fine grained Moina Sandstone. To the east of the lease the Moina was faulted-out, or deposited only to the west as a result of syn-depositional faulting.

Overlying the Moina Sandstone is Ordovician Gordon Limestone, assumed to be at least 700m thick within the licence. The dark grey limestone contains various facies including a basal bioclastic argillite and oolite which has been pervasively dolomitized and sideritized.

The Lord Siltstone, a fine grained argillaceous unit, forms a marker throughout the Gordon Limestone. An Upper Dolomite unit is recognized in the Zeehan area.

There are occasional outcrops of limestone visible in road cuttings and the limestone has been strongly weathered to a depth of several hundred metres. The top 20m is highly weathered to form an undulating surface that has been infilled by organic material (peat) and "slumped" blocks of limestone. The peat is overlain by up to 8m (usually <2m) of hard Moina Sandstone gravel that has shed off the escarpment from the southeast and a surface veneer of swampy peat.

The Gordon Limestone is conformably overlain by Silurian Crotty Quartzite that dips steeply northeast.

The Owen Conglomerate, Moina Sandstone and Crotty Quartzite form 70-90m high escarpments either side of the low-lying weathered limestone.

The sediments are folded around axes trending NW and cut by a series of NW-trending faults (Figure 3).

## **6 PREVIOUS EXPLORATION**

Previous exploration has been summarised in a previous annual report (Lewis, 2006).

## **7 MINERAL OCCURENCES**

This summary was partly compiled from Russell and Tear, 1996.

Previous explorers have interpreted zinc-lead mineralisation within the Gordon Limestone to be pre-Devonian in age and unrelated to the Tabberabberan Orogeny (ie, in contrast to most of the Zeehan silver-lead field). The Gordon Limestone was deposited at the end of a period of major tectonic activity that produced the Mount Read Volcanics. Hydrothermal systems may have continued to emit metals into the Gordon Limestone via basement and syn-sedimentary faults.

Five zones within the Gordon Limestone have been recognised as targets for zinc-lead mineralisation.

- Stratabound at the lower limestone-sandstone contact. This zone is characterised by carbonaceous and/or ferruginous clays less than 50m thick above the contact with the Moina Sandstone. It can be overlain by a massive siderite zone less than 25m thick.
- Stratabound at the upper limestone-quartzite contact. This zone is typically within the Upper Dolomite Unit.
- Stratabound within a brecciated (possibly syn-sedimentary) and/or sideritized unit in the middle of the limestone.
- Structurally controlled discordant mineralisation. This can occur throughout the limestone sequence and may be the late-stage filling of brittle fractures.
- Surficial peat hosted: eg, at the Grieves prospect, the peat layer beneath the sandstone gravel contains significant values of zinc in zones directly overlying the limestone-hosted oxide mineralized zone. Recent work has shown the metals occur within the clays as fine colloform sphalerite and galena, apparently actively depositing within the organic carbon and “growing” in-situ (Purvis, 2006).

## 7.1 Grieves

Mineralisation at Grieves consists of two zones:

- Near surface peat-hosted sphalerite overlying the Gordon Limestone, at the base of the escarpment formed by Moina Sandstone.
- Sphalerite and minor galena partially oxidised to zinc oxides, carbonates and silicates to a depth of 100 to 200m. The best grades to date are from the lower limestone/sandstone contact. The peat resource occurs above this zone.

A JORC-compliant Inferred Resource was calculated by Tracie Burrows in December 2005 for the peat-hosted zinc of 409, 000t @ 3.9% Zn (Burrows, 2005). The resource is made up of three blocks as detailed below:

Table 1: Surficial Zinc Inferred Mineral Resource (Burrows 2005)

<b>Block</b>	<b>Tonnes</b>	<b>Zn (%)</b>
North	164 000	3.2
Central	65 000	1.1
South	180 000	5.6
<b>Total</b>	<b>409 000</b>	<b>3.9</b>

The above assumed a density of 1.9t/m<sup>3</sup> (i.e. the density of dry clay). This tonnage decreases to 337,000t using a value of 1.4t/m<sup>3</sup>, (i.e. the average wet or in-situ value of the Zinifex pits).

## 7.2 Other Prospects

Other prospects within the lease are summarised in the table below.

Table 2: Prospects within EL47/2004

Prospect	Description	Intercepts (% Zn)	
South Grieves	Middle zone of Gordon Limestone; <20m vertical depth	ZWG1	11.8m @ 6
		ZWG22	0.8m @ 17.5
		ZWG26	1.9m @ 7.3
		ZWG26	1m @ 6.9
Myrtle	Middle zone of Gordon Limestone, associated with a dolomitization	ZM1008	3m @ 6.7
		ZM1008	6m @ 4.3
		ZWM18	7.1m @ 2.4
		ZM185	0.6m @ 14.9
Baura	Upper dolomite unit	ZG402	2.5m @ 3
Firewood Siding	Upper dolomite unit	ZF37	10m @ 0.38
Rose Valley	Silicified carbonate breccia	defined by 14 wacker samples, with max of 242ppm Zn	

## 8 WORK COMPLETED

### 8.1 Metallurgical Testwork – Rogers Chemical Engineering Pty Ltd

Rogers Chemical Engineering Pty Ltd (RCE), namely Frank Rogers and Dr. Neil Allen, was contracted to determine if the zinc-in-peat could be economically extracted, and if so to proceed to the construction of a pilot plant. RCE completed various metallurgical tests using bulk 30-40kg samples that were collected from the peat horizon at Grieves prospect in early 2007.

The tests used a wide range of acid levels, time periods, and temperature and were completed under both static and agitated conditions.

The main conclusions from a report completed in November 2007 (Appendix 1) are summarised below:

- High variances in the percentage of recovered zinc;
- Zinc is dissolving and re-forming other silica-based complexes; and
- A high consumption of acid.

Test work continued in January 2008 to see if by controlling the pH it was possible to

- to minimise the consumption of acid; and
- promote silica into solution and remove it.

Minimal amounts of sulphuric acid were used to achieve a pH of 2 required to keep any soluble silicon in solution. These tests indicated an approximate 15% loss of zinc with the silicon removal.

Test notes and reports by RCE and Dr Allen are included in appendix 2.

## **8.2 Metallurgical Testwork – Robert Groenewoud**

Between late 2007 and March 2008, Robert Groenewoud was contracted to conduct further tests to replicate those by RCE and to determine zinc extraction rates at various acid concentrations. These tests were designed to determine the following:

1. Will leaching at 90°C and 20g/L acid result in a high zinc recovery in a reasonable time?
2. Will the recovery of acid by pre-leaching the ore with ion exchange bleed result in excessive zinc loss, or create a high recirculating load of aluminium or silica?
3. Is the ion exchange specific for zinc without passing elements, which affect the efficiency of electrolysis, or which affect the quality of the zinc?

Three tests were completed on Test#95 by RCE (details of the original test are including in appendix 1):

- Test 1: Repeat test #95
- Test 2: As per test 1, without stage1 and the wash stage
- Test 3: As per test 2, but at 60°C

The purpose of Tests 1 and 2 were to determine if stage 1 and the wash stage in the RCE tests are necessary to achieve the high zinc recovery. Tests 2 and 3 were designed to determine if the same extraction can be obtained at a higher temperature, but in a much shorter time. All tests were to include analysis for Zn, Fe, Mg, Al, Ca, Pb, Mn and SO<sub>4</sub>.

The tests were completed by Analytical Service Tasmania and the report is included in appendix 3.

The results from test 3 indicate that the extractions of Al, Fe, and Mg were similar to those in test 2. The extraction of zinc was approx 60% based on both the solution and solids analysis which is a considerable improvement on the extraction of zinc in test 2. Acid usage was 429kg/tonne ore.

Notes and communication from Robert Groenewoud is included in appendix 4.

## **9 ENVIRONMENTAL**

No ground disturbing activities were completed within the reporting period.

## 10 EXPENDITURE STATEMENT

Total expenditure on EL647/2004 for the year period 1<sup>st</sup> January until 31<sup>st</sup> December 2008 is \$40,731.

	\$
Mineral Processing	11 829
Geochemistry/Assays	1 017
Geophysics	1 921
Geology	6 533
Consultants:Metallurgy	4 675
Field Expenses/Consumables	84
Tenement Administration	306
Travel/Accom/Vehicles	1 613
Employee/Office Costs	9 050
Subtotal	37 028
10% Admin	3 703
<b>Total</b>	<b>\$ 40 731</b>

## **11 REFERENCES**

Burrows, T. 2006. Estimate of Work Required to Upgrade Resources: Grieves Prospect. Report by AMC Consultants for Icon Resources Ltd.

Lewis, R. 2006. South Eastern Resources Pty Ltd, EL47/2004 Henty Road First Annual Report 2005-2006.

Purvis, J.G. 2006. Sampling and Metallurgical Testing of Surficial Zinc Clays, Grieves, EL47/2004, Western Tasmania, J.G. Purvis & Associates Pty Ltd.

Russell, S.A.J., Tear, S.J. 1996. Annual Report P.E. November 1996 - EL 34/88 - Zeehan No. 2.

# APPENDIX 1



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## **Progress Report on Grieves Siding Testwork for October – November 2007**

### **1. Laboratory Processing.**

Testwork during the processing period was frustrating due to the variances in recovered zinc. A range of acid levels were tried together with various quantities of sodium hydroxide in order to see if consistent results could be obtained. Also the trials were conducted over a range of time periods to see if there was some benefit from allowing long reaction times. The ore was checked in both static and agitated environments again to check for variations. The temperature of the process solution was altered using the exothermic reaction of the higher strength acid together with varying pulp densities.

In summary it appeared that the zinc was dissolved and reformed into other complexes during this process. It is pretty well confirmed from the testwork that the offending complexes are silica based and that the zinc form bonds to or with this silica during the various reagent level tests. Also it became evident that the consequential pH changes had some bearing on this erratic behaviour.

In researching “The Extractive Metallurgy of Zinc”, R. J. Sinclair describes the problem. Although he is looking at a concentrate the same problems appear in the testwork on the ore. The pH of the ore in the ground ranges from 2.4 to 5.5 depending upon its location. This covers the range that extends from the high stability point of orthosilicic acid to its point of least stability. That is, it covers the range where orthosilicic acid is breaking down and precipitating silica from solution. It is reasonable therefore to run tests to see if by controlling the pH to promote the movement of silica into solution it could be removed from complicating the zinc recovery process and reduce acid consumption.

## **2. Acid costs.**

Based on the results to date the acid cost appears to be the greatest impediment to a viable operation. Considerable effort has been undertaken to source other suppliers in Korea and India. We are currently awaiting a response.

Also to give some incentive to acid suppliers we have added the planned requirements of Proto Resources and Alichem Limited which takes the annual requirement to a level approaching 250,000 tonnes, a sizeable contract for any supplier.

The target price for the acid is in the \$50.00 per tonne range for 98% sulphuric acid. Apparently the demand from South America for a fertiliser base is causing the price increase. This however seems in conflict with information received stating that Korean suppliers were looking to subsidise the freight cost in order to get rid of the excess.

## **3. Summary and Recommendations.**

The work during the reporting period was slowed down due to the sudden illness of Dr Neil Allen's wife who was diagnosed with an aggressive cancer. Neil was obviously traumatised by this discovery and so I thought it only reasonable to allow him some time to deal with the situation. Neil has stoically soldiered on with work when he was not at the hospital and it appears that the comprehensive work done over the past two months has shown some light on the reasons for the high acid consumption.

The costs of the works for the period are well below the budgeted \$24,000 for October to \$16,000 and it is recommended that testwork be extended to the end of the current year in order to reach some finality to the process options.

This extension would result in an estimated additional cost of around \$32,000.

We believe that there is a reasonable prospect of success in achieving economic recovery of the zinc. The pH control tests will be critical in this process together with achieving an acid price in the \$50 – 70 per tonne range.

Frank Rogers  
23<sup>rd</sup> November 2007

## APPENDIX 2

## Test 86 summary

### *Overall plan*

This test sought to recover the zinc by initially moving it into solution with sulphuric acid, and then precipitating it as zinc hydroxide. The latter involved a two-stage precipitation, with the first precipitation aimed at removing most of the silicon.

stage 1	50g of combined dry sample in 500 ml of 5% H <sub>2</sub> SO <sub>4</sub> (25 ml of 98% acid, - or 45 g of 98% acid) digested over 10 days. Then solution filtered off.
stage 2	Solution partially neutralised to pH 3.3 using 10% NaOH. Resulting precipitate filtered off. Precipitate intended for tailings.
stage 3	Remaining solution neutralised to pH 6.6 using 10% NaOH. Resulting precipitate intended to contain most of the zinc.
stage 4	Remaining solution neutralised to pH 11 with 10% NaOH. This was a check to confirm that most zinc had been removed in previous precipitate.

### *Overall results*

While the digestion appeared to go well (estimated over 90% Zn recovery to solution), a significant percentage of the zinc (about 50%) appeared to “get lost” somehow with stage 2. Stage 3 balanced out well based on remaining zinc. Stage 4 confirmed that nearly all the stage 2 zinc had been removed at stage 3.

However, acid usage was very high, at 90% of dry ore weight. The amount of NaOH needed to precipitate the zinc was also very high, at about half the dry ore weight.

Later tests (tests 87, 89 and 90) did indicate that a lower acid concentration could be used, but indicated that it would probably not be less than about 60% of dry ore weight. This approach to the zinc recovery was abandoned in further tests.

### *Test procedure details*

#### **Acid digestion**

50g of pulverised combined dry sample (@ an estimated 7% Zn) was stirred into 500 ml of 5% H<sub>2</sub>SO<sub>4</sub> (i.e. 25 ml of 98% acid, - or 45 g of 98% acid).

A hotplate-magnetic-stirrer was used to maintain the temperature at approximately 30°C. The hotplate-stirrer was operated only during the day, and was turned off during the evenings (I did not trust the hotplate-stirrer, - with good reason as it later turned out).

Temperatures during the night were not monitored, but would have been generally below 15°C.

For two hours prior to sampling (during the afternoons) the hotplate-stirrer was turned off to allow the sediment to settle.

After 6 days the sample was removed from the hotplate-stirrer, and stirring was only occasional by hand-held stirring rod. Analysis was continued for a further 4 days.

Zinc recovery to solution appeared to be about 90% after 6 days, - rising, eventually, to above 95% after 10 days.

The Zinc acid solution from the above digestion (after 10 days) was recovered by filtration. This amounted to 275 ml of solution which analysed at 1.2% Zn, implying a weight of 3.3g of Zn in solution. Considering that some Zn has been lost in assays over the 10 days of the above digestion, I think the head assay, assumed from a previous analysis on the pulverised sample, was a bit lower than the sample used here.

The 275 ml of solution was neutralised in 3 steps.

### **First stage neutralisation**

The 275 ml of solution (@ 1.2% Zn) was neutralised to pH 3.3 using 10% NaOH. The neutralisation was done by pipette, while stirring (magnetic stirrer), using 2 ml alkali intervals and then, after pH 2 using 1 ml alkali intervals.

At the completion of the first neutralisation, the solution contained a light brown precipitate. Total volume after the neutralisation was 419 ml. After filtration 260 ml of light brown solution was obtained. The other 159 ml remained as precipitate, plus solution trapped in the precipitate.

The resulting solution analysed at 0.37% Zn  
The washed precipitate weighed 2.25g @ 0.71% Zn (0.016g Zn)

Unfortunately the water from washing the precipitate was not analysed. But if we assume that there was about 415 ml of solution (including that lost in the washing), the solution analysis implies a total of 1.5g of Zn in the solution.

There is something very wrong here. This is only half the amount of Zn there should be. What has happened to the rest?

### **Second stage neutralisation**

250 ml of the above solution (@ 0.37% Zn) was then used for the second stage neutralisation to pH 6.6. This produced a light blue/green precipitate, which was filtered off, leaving 190 ml of very clear colourless solution.

The resulting solution analysed at 0.014% Zn. (0.027g Zn for the 190 ml)  
The washed precipitate weighed 2.85g @ 35.1% Zn (1.0g Zn)  
The wash water from the precipitate (160 ml) gave 92 ppm (0.015g Zn)

The weight of the wet precipitate was 55g, which dried out to 2.85g, so the volumes are pretty well accounted for (allowing perhaps 5 ml being soaked up by the filter paper). The total Zn implied by the analyses is 1.04g. The start solution analysis implies 0.93g was available to begin with. This discrepancy is within analysis error (+/- 5%) for the precipitate and start solution.

Things balance for this stage anyway.

### **3<sup>rd</sup> stage neutralisation**

180 ml of the solution (@ 0.014% Zn) from the previous neutralisation was then taken up to a pH of just above 11 with further NaOH. This produced a further precipitate, which was filtered off and analysed.

The resulting solution analysed at 0.0001% Zn. (negligible Zn)  
The washed precipitate weighed 1.6g @ 1.6% Zn (0.026g Zn)  
Precipitate wash water (1000 ml), Zn below detectable

This stage balances very well with the previous.

### **Sediment analysis**

The sediment remaining after the acid digestion was dried and weighed, giving 42.1g. There has been a weight loss of about 7.9 g, which matches reasonably well, given weight errors in scraping the precipitate from the filter papers, with the total precipitate weight (6.7 g).

The 42.1 g of dried sediment was then placed into 100 ml of aqua regia for analysis, but a very delayed and very sudden and violent reaction (after I concluded nothing much was going to happen) spread the sample all over a safety tray in the fume cupboard. No sensible analysis was then possible on this sediment.

The unfortunate thing here is that it looks like about half a tonne of NaOH would be required for every tonne of ore. This would be too expensive.

## Samples sent to Hobart 22<sup>nd</sup> January 2008

### Test 95

1. **Stage 1 solution** after attempt to place silicon into solution at pH 2. This is the solution decanted at the completion of stage 1.
2. **Stage 1 wash solution**. After the initial decanting the slurry was brought back to 4 litres volume and again lowered to pH 2 with H<sub>2</sub>SO<sub>4</sub> (4.5 g acid needed). This is the decanted solution after stirring and allowing to settle overnight.
3. **Stage 2 slurry**. This is the complete slurry from the zinc digestion stage, - 19 days after adding the acid. Solution Zn assay on 21<sup>st</sup> Jan, after heating from 11<sup>o</sup>C to 34<sup>o</sup>C over 2.5 hours, was 0.46% Zn.
4. **Stage 2 solution sample** after 9 days digestion at ambient temperature. This represents the highest recovery of Zn to solution, and is the small solution sample in the test tube. It assayed at 0.54% Zn on 12<sup>th</sup> Jan.

### Test 96 slurry

This slurry was prepared on 3<sup>rd</sup> December.

500 g wet combined sample ground in 500 ml water in small rod mill, then screened at 2 mm and originally made up to 4 litres of slurry. The volume has since been reduced by evaporation.

No acid was added. The sample was intended for a repeat of the “failed” Hobart higher temperature digestion, but this was not carried out.

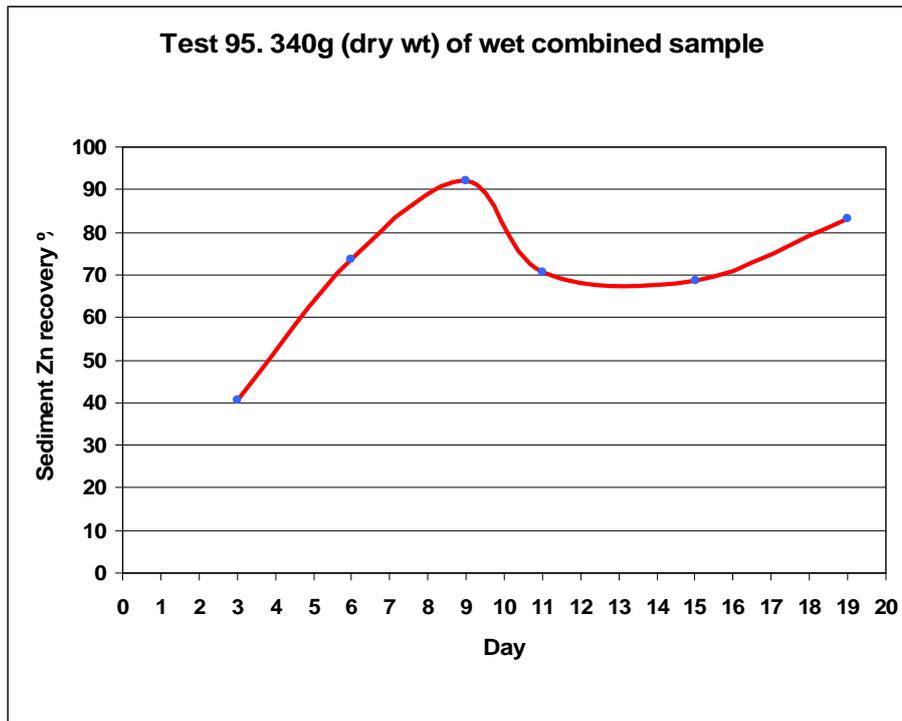
Also, taped to the container, are the larger (>2mm) particles.

## Further comments on Test 95

The day 6 and day 9 assays on the test 95 solution coincided with some of the hottest weather we had so far this summer. The lower two assays that followed coincided with very much cooler weather. There was therefore the suspicion that the digestion was temperature dependent, possibly due to a reversible reaction involving some remaining silicon. It had been intended to follow this up by heating and cooling the slurry and seeing if the assays followed the temperature variations.

The requirement to send the slurry to Hobart ended this idea, but some brief heating was tried today (21<sup>st</sup> January).

The slurry temperature at 9.30 am was 11<sup>o</sup>C. At 12 midday the hotplate had increased it to 34<sup>o</sup>C, after which the sample was allowed about an hour to settle and then sampled. The zinc assay did increase over the previous two assays, as shown in the graph.



However, I think the results of this quick test are just suggestive.

## Record of test work (January 2008)

(Tuesday 1<sup>st</sup> January to )

### Test 95

500g of wet combined sample was ground in 500 ml of water. The resulting slurry was screened at 2mm and then made up to 4 litres of slurry. 10% sulphuric acid was then added to bring the pH down to 2. The dry weight of the 500g used in the test was approximately 340g, with a density of 2.1 g/cm<sup>3</sup> (a solids volume of 162 cm<sup>3</sup>).

The sample was continuously stirred during the day by a rotating stainless steel paddle. The pH was monitored by periodically removing a small sample in a beaker. After measuring, the test sample was returned to the bucket.

A total of 44.1g of 98% H<sub>2</sub>SO<sub>4</sub> had been added to the slurry to bring the pH down to 2 at the end of November 2007. The resulting solution was analysed for Zn, giving 600 ppm for 3838 ml of solution. This amounts to a removal of 10% to 11% of the available Zn from the ore.

This test had been ceased at this stage, in preparation for the major test designed by Bob.

When again looked at on 1<sup>st</sup> January, and after adding water to bring the slurry volume back to 4 litres (evaporation over the past month had lowered the volume), the pH had risen to 2.7. Note: the pH meter was calibrated with 0.01N HCl.

A further 9g of 98% acid was required to bring the pH back down to 2, giving a total acid usage so far of 53.1g (15.6% of dry ore weight).

The slurry was then let stand overnight before commencing to draw off the solution (hopefully containing most of the soluble silicon). This amounted to 3.2 litres of solution. The slurry was then made up again to 4 litres with water, and 25 ml of 10% acid was needed to bring the pH back to 2.02 to avoid any remaining soluble silicon from coming out of solution. After several hours stirring, this was allowed to settle overnight before again decanting the solution (approx 3.2 litres). Note that the second 25 ml addition of 10% acid has not been included in the total acid usage.

Up till this point there was no obvious smell of H<sub>2</sub>S.

Both decanted solutions were retained for later analysis.

Initial 3.2 litres of solution gave 900 ppm Zn (indicating 2.9g Zn in the 3.2 litres)  
(and 3.67g in the 4.1 litres of total solution)

Wash solution of 3.2 litres gave 230 ppm Zn (indicating 0.74g Zn in the 3.2 litres)  
(and 0.93g in the 4.1 litres of total solution)

Therefore the total Zn extracted by the silicon removal is about 3.8g out of an estimated total Zn of about 25.8g, based on the test 94(b) Zn balance. This indicates approximately a 15% loss of Zn to the silicon removal.

500 ml of 10% (90g of 98%) H<sub>2</sub>SO<sub>4</sub> was then added to the slurry, and stirred for about 4 hours before allowing to settle overnight. At this stage the acid usage was

Removal of silicon	53g	
Zn Digestion	90g	
Total	143g	(42% of dry ore weight)

A strong smell of H<sub>2</sub>S was then evident.

After 3 days digestion, with constant stirring during the day and sample settling overnight, at an average of about 20<sup>0</sup>C; -

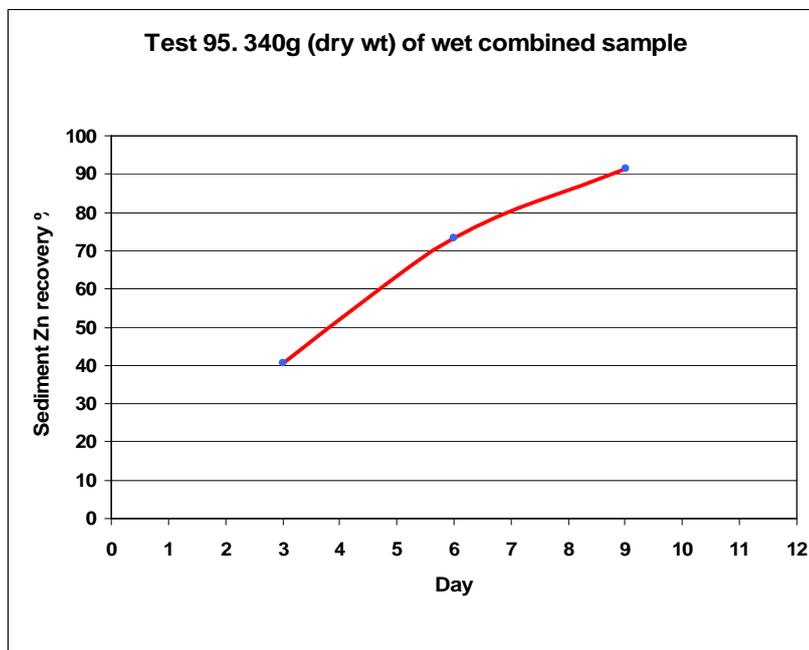
Zn recovery (@ 0.21%) - approximately 40% of Zn remaining in sediment.  
 - approximately 34% of total sample Zn.

After 6 days digestion (volume reduced to 3838 ml, - due to evaporation)

Zn recovery (@ 0.42%) - approximately 73% of Zn remaining in sediment.  
 - approximately 62% of total sample Zn.

After 9 days digestion (volume reduced to 3638 ml, due to evaporation)

Zn recovery (@ 0.55% Zn) - approximately 91% of Zn remaining in sediment.  
 - approximately 78% of total sample Zn.



## APPENDIX 3



# ANALYTICAL SERVICES TASMANIA

18 St Johns Avenue, New Town, Tasmania, Australia, 7008  
New Town Laboratory

Telephone: (03) 6230 7000 Fax: (03) 6230 7001 Email: astproduction@environment.tas.gov.au



<p><b>To:</b> John Bishop Icon Resources <b>Address</b> 38 George St, North Hobart, 7000</p> <p><b>Fax No:</b> <b>Phone:</b> 62298612 <b>Mobile:</b></p>	<p><b>Date:</b> 12-Mar-08 <b>Pages:</b> 4 (including this one)</p> <p><b>From:</b> Amanda Freeman</p> <p><b>Fax No</b> (03) 6230 7001 <b>Phone:</b> (03) 6230 7000</p>
--	--

## Order No

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Following is a reissue of report number 34454. The new report, issue number 2, contains contains Changes/Corrections.  
Please destroy issue number 1.

The original was sent to:

Bob Groenewoud, Icon Resources, PO Box 2001 North Hobart TAS 7002.

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Thank you.

# ANALYTICAL SERVICES TASMANIA

## New Town Laboratory

18 St Johns Avenue, New Town, Tasmania, Australia, 7008

New Town, Tasmania, 7008

Telephone: (03) 6230 7000 Fax: (03) 6230 7001

Email: [astproduction@environment.tas.gov.au](mailto:astproduction@environment.tas.gov.au)



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## Laboratory Report

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**Report No: 34454**      **Issue No: 2**      **Report Date 12-Mar-2008 8:55**

**Status:** Full Report with Changes/Corrections

### Site Description:

**Received:** 22-Jan-08      **to** 28-Jan-08

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**Submitted to:** New Town Laboratory

**Submitted By:** Bob Groenewoud

### Client Order No:

**Report To:** Bob Groenewoud

**Client:** Icon Resources

**Address:** 38 George St, North Hobart TAS 7000

---

The tests, calibrations or measurements covered by this document have been performed in accordance with NATA requirements which include the requirements of ISO/IEC 17025 and are traceable to national standards of measurement.

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Samples analysed as received.



## ANALYTICAL SERVICES TASMANIA

**Report No:** 34454    **Issue No:** 2    **Report Date:** 12-Mar-2008 8:55

Method	Analyte	Units / Sampled On :	Lab.No.:	121619	121620	121621	121622	121624	121627	122327
			Sample Id.:	T95 Stage 1	T95 Stage 1decant	T95 After 9 days	T95 Slurry	T95 Slurry Solids	Original Ore	Icon Resources
1001-Water	pH			2.2	2.2	<b>1.6</b>	1.4			
1103-Water	Sulphate	mg/L		8040	2980	48100	34900			
1301-Water	Al Total	µg/L		1150000	309000	465000	446000			
	Cu Total	µg/L		231	154	52	39			
	Fe Total	µg/L		680000	230000	392000	1630000			
	Pb Total	µg/L		7440	4500	2390	3150			
	Zn Total	µg/L		759000	227000	3080000	3020000			
1302-Water	Ca Total	mg/L		398	498	509	491			
	Mg Total	mg/L		621	157	72.8	62.9			
2301-Soil	Al	mg/kgDMB						4150	23000	<b>19900</b>
	Cu	mg/kgDMB						629	124	<b>127</b>
	Fe	mg/kgDMB						13600	19900	<b>15100</b>
	Pb	mg/kgDMB						9250	6960	<b>6390</b>
	S*	mg/kgDMB						51500	43800	<b>46900</b>
	Zn	mg/kgDMB						32500	75000	<b>69700</b>
2309-Soil	Ca	mg/kgDMB						6400	17800	<b>9190</b>
	Mg	mg/kgDMB						360	10900	<b>10400</b>

\* NATA accreditation does not cover this analyte.

Results in **BOLD** are those that have changed or been added since the report was last issued

# ANALYTICAL SERVICES TASMANIA

Report No: 34454 Issue No: 2 Report Date: 12-Mar-2008 8:55

**Test Method(s) :** **Test Date**

## Inorganic Testing

1001-Water:	pH in Water by APHA Method 4500-H Work Conducted at: New Town	07-Feb-2008
1103-Water:	Anions by Ion Chromatography APHA Method 4110C Work Conducted at: New Town	12-Feb-2008
1301-Water:	Metals in Water by APHA Method 3030/3120 Work Conducted at: New Town	13-Feb-2008
1302-Water:	Major Cations in Water by APHA Method 3030/3120 Work Conducted at: New Town	13-Feb-2008
2301-Soil:	Metals in Soil, Sediment & Dust by ICPAES Work Conducted at: New Town	15-Feb-2008
2309-Soil:	Major Cations in Soil, Sediment & Dust by ICPAES Work Conducted at: New Town	15-Feb-2008

Authorised By:



Damien Norman  
Section Head - Inorganic



Accreditation No. 5589

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\* NATA accreditation does not cover the performance of this service.

## APPENDIX 4

## R. Groenewoud Email communication: 14 January 2008

The last 2 tests at AST were done on 90g wet sample each plus about 10g for initial solid analysis. About 100g per test would suffice. More would be comfortable if additional initial solid analysis were required for confirmation later on.

Test #86 was carried out on 50g dried sample in 500ml acid containing 25g 98% H<sub>2</sub>SO<sub>4</sub> at upto 27C. The 2nd AST test used 90g wet sample (55.62g dry) in 300ml (330 total) to which 21.5g 98% H<sub>2</sub>SO<sub>4</sub> was added in 4stages. The temperature was kept close to 90C. The final acidity in the AST test was approx. 22g/L, whereas test #86 would have been approx. 18g/L. The latter was lower, because of the larger volume and the higher zinc dissolution.

A closer look at the graph of test #86 indicates a zero order rate reaction, which is typical of bioleaching. A similar graph is shown in the article '**Bioleaching of Zinc Sulphide (ZnS) Ore using *Thiobacillus ferrooxidans***' (1st article when googling, **bioleaching zinc**). The bacterium used in this was *Thiobacillus ferrooxidans*, which operates in an aerobic environment. Test #86 would have been anaerobic, therefore different bacteria would have been at work.

If this hunch is correct, it is important to identify the bacterium . So I suggest Neil does another test and all samples are analysed by AST, including for bacterial analysis. If this is correct we may have a breakthrough. However, where did the bugs come from? Were they already in the ore or did they come with the water used in test #86?

In the meantime I suggest, if the samples are available, that AST analyses a sample of test #86 leach liquor, and a residue sample.

## R. Groenewoud Email communication: 5 March 2008

I've attached the calculation results for the 3 tests.

My preliminary conclusions are as follows,

### Test#1,

This test was carried out at a final pH 2.3, which was an acidity of 1.35g/L ( the intention had been to keep the acidity between 5 and 10g/L), and approximately 85C.

This test indicates that it is possible to recover acid at approximately 120kg/tonne ore, with a zinc extraction of 8%.

If it is possible to recover the zinc extracted in the acid recovery stage in a separate Ion Exchange column, then a higher acid recovery could be possible.

( Note: Only acid that is regenerated in IE, and that, which makes up the acidity of the solution can be recovered. All other acid, which produces metal sulphates can not be recovered. It is therefore important to physically remove any dolomite or limestone to lower the magnesium and calcium in the feed (This is what Neil did in test#95 by removing the >2mm particles.) By far the biggest consumer of acid is aluminium which is present in felspar and clay.)

### Test#2,

This test was carried out at approximately 20g/L and 90C over a 5hour period. Acid usage was 385kg/tonne ore.

The zinc extraction was rapid for the 1st 30minutes, but then slowed rapidly coming to an almost stand still after 5hours, at which the extraction was only 23.5%.( calculations suggest that the maximum zinc extraction would have been 24.2% after 7hours.)

The only explanation I can give is that at 90C, which is close to the sulphur melting point, the particles become coated with sulphur, which is produced according to the following reaction  $ZnS + Fe_2(SO_4)_3 = ZnSO_4 + FeSO_4 + S$ . Pyrite  $FeS_2$  reacts rapidly with more than 50% Fe extraction in the 1st 30minutes. This produces  $FeSO_4$ , which oxidises to  $Fe_2(SO_4)_3$ . It may be reasonable to assume that at a lower temperature this slow down of the zinc extraction may not happen. I therefore suggest we repeat this test at 60C.

### Test#3

This test consisted of a full analysis of the solutions and solids from test#95 carried out by Dr. Neil Allen. This test#95 was carried out in 2stages, the 1st stage at pH2, then a decant wash, and then a 2nd stage 19days later at mostly ambient temperature with a short period, 2or 3hours at 34C, over a 9day period at a final acidity of 14g/L(calculated). The AST analysis were done about another month later. The concentrations of the metals in the 9day and final solution samples were similar except in the case of iron. The iron analysis of the final solution is suspect, because it results in a metal balance of several hundred %.

The results indicate that the extractions of Al, Fe, and Mg were similar to those in test#2. The extraction of Zn was approx 60% based on both the solution and solids analysis. This is a considerable improvement on the extraction of Zn in test#2. Acid Usage was 429kg/tonne ore.

**Test 1**

**Head Assays of Pit Samples**

Pit No.	Zn %	Pb %	Fe %	Mg %	Al %	Ca %	Na %	total S %	sulphide %	calc.SO4 %	calc.CO3 %	SiO2 %	Mn ppm	Cu ppm	Cd ppm	As ppm
115	2.84	0.06	0.79	0.10	1.16		0.10	2.63	1.77	2.58	0.10	32.80	20	40	10	<50
170	2.53	0.10	4.01	1.24	0.77		0.09	4.94	3.72	3.66	7.55	35.00	300	20	20	<50
181	17.50	1.45	2.10	0.03	2.27		0.03	11.50	10.30	3.60	0.15	9.16	10	50	330	250
264	2.69	0.09	4.33	0.57	0.87		0.06	6.89	4.04	8.55	7.50	3.02	140	20	20	<50
368	4.98	1.63	1.10	0.11	1.21		0.68	3.74	2.93	2.43	4.50	22.90	<10	290	130	100
<b>Average %</b>	<b>6.11</b>	<b>0.67</b>	<b>2.47</b>	<b>0.41</b>	<b>1.26</b>		<b>0.19</b>	<b>5.94</b>	<b>4.55</b>	<b>4.16</b>	<b>3.96</b>	<b>20.6</b>	<b>94</b>	<b>84</b>	<b>102</b>	<b>70</b>

AST calculated **7.34 1.29 1.80 0.96 2.42 1.89** **5.86** **5.20** **112 60 89 64**

Calculated S is based on ZnS, FeS2, and PbS.  
Calculated CO3 is based on CaCO3 and MgCO3.

**Experimental Results.**

Analyte	1470	1910	2250	2080	2260
minutes	30	60	120	180	240
Al	789	737	428	773	714 mg/L
Ca	1470	1910	2250	2080	2260 mg/L
Cu	<1	<1	<1	<1	<1 mg/L
Fe	1320	1740	1700	1810	1810 mg/L
Mg	1460	1620	1600	1810	1830 mg/L
Mn	18.7	24.8	23.8	25.2	25.7 mg/L
Pb	33	37.4	37.3	31.3	36 mg/L
S	9650	11600	13500	13200	13000 mg/L
Zn	992	1140	1060	1130	1130 mg/L

Max Ca solubility is 620mg/L

**Solids**

	initial		final		solids			solution	
					initial calc	final calc	dissolved		
As	64	66 mg/kg	3.52	3.14	0.38				mg
Cd	89	96 mg/kg	4.89	4.57	0.32				mg
Co	29	10 mg/kg	1.59	0.48	1.12				mg
Cr	53	51 mg/kg	2.91	2.43	0.49				mg
Cu	60	81 mg/kg	3.30	3.86	-0.56				mg
Mn	112	23 mg/kg	6.16	1.10	5.06				mg
Ni	70	30 mg/kg	3.85	1.43	2.42				mg
Pb	12900	7330 mg/kg	0.71	0.35	0.36	0.011			g
Zn	73400	69000 mg/kg	4.04	3.29	0.75	0.334			g
Mg	9580	1640 mg/kg	0.53	0.08	0.45	0.542			g
Fe	18017	11137 mg/kg	0.99	0.53	0.46	0.536			g
Al	24192	19970 mg/kg	1.33	0.95	0.38	0.211			g
Ca	18879	18772 mg/kg	1.04	0.89	0.14	0.669			g

**Calculations**

	initial	final	wet weight	moisture	dry weight
liquor	300	235 ml	g	%	g
moisture	35	29 ml	ore	90	38.9
4hr sample	5 ml	5 ml	final solids	76.8	38
other samples	335	269 ml	change g		7.37
		20 ml	change %		13.4
Evaporation		46 ml			
Over 4hours		11.5 ml/hour			

**Zn extraction calculations**

without adjustments 8.1 %  
with sampling and dilution corrections 8.3 %

**Acid requirement for maximum extraction**

Zn	110.0 kg/tonne
Fe	31.6 kg/tonne
Pb	6.1 kg/tonne
Al	87.9 kg/tonne
Mg	38.6 kg/tonne
Ca	46.3 kg/tonne

Extraction 'adjusted for % metal balance

Al	15.9	18.6
Fe	54.1	53.7
Mg	102.8	98.6
Mn	123.5	98.5
Zn	8.3	9.3

Ca and Pb can only be estimated, because their sulphates have low solubilities

**Acid Recovery**

based on 7g acid addition 118.6 kg/tonne

Note: Acid can only be recovered from Zn in solution by IE or SX, and from acidity in solution. Acid used to produce of the other metal sulphates can not be recovered

**Assumptions:**

- Mg is MgCO3.
- Ca is CaCO3
- Zn, Pb and Fe are sulphides, Fe is FeS2.
- Al is as Al2O3

Pit No.	Zinifex S in sulphides ZnS,PbS, FeS2	
115	2.63	2.30
170	4.94	5.85
181	11.5	11.19
264	6.89	6.29
368	3.74	3.95
average	5.94	5.92

**Correction for evaporation**

evap.factor	0.98283	0.9654	0.92998	0.893964	0.857305
minutes	30	60	120	180	240
Al	775	712	398	691	612 mg/L
Ca	1445	1844	2092	1859	1938 mg/L
Cu					mg/L
Fe	1297	1680	1581	1618	1552 mg/L
Mg	1435	1564	1488	1618	1569 mg/L
Mn	18	24	22	23	22 mg/L
Pb	32	36	35	28	31 mg/L
S	9484	11199	12555	11800	11145 mg/L
Zn	975	1101	986	1010	969 mg/L

**Correction for evaporation + dilution**

(Assuming Mn is maximum after 1hour and remains constant at 22.7mg/L)

minutes	30	60	120	180	240
Al	775	675	408	696	631
Ca	1445	1748	2146	1874	1996
Cu	0	0	0	0	0
Fe	1297	1593	1621	1630	1599
Mg	1435	1483	1526	1630	1616
Mn	18.4	22.7	22.7	22.7	22.7
Pb	32.4	34.2	35.6	28.2	31.8
S	9484	10618	12876	11890	11482
Zn	975	1043	1011	1018	998

**Acid Balance**

	After 2hours and 6g acid		After 4hours and 7g acid	
	grams	H2SO4 g	grams	H2SO4 g
Al	0.137	0.75	0.211	1.15
Ca	0.62	1.53	0.62	1.53
Cu	0.000		0.000	
Fe	0.543	0.95	0.536	0.94
Mg	0.511	2.07	0.542	2.19
Mn	0.008	0.014	0.008	0.014
Pb	0.43	0.201	0.43	0.201
S	4.314		3.847	
Zn	0.339	0.51	0.334	0.50
	total g	6.01	total g	6.52
			pH2.3	0.45
			total g	6.97
SO4 in solution g	4.70		SO4 in solution g	5.64

**Metal balances %**

Analysis as received		Analysis corrected for evaporation and dilution.		Average concentrations for 1st four samples	
Al	85.4 %	Al	86.0 %	Al	682 mg/L
Ca	142.0 %	Ca	144.9 %	Ca	1928 mg/L
Cu		Cu		Cu	0 mg/L
Fe	100.7 %	Fe	102.9 %	Fe	1643 mg/L
Mg	104.3 %	Mg	108.7 %	Mg	1623 mg/L
Mn	125.4 %	Mn	130.5 %	Mn	23 mg/L
Pb	50.5 %	Pb	50.6 %	Pb	35 mg/L
S		S		S	11988 mg/L
Zn	88.6 %	Zn	89.0 %	Zn	1081 mg/L

The large discrepancy in the elements that have a large proportion in the residue is due to insufficient mixing of the residue before sampling.

Max Ca solubility is 620mg/L, Ca metal balances are then 101.6% and 102.2%

Assumes Ca & Pb had reacted 60% after 2hours and 60% after 4hours.

Final pH2.3 is 1.35g/L H2SO4

**Test 2 AST Results**

Analyte	30	60	120	180	240	300	Solids	initial
minutes	30	60	120	180	240	300		
Al	2.65	3.13	2.88	2.67	3.23	2.94 g/L	As	64
Ca	1.52	1.69	1.53	1.24	1.95	1.26 g/L	Cd	89
Cu	<5	<5	<5	<5	<5	<5 mg/L	Co	29
Fe	1.72	1.88	1.74	1.62	1.90	1.70 g/L	Cr	53
Mg	1.34	1.43	1.33	1.26	1.50	1.34 mg/L	Cu	60
Mn	18.3	19.5	18.5	16.9	20.2	17.8 mg/L	Mn	112
Pb	35.7	38.2	34.5	31.1	40.3	34.8 mg/L	Ni	70
S						g/L	Pb	12900
Zn	1.48	1.94	2.15	2.26	2.91	2.73 g/L	Zn	73400
							Mg	9580
							Fe	18017
							Al	24192
							Ca	18879
								67200

	wet weight	moisture	dry weight
	g	%	g
ore	90.09	38	55.9
final solids	76.8	38	47.62

If it is assumed that the Mg concentration is at a maximum within the 1st hour and remains constant during the remainder of the test, then fluctuations in volume and errors in dilution prior to analysis should be evened out.

volume	0.334	0.325	0.315	0.355	0.325	0.325	0.325
--------	-------	-------	-------	-------	-------	-------	-------

**Dilution factors**

minutes	30	60	120	180	240	300	
sampling	1.000	1.015	1.031	1.047	1.064	1.081	
volume change	0.973	0.943	1.063	0.973	0.973	0.973	
combined	0.973	0.957	1.096	1.019	1.035	1.052	average
Al	2.58	3.00	3.16	2.72	3.34	3.09	3.06
Mg	1.30	1.37	1.46	1.28	1.55	1.41	1.41
Mn	17.8	18.7	20.3	17.2	20.9	18.7	19.16

**Extraction**

	liquor %	solids %
Al	76.7	
Ca		
Cu		
Fe	59.6	
Mg	88.3	
Mn	100.3	
Pb		
Zn	23.48	21.95

**Acid Usage**

acid added	21.5 g
	384.9 kg/tonne ore

Analyte	30	60	120	180	240	300	acid g	at 30min	1hour	2hour	3hours	4hours	5hours	
minutes	30	60	120	180	240	300		15	19.5	19.5	21.5	21.5	21.5	
volume litre	0.334	0.325	0.315	0.355	0.325	0.325	liquor litre	0.334	0.334	0.334	0.334	0.334	0.334	
dilution	0.973	0.957	1.096	1.018	1.034	1.050		SO4 g	SO4 g	SO4 g	SO4 g	SO4 g	SO4 g	
Al	2.58	3.10	3.06	3.00	3.05	3.10 g/L	Al	13.77	16.54	16.36	16.01	16.27	16.58	
Ca	1.48	1.67	1.63	1.39	1.84	1.33 g/L	Ca	3.55	4.02	3.91	3.34	4.42	3.19	
Cu	<5	<5	<5	<5	<5	<5 mg/L	Cu							
Fe	1.67	1.86	1.85	1.82	1.79	1.79 g/L	Fe	2.88	3.20	3.18	3.13	3.08	3.09	
Mg	1.30	1.41	1.41	1.41	1.41	1.41 g/L	Mg	5.15	5.59	5.59	5.59	5.59	5.59	
Mn	17.81	19.29	19.67	18.97	19.05	18.79 mg/L	Mn	0.03	0.03	0.03	0.03	0.03	0.03	
Pb	34.74	37.78	36.69	34.91	38.00	36.73 mg/L	Pb	0.02	0.02	0.02	0.02	0.02	0.02	
Zn	1.440	1.919	2.286	2.537	2.744	2.882 g/L	Zn	2.12	2.82	3.36	3.73	4.03	4.23	
pH dilution/10	1.96	1.70	1.76	1.65	1.71	1.70	total SO4 g/L	27.52	32.21	32.45	31.85	33.44	32.73	
H2SO4 g/L	11.7	21.7	19.1	24.2	21.5	21.44	H2SO4 g/L	19.6	28.5	28.1	34.0	33.5	33.0	
adjusted for sampling and dilution							Ca	7.74	7.74	7.74	7.74	7.74	7.74	
	11.4	20.8	20.9	24.7	22.3	22.6	Pb	1.02	1.02	1.02	1.02	1.02	1.02	
							H2SO4 g/L calculated	10.8	19.7	19.3	25.2	24.7	24.2	124.0
							H2SO4 g/L from pH	11.4	20.8	20.9	24.7	22.3	22.9	122.9

### Zinc Extraction Rate

minutes	0	30	60	120	180	240	300
Zn g/L	0	1.44	1.92	2.29	2.54	2.74	2.88

maximum extraction occurs after 7hours at 2.976g/L, which is 24.2 %

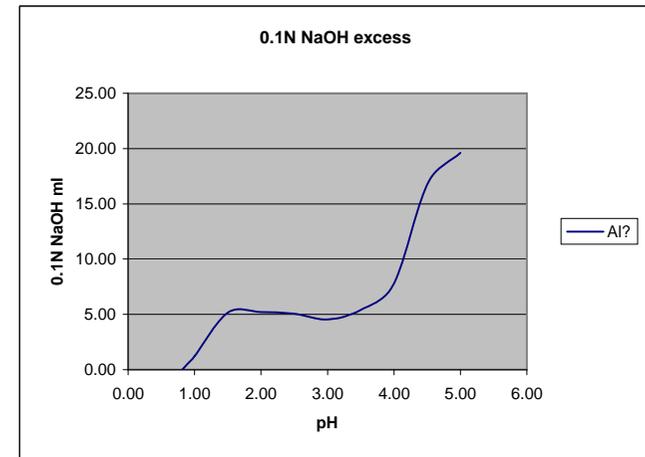
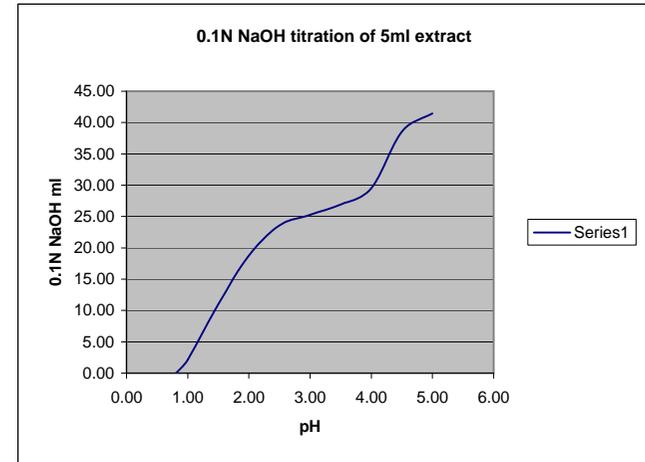
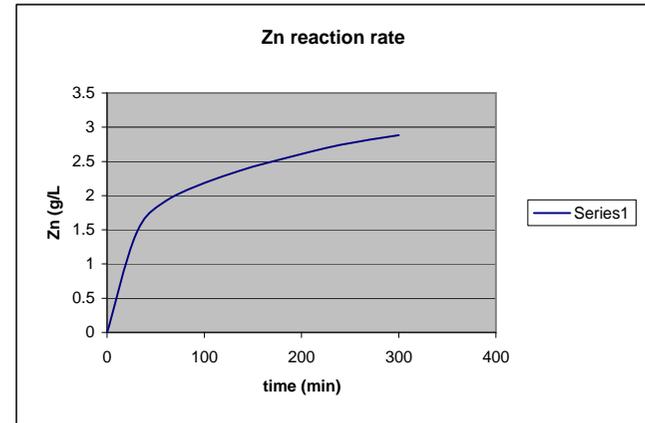
### 0.1N NaOH Titration

pH	0.1N NaOH	ml	SO4 g/L	H2SO4	SO4	0.1N NaOH required		pH	Al?
						H2SO4			
0.81	0.00	5.00	52.12	21.44	31.12	0.00	0.81	0.00	0.00
1.00	2.15	7.15	36.45	14.35	22.39	0.94	1.00	1.21	
1.50	11.06	16.06	16.23	4.87	11.46	5.91	1.50	5.15	
2.00	18.77	23.77	10.96	1.72	9.28	13.55	2.00	5.22	
2.50	23.58	28.58	9.12	0.576	8.55	18.52	2.50	5.06	
3.00	25.25	30.25	8.61	0.188	8.43	20.72	3.00	4.53	
3.50	26.90	31.90	8.17	0.059	8.11	21.49	3.50	5.41	
4.00	29.55	34.55	7.54	0.018	7.53	21.75	4.00	7.80	
4.50	38.60	43.60	5.98	0.0050	5.97	21.83	4.50	16.77	
5.00	41.48	46.48	5.61	0.0016	5.61	21.86	5.00	19.62	

2.94 g/L Al                      16.3 ml 0.1N NaOH                      adjusted for dilution and sampling  
 21.8 g/L H2SO4                22.2 ml 0.1N NaOH  
 total                                38.6 ml 0.1N NaOH

	g/L	activity coefficient	activity moles/L
pH	5.00	1	0.0000
total SO4	5.61	0.440	0.0257
2nd proton dissociation constant K      0.0120			
1st H	0.000	100.00 %	
2nd H	-0.00001	-36.29 %	
total H	0.00001	0.00001	
H2SO4	0.00154		
SO4	5.608		

	g/L	activity coefficient	activity moles/L
H2SO4	2.15	1	0.0219
added SO4	3.1	0.500	0.0162
	5.216		
2nd proton dissociation constant K      0.0120			
1st H	0.022	100.00 %	
2nd H	-0.0019	-8.70 %	
total H	0.02001		
pH	1.699		



**ANALYTICAL SERVICES TASMANIA**

Report No: 34335 Issue No: 1 Report Date: 10-Jan-2008 17:10

Method	Analyte	Units / Sampled On :	Lab.No.:					
			120941	120942	120943	120944	120945	120946
Sample Id.:			Extract 30min	Extract 60min	Extract 120min	Extract 180min	Extract 240min	Extract 300min
1311-Water	Al Total	µg/L	2650000	3130000	2880000	2670000	3230000	2940000
	Ca Total	mg/L	1520	1690	1530	1240	1950	1260
	Cd Total	µg/L	<500	<500	<500	<500	<500	<500
	Co Total	µg/L	3300	3220	3000	3190	3520	3010
	Cr Total	µg/L	<5000	<5000	<5000	<5000	<5000	<5000
	Cu Total	µg/L	<5000	<5000	<5000	<5000	<5000	<5000
	Fe Total	µg/L	1720000	1880000	1740000	1620000	1900000	1700000
	K Total	mg/L	<250	<250	<250	<250	<250	<250
	Mg Total	mg/L	1340	1430	1330	1260	1500	1340
	Mn Total	µg/L	18300	19500	18500	16900	20200	17800
	Na Total	mg/L	<250	<250	<250	<250	<250	<250
	Ni Total	µg/L	8210	8390	8460	7500	9290	8020
	Pb Total	µg/L	35700	38200	34600	31100	40300	34800
	Zn Total	µg/L	1480000	1940000	2150000	2280000	2910000	2730000

**Test 95**

1. Stage 1 solution after attempt to place silicon into solution at pH 2. This is the solution decanted at the completion of stage 1.
2. Stage 1 wash solution. After the initial decanting the slurry was brought back to 4 litres volume and again lowered to pH 2 with H2SO4 (4.5 g acid needed). This is the decanted solution after stirring and allowing to settle overnight.
3. Stage 2 slurry. This is the complete slurry from the zinc digestion stage, - 19 days after adding the acid. Solution Zn assay on 21st Jan, after heating from 11C to 34C over 2.5 hours, was 0.46% Zn.
4. Stage 2 solution sample after 9 days digestion at ambient temperature. This represents the highest recovery of Zn to solution, and is the small solution sample in the test tube. It assayed at 0.54% Zn on 12th Jan.

**Test 96 slurry**

This slurry was prepared on 3rd December.  
 500 g wet combined sample ground in 500 ml water in small rod mill, then screened at 2 mm and originally made up to 4 litres of slurry. The volume has since been reduced by evaporation.  
 No acid was added. The sample was intended for a repeat of the "failed" Hobart higher temperature digestion, but this was not carried out.  
 Also, taped to the container, are the larger (>2mm) particles.

**Test 95 Analysis**

ID		stage 1 decant	decant wash	9days	slurry		slurry slds %	feed slds %	balance %		dry solids g	moisture g	"washed" dry solids g	%
SO4	pH	2.2	2.2		1.4	1.34								
	g/L	8.04	2.98	48.1	34.9	S	5.15	4.38			10.62	2.07	8.56	4.15
H2SO4	g/L	1.26	0.78		7.95							6.20		
Al	g/L	1.15	0.309	0.465	0.446	Al	0.415	2.3	93.8		0.86	0.12	0.74	0.357
Cu	mg/L	0.231	0.154	0.052	0.039	Cu	0.063	0.0124			0.13	0.00	0.13	0.063
Fe	g/L	0.68	0.230	0.392	1.63	Fe	1.36	1.99	116.8		2.81	0.11	2.70	1.31
Pb	mg/L	7.44	4.5	2.39	3.15	Pb	0.925	0.696			1.91	0.00	1.91	0.925
Zn	g/L	0.759	0.227	3.08	3.02	Zn	3.25	7.5	86.8		6.70	0.81	5.89	2.86
Ca	g/L	0.398	0.50	0.509	0.491	Ca	0.64	1.78	103.9		1.32	0.13	1.19	0.576
Mg	g/L	0.621	0.157	0.073	0.063	Mg	0.036	1.09	76.4		0.074	0.017	0.057	0.028

**Analysis impossibly high.** Low Mg metal balance due to removal of >2mm particles.  
 Feed analysis shown includes >2mm particles.

**Feed Solids**

340 g

**Slurry solids**

wet g	dry g	moisture
475	206	269
678	294	383

**Slurry Solids calculated from**

AST	201
Pb balance	251
Test2	272

**Zn recovery**

solution	56.3 %
solids	69.5 %

**SO4 dissolution**

	stage 1 decant	decant wash	decant wash % into slurry	9days SO4	slurry SO4	SO4 increase in 2nd stage	Acid required for Ca and Pb. g
	SO4	SO4	2.76				PbSO4 1.12
Al	6.14	1.65	0.17	2.48	2.38	2.31 g/L	CaSO4 14.83
Cu							total 15.95
Fe	1.17	0.40	0.03	0.67	0.67	0.64 g/L	1st stage Ca & Pb 4.61
Pb							wash
Zn	1.11	0.33	0.03	4.52	4.43	4.49 g/L	Ca & Pb 4.92
Ca	0.96	1.20	0.48	1.22	1.18	0.74 g/L	
Mg	2.45	0.62	0.07	0.29	0.25	0.22 g/L	2nd stage
total	11.83	4.19	0.78	9.19	8.91	8.41 g/L	Ca & Pb 6.42
H2SO4	12.07	4.28	0.80	9.38	9.10	8.58 g/L	

**Used 9day** analysis

### Acid calculation

acid	added	removed	remained		
1st stage	52.0	47.2	4.83588	1.26 g/L	added 53.1g 98% acid,
decant			4.83588	1.26 g/L	
wash	12.6	4.03	2.99	0.78 g/L	3.20 litres made up to 4L slurry
2nd stage	88.7	34.0	54.65	14.24 g/L	3.20 litres removed added 90g 98% acid, and made up to 4L slurry

If SO<sub>4</sub> in 9day sample is 48.1g/L, and 1/10 dilution sample has pH1.6, then acidity is 25.1g/L. This is 11.2g/L more than that calculated, which is an additional 44g 98% acid. This means that if both the SO<sub>4</sub> analysis and the pH are correct then the acid addition to the 2nd stage must have been an extra 44g 98%acid.

### SO<sub>4</sub> calculation

SO <sub>4</sub>	added	removed	remained		
1st stage	50.0	0.85	49.1		added 53.1g 98% acid.
decant			49.1	12.80 g/L	
wash	12.3	40.9	19.6	5.12 g/L	3.20 litres addition for pH adjustment etc.
2nd stage	87.9	16.4	88.6	23.1 g/L	3.20 litres removed added 90g 98% acid, and made up to 4L slurry

### Al, Fe, Zn, Ca and Mg calculations

	added	removed	remained	total	
1st stage and decant					
Al	4.41			4.41	1.150 g/L
Fe	2.61			2.61	0.680 g/L
Zn	2.91			2.91	0.759 g/L
Ca	1.53			1.53	0.398 g/L
Mg	2.38			2.38	0.621 g/L
wash	g	g			
Al	0.45	3.68	0.73	1.19	0.309 g/L
Fe	0.45	2.18	0.43	0.88	0.230 g/L
Zn	0.39	2.43	0.48	0.87	0.227 g/L
Ca	1.66	1.27	0.25	1.91	0.498 g/L
Mg	0.21	1.99	0.40	0.60	0.157 g/L
"9days"	g	g	g		
Al	1.49	0.99	0.20	1.69	0.465 g/L
Fe	1.28	0.74	0.15	1.43	0.392 g/L
Zn	11.06	0.73	0.14	11.21	3.080 g/L
Ca	1.53	1.59	0.32	1.85	0.509 g/L
Mg	0.17	0.50	0.10	0.27	0.073 g/L

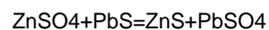
### Extraction

	extraction %			for 100%	
	1st stage	wash	2nd stage	total %	balance
Al	56.4	5.8	19.1	81.3	86.7
Fe	38.6	6.6	18.9	64.1	54.9
Zn	11.4	1.5	43.4	56.3	64.9
Ca	25.2	27.4	25.3	78.0	75.0
Mg	64.3	5.6	4.5	74.3	97.3

### Acid Usage

	acid usage g			total %
	1st stage	wash	2nd stage	
Al	24.05	2.46	8.14	22.60
Fe	4.58	0.79	2.24	4.96
Zn	4.37	0.58	16.57	14.04
Ca	3.74	4.06	3.76	7.54
Mg	9.60	0.83	0.67	7.24
additional for CaSO <sub>4</sub> &PbSO <sub>4</sub>	0.86	0.86	2.66	2.86
total	47.20	9.59	34.05	
acidity	4.84	2.99	54.7	40.75
used	52.04	12.6	88.7	100.00

Acid added 153.3 g  
 removed in decants 5.4 g  
 nett 147.9 g  
 435.1 kg/tonne ore



lithopone-paint pigment

Pit No.	Zn %	Pb %	Ag g/t	Cu ppm	As ppm	Fe %	Ba ppm	Au g/t	Mn ppm	Cd ppm	Na ppm	Mg ppm	Al %	SiO <sub>2</sub> %	S total %	S total Repeat%	Sulphide S %	Calc. SO <sub>4</sub>	Total Carbon	Non-carbonate	Calc. CO <sub>2</sub> %	Elemental	Calc.	Dry Solid SG	Bulk Density	Moisture	Ca %	
																			%	Carbon %		Carbon %	Organic Carbon %		Wetcake	Content %		
115	2.84	0.058	2	40	<50	0.79	40	<0.1	20	10	1010	1040	1.16	32.8	2.63	2.38	1.77	2.58	16.7	16.6	0.1	2.32	14.3	2.23	1.29			
170	2.53	0.103	2	20	<50	4.01	30	<0.1	300	20	890	12400	0.77	35	4.94	4.6	3.72	3.66	7.79	6.28	7.55	1.89	4.39	2.62	1.56			
181	17.5	1.45	3	50	250	2.1	<10	<0.1	10	330	250	320	2.27	9.16	11.5	10.7	10.3	3.6	16.8	16.5	0.15	1.55	15	2.31	1.55	42.9		
264	2.69	0.088	3	20	<50	4.33	<10	<0.1	140	20	590	5710	0.87	3.02	6.89	6.37	4.04	8.55	27.8	26.3	7.5	16.3	10	1.8	1.32			
368	4.98	1.63	12	290	100	1.1	10	<0.1	<10	130	6800	1110	1.21	22.9	3.74	3.63	2.93	2.43	21.5	20.6	4.5	10.6	10	2.1	1.51			

## Zinc Leach Test # 2 Procedure

The aim of the test work is to determine the rate of dissolution of Zn in 5 -10g/L H<sub>2</sub>SO<sub>4</sub> at about 90C, and to determine the acid requirement.

Procedure:

To 120g of sample in a 1L conical flask with stirrer, add 400ml of distilled water, and heat to 80C. Add 8g 98% H<sub>2</sub>SO<sub>4</sub> to the slurry. (initial acidity approx.18g/L)

After 30minutes, 1, 2, and 3 hours take a 5ml sample, dilute to 50ml, filter, and analyse for Zn, Fe, and H<sub>2</sub>SO<sub>4</sub>.

Aim to maintain acidity at about 5 to10g/L, by adding 1g H<sub>2</sub>SO<sub>4</sub> to the slurry for every 2g/L the acidity is less than 10g/L H<sub>2</sub>SO<sub>4</sub>. Record acid additions. Aim to maintain the temperature at about 90C. Record temperature before taking each sample.

After 4 hours filter the pulp, and wash and dry the solids. Keep the liquor and wash separate and record the volumes. Record the wet and dry weights of the solids. Analyse liquor for Zn and H<sub>2</sub>SO<sub>4</sub>. Analyse the wash liquor, and the solids for Zn. Keep the final liquor and solid samples.

Note: To ensure that only H<sub>2</sub>SO<sub>4</sub> is determined in the titration an end point at pH3 is required for the diluted solutions, to prevent precipitation of ferric hydroxide and silica.

Analysis required,

Ore sample: Zn, Fe, Al, Mg, Cu, Ca, Pb, S.

Final leach solution: Zn, Fe, Al, Mg, Cu, Si.

Final solids: Zn, Pb.

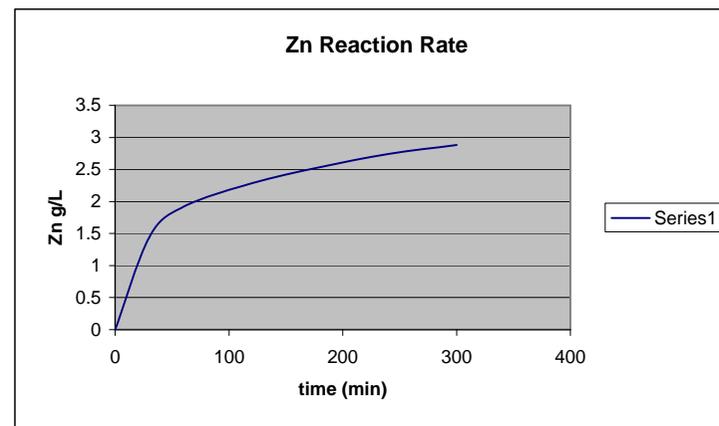


Pb	34.74	37.78	36.69	34.91	38.00	36.73 mg/L	Pb	0.02	0.02	0.02	0.02	0.02	0.02
Zn	1.44	1.92	2.29	2.54	2.74	2.88 g/L	Zn	2.12	2.82	3.36	3.73	4.03	4.23
pH dilution,	1.96	1.70	1.76	1.65	1.71	1.70	total SO4 g/L	27.52	32.21	32.45	31.85	33.44	32.73
H2SO4 g/L	11.7	21.7	19.1	24.2	21.5	21.44	H2SO4 g/L	19.6	28.5	28.1	34.0	33.5	33.0
adjusted for sampling and dilution							Ca	7.74	7.74	7.74	7.74	7.74	7.74
	11.4	20.8	20.9	24.7	22.3	22.6	Pb	1.02	1.02	1.02	1.02	1.02	1.02
							H2SO4 g/L calculated	10.8	19.7	19.3	25.2	24.7	24.2
							H2SO4 g/L from pH	11.4	20.8	20.9	24.7	22.3	22.9

Zinc Extraction Rate

minutes	0	30	60	120	180	240	300
Zn g/L	0	1.44	1.92	2.29	2.54	2.74	2.88

maximum extraction occurs after 7hours at 2.976g/L, which is 24.2 %



**Test 95 Analysis**

ID		stage 1 decant	decant wash	9days	slurry		slurry sls %	feed sls %	balance %		dry solids g	moisture g	"washed" dry solids g	%
SO4	pH	2.2	2.2		1.4	1.34								
	g/L	8.04	2.98	48.1	34.9	S	5.15	4.38			10.62	2.07	8.56	4.15
H2SO4	g/L	1.26	0.78		7.95							6.20		
Al	g/L	1.15	0.309	0.465	0.446	Al	0.415	2.3	93.8	Al	0.86	0.12	0.74	0.357
Cu	mg/L	0.231	0.154	0.052	0.039	Cu	0.063	0.0124		Cu	0.13	0.00	0.13	0.063
Fe	g/L	0.68	0.230	0.392	1.63	Fe	1.36	1.99	116.8	Fe	2.81	0.11	2.70	1.31
Pb	mg/L	7.44	4.5	2.39	3.15	Pb	0.925	0.696		Pb	1.91	0.00	1.91	0.925
Zn	g/L	0.759	0.227	3.08	3.02	Zn	3.25	7.5	86.8	Zn	6.70	0.81	5.89	2.86
Ca	g/L	0.398	0.50	0.509	0.491	Ca	0.64	1.78	103.9	Ca	1.32	0.13	1.19	0.576
Mg	g/L	0.621	0.157	0.073	0.063	Mg	0.036	1.09	76.4	Mg	0.074	0.017	0.057	0.028

**Analysis impossibly high.**

Low Mg metal balance due to removal of >2mm particles.

Feed analysis shown includes >2mm particles.

**Feed Solids**

340 g

**Slurry solids**

wet g	dry g	moisture	
475	206	269	56.6 %
678	294	383	56.6

**Slurry Solids calculated from**

AST	201
Pb balance	251
Test2	272

**Zn recovery**

solution	56.3 %
solids	69.5 %

**SO4 dissolution**

	stage 1 decant SO4	decant wash SO4	decant wash % into slurry 2.76	9days SO4	slurry SO4	SO4 increase in 2nd stage
Al	6.14	1.65	0.17	2.48	2.38	2.31 g/L
Cu						
Fe	1.17	0.40	0.03	0.67	0.67	0.64 g/L
Pb						
Zn	1.11	0.33	0.03	4.52	4.43	4.49 g/L
Ca	0.96	1.20	0.48	1.22	1.18	0.74 g/L
Mg	2.45	0.62	0.07	0.29	0.25	0.22 g/L
total	11.83	4.19	0.78	9.19	8.91	8.41 g/L
H2SO4	12.07	4.28	0.80	9.38	9.10	8.58 g/L

**Acid required for Ca and Pb.**

	g
PbSO4	1.12
CaSO4	14.83
total	15.95
1st stage	
Ca & Pb	4.61
wash	
Ca & Pb	4.92
2nd stage	
Ca & Pb	6.42

**Used 9day analysis**

**Acid calculation**

acid	added	removed	remained		
1st stage	52.0	47.2	4.83588	1.26 g/L	added 53.1g 98% acid,
decant			4.83588	1.26 g/L	
wash	12.6	4.03 9.6	2.99	0.78 g/L	removed 3.20 litres made up to 4L slurry
2nd stage	88.7	34.0	54.65	14.24 g/L	removed 3.20 litres added 90g 98% acid, and made up to 4L slurry

If SO4 in 9day sample is 48.1g/L, and 1/10 dilution sample has pH1.6, then acidity is 25.1g/L. This is 11.2g/L more than that calculated, which is an additional 44g 98% acid. This means that if both the SO4 analysis and the pH are correct then the acid addition to the 2nd stage must have been an extra 44g 98%acid.

**SO4 calculation**

SO4	added	removed	remained		
1st stage	50.0	0.85	49.1		added 53.1g 98% acid.
decant			49.1	12.80 g/L	
wash	12.3	40.9 0.85	19.6	5.12 g/L	removed 3.20 litres addition for pH adjustment etc.
2nd stage	87.9	16.4 2.61	88.6	23.1 g/L	removed 3.20 litres added 90g 98% acid, and made up to 4L slurry

**Al, Fe, Zn, Ca and Mg calculations**

	added	removed	remained	total	
1st stage and decant					
Al	4.41			4.41	1.150 g/L
Fe	2.61			2.61	0.680 g/L
Zn	2.91			2.91	0.759 g/L
Ca	1.53			1.53	0.398 g/L
Mg	2.38			2.38	0.621 g/L
wash	g	g			
Al	0.45	3.68	0.73	1.19	0.309 g/L
Fe	0.45	2.18	0.43	0.88	0.230 g/L
Zn	0.39	2.43	0.48	0.87	0.227 g/L
Ca	1.66	1.27	0.25	1.91	0.498 g/L
Mg	0.21	1.99	0.40	0.60	0.157 g/L
"9days"	g	g	g		
Al	1.49	0.99	0.20	1.69	0.465 g/L
Fe	1.28	0.74	0.15	1.43	0.392 g/L
Zn	11.06	0.73	0.14	11.21	3.080 g/L
Ca	1.53	1.59	0.32	1.85	0.509 g/L
Mg	0.17	0.50	0.10	0.27	0.073 g/L

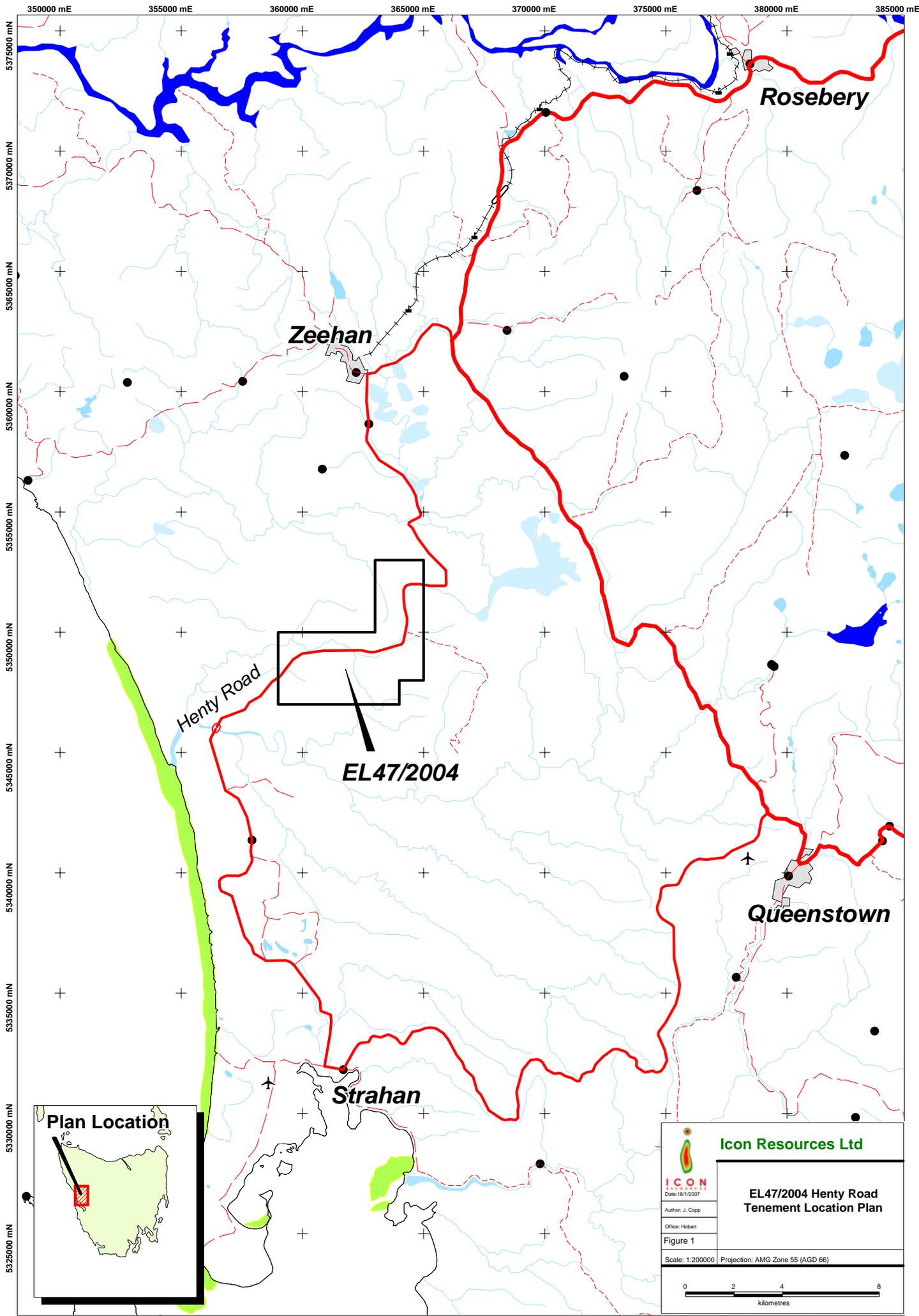
**Extraction**

	extraction %			for 100% balance	
	1st stage	wash	2nd stage	total %	
Al	56.4	5.8	19.1	81.3	86.7
Fe	38.6	6.6	18.9	64.1	54.9
Zn	11.4	1.5	43.4	56.3	64.9
Ca	25.2	27.4	25.3	78.0	75.0
Mg	64.3	5.6	4.5	74.3	97.3

**Acid Usage**

	acid usage g			
	1st stage	wash	2nd stage	total %
Al	24.05	2.46	8.14	22.60
Fe	4.58	0.79	2.24	4.96
Zn	4.37	0.58	16.57	14.04
Ca	3.74	4.06	3.76	7.54
Mg	9.60	0.83	0.67	7.24
additional for CaSO4&PbSO4	0.86	0.86	2.66	2.86
total	47.20	9.59	34.05	
acidity	4.84	2.99	54.7	40.75
used	52.04	12.6	88.7	100.00

Acid added 153.3 g  
 removed in decants 5.4 g  
 nett 147.9 g  
 435.1 kg/tonne ore



**Icon Resources Ltd**

**EL47/2004 Henty Road Tenement Location Plan**

Figure 1

Scale: 1:200000 Projection: AMG Zone 55 (AGD 66)

0 2 4 8  
kilometres

**Zeehan**

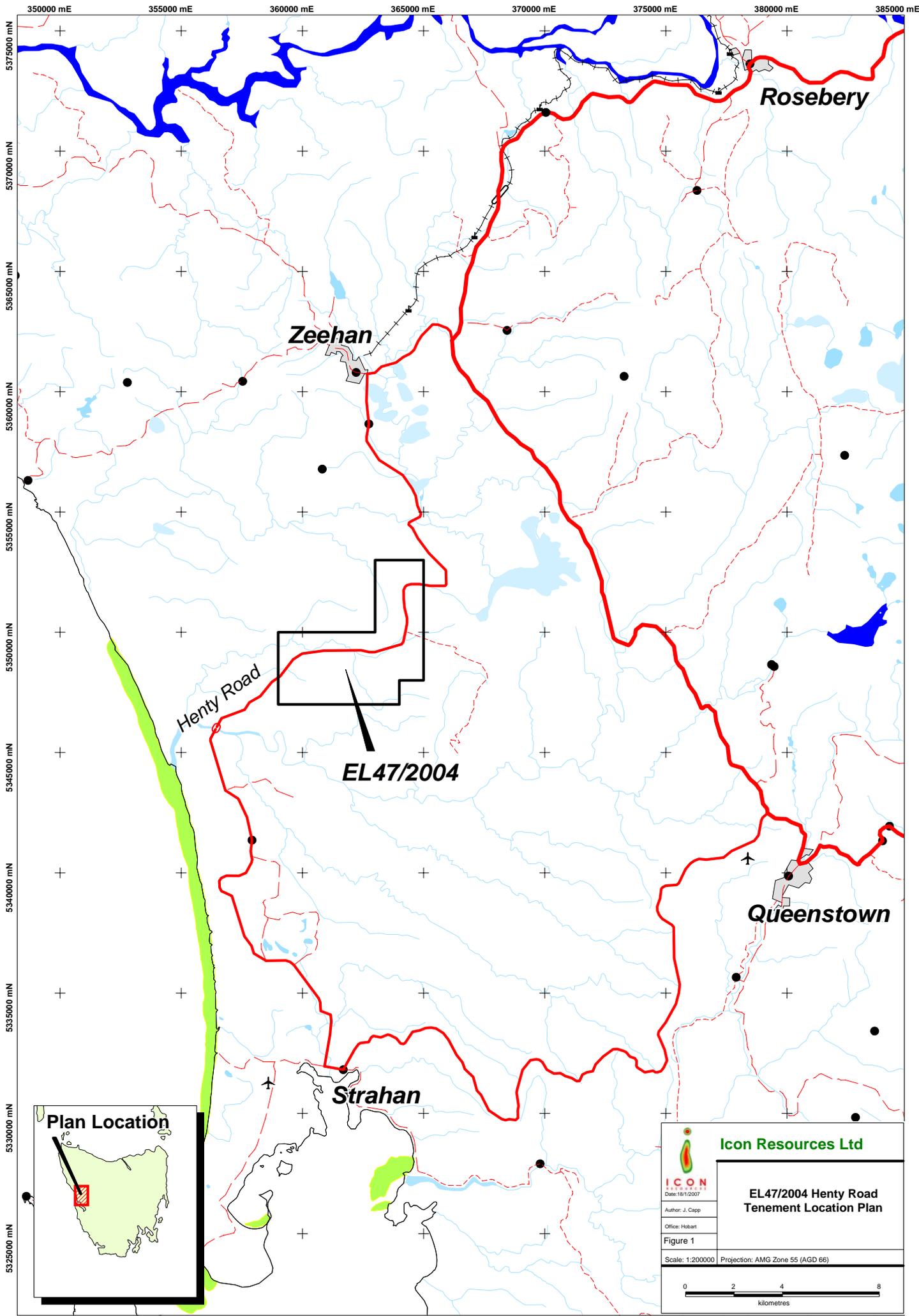
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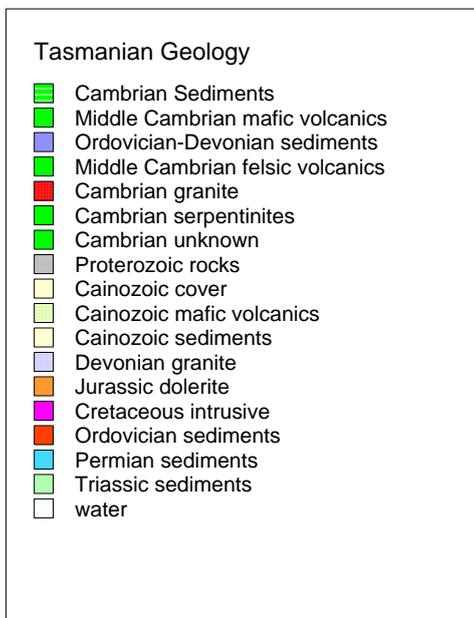
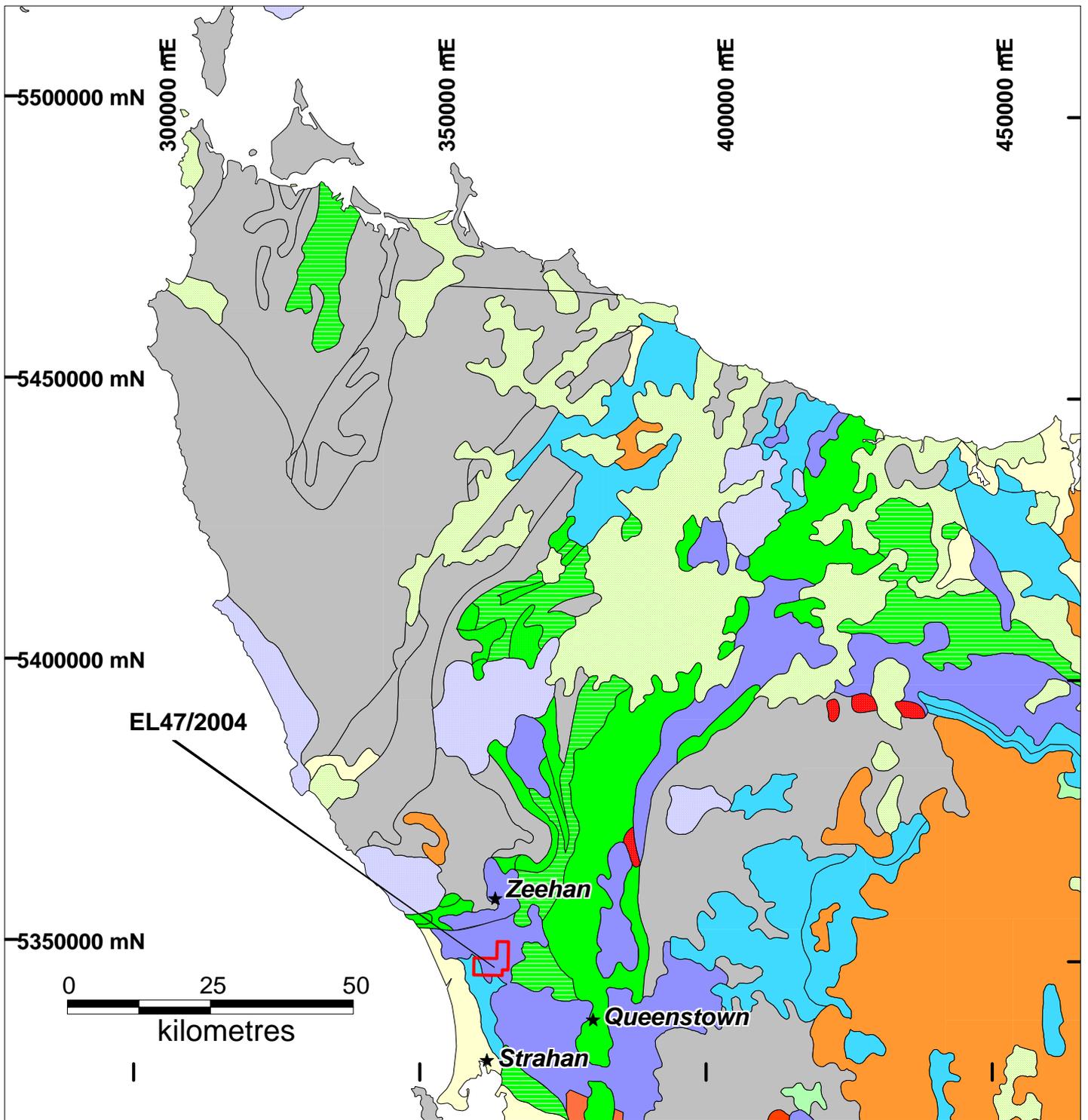
**Queenstown**

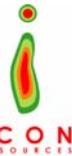
**Strahan**

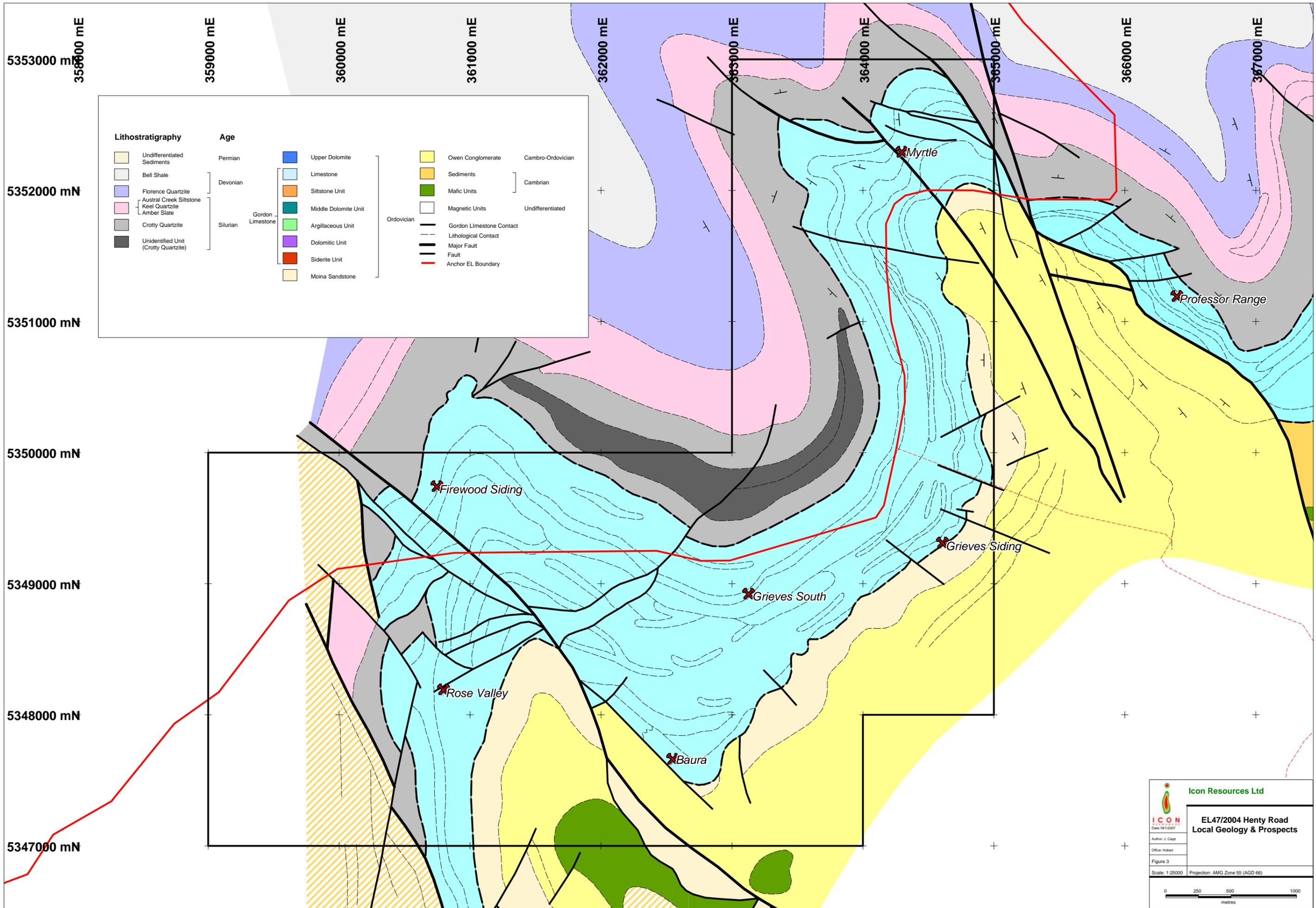
Henty Road

**EL47/2004**





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	Date: Author: J. Capp Office: Hobart	
<b>Figure 2</b>		
Projection: AMG Zone 55 (AGD 66)		



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**EL47/2004 Henty Road  
Local Geology & Prospects**

Author: J. Clapp  
 Office: Hobart  
 Figure 3  
 Scale: 1:25000 Projection: AMG Zone 55 (AGD 66)