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## **JERVOIS MINING: YOUNG NI/CO LATERITE PREFEASIBILITY STUDY TESTWORK REVIEW**

by  
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## 1. INTRODUCTION

This paper reviews the results of prefeasibility metallurgical testwork carried out during the period June to October 2000 on composite samples of low-grade, ROM and high-grade Ni/Co laterite materials from the Jervois Mining Young, NSW, nickel/cobalt laterite resource. The low-grade testwork was carried out by Metcon Laboratories and the ROM and high-grade testwork by Hydrometallurgical Research Laboratories (HRL). Sighter resin-in-pulp testwork was carried out by Clean TeQ and mineralogical work was carried out by Pontifex & Associates.

### Low Grade Testwork:

The purpose of the low-grade testwork was to identify whether there was potential for upgrading Young low-grade material by size separation and/or flotation. Neutralisation and settling characteristics were also evaluated. Magnesite, limestone and magnesia precipitation of Ni/Co hydroxide were assessed. Carbonate precipitation of magnesite from barren neutral solutions was also briefly evaluated. The low-grade composite was prepared from air core composite samples from the Thuddungra East and West Arms, which represent approximately 46% and 33% respectively of a 70 MT indicated and inferred low-grade resource.

### High Grade and ROM Testwork:

The purpose of the high-grade and ROM test work was to conduct an updated pressure leach characterisation of high-grade and ROM nickel/cobalt laterite samples from the Young resource. The samples were intended to be more representative than earlier high-grade preliminary RC drill hole and composite samples tested by HRL in September 1999. The work encompassed a program of ore characterization, feed upgrading, feed thickening, pressure acid leaching and leach residue thickening. Sighter resin-in-pulp testwork was also carried out.

Three samples representing the Ardnaree, West Thuddungra and East Thuddungra Areas and one sample representing a High-Grade composite from all three areas were evaluated.

The samples from the Ardnaree, West Thuddungra and East Thuddungra areas represented approximately 26%, 49% and 18% respectively of 126 MT of indicated and inferred resources of plus 0.6% Ni equivalent, with a total grade of 0.96% Ni equivalent. The sample of High Grade represented the combination of approximately 34%, 29% and 22% of the indicated and inferred resources at plus 0.8% Ni equivalent from each of the three areas, at a total grade of 1.26% Ni equivalent in a total resource of 44 MT.

### Data Sources:

Information for this paper was drawn from the following reports prepared for Jervois Mining:

- Young Low-Grade Ni/Co Laterite Prefeasibility Study Testwork Review, IPDS January 2001
- Young ROM & High Grade Ni/Co Laterite Prefeasibility Study Testwork Review, IPDS January 2001
- Evaluation for Upgrading Ni/Co Low Grade Laterite Ore, Report No. 00886, Metcon, October 2000
- Prefeasibility Metallurgical Program, Phase 1: Pressure Acid Leach characterisation of Metallurgical composites, Report No. 0697, Hydrometallurgical Research Laboratories, October 2000 and Addenda
- Composite Samples for Metallurgical Samples Collection, Douglas McKenna & Partners, May 2000
- Mineralogical Reports: No 8017 September 2000 and No 7995 July 2000, Pontifex and Associates
- Investigation Report, Extraction of Ni & Co using Clean-iX™ Resin Technology, Clean TeQ, Oct 2000

## 2. SUMMARY

### 2.1 Low Grade Testwork

A low-grade composite sample representing approx. 80% of a 70 Mt low-grade Ni/Co laterite resource at Young, NSW, was prepared from selected one metre intervals of 0.3 to 0.6% Ni equivalent grade taken from 11 air core drill holes within the East and West Thuddungra areas of the Young resource.

Ore types within the composite included approximately 36% hematite, 50% limonite and 14% saprolite. Sample depths ranged from 21 to 63m below surface.

The composite assayed 3990 ppm Ni, 410 ppm Co, 43 ppm Sc, 8760 ppm Cr, 1490 ppm Mn, 240 ppm Zn, 72 ppm Cu, 32.5% SiO<sub>2</sub>, 28.5% Fe, 6.3% Al, 2.85% Mg, 0.3% Ca and 0.12% C.

Moisture content was 20% and SG of material dried for 36 hours at 50C and crushed to minus 2mm was 2.9.

Upgrading of the low-grade composite by flotation or screening was largely unsuccessful.

Acid-neutralisation investigations showed that the acid neutralising capacity of the low-grade material was limited to pH 1.4, even when using finely ground material with hot, 50 gpl free acid, synthetic HPAL solution. Additional neutralising materials such as finely ground magnesite or limestone were found to be necessary for achieving higher pH levels during acid neutralisation.

Finely ground magnesite was found to partially precipitate nickel from hot synthetic HPAL leach solutions between pH 4.5 and pH 7.0, but to have no affect on cobalt.

Finely ground limestone was found to fully precipitate nickel and cobalt from hot synthetic HPAL leach solution from pH 4.5 to pH 7.0.

Magnesium and calcium carbonate precipitation from spent sulphate solutions at pH 8.5 by 50 psig carbon dioxide gaseous overpressure was found to be technically feasible, but requiring pressure filtration to prevent redissolution of carbonate on release of carbon dioxide overpressure to atmospheric pressure.

Preliminary settling tests on finely ground low-grade ore indicated similar settling rates (5m/h) for both diluted fresh ore (pH 7.8, 4% solids) and partially neutralised ore (pH 2.3, 4% solids) at Magnafloc 358 dosages of 10 and 34 g/t respectively. Ultimate densities of freshly ground ore and partially neutralised ore were expected to be in the range of 35-45% solids.

Mineralogical studies showed that nickel in low-grade ore was difficult to identify as a specific mineral phase and that intergrowth of most of the mineral phases prevented effective recovery of nickel by either magnetic separation or flotation.

Recommendations for future work included finer grinding of low-grade ore to attempt to improve free acid neutralising capacity and reaction rate and finer grinding of magnesite for enhanced acid neutralisation and Ni/Co hydroxide precipitation

### 2.2 High Grade and ROM Testwork

One high-grade composite (High Grade) and three ROM composites (Ardnaree, East Thuddungra and West Thuddungra) were prepared from one metre intervals of plus 0.6% Ni equivalent within 100 air core drill holes within the Ardnaree and East and West Thuddungra areas. These three areas are understood to account for approximately 92% of a 126 Mt ROM laterite resource at Young at a cut-off of 0.6% Ni equivalent.

The High Grade and Ardnaree composites were generally similar in ore type, consisting primarily of weathered serpentinite 35-42%, saprolite 29-31% and limonite 26-32%. Hematite, fresh serpentinite and other clay were low at 0-3%.

The East Thuddungra and West Thuddungra composites had significantly more hematite (6-10%), slightly less weathered serpentinite (10-24%), and much more extreme levels of both limonite (18 - 58%) and saprolite (21-48%). Clay and fresh serpentinite were low at 1-3%.

Compositions ranged from 0.64 to 1.01% Ni, 0.068 to 0.083% Co, 27 to 41 ppm Sc, 23.2 to 37.3% Fe<sub>2</sub>O<sub>3</sub>, 6.2 to 10.8% Al<sub>2</sub>O<sub>3</sub>, 5.9 to 11.1% MgO and approx. 28 to 43% SiO<sub>2</sub>.

The results of the testwork provided general confirmation of earlier high-pressure acid leach characteristics on less representative samples, though with marginally lower nickel and cobalt recoveries due to lower head grades and/or more complex mineralogy. Nickel recoveries were 93-95% for three ore types and 86% for a fourth. Cobalt extractions were 94-96%.

Feed size was 100% passing 210 microns on minus 2mm feed material. Leach time was 30 to 90 minutes and leach temperature 250-265C.

Slightly higher acid additions were found necessary to offset a tendency for higher leach residues on all composites. Acid additions ranged from 0.33 to 0.48 t/t HPAL feed.

Leach impurity levels were favourably low, including iron (1-1.3 gpl), aluminium (0.7-2.5 gpl) and chrome (<100ppm). No oxidized chromium was present.

Feed and leach residue settling characteristics were found to be reasonable, with potential for further improvement.

A preliminary economic assessment of nickel recoveries versus acid consumption suggested that optimum recoveries were slightly below maximum recoveries achieved in the testwork when using a conventional wash circuit for nickel recovery.

Preliminary resin-in-pulp testwork on HPAL high-grade residue pulp indicated the possibility of higher metal recoveries, higher intermediate product purity and a simpler front end processing circuit when using RIP processing rather than conventional CCD washing, acid neutralisation, iron precipitation and hydroxide precipitation.

Recommendations for future prefeasibility testwork included economic studies of process options, follow up confirmatory tests on resin-in-pulp processing, electron microprobe analysis of leach residue samples, a review of the acid demand of the plus 2mm reject fraction, investigation of the wetting behaviour of the different ore types, additional thickening tests, comparative costs of neutralisation for low cost on-site neutralising agents and testwork on the most likely plant feed materials and process water available in the early years of mine life.

## PART A – LOW GRADE TREATMENT

- Size by Size Analysis – Total low-grade ore, Section 3
- Flotation – Minus 2mm Low-grade Composite, Section 4
- Low-grade, Limestone and Magnesite Neutralisation, Section 5
- Low-grade Mineralogy, Section 6
- Conclusions, Section 7

### 3. SIZE BY SIZE ANALYSIS – TOTAL LOW GRADE ORE

**Total Ore Composite:** A total ore composite was prepared from four Jervois supplied low-grade feed samples. Preparation included separate treatment of each of the four feed samples by drying at 50C for 36 hours, dry screening each product at 2", bottle rolling the plus 2" coarse size agglomerates and combining the resulting minus 2" material into a single composite.

The head analyses of Total Ore are shown below in Table 1. The analyses compared favourably with individual drill hole samples used in earlier Amdel low-grade testwork on the Young resource.

**Table 1: Head Analysis of Jervois Young Low-grade Ni/Co "Total Ore" Composite, compared with earlier Amdel Low-grade Drill Hole Samples**

Assay	Units	Metcon Total Ore*	Amdel YA184	Amdel YA 226
		East/West Thuddungra	East Thuddungra	West Thuddungra
Ni	ppm	3990	3200	3600
Co	ppm	410	500	500
Sc	ppm	43	50	40
Zn	ppm	240		
Cu	ppm	72		
Cr	ppm	8760	8500	12400
Mn	ppm	1490		
Fe	%	28.5	28.8	28.3
SiO <sub>2</sub>	%	32.5	28.1	30.6
Mg	%	2.85		
Total C	%	0.124		
SG solids	g/cm3	2.855		

\* after drying at 50C for 36 hours

**Size by Size Analysis:** Preparation of Total Ore for size by size analysis involved thorough separation of all agglomerates into individual particles by a combination of:

- wet screening at 150 microns
- reprocessing the plus 150 micron size fraction by slurry agitation for 30 minutes to break down the weak agglomerates formed during earlier drying at 50C
- wet screening of the combined plus 150 micron reprocessed material and the earlier minus 150 micron screen undersize material into thirteen size fractions ranging from minus 25 micron bottom size to plus 9500 micron top size.

Size by size assay results indicated that there appeared to be little opportunity for major upgrading of the low-grade ore by rejection of either silica or Mg into a coarse size fraction. In this sense the Young low-grade ore would appear to differ significantly from higher grade laterite resources at Syerston, Cawse or Ravensthorpe, where the order of 30% or higher upgrading is understood to be achieved by screening.

Specific characteristics of the Young low-grade material included:

- over 42% of the weight fraction and 48% of the Ni distribution in the minus 25 micron size fraction (i.e. only minor upgrading of Ni in the fines fraction)
- only 26% of the Co distribution in the minus 25 micron size fraction (i.e. significant depletion of Co in the fines fraction)
- a steady increase in Ni grade with decreasing particle size, from 1970 ppm Ni at plus 9500 microns to 4460 ppm Ni at minus 25 microns (i.e. Ni grade doubling from coarse to fine size)
- a maximum Co grade of 680 ppm Co at 212 microns, decreasing with both coarser and finer particle size to 380 ppm Co at 9500 microns and to 255 ppm Co at minus 25 microns
- a maximum Fe grade of 41.8% Fe at 1180 microns, decreasing to 27.7% Fe at 9500 microns and 21.8% Fe at minus 25 microns
- a minimum Mg grade of 1.88% Mg at 1180 microns, increasing to 3.9% Mg at 9500 microns and 3.1% Mg at minus 25 microns
- an essentially constant Al grade at 5.0 to 6.8% Al over all size ranges, except for an increase to 8.7 - 13.2% Al in the plus 4750 - 9500 micron range
- a maximum SiO<sub>2</sub> grade of 37-38% SiO<sub>2</sub> from 75 to 150 microns, decreasing to 22% SiO<sub>2</sub> at 9500 microns and 34% SiO<sub>2</sub> at minus 25 microns
- broadly similar Ni, Mg, Al and SiO<sub>2</sub> distributions in the coarse and fine ends of the size range, including 43-49% distribution in the minus 25 micron size fraction and 10-18% distribution in the plus 2000 micron size fraction
- generally similar Co and Fe distributions in the coarse and fine ends of the size range, with 27-34% in the minus 25 micron fraction and 13-19% in the plus 2000 micron fraction

**Screen Upgrading:** The best upgrading that could be achieved by screening the low-grade material appeared to be rejection of plus 2mm material containing 2509 ppm Ni. This theoretically left behind a slightly upgraded product containing 90.8% of the feed nickel and 84.3% of the feed cobalt in a product assaying 4098 ppm Ni and 400 ppm Co. This upgrading was adopted for preparation of the feed for flotation testwork. However the upgrading was not as efficient as expected due to the lower efficiency of dry screening than wet screening. Calculated product assays and metal distributions for upgrading of low-grade ore by 2mm wet screening are shown in Table 2 below.



**Table 2: Assay and Metal Distribution of Wet Screened Minus 2mm Fraction versus Feed Material and Wet Screened Plus 2mm Material – Young Low-Grade “Total Ore” Composite**

Size	Wt	Assay						Distribution					
		Ni	Co	Fe	Mg	Al	SiO <sub>2</sub>	Ni	Co	Fe	Mg	Al	SiO <sub>2</sub>
micron	%	ppm	ppm	%	%	%	%	%	%	%	%	%	%
-2mm	85.8	4098	400	25.8	2.8	6.2	33.2	90.8	84.3	81.2	87.7	82.2	90.5
+2mm	14.2	2509	450	36.1	2.3	8.2	21.1	9.2	15.7	18.8	12.3	17.8	9.5
Feed*	100.0	3872	407	27.3	2.7	6.5	31.5	100.0	100.0	100.0	100.0	100.0	100.0

Data Source: Recalculated from Metcon Report 00886, Table 3, using calculated feed assay from Table 3

**Size Fractions for Mineralogy:** Four different size fractions were chosen to provide a broad range of sizes and different Ni, Co and Fe grades for mineralogical evaluation by Pontifex and Associates:

- +4mm, representing a coarse size fraction
- –2mm +1.18mm, representing 5.3 wt% at 3250 ppm Ni, 600 ppm Co and 41.8% Fe
- -106 micron +75 micron, representing 5.6 wt% at 3940 ppm Ni, 465 ppm Co and 24.2% Fe
- –25 micron, representing 42.3 wt% at 4460 ppm Ni, 255ppm Co and 21.8% Fe

**Grinding Versus Attrition Scrubbing:** Refer to Flotation Section 4.

#### 4. FLOTATION – MINUS 2MM LOW GRADE COMPOSITE

**Feed Preparation:** Dry screening of total ore composite at minus 2mm was used to prepare feed material for the flotation test work. This involved an initial dry screening at 2mm, bottle rolling the plus 2mm oversize consisting of clay pieces and agglomerates, and dry rescreening of the bottle rolled product. The plus 2mm material remaining after the second screening was weighed, sub-sampled and wet screened to determine the efficiency of dry screening.

The dry screening assay results showed that dry screening resulted in:

- Unwanted upgrading of the plus 2mm product in both weight (from 14 to 26 wt%) and Ni grade (from 2509 to 3459 ppm Ni)
- No change in the plus 2mm Co grade (450 ppm Co from wet screening versus 455 ppm Co from dry screening)
- Unwanted reduction of the minus 2mm product weight reporting to flotation feed (from 86 wt% to 74 wt%)
- Little change in the minus 2mm flotation feed Ni grade (from 4098 to 4090 ppm Ni) or the minus 2mm flotation feed Co grade (from 400 to 405 ppm Co)

Wet screening of the plus 2mm fraction from dry screening resulted in:

- about half the weight of the plus 2 mm dry material being retained in the plus 2mm wet fraction
- the grade of the plus 2 mm wet fraction being almost identical to the grade of the original plus 2mm wet fraction (2440 ppm Ni, 450 ppm Co and 38.4% Fe versus 2509 ppm Ni, 450 ppm Co and 36.1% Fe)
- about half the plus 2 mm dry material reporting to the minus 2mm wet fraction
- the grade of the minus 2mm wet fraction being almost identical to the Ni grade but higher than the Co grade in the original minus 25 micron wet screen fraction (4550 ppm Ni, 460 ppm Co and 24.5% Fe versus 4460 ppm Ni, 255 ppm Co and 21.8% Fe)

Although there were slight differences in head assays of total ore between the wet screening and dry screening testwork, these results suggested that:

- approximately 30% of the nickel bearing slimes in the minus 25 micron wet fraction appear to have adhered to coarse particles in the dry feed
- the nickel slimes were not removed from the plus 2mm material by dry screening and required wet screening at minus 2mm for efficient removal
- cobalt behaviour differed from nickel behaviour in that there was no apparent upgrading of cobalt in the minus 2mm fines fraction with wet screening
- the weight yield from minus 2mm wet screening was about 12% higher than from minus 2mm dry screening, even after bottle rolling the coarse oversize (85.8% versus 73.9%).

- assays for the minus 2mm wet fraction were essentially the same as those for the minus 2mm dry fraction (4142 ppm Ni, 405 ppm Co, 25.8% Fe, 2.8% Mg, 6.25% Al and 35.2% SiO<sub>2</sub> from wet screening versus 4090 ppm Ni, 405 ppm Co, 26.9% Fe, 2.8% Mg, 6.25% Al and 33% SiO<sub>2</sub> from dry screening), due to the large weight of intermediate size material of constant grade

Wet screening was clearly a highly efficient way to wash the coarse ore free of Ni-rich slimes and to shift up to 12% of the original feed weight in the form of these slimes from the plus 2mm size fraction to the minus 2mm size fraction.

**Trial grinds:** Trial grinds showed that it took about 3 minutes per kilogram of ore to grind minus 2mm dry screen undersize to a P80 of 106 microns, the feed size used for flotation testwork. The trial grinding tests were carried out on 1kg of material in a stainless steel rod mill at 40% solids at grind times of 2 and 4 minutes.

**Intensive Attrition Scrubbing:** A sample of dry screened –2mm material was subjected to high intensity attrition scrubbing in a modified Denver D12 flotation machine at 20% solids and 10 minutes. The attritioned sample was wet screened at 25 micron, with the plus 25 micron fraction sized over 212, 75 and 38 micron screens. The undersize was sized through a high-pressure mini cyclone at the Ian Wark Institute in South Australia to achieve separations at approx. 16, 1 and 5 micron. Size by size assays were determined for Ni, Co, Fe, Mn, Mg and Al.

**Flotation Feed Grind P80 of 106 Microns:** A sample of dry screened minus 2mm material was ground to a p80 of 106 microns. The ground sample was wet screened and cycloned under the same conditions as the intensive attrition scrubbed product. Size by size assays were determined as for the attrition scrubbed material.

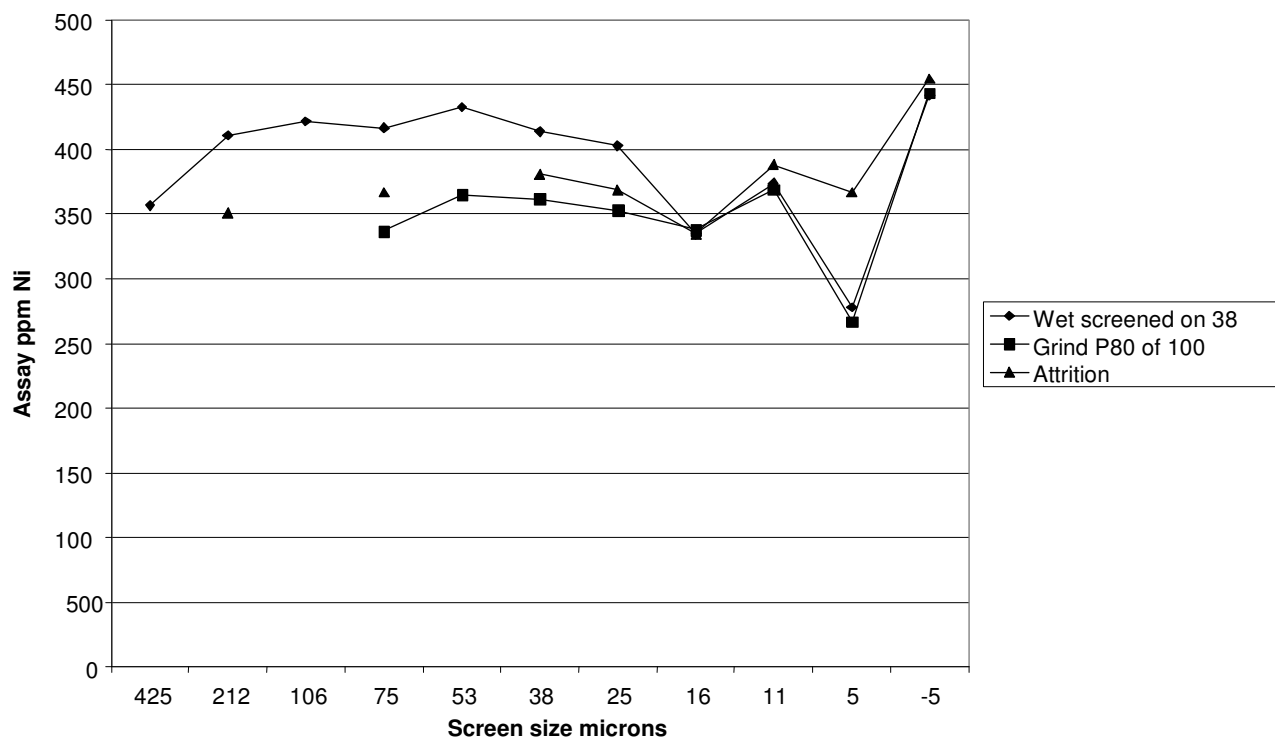
**Effect of Scrubbing versus Grinding on Size by Size Assays:** Both grinding to a p80 of 106 microns and intensive attrition scrubbing of minus 2mm dry screen material had similar physical effects on the finer size fractions:

- increased the percentage of material below 25 microns from 24% to approx. 60%
- increased the percentage of nickel below 25 microns from 24% to approx.60%
- largely unchanged the assay of the 25 micron fraction from 3900-4300 ppm Ni
- largely unchanged the assay of the minus 5 micron fraction which remained at 4400-4600 ppm Ni, 195-225 ppm Co, 6.6-7.3% Al, 2.7-3.2% Mg

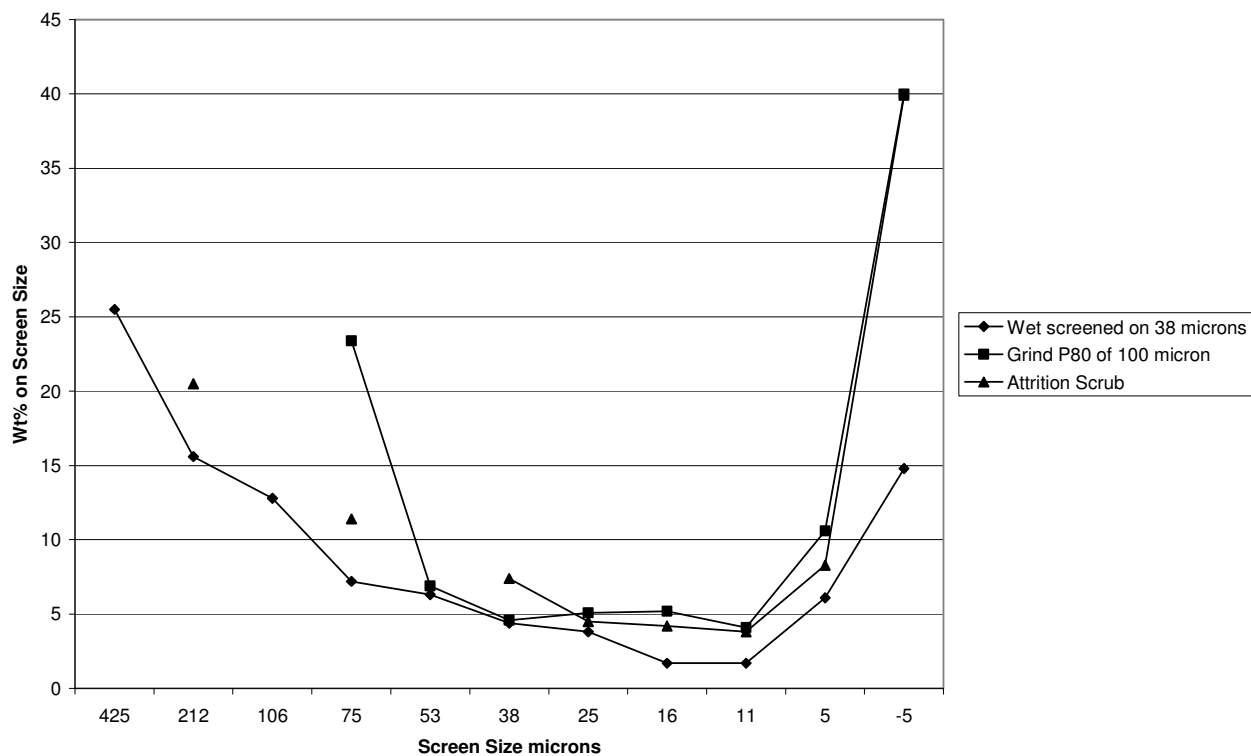
The hoped for effect of finding a Ni assay in the finest size fraction (minus 5 micron) of say more than twice the average assay in the "Total Ore" composite of 0. 4% Ni did not eventuate. However the results did show that a light grind was equivalent to intensive attrition scrubbing in generating a similar wet size-assay distribution and in maximising the grade of the minus 25 micron fraction.

Figs 1 and 2 show plots of Ni assay and wt% in specific size fractions versus particle size for different scrubbing and grinding pretreatments. Again, both scrubbing and grinding increased the quantity of slightly higher grade fine material. As an example, the minus 5 micron fraction increased from about 17 wt% in the untreated minus 2mm dry screen feed material to about 45 wt% in the scrubbed and/or attritioned product. There was no appreciable change in grade of the minus 5 micron fraction with different pretreatments, and the grade ranged from 4420 to 4550 ppm Ni in all cases. The sharp increase in grade between the minus 11 micron and minus 5 micron cut size (from about 2700 ppm Ni to about 4500 ppm Ni) suggested that a higher Ni grade might possibly occur at a particle size of below 5 microns. However the existence of material at such a fine size may be of little practical significance because the weight percentage of this material would appear to be quite small (say 25 wt% at best), and recovery would be difficult.

**Fig 1: Effect of Particle Pretreatment Method on Young Low Grade Ni Assay versus Screen Size**



**Fig 2: Effect of Particle Pretreatment Method on Wt% Versus Screen Size: Young Low Grade**



**Flotation Results:** There was essentially no difference in rougher flotation results from the three flotation tests carried out using:

- a new oxide collector KXM (recently renamed “AM2”) in conjunction with sodium silicate as a dispersant, sodium carbonate for pH control at pH 8.8-8.9, preflotation using MIBC at natural pH 8.1, 5 minutes prefloat time and 5+5+10 minutes rougher flotation time
- a conventional collector AM-15 from Clariant in conjunction with starch and sulphuric acid for pH control at pH 3.2 (460g/t acid initially and three doses of 200 g/t each subsequently), initial conditioning using MIBC at natural pH, 5 minutes prefloat time and 5+5+10 minutes rougher flotation time
- reverse flotation to remove silica minerals using Flotigam ENA from Clariant in conjunction with sodium silicate as dispersant, sodium carbonate for pH control at pH 9.2-9.3, conditioning and prefloat stage using MIBC at natural pH, 5 minutes prefloat time and 5+5+5 minutes rougher flotation time.

The second set of conditions corresponded as close as possible to the most successful of a range of laboratory flotation tests carried out on a laterite ore in India.

In all cases there was a relatively constant rougher concentrate grade of approx. 4300-4500 ppm Ni and rougher recoveries of 15 to 33% Ni. Combined prefloat plus rougher grades varied from 4200- 4300 ppm Ni at recoveries of 23 to 40% Ni, with the prefloat containing 6-7% Ni at a slightly lower Ni grade than the overall concentrate.

**Slimes flotation:** The close similarity of rougher concentrate grade to the grade of slimes recovered from wet screening suggests that the flotation tests were in fact only recovering slimes, and that the rate of recovery of nickel was a function only of the slow rate of slimes entrapment and flotation. The maximum nickel recovery by flotation of 33% was significantly below the maximum possible nickel recovery of 58%, based on a grade of 4000-4460 ppm Ni in the minus 53 micron fraction obtained by wet screening fraction of Total Ore. These results suggest that flotation was ineffective under the conditions evaluated and that further flotation work was unlikely to be successful, unless there were a more selective system available for higher recovery of high-grade slimes from this ore.

## 5. LOW GRADE, LIMESTONE AND MAGNESITE NEUTRALISATION

Neutralisation tests were carried out on synthetic acidic solutions containing either free acid only or in a limited number of cases the approximate composition of a synthetic HPAL solution.

### Neutralisation Materials:

Materials used for neutralisation were:

- Young low-grade laterite ore,
- general agricultural limestone (in the absence of limestone being available from the Young site)
- various grades of Causmag low to high-grade magnesite including:
  - ROM magnesite
  - -25 mm screen fines magnesite
  - magnesite mud from pond
  - No 2 crusher stockpile magnesite
  - 2AS high-grade ground magnesite

The objective in assessing the behaviour of on-site low-grade ore and nearby magnesite was to establish the relative effectiveness of relatively low cost site materials for free acid neutralisation. The purpose of including limestone was to use the behaviour of limestone as a reference basis.

High-grade Causmag caustic calcined magnesia, grade SGW, was also evaluated as a precipitating agent for nickel and cobalt hydroxide from synthetic HPAL solution after acid neutralisation and iron precipitation.

The Causmag magnesite and caustic magnesia products were from Causmag, a subsidiary of Orissa of India, and a major local magnesite and magnesia producer located adjacent to the Jervois Young laterite site. Causmag is understood to have rights to magnesite overlying certain areas of the Young laterite and adjoining mine sites.

The neutralisation materials were evaluated at both coarse (-1 to -2mm) and fine sizes (generally 80% passing 53 micron) and at low (20C) and high solution temperatures (90C).

Head assays showed that compared with a Causmag ROM magnesite assay of 83.3%  $\text{MgCO}_3$  and 0.74%  $\text{CaCO}_3$ , the Causmag waste magnesite products varied from a low of 22.7%  $\text{MgCO}_3$  and 0.5%  $\text{CaCO}_3$  (-25mm screen fines) to a high of 55.2%  $\text{MgCO}_3$  and 0.74%  $\text{CaCO}_3$  (mud from pond). The high-grade product, 2AS high-grade magnesite, contained 98.2%  $\text{MgCO}_3$  and 0.5%  $\text{CaCO}_3$  (based on Causmag assays).

Based on Ca, Mg and total C assays, low-grade Young ore ("Total ore") was deduced to contain no magnesite (0.0%  $\text{MgCO}_3$ ), very little limestone (0.3%  $\text{CaCO}_3$ ) and largely a combination of free silica, magnesium silicate, iron hydroxide and aluminium hydroxide. Low-grade assays were 31.5%  $\text{SiO}_2$ , 27.3% Fe, 6.52% Al, 2.85% Mg, 0.298% Ca, 0.124% total C, 0.387% Ni and 0.04% Co.

With the virtual absence of any magnesite or limestone from the low-grade ore, these assay results suggested that the neutralisation capacity of low-grade material would be limited by the ability of magnesium silicate, ferrous hydroxide and aluminium hydroxide to react with acid in the HPAL discharge solution.

It was already known from the earlier Amdel hot atmospheric leach test work that the low-grade would consume large quantities of acid at 90C. However, it was not known whether the ore would react with acid concentrations of below 50gpl  $H_2SO_4$ , as expected in typical HPAL discharge slurry produced by treatment of high-grade Jervois ore by HPAL.

## Neutralisation Test Results

### 5.1.1 50 gpl free acid barren feed solution neutralisation: $MgCO_3$ , $CaCO_3$ and Total Ore

**Room Temperature and Coarse to Fine Grind Size:** At room temperature, agricultural limestone was found to be highly reactive at the purchased feed size of 95% passing 850 microns. In particular it showed visual effervescence of carbon dioxide and an increase in solution pH to slightly over pH 4.0 within 2 hours.

ROM magnesite at a much coarser particle size of 100% passing 2 mm showed no visual reaction and failed to raise the pH to above 1.0 in two hours.

Total ore at 100% passing 2 mm achieved a pH of less than 1.0 after 4 hours of reaction.

When the grind sizes of both the limestone and magnesite were reduced to 100% passing 75 micron, both materials became significantly more reactive. The limestone reached a pH of 7.7 after 3 hours, but the magnesite was still slow, achieving only pH 9.1 after 42 hours. Essentially stoichiometric reagent additions were used in both cases. It was decided to use the combination of fine particle size and a high reaction temperature of 90C to determine if reaction kinetics might improve under more closely simulated HPAL discharge conditions.

### 90C Reaction Temperature and Fine Grind Size:

**Magnesite:** Various grades of Causmag magnesite, ground to a p80 of 53 micron, were found to react rapidly with a hot 50 gpl acid solution when a one-off near stoichiometric addition of magnesite was made to a known volume of solution. The higher grade (53-83%  $MgCO_3$ ) magnesite materials (ROM, mud from pond and No 2 crusher reject) reached pH 5-6.0 within 10-30 minutes and pH 7.0 -7.1 within 50-70 minutes, based on 1.0-1.2 times stoichiometric addition for 50 gpl acid. The lower grade -25mm screen fines (22.7%  $MgCO_3$ ) reacted more slowly, taking almost 4 hours to reach pH 4.9 and not climbing to higher than pH 6.3 after 4.5 hours. The low final pH was attributed to insufficient excess magnesite being provided (only 102% of stoichiometric magnesite requirement was used).

**Limestone:** Purchased agricultural limestone, ground to a p80 of 53 micron, reacted much faster than magnesite with the hot acid feed solution, reaching pH 7.6 in 1 minutes and pH 8.2 in 2 minutes at an limestone addition of approx. 110% of stoichiometric.

**Total Low-grade Ore:** Ore ground to 100% passing 2 mm yielded pH 1.1 and 14 gpl acid after 4 hours of reaction with 50 gpl acid. By contrast ore ground to a p80 of 75 micron yielded pH 1.1 and 14 gpl free acid after 4 hours of reaction. Ore ground to a p80 of 53 microns yielded pH 1.6 and 4 gpl free acid after 4 hours of reaction. These results suggested that low-grade ore needed to be ground to at least a p80 of 53 microns to be reactive. The effects of a finer grind size were not investigated in the current test program, but are considered to be worth evaluating in future tests due to the potential for greater neutralising capacity at finer grind size.



100 g ore was added to 200 ml of acid solution in a single addition at the start of two of the tests and 300 g to 600 ml in one other test. The reaction was carried out in a glass beaker with agitation provided by a stainless steel stirrer.

Reaction products from the neutralisation of free acid with total ore included soluble Mg (14-19% dissolution), Fe (2-4% dissolution), Al (5-8% dissolution), Ni (possibly 10-15% dissolution) and Co (possibly 30-50% dissolution).

**Sulphate distribution:** Assuming that all iron in solution were present as ferric sulphate, an evaluation of the sulphate tied up as free acid,  $\text{MgSO}_4$ ,  $\text{Al}_2(\text{SO}_4)_3$ ,  $\text{NiSO}_4$ ,  $\text{CoSO}_4$  and  $\text{Fe}_2(\text{SO}_4)_3$  indicated that over the range of free acid neutralisation at 90C from 50 to 14 gpl, there appeared to be generally:

- only 20 to 24%  $\text{SO}_4$  present as fully neutralised  $\text{MgSO}_4$
- 15 to 40%  $\text{SO}_4$  present as free acid, which would still require subsequent limestone addition for neutralisation
- 40 to 60%  $\text{SO}_4$  present as a 50:50 mixture of  $\text{Al}_2(\text{SO}_4)_3$  and  $\text{Fe}_2(\text{SO}_4)_3$ , which would require subsequent further limestone addition for neutralisation
- less than 1%  $\text{SO}_4$  as Ni and Co
- an apparent 20% decrease of total sulphate in solution with a decrease in free acid from 18 to 14 gpl, possibly due to some type of sulphate precipitation in the solid phase
- an apparent increase from 15% to 19% Mg dissolution, 5 to 8% Al dissolution and 2 to 4% Fe dissolution with a decrease in grind size from 100% passing 2 mm to 80% passing 75 micron

These results suggest that low-grade ore might be of only limited use in free acid neutralisation, due to simultaneous leaching and take up of free acid by iron and aluminium at the same time as sulphate consumption by magnesium in the ore. It seems that possibly 40 to 60% of the reduction in free acid was simply due to leaching of iron and aluminium, which will in turn require later addition of limestone for subsequent neutralisation. Thus, effective free acid neutralisation by ore might only be of the order of the 20 to 24% due to magnesium sulphate formation.

**Ni and Co Leaching:** The apparent simultaneous leaching of up to 15% nickel and up to 50% cobalt during neutralisation of a simple 50 gpl acid solution (to approx. 14 gpl acid) with low-grade ore (0.39% Ni and 0.04% Co) appeared to be beneficial to project economics. However, the variability of results suggested further tests needed to be carried out to confirm the reliability of the current data.

**Thickener area issues:** The issue of settling rate and required thickener area for low-grade ore thickening and washing after partial free acid neutralisation was considered briefly by preliminary settling tests on natural and partly leached low-grade ore. The thickener size for separation of pregnant liquor from partly reacted low-grade ore would obviously need to be much larger than that required for use magnesite or limestone alone, due to the much larger tonnage throughput requirements for use of low-grade ore rather than magnesite or limestone. A combination of low-grade ore followed by magnesite or limestone appeared to be the preferred alternative, if the additional nickel recovery is sufficient to pay for the extra thickener requirements.



**Resin-in-pulp:** Separate testwork on the use of resin-in-pulp on pH 4-4.5 HPAL slurry from Young high-grade ore leaching indicated that there was potential for resin-in-pulp to be successfully used instead of thickening and CCD washing for nickel recovery from high-grade ore –see Part B of paper. Whether or not resin-in-pulp processing might be more economic with low-grade ore for acid neutralisation instead of only limestone or magnesite has yet to be assessed. However the possibility of using resin-in-pulp would certainly improve the chance to use low-grade ore for part of the neutralisation requirements without the penalty of high additional thickener costs for solid-liquid separation.

### 5.1.2 50 gpl free acid synthetic HPAL solution neutralisation: $\text{MgCO}_3$ versus $\text{CaCO}_3$

A synthetic HPAL feed solution containing approx. 50 gpl free acid, 0.4 gpl Ca, 8.0 gpl Mg, 0.7 gpl Al, 0.2 gpl Co, 2.0 gpl Ni, 2.4 gpl Fe T (83%  $\text{Fe}^{+3}$ ) and 0.03 gpl Cr was prepared from sulphate salts and sulphuric acid. The solution colour was light green.

Causmag ROM magnesite and agricultural limestone, both ground to 100% passing 75 micron, were used as neutralising agents.

Procedure for neutralising agent addition to hot solutions (90C) consisted of three stages:

- one-off addition of neutralising agent to the feed solution to achieve approx. pH 4.0 and filter off solution left after reaction
- raise pH to 7 if possible, by adding further neutralising agent and filter off solution
- raise final pH to pH 8.5 or as high as possible on residual solution

The pH change with neutralising agent addition was followed and solution samples were taken over the complete pH range achieved.

Results, as shown by Figs 3 and 4 attached, showed that:

- both  $\text{MgCO}_3$  and  $\text{CaCO}_3$  as neutralising agents separately achieved pH 7.0 when added to the synthetic HPAL solution
- both reagents achieved broadly similar profiles of iron, chromium and aluminium concentration in solution versus pH
- impurity levels in solution at pH 6.0-6.2 for  $\text{MgCO}_3$  neutralisation and from pH 5.8-6.0 for  $\text{CaCO}_3$  neutralisation were <100 ppm Fe, <70 ppm Al, <1ppm Cr
- $\text{MgCO}_3$  did not precipitate Co but appeared to rather strangely precipitate approx. 30% Ni from pH 6.2 to 7.0, suggesting that stage-wise and/or counter current addition of  $\text{MgCO}_3$  might be needed to try to limit the amount of excess  $\text{MgCO}_3$  present above pH 6.2 and therefore the extent of Ni precipitation in this region
- $\text{CaCO}_3$  started to precipitate Co and Ni above pH 5.0 with the precipitation being essentially complete by pH 7.0. This was considered to be rather unusual, considering that limestone is traditionally regarded as having no neutralisation capacity above pH 4.0. This step was considered novel.
- Mg solubility in solution rose to approx. 25 gpl Mg in the case where magnesite was used as the neutralising agent, versus only approx. 9.5 gpl Mg where limestone was used as the neutralising agent

Fig 3: Synthetic HPAL Solution Concentration Versus pH (MgCO<sub>3</sub> Addition at 90C)

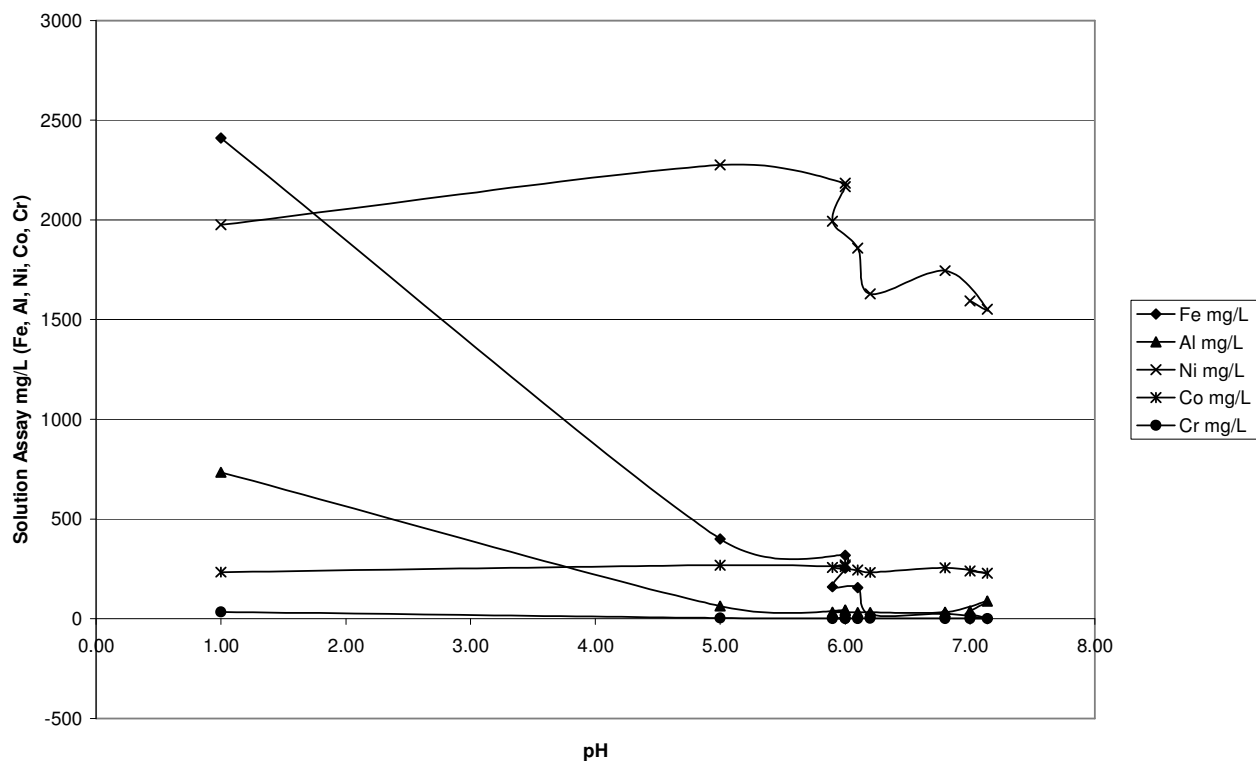
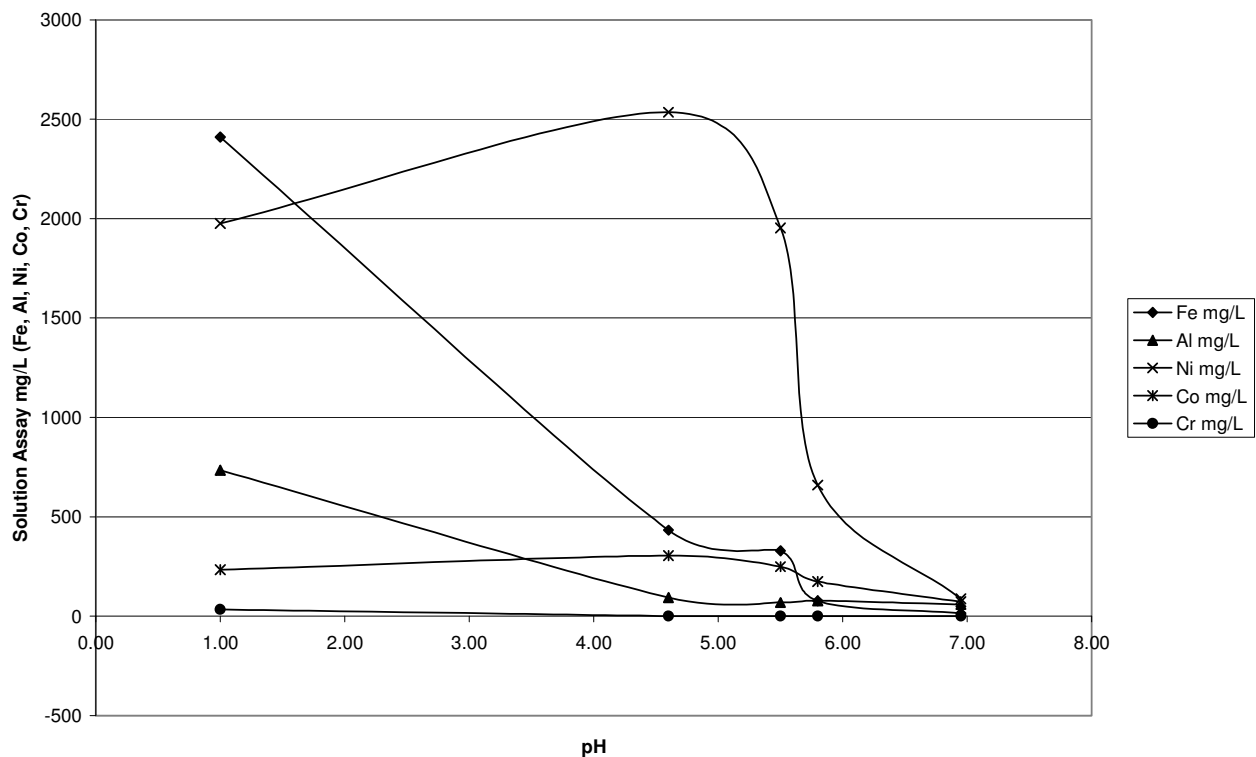


Fig 4: Synthetic HPAL Solution Concentration Versus pH ( CaCO<sub>3</sub> Addition at 90C)



No precipitate samples were taken to determine the precipitate compositions in the above tests, but samples were taken from later tests.

### 5.1.3 50 gpl free acid HPAL feed solution neutralisation: Sequential Total Ore, $\text{MgCO}_3$ and $\text{MgO}$

The purpose of this test at 90C was threefold:

- firstly to achieve sequential free acid neutralisation with ore, followed by  $\text{MgCO}_3$  addition to pH 4.5 while aerating the solution to oxidise ferrous iron to ferric iron
- secondly to raise the pH to 7 with  $\text{MgCO}_3$  to precipitate the majority of Ni and Co from solution
- thirdly to raise the pH above 8 using  $\text{MgO}$  to remove any residual Ni and Co in solution.

Results showed:

- little or no extraction of Ni and Co from the low-grade ore used for acid neutralisation to 19 gpl acid (in contrast to Metcon's estimation of 15% Ni extraction and 43% Co extraction, but this is difficult to reconcile with the limited change in solution composition over the duration of low-grade ore addition to a free acid concentration of 19 gpl)
- possibly up to 15% precipitation of Ni, 0% precipitation of Co, 15% Al precipitation and 98% Fe precipitation, after addition of ROM  $\text{MgCO}_3$  to lower the free acid from 19 gpl acid to 0 gpl acid (pH 3.8)
- precipitation between pH 4.3 (3.8?) and 6.9 by  $\text{MgCO}_3$  of 34% of the remaining Ni, 24% of the remaining Co, plus all residual iron and most of the residual Al in solution into a precipitate assaying 2.9% Ni, 0.17% Co, 0.3% Fe, 0.8% Al and 25.5% Mg.
- production of an intermediate precipitate grade at pH 6.9 by  $\text{MgCO}_3$  that was:
  - substantially higher in grade than Jervois low-grade ore (0.39% Ni, 0.04% Co, 32%  $\text{SiO}_2$ , 28.5% Fe, 6.3% Al and 2.9% Mg)
  - substantially higher in grade than Jervois high-grade ore (approx. 1.0% Ni, 0.07% Co, 18.9% Fe, 3.31% Al and 5.9% Mg).
  - potentially capable of further improvement in grade by staged addition of  $\text{MgCO}_3$  and staged separation of the resulting precipitate, if warranted.
- precipitation between pH 6.9 and 8.6 by  $\text{MgO}$  addition of 89-99% of the residual Ni and 83-6% of the residual Co in solution to form a precipitate assaying 23.5%Ni, 3.6% Co, 0.10% Fe, 0.22% Al and 7.2% Mg and capable of being upgraded by two-stage  $\text{MgO}$  precipitation if desired.

### 5.1.4 Recovery of Ca and Mg by Sodium Carbonate and $\text{CO}_2$ Addition

**Sodium Carbonate Addition to Magnesium Sulphate:** Sodium carbonate addition as a 0.25 molar solution (pH 11.6) to a 0.25 molar magnesium sulphate solution (5.4 gpl Mg, pH 6.3) resulted in:

- Precipitation of 94% of the magnesium (to 0.3 gpl Mg) at ambient conditions
- Precipitation of 99.6% of the magnesium (to 0.019 gpl Mg) at 90C.

The precipitate appeared to be stable at above pH 9.0 corresponding to the addition of 4 ml of sodium carbonate. The pH after addition of 100 ml of sodium carbonate, and presumably after completion of the reaction with 100 ml of magnesium sulphate, was pH 10.6

Addition of 50gpl acid solution (0.5 molar) to the resulting precipitate slurry at pH 10.6 under ambient conditions resulted in the commencement of redissolution of the precipitate when the pH was below pH 9. Redissolution reached approx. 50% at a pH 8.1 for 72.5 ml acid addition, suggesting that the reverse reaction at ambient conditions was slower than the forward reaction.

**Sodium Carbonate Addition to Calcium Sulphate:** Sodium carbonate addition as a 0.25 molar solution to a saturated calcium sulphate solution (0.6 gpl Ca at ambient temperature) resulted in:

- Precipitation of 99% of the calcium (to 0.007 gpl Ca) at ambient conditions
- Precipitation of 99.5% of the calcium (to 0.003 gpl Ca) at 90C.

The addition of sodium carbonate to solutions of magnesium sulphate and calcium carbonate showed that high soluble carbonate concentrations led to close to stoichiometric precipitation of magnesium and calcium.

**Carbon Dioxide Gas Addition to Neutral Liquor under Pressure:** Carbon dioxide was added to a neutralised synthetic HPAL solution at 90C, pH 7.0. Carbon dioxide overpressure was maintained at 20 to 100 psig. There was apparent production of copious amounts of a white precipitate in samples of liquor withdrawn from the autoclave when the gas over-pressure was above 50 psig. With pressure release, the precipitate apparently redissolved.

The conclusion from the carbon dioxide pressure precipitation tests is that  $MgCO_3$  and  $CaCO_3$  can apparently be recovered from neutralised HPAL solutions by carbon dioxide precipitation of insoluble carbonate at greater than 50 psig over-pressure. However, recovery of  $MgCO_3$  for recycle and re-use back in the acid neutralisation reaction would require pressure filtration of the gaseous precipitation product. In addition, carbon dioxide would need to be recovered for recycle from the resulting liquor. After pressure release the liquor had a pH of approximately 5-5.2.

The possibility of separate precipitation and recovery of  $NiCO_3$  and  $CoCO_3$  as final products by carbon dioxide overpressure before  $MgCO_3$  or  $CaCO_3$  precipitation was not investigated. However it was noted that precipitate recovery would require pressure filtration to prevent carbonate redissolution at atmospheric pressure. If gas precipitation were proven to be successful, it might avoid the need and high cost for MgO addition. However, the additional costs of gas handling, autoclave precipitation of carbonate and pressure filtration would be significant. A preliminary desktop study of the feasibility of carbon dioxide gaseous precipitation of nickel and cobalt carbonate from solution was considered to be desirable if gaseous carbonate precipitation were to be evaluated further.

#### 5.1.5 Settling of Ground Fresh Ore and Ground Acidic Partially Reacted Total Ore

The slurry resulting from the reaction of finely ground total low-grade ore (p80 53 micron) with HPAL solution at 90C achieved a final free acidity of approximately 8-14 gpl free acid (Test 16). This slurry was allowed to settle at 90C without flocculant addition. Feed percent solids were 32%.

Samples of freshly ground total ore and a separate sample of partly neutralised ground ore from Test 17 were also sent to Ciba Specialty Chemicals for flocculant screening and settling tests.

The solids density from the partial neutralisation settling test without slurry dilution and without flocculant addition was approx. 40% solids after 3 hours, 44% solids after 18 hours and 45% solids after 96 hours.

Preliminary testwork by Ciba on a range of flocculants including Magnafloc 800HP, 351, 358, 1011, 338 and DPI-8468 showed that Magnafloc 358 was the most responsive flocculant. Performance characteristics of Magnafloc 358 were as follows:

- less than 10 g/t addition to achieve a settling rate of 5 m/h on pH 7 freshly ground ore
- approx. 34 g/t addition to achieve a settling rate of 5 m/h on partially neutralised pH 2 ore
- intermediate settled solids densities of approx. 30% solids at pH 7.8 and 24% solids at pH 2.3, after 20 minutes settling from an initial diluted solids density of about 4% solids
- intermediate settled solids turbidity of approx. 250 NTU for both freshly ground and partially neutralised ore compared with non-flocculated feed turbidities of 176 NTU for freshly ground ore and 429 NTU for partially neutralised pH 2 ore
- maximum settling rates were much higher for freshly ground ore than for partially neutralised ore, being up to 70 to 105 m/h with Magnafloc 358 and 1011 on pH 7 freshly ground ore and 14 m/h with Magnafloc 358 on pH 2 partially neutralised ore

Ultimate density for flocculated material has yet to be established but may at best be of the order of 35-40% solids.

## 6. LOW GRADE MINERALOGY

Gross mineralogy work consisting of XRD and optical microscope “first pass” work was carried out by Pontifex and Associates on the following selected size fractions of low-grade ore:

- Size +4.7 mm – optical microscopy
- Size +1.18mm - optical microscopy and XRD
- Size +75 micron - optical microscopy and XRD
- Size –2.5 micron – XRD only

Photomicrographs of particles in different size ranges and interpretation of the x-ray and optical data indicated that:

- Separate specific Ni-bearing phases were not detectable in the Young low-grade material
- The main mineral phases were smectite, goethite, maghemite, hematite, lesser serpentine, chlorite, talc, quartz and gibbsite
- Most of the mineral phases were impure due to intricate mixing
- Ni was expected to occur in minor amounts in minor serpentine, talc, smectite, goethite
- Ni was also expected to occur in intricately mixed secondary oxide and secondary silicate impurities in maghemite and hematite
- Smectite and possibly goethite were expected to be the most likely phases to host most of the Ni, but this could not be proven by the study undertaken to date
- A mixture of vermiculite (60%), smectite subspecies montmorillinite, saponite and nontronite (collectively 6%), goethite (34%) and hematite was shown as having the potential to account for the SiO<sub>2</sub>, Al, Fe and Mg assays reported in the six finest size assays of low-grade ore.

Suggestions by Pontifex and Associates for upgrading the low-grade ore were confined to the possibility of producing a magnetic fraction from the iron oxides maghemite, magnetite and hematite. It was considered that there might be a chance that the magnetic fraction could be enriched in nickel. However, earlier work by Amdel had already shown that there was no upgrading of low-grade ore under the magnetic separation conditions evaluated so far. Consequently, no further evaluations of the possibilities for magnetic separation and upgrading were made during the current program.

The intricate intermixing of minerals of all types at all size ranges suggested that flotation for nickel recovery or gangue rejection was unlikely to be successful. This unfortunately proved to be the case, based on the results of testwork carried out to date.

## 7. CONCLUSIONS

### JERVOIS YOUNG LOW GRADE ORE TREATMENT

1. Upgrading of the ore by coarse wet screening was largely unsuccessful
2. A light grind was found to be equivalent to intensive attrition scrubbing in maximising the weight fraction of the Ni-rich minus 25 micron wet screened size fraction.
3. Rougher flotation was unsuccessful and yielded at best a rougher upgrading of 7% at recoveries of 23 to 40%, compared with an upgrading of 12% and recovery of 58% in the minus 53 micron wet screened fraction of Total Ore.
4. Fine grinding and high reaction temperatures markedly increased the reactivity of low grade ore, magnesite and limestone
5. The low-grade ore had only a limited neutralisation capability.
6. Above stoichiometric addition of finely ground magnesite precipitated a significant part of the nickel from a synthetic HPAL solution but none of the cobalt at pH 4-7 and 90C, yielding a precipitate of 2.9% Ni, 0.2% Co, 0.3% Fe, 0.8% Al and 25.5 % Mg.
7. Finely ground limestone precipitated all nickel and cobalt from a synthetic HPAL solution between pH 4.5 and 7.0.
8. Above stoichiometric addition of finely ground magnesite precipitated virtually all remaining nickel and cobalt at pH 7.0-8.6, yielding a precipitate of 23.5% Ni, 3.6% Co, 0.1% Fe, 0.2% Al and 7.2% Mg
9. Carbon dioxide gaseous over-pressure of 50 psig appeared to precipitate magnesium carbonate from a synthetic pH 8.0 HPAL Ni-depleted neutral spent solution, but solids recovery problems under pressure filtration appear formidable. Additional limestone or magnesite addition would also be needed to neutralise carbonic acid formed from gas addition.
10. Natural fresh ground low-grade ore settled more rapidly than acidified (pH2) freshly ground low-grade ore
11. Flocculants significantly aided settling when added at approx. 4% solids density, with the best flocculant identified by a Ciba Specialty Chemicals flocculant screening test being Magnafloc 358
12. Intermediate underflow densities were higher for neutral ore (30% solids) than for acidic slurries (24% solids)
13. The mineralogy was extremely complex with intricate mixing of the impure main mineral phases of smectite, goethite, maghemite, hematite, lesser serpentine, chlorite, talc, quartz and gibbsite
14. The benefits of treating low grade ore economically appear to be formidable, unless leached in conjunction with high grade

## HIGH GRADE ORE TREATMENT

- Previous Testwork, Section 8
- Sample Preparation, Section 9
- Pressure Acid Leach Response, Section 10
- Leach Residue Thickening, Section 11
- Mineralogy, Section 12
- Resin-In-Pulp, Section 13
- Conclusions, Section 14
- Recommendations, Section 15



## 8. PREVIOUS HG TESTWORK

Previous HRL work on Young High Grade RC drill core and composite samples in September 1999 consisted of preliminary metallurgical characterisation tests on eight composites from eight drill holes from the Thuddungra Mines Road, Ardnaree, Shangrila and Redlands areas of the Young resource. Grades varied from 0.7 to 1.2% Ni, 0.03 to 0.15% Co, 3 to 12% MgO and 2 to 12%  $\text{Al}_2\text{O}_3$ .

Preliminary high-pressure acid leach tests were carried out on three nominated composites, a limonitic composite (Thuddungra East Sample A), a limonitic saprolite composite (Ardnaree Sample B) and a saprolite composite (Ardnaree Sample C). Each composite comprised the minus 2mm fractions of either two or three of the eight individual air-core drill-hole composites. Grades of the three composites varied from 0.88 to 1.09% Ni, 0.09 to 0.13% Co, 7.7 to 8.3% MgO, 3.9 to 7.0%  $\text{Al}_2\text{O}_3$ , 31.5 to 36.1%  $\text{Fe}_2\text{O}_3$  and 34.7 to 35.8%  $\text{SiO}_2$ .

Results showed:

- Rejection of plus 2mm coarse material increased the Ni head grade by up to 10%
- Ni grade was generally uniform across all sizes for two of the composites and exhibited an increasing grade with finer size for the third composite
- Co distribution consisting generally of a broad higher concentration band between 20 micron and 1,200 microns
- Generally coarser particle size distribution than for typical tropical laterites
- Nickel recoveries increasing with residual free acid concentration
- Nickel grades in leach residues of 0.05 to 0.06% Ni at residual free acid concentrations of 54 to 57 gpl
- Acid additions ranging from 260 to 410 kg/t ore
- Slow leach kinetics at low acid addition levels, possibly due to a high proportion of coarse material in the -212 micron feed material
- Faster leach kinetics for Ardnaree Sample C (partially limonitic) and East Thuddungra Sample A (mixed limonite/saprolite) than for Ardnaree Sample B (saprolitic)
- Favourably low Fe and Cr uptake during leaching
- Magnesium extractions of 60 to 81%
- Low levels of ferrous and chrome in solution
- No impurity elements of any concern in solution
- Up to 93-96% Ni and Co extraction after 90 minutes leaching at acid additions of 290 to 410 kg/t ore, depending on composite composition

## 9. HG AND ROM SAMPLE PREPARATION

**Composite Preparation:** In the current test program a separate composite was prepared for each of the three Jervois selected area samples (Ardnaree, East Thuddungra and West Thuddungra) and a fourth composite was prepared for a separate High Grade feed sample. Preparation method for each composite included:

- combining all components of a given composite into a total composite of approx. 100kg
- separate dry blending of each composite in a rotary blender
- splitting off a 20-25 kg metallurgical sample by cone and quartering
- saving the unused 75 kg of each composite in a sealed plastic bag.

The dry coned samples were combined and processed by wet blending the entire sample.

The + 2mm oversize was removed and size by size assays determined for the total size range of the original feed material for each composite. Only the minus 2mm fraction was used for subsequent testwork, in view of the generally lower grade or higher magnesium level of this fraction. Sample SG of the minus 2mm material was measured by air pycnometer.

Each of the four minus 2mm feed composites was kept in wet slurry form throughout the test campaign of approx. 12 weeks.

**Comparison with Earlier Feed Samples:** Assays of the current feed composites are compared with assays from the earlier testwork composites in Table 3:

TABLE 3 EARLIER AND CURRENT METALLURGICAL SAMPLE SIZE, SG AND ASSAY COMPARISON: JERVOIS YOUNG ROM AND HIGH GRADE COMPOSITES

Ore Sample	p80 micron	SG g/cc -2mm	Ni %	Co %	Sc ppm	MgO %	Al <sub>2</sub> O <sub>3</sub> %	Fe %	Cr %	Mn %	SiO <sub>2</sub> %	Co/Zn	Co/Cu
<b>Earlier Composites: HRL 0675</b>													
East Thudd A (1)	163		0.99	0.125		8.3	7	22.1	0.77	0.45	35.3	2.6	13.9
Ardnaree B (2)	137		0.88	0.092		7.7	3.9	25.3	1.1	0.49	34.7	2.6	3.8
Ardnaree C (3)	180		1.09	0.094		7.7	6.5	21.9	0.43	0.47	35.8	3.0	90.8
<b>Current Composites: HRL 0697</b>													
High Grade (4)	180	3.05	0.99	0.069	26	10.5	6.1	18.9	0.65	0.3	38.4	1.4	11
Ardnaree (5)	420	3.1	0.78	0.066	27	11.1	7	17.9	0.55	0.36	41.8	2	5.7
East Thudd (6)	190	3.15	0.61	0.081	39	5.5	9.9	25.5	0.78	0.31	28.8	2.7	11.8
West Thudd (7)	135	2.95	0.73	0.083	28	14.1	7.4	16.6	0.57	0.29	38.7	2.5	11.2
1) limonitic		4) weathered serpentinitic limonitic saprolite											
2) limonitic saprolite		5) weathered serpentinitic limonitic saprolite											
3) saprolite		6) Saprolitic limonite		7) weathered serpentinitic limonitic saprolite									

Current feed composites were generally lower in nickel, lower in cobalt and higher in MgO and Al than earlier test composite samples. In particular:

- the lowest grade of the new feed composites (East Thuddungra) was approx. 30% lower in Ni, Co and MgO and 70% higher in Al than the previous lowest grade composite (Ardnaree Sample B)

- the highest grade of the new feed composites (High Grade) was approx. 10% lower in Ni, 25% lower in Co, 35% higher in MgO and 70% higher in Al than the previous highest grade composite (Ardnaree Sample C)

The head analyses of the minus 2mm material used for pressure leaching are shown below in Table 4. The analyses were up to 10% higher in nickel grade than the original feed samples.

**Table 4: Head Analyses of Jervois Young Minus 2mm ROM and High Grade Ni/Co Composites used as Feed to Pressure Leach**

Assay	High Grade	Ardnaree	West Thuddungra	East Thuddungra
No. samples	6	2	3	2
Ni, %	0.99-1.06	0.78	0.709 – 0.749	0.63 – 0.66
Co, %	0.074 - 0.078	0.074	0.086 – 0.0895	0.082 – 0.085
Sc, %	0.0026 – 0.0028	0.0027	0.0027 – 0.0029	0.0039 – 0.0041
Zn, %	0.024 – 0.035	0.029	0.028-0.032	0.025-0.041
Cu, %	0.0058 – 0.0096	0.0068	0.008 – 0.064	0.005 – 0.006
Cr, %	0.76 – 0.85	0.62	0.639 – 0.747	0.80 – 0.85
Mn, %	0.308 – 0.329	0.4	0.304 – 0.315	0.32 - 0.33
Fe, %	19.6 – 21.6	18.4	28.8	28.3
Si, %	17.1 – 18.0	32.5	28.1	30.6
MgO, %	10.1 – 10.5	10.4	13.2 – 13.8	5.3 – 5.7
Al <sub>2</sub> O <sub>3</sub> , %	6.24 – 6.58	6.8	7.5 – 7.9	10.1 – 10.7

### Size by Size Analysis:

Specific characteristics of the Young ROM and High-Grade materials, which are detailed in the HRL report, include:

- Ardnaree was the coarsest and West Thuddungra the finest material, with size and weight characteristics being:

	High Grade	Ardnaree	East Thuddungra	West Thuddungra
P80, micron	180	420	190	135
Wt% less 20 micron	47.8	46.6	52.2	53.1
Wt% >2mm	10.9	16.3	9.2	6.9

- Nickel grades generally declined with increasing size, although in three cases (High Grade, East Thuddungra and West Thuddungra) the decline in grade did not start until a size of 595 micron was reached

- Cobalt grades showed a maximum of between 0.14 and 0.19% Co in the size range from 100 or 215 micron to 595 micron, minimum grades of between 0.01 and 0.05% Co in the minus 20 micron and +2mm size bands and a maximum-to-minimum Co grade ratio of between 3 and 9
- Magnesium grades were relatively constant with increasing particle size until a size of about 1.5 to 2mm, where they tended to increase with particle size. However, in the case of East Thuddungra they fell about 10% from 2 mm to 4.7mm before recovering the same 10% from 4.7 mm to 9.5 mm
- Silicon often displayed similar constant grade behaviour as magnesium up to a certain size, but thereafter decreased in grade with increasing size, in the reverse direction to magnesium
- Aluminium levels were relatively constant with particle size for High Grade and Ardnaree, but increased sharply above 2mm with increasing size for East Thuddungra.
- Iron levels tended to fall with increasing size for High Grade and Ardnaree, but either rose slowly with increasing size (East Thuddungra) or passed through a maximum with increasing size (West Thuddungra)
- Scandium tended to follow aluminium behaviour with size

**Feed Thickening:** Discouraging results from early thickening testwork in shallow settling cylinders in the current testwork program gave rise to concerns about the possibility of major changes in solids settling characteristics due to standing as a wet slurry for a long periods of time (4-5 weeks in this case). However, it was subsequently found that the use of deep bed thickening overcame these settling problems.

Underflow densities of the order of 40% solids were achieved in deep bed feed thickening of High Grade, East Thuddungra and Ardnaree in 10 litre settling cylinders after dilution to 3-4% solids and addition of 70-100 gpt Magnafloc 358. Coagulant LT-35 addition ahead of flocculent addition was required with East Thuddungra.

At a Magnafloc 358 cost of A\$6.58/kg (US\$ 4.00/kg) for bulk delivery to site, the Magnafloc 358 costs were of the order of A\$ 0.50 – 0.66 (US\$ 0.30 – 0.40) per t ore for 70-100 gpt usage.

All thickener feed samples were prepared by screening to recover minus 2mm material and grinding to 100% passing 212 micron.

Magnafloc 155 was also tested on High Grade and Ardnaree, with broadly similar performance to Magnafloc 358.

West Thuddungra did not achieve better than 28% solids density even with combined high flocculent and coagulant dosages and more work remains to be done on the settling behaviour of this material.

Limited unit settling area data collected to date suggests preliminary design requirements of possibly 0.2 – 2.4 m<sup>2</sup>/tpd solids at 30% solids and 0.4 – 2.6 m<sup>2</sup>/tpd at 40% solids for High Grade, East Thuddungra and Ardnaree, depending on ore type. Unit settling areas for West Thuddungra were not capable of being determined at 30-40% solids due to the poor settling characteristics of this material under the conditions investigated so far.

Saline water did not help to increase underflow densities or overflow solution clarity.

The possibility of avoiding feed thickening altogether and feeding the high pressure leach directly from milling without any dilution of the mill discharge slurry for size separation merits investigation. This option could save significant capital cost, as presumably has already been demonstrated at the Murrin Murrin nickel laterite plant in WA.

**Screen Upgrading:** The best upgrading that could be achieved by screening the ROM or High Grade material containing 0.6 to 1.0% Ni, 0.07 to 0.08% Co and 3.3 to 8.5% Mg appeared to be that obtained by rejection of plus 2mm material. This material contained the lowest grade nickel (0.4 to 0.6% Ni) and the highest grade magnesium (5.0 to 13% Mg). This left behind a modestly upgraded product containing 84 to 93% of the feed weight, 89 to 95% of the feed nickel and 92 to 97% of the feed cobalt and assaying 0.6 to 1.0% Ni, 0.07 to 0.09% Co and 3 to 8% Mg. This upgrading was adopted for preparation of the feed for high pressure acid leaching. Calculated product assays and metal distributions for upgrading of ROM and High-Grade ore by 2mm wet screening are shown in Table 5 below.

**Table 5: Assay and Metal Distribution of Wet Screened Minus 2mm Fraction versus Feed Material and Wet Screened Plus 2mm Material – Young ROM and High Grade Composites**

		Assay						Distribution					
Size	Wt	Ni	Co	Fe	Mg	Al	SiO <sub>2</sub>	Ni	Co	Fe	Mg	Al	SiO <sub>2</sub>
micron	%	%	ppm	%	%	%	%	%	%	%	%	%	%
<b>High Grade:</b>													
-2mm	89.1	1.033	740	19.9	6.2	3.3	35.9	92.8	96.1	93.8	87.1	90.8	83.3
+2mm	10.9	0.653	247	10.8	7.5	2.7	58.8	7.2	3.9	6.2	12.9	9.2	16.7
Feed	100.0	0.992	690	18.9	6.3	3.2	38.4	100.0	100.0	100.0	100.0	100.0	100.0
<b>Ardnaree:</b>													
-2mm	83.7	0.824	744	19.5	6.5	3.7	39.2	88.6	94.3	91.4	81.2	84.4	78.5
+2mm	16.3	0.544	231	9.4	7.7	3.5	55.1	11.4	5.7	8.6	18.8	15.6	21.5
Feed	100.0	0.779	660	17.9	6.7	3.7	41.8	100.0	100.0	100.0	100.0	100.0	100.0
<b>West Thuddungra:</b>													
-2mm	93.1	0.743	865	29.2	8.1	4.0	38.4	95.3	97.0	95.7	88.7	94.6	92.4
+2mm	6.9	0.495	361	17.7	14	3.1	42.6	4.7	3.0	4.3	11.3	5.4	7.6
Feed	100.0	0.726	830	28.4	8.5	3.9	38.7	100.0	100.0	100.0	100.0	100.0	100.0
<b>East Thuddungra:</b>													
-2mm	90.8	0.629	822	24.8	3.2	5.1	29.3	93.6	92.1	89.3	87.0	88.5	92.5
+2mm	9.2	0.424	696	29.3	4.7	6.5	23.5	6.4	7.9	10.7	13.0	11.5	7.5
Feed	100.0	0.610	810	25.2	3.3	5.2	28.8	100.0	100.0	100.0	100.0	100.0	100.0

Data Source: Recalculated from Size by Size Assays, HRL Report 0697, Appendix 1

Upgrading ratios from feed to minus 2mm fractions, as shown in Table 6 below, indicate limited upgrading of Ni and Co and limited downgrading of magnesium. Specific changes were 2 to 6% upgrading for Ni, 1 to 13% upgrading for cobalt and 2 to 16% downgrading for magnesium, with most changes being 7% or less. Recognising that the reactivity of magnesium in the coarse rejects fraction may not necessarily have been any different from that in the fine accept fraction, in hindsight the benefits of upgrading would appear to have been marginal, except for Ardnaree.

TABLE 6: UPGRADING RATIOS FROM FEED TO MINUS 2MM SIZE FRACTION

	Ni	Co	Mg
High Grade	1.04	1.07	0.98
Ardnaree	1.06	1.13	0.84
West Thuddungra	1.02	1.04	0.95
East Thuddungra	1.03	1.01	0.97

**Size Fractions for Mineralogy:** Three different size fractions of High Grade composite were chosen for mineralogical assessment by Pontifex and Associates to provide a broad range of sizes and different Ni, Co and Fe grades for mineralogical evaluation. Unfortunately the coarser sample, -2mm, was very fine due to supply of material that had been ring ground for assay. Nevertheless, the weight fractions and assays of the materials evaluated by Pontifex & Associates were as follows:

- -2mm, representing 89.1 wt% of the original feed composite and 100% of the autoclave feed sample, assaying 9920 ppm Ni, 690 ppm Co and 18.9% Fe, and consisting of a sample of ring ground material used for assay
- -100 micron + 75 micron, representing 3.9 wt% of a narrowly sized fraction at 10500 ppm Ni, 1310 ppm Co, 6.4% Mg, 2.9% Al, 20.5% Fe, 0.57% Mn and 1.0% Cr
- -20 micron, representing 47.8 wt% at 10800 ppm Ni, 400ppm Co and 20.3% Fe

## 10. HG AND ROM PRESSURE ACID LEACH RESPONSE

**Leach Kinetics:** Leach kinetics, based on the relative leach recoveries reported at 30 minutes and 90 minutes, indicated that:

- Higher acid additions (+0.04 to 0.07 t/t ore) and generally higher associated residual acid concentrations (variable but up to 8 to 15 gpl additional acid after 60 to 90 minutes) appear to have been more beneficial than finer grind size (75 micron versus 212 micron)
- Mineralogy differences associated with low levels of magnesium extraction were suspected to be the cause of lower initial nickel extractions achieved with High Grade, West Thuddungra and East Thuddungra, but it was felt that this needed to be checked by mineralogical evaluation of selected size fractions of leach feed and tails
- An increase in leach temperature from 250 to 265C appeared to be more favourable to nickel extraction than an increase in acid addition of 0.05 to 0.07 t/t ore

**Leach economics:** Incremental leach economics have been evaluated below in Table 7 at an estimated acid cost of US\$50/t acid. Actual site costs may be higher or lower, depending on various site factors. The +0.04 to 0.07 tonne of additional acid required per tonne of ore to achieve up to 3% additional nickel recovery would incur an incremental acid cost of US\$1.50 – 3.50/ t ore, which is equivalent to US\$3.9-5.0/lb Ni for Ardnaree and High Grade. This incremental cost would be uneconomic.

Economic nickel recoveries are expected to range from the high 80's to low 90' s, or potentially 3-4% lower than the maximum recoveries reported by HRL. Further economic evaluation of nickel production costs for the most likely acid and limestone costs applicable at Young is advisable.

TABLE 7: INCREMENTAL ACID ADDITION COSTS AT MAXIMUM NICKEL RECOVERIES

	Head Grade % Ni	Acid Addn t/t ore	Acid Cost US\$/t ore	Leach Extn %	Ni Extn kg/t ore	Acid addn kg/kg Ni	Acid ddn US\$/lb Ni
High Grade	1.05	0.41	20.5	89	9.35	43.9	1.00
	1.05	<u>0.48</u>	24	92	<u>9.66</u>	<u>49.7</u>	<u>1.13</u>
	Increment	0.07	3.50		0.32	222.2	5.04
Ardnaree	0.78	0.38	19	92	7.18	53.0	1.20
	0.78	0.42	21	95	<u>7.41</u>	<u>56.7</u>	<u>1.29</u>
	Increment	0.04	2.00		0.23	170.9	3.88
West Thudd	0.65	0.43	21.5	87	5.66	76.0	1.72
	0.65	0.46	23	86	<u>5.59</u>	<u>82.3</u>	<u>1.87</u>
	Increment	0.03	1.50		-0.06	-461.5	-10.47
Notes:							
Acid cost, US\$/t acid		50					
Grind size, microns		-212					
Leach temp, C		250					



**Size Analysis of Leach Residue:** Size by size analysis of West Thuddungra leach residue from Test SS14-26, which had a higher than desirable Ni and Co residue analysis (0.13% Ni and 0.008% Co), was carried out in supplementary testwork after completion of the Main Testwork Report. The purpose of this work was to try to assist in understanding the mineralogy of the different residue size fractions and to thereby determine the most likely conditions for better leach extraction.

Residue assays for Ni, Co and Mn in four consecutive size ranges between minus 20 microns and plus 53 microns were compared with the corresponding size assays of the feed sample, prior to grinding the sample to minus 212 microns. The analysis shown below in Table 8, indicated:

- Broadly similar residue:feed assay ratios within each size fraction for Ni (0.165 to 0.22), Co (0.099 to 0.147) and Mn (0.237 to 0.278)
- A relative insensitivity of residue:feed assay ratio of particle size between minus 20 microns and plus 53 microns
- A reaction mechanism that was believed more likely to indicate chemical reaction controlled rather than diffusion or mass transfer controlled reaction.

**Table 8: Residue-to-Feed Assay Ratios, West Thuddungra Leach Residue**

Size range, micron	Nickel Assay, % Ni			Cobalt Assay, % Co			Manganese, % Mn		
	Residue	Feed*	Ratio	Residue	Feed*	Ratio	Residue	Feed*	Ratio
+53	0.153	0.729	0.210	0.014	0.142	0.099	0.123	0.52	0.237
+38	0.150	0.744	0.202	0.013	0.0965	0.135	0.087	0.361	0.241
+20	0.153	0.695	0.220	0.014	0.0955	0.147	0.096	0.345	0.278
-20	0.126	0.762	0.165	0.007	0.0485	0.144	0.037	0.149	0.248

\* Feed assay minus 2 mm size fraction before grinding to minus 212 micron, Data from HRL File Note 16 Nov 2000

Additional data showed that most of the leaching problem resided in the minus 20 micron size fraction, as this size range contained over 80% of the residue weight and 70-80% of the nickel and cobalt distributions. Specific differences between the fine and coarse size ranges included:

- Little difference in assays and weight fractions of Ni, Co and most impurities in three consecutive size ranges between plus 20 microns and plus 53 microns
- Major difference in assays and weight fractions between the coarser plus 20 micron size range and the finer minus 20 micron size range including:
  - A decrease of 20 to 50% in Ni, Co, Mg, Mn, Zn, Cr and SiO<sub>2</sub> assays in the finer size range
  - A 20 to 250% increase in Fe and Al assays in the finer size range
  - A 5 fold increase in weight fraction in the finer size range
  - A 2 to 15 fold increase in metal distribution in the finer size range

It was concluded that the solution to the problem of high nickel residue assays might require a combination of future more severe leaching conditions such as:

- Higher temperature to increase kinetics of leaching and slightly increase Ni and Co extractions, although possibly at the penalty of a marginally higher acid consumption



- Higher free acid concentration to increase slightly the kinetics of leaching and slightly increase Ni and Co extraction, although probably at the penalty of marginally higher acid consumption
- Finer grind size to increase kinetics of leaching and slightly increase Ni and Co consumption, although probably at the penalty of a significant but unavoidable increase in impurity extraction

**Resin-in-Pulp Recovery Benefit:** As part of an investigation into possible process routes which could potentially result in lower plant capital costs, one set of preliminary resin-in-pulp tests was carried out by Clean TeQ using proprietary resins to process a slurry sample of leached High Grade pulp. This test achieved significantly higher nickel recoveries than the HRL batch leach recovery results –see Section 14. If higher recoveries can be confirmed in later resin-in-pulp tests, there would appear to be a strong case to seriously consider resin-in-pulp processing rather than conventional CCD wash thickener processing of the leach pulp.

**Limestone/Magnesite Unit Costs:** Local limestone sources and costs in the Young project area need verification, since there were apparently no readily known in-situ limestone resources at Young at the time of this testwork. Imported limestone from Marulan in Southern NSW would currently be expected to have a cost closer to US\$40/t rather than currently assumed US\$5/t for local limestone cost. Costs for use of low-grade magnesite, of which there are understood to be ample local in-situ supplies at Young, need to be confirmed, since magnesite would currently appear to be the lowest cost neutralising agent at Young.

**Liquor Analyses:** An encouraging feature of the current Young HPAL testwork results and also earlier Young HPAL testwork results was the continuing low levels of iron, aluminium and chrome present in the leach liquor. Total iron and aluminium levels in solution varied from 1-3.3 gpl for iron and 0.7-2.5 gpl for aluminium during the current pressure acid leach series. Soluble chrome levels were less than 100 ppm, with Redox levels indicating the absence of soluble chrome species. This data suggest that the removal of impurities from nickel laterite high-pressure acid leach solutions should be relatively straightforward. Thus the quantity of neutralising agent needed for pH adjustment ahead of intermediate product precipitation should mainly be for the neutralisation of excess acid in the leach discharge rather than for the additional neutralisation of substantial quantities of iron and aluminium.

Soluble magnesium levels in the HPAL liquor were of the order of 16 to 26 gpl Mg, reflecting the relatively high head assays of 6 to 13% MgO and magnesium extractions of 45 to 75%, depending on the particular composite being leached.

**Mass Balances:** Mass balances for metal extraction in most laboratory tests were generally of the order of 90 to 110%, with occasional excursions to 80 or 120% for aluminium. These balances were considered to be acceptable.

**Atmospheric Diagnostic Leaching:** An atmospheric diagnostic leach used by HRL to evaluate the likely acid consumption of mainly tropical laterites was evaluated on the Young laterite composites. Diagnostic leaches were conducted under the conditions of temperature 95-100C, time 120 minutes, 17 gpl solids and 150 gpl HCl initial acidity.

It was found that the diagnostic leach constantly underestimated magnesium extraction compared with pressure leach conditions, indicating that Young laterites were more refractory to acid attack than tropical nickel laterites. Table 8 shows the difference in magnesium extraction, which was close to 22-36% extraction over the range of feed MgO values from 5.5 to 14.1% MgO.

**Table 8: Comparison of Magnesium Atmospheric Diagnostic Leach Extractions with PAL Extractions, Young Ni/Co Laterite Project**

Ore Type	Feed MgO %	Magnesium Extraction %		
		PAL*	Diagnostic Leach*	Difference
High Grade	10.5	69-75	43	26-32
Ardnaree	11.1	45-52	23	22-29
East Thuddungra	5.5	71-82	55	16-27
West Thuddungra	14.1	69-75	36	33-39

## 11. HG AND ROM LEACH RESIDUE THICKENING

Leach residue thickening tests were conducted under the same conditions as the feed thickening tests, namely 3-5 wt% solids, 10L scale settling cylinders equipped with a slow speed rakes and ambient temperature. However, instead of using water for solids dilution, an artificial leach liquor containing 2.8 gpl Ni, 0.2 gpl Co, 1.9 gpl Fe, 21 gpl Mg, 1.2 gpl Mn, 0.8 gpl Al and pH 2.0 was used. The dilution was applied to leach residue filter cakes to produce the required feed slurry density for the thickening tests. The solution composition was determined by HRL from METSIM based CCD simulation.

Scouting tests were used to select the best overall flocculents, which were found to be Magnafloc 800HP (non-ionic) and Magnafloc E10 (slightly anionic). Magnafloc 800HP resulted in a clearer supernatant solution and was selected as the preferred flocculent. Magnafloc 455 also performed well but required a higher flocculent dosage.

All leach residues except West Thuddungra flocculated easily at additions of the order of 100 gpt. West Thuddungra was difficult to thicken even at high flocculent dosages of 130-140 gpt.

Eight thickening tests were carried out including 4 on High Grade (-212 microns), 2 on West Thuddungra (-212 and -105 microns) and one each on Ardnaree (-212 microns) and East Thuddungra (-105 microns). Underflow densities were 36-37% solids for High Grade, 30% for Ardnaree, 31% for East Thuddungra and 28-29% for West Thuddungra. Higher densities are expected for Ardnaree, for which the present densities were limited by the restricted amount of sample available for testing.

Thickening rates at an underflow density of 28 wt% solids were calculated by HRL using the Wilhelm-Naide technique corrected for depth. Rates were highest for High Grade ( $0.44 \text{ m}^2/\text{tpd}$ ), followed by East Thuddungra ( $0.79 \text{ m}^2/\text{tpd}$ ), Ardnaree ( $2.2 \text{ m}^2/\text{tpd}$ ) and West Thuddungra ( $2.5 \text{ m}^2/\text{tpd}$ ). Rates were slower at higher solids densities. Further improvements in thickening rates are expected from additional testwork.

## 12. HG FEED MINERALOGY

Gross mineralogy work consisting of XRD and optical microscope “first pass” work was carried out by Pontifex and Associates on the following selected size fractions of High-Grade ore:

- Size -2 mm – optical microscopy, XRD and probe analysis
- Size +75 micron to -100 micron - optical microscopy and XRD
- Size –20 micron – XRD only

Photomicrographs obtained for the –100 +75 micron size fraction indicated the dominance of Fe-rich particles, ranging from limonite-goethite stained clay and other grains through to goethite.

General results of the x-ray and optical mineralogy are shown in Table 9a.

In general, it would seem that:

- smectite was present as co-dominant or dominant mineral in all size fractions, but particularly in the –20 micron fraction
- lizardite was present as a minor to co-dominant depending on the size fraction
- chlorite was present as a minor to co-dominant depending on the size fraction
- goethite was present as a minor to co-dominant depending on the size fraction
- talc was present as a minor to co-dominant depending on the size fraction
- quartz was present as trace to co-dominant depending on the size fraction
- chromite was present as minor to co-dominant depending on the size fraction
- XRD provided higher weight fraction indications than optically for unknown reasons

TABLE 9A: OPTICAL MINERALOGY OF YOUNG HIGH GRADE SIZE FRACTIONS - PONTIFEX

Mineral	-2mm reground			-100 +75 micron		-20 micron
	XRD	Optical coarse	Optical fine	XRD	Optical	XRD
lizardite (serpentine)	co-dom.	present	present <sup>2</sup>	trace	5*, 10%	minor (1-4%)
chlorite	co-dom	present	present <sup>2</sup>	co-dominant	10%	minor (1-4%)
limonitic clay			present <sup>3</sup>			
smectite	co-dom		present	co-dominant	20%	<b>dominant</b> (>60%)
magnetite				co-dominant	8%	
maghemite	sub-dm					subdominant (10-30)
goethite	sub-dm		present	minor	10%	subdominant (10-30)
goethite+/- hematite		present				

talc	minor	present	present <sup>2</sup>	co-dominant	5%*,15%	minor (1-4%)
hematite	minor			trace		minor (1-4%)
hematite/maghemite		present	present <sup>1</sup>			
limonite+/hematite			present			
quartz	minor	present	present <sup>1</sup>	co-dominant	7%	trace (<2%)
actinolite/tremolite	trace	present		trace	7%	trace (<2%)
magnesite				trace		
chromite		present		co-dominant	3%	
chromite/magnetite			present <sup>1</sup>			

\* limonite stained, 1 = relatively fairly discrete grains, 2= relatively discrete flakes, comminuted particles, 3= smectitic?

### 13. HG RESIN-IN-PULP

Resin-in-pulp processing of leach residue slurry was evaluated on a preliminary basis for Young High-Grade composite leach slurry to determine whether resin-in-pulp might provide higher recoveries of nickel and cobalt than a more conventional CCD circuit.

The results of preliminary sighter resin testwork by Clean TeQ of Dandenong, Australia using Clean TeQ's proprietary resin, Clean-iX™, on a high pressure acid leach slurry produced by HRL from batch pressure leaching of Young High Grade ore are described below.

The initial RIP results appear encouraging. Subject to resolution of leach residue assay discrepancies and clarification of general resin supply and other process development issues, further resin-in-pulp testwork would appear to be worthwhile.

#### Testwork Procedure:

The testwork procedure adopted by Clean TeQ consisted of:

- starting with a leach residue pulp for Young "High Grade" prepared by HRL in Brisbane
- heating the pulp to 90C
- precipitating out the iron in the pulp by pH adjustment somewhat similarly to other plants (but using magnesite supplied by IPDS from Young instead of limestone, and a leach pulp feed instead of a leach solution feed)
- contacting the pulp with 5 successive stages of fresh resin to remove the nickel and cobalt
- stripping the resin from stage 1 with dilute sulphuric acid
- assaying the liquor and solids from the feed pulp, the neutralised pulp, each stage of contact and also assaying the strip liquor

The above resin-in-pulp treatment process differs from the Cawse and Bulong laterite leach plants where the solution is separated from the pulp. Either a nickel/cobalt hydroxide is precipitated as at Cawse or direct SX is used to produce a cobalt sulphide and concentrated feed liquor for nickel electrowinning as at Bulong.

Pulp neutralisation carried out by Clean TeQ was at 90C, which is at a higher temperature than the standard conditions normally used in the Bulong and Cawse circuits. Clean TeQ advised that their resin was robust and could stand high temperatures.

#### Testwork Results

The initial Clean TeQ sighter RIP laboratory test achieved the following results, based on Analabs assays for feed solution and solids, ALS assays for product solutions, adjusted ALS assays for product solids and Analabs check assays for RIP feed solids:

#### Iron Removal:

1. Virtually complete removal of Fe<sup>+++</sup> iron and almost complete removal of Fe<sup>++</sup> iron was achieved by magnesite addition to pH4, as indicated by the following changes in composition of feed solution to and product solution from neutralisation:
  - Solution in feed pulp to neutralisation: 1900-2260 ppm FeT (approx. 50-80% Fe<sup>++</sup>, HLS data)

- Solution in product pulp from neutralisation: 130 ppm  $\text{Fe}^{++}$ , <1 ppm  $\text{Fe}^{+++}$
- 2. Partial precipitation of nickel and cobalt occurred during neutralisation, as indicated by the following solution composition changes during neutralisation to pH 4:
  - Solution in feed pulp to neutralisation: 3600 – 4080 ppm Ni, 260 – 305 ppm Co
  - Solution in feed pulp from neutralisation: 2730 ppm Ni, 221 ppm Co
- 3. There was apparently no special requirement for a ferrous oxidation stage at the same time as pulp neutralisation ahead of the RIP circuit, indicating that:
  - oxidation of  $\text{Fe}^{2+}$  (1100 ppm in feed) to  $\text{Fe}^{3+}$  was essentially 90% complete (to 130 ppm  $\text{Fe}^{2+}$ ) without special aeration provisions
  - the pulp was somehow still aerated without air actually being sparged into the pulp
- 4. Virtually complete precipitation of  $\text{Fe}^{3+}$  iron occurred from solution at pH 4.0 and 90C after 90 minutes contact with magnesite
- 5. Magnesite addition of 70 g per litre of pulp appears to correspond to almost 100% in excess of stoichiometric requirements for the combination of 40.4 gpl acid, 1.9 gpl iron and 1.3 gpl aluminium neutralisation. This usage should be capable of significant reduction.

#### RIP Results:

##### *Graphs of solutions and solid assay profiles:*

6. Fig. 5 at the end of this section shows a plot of the decrease in Ni and Co solution concentration from feed pulp through pulp neutralisation and during each of the 5 RIP contact stages. It can be seen that the solution composition decreased after pH adjustment with magnesite to pH 4, presumably due to excess magnesite usage and resulting in the apparent temporary 30% initial precipitation of nickel and cobalt. It can also be seen that there is virtually complete recovery of all aqueous Ni and Co in five loading stages.
7. Fig. 6 shows similar data to Fig. 5 but for the Ni and Co solid assays rather than the Ni and Co solution assays. Surprisingly the Ni and Co solid concentrations do not increase due to the decrease in soluble Ni and Co concentration in solution when magnesite is first added to the feed pulp. Rather, the Ni concentration decreased 10% and the Co concentration increased 10%. Solid assay errors may be possible for this anomaly. Thereafter, the Ni on solids decreased approximately 60% and the Co in solids decreased approximately 40% during five stages of RIP contacting.

As discussed elsewhere, the decrease in leach residue solids assays for Ni and Co during RIP appears to be evidence of additional Ni and Co leaching during RIP processing. The decrease in Ni leach residue assay was equivalent to approx. 45% of the head grade and the decrease in the Co leach residue assay equivalent to a 60% decrease of the cobalt head grade.

This 45-60% additional leaching of solids during RIP contact appeared to be a highly significant advantage for RIP processing over conventional CCD washing for nickel recovery.

8. Fig 7 shows the major upgrading of Ni and Co from feed solution to RIP strip solution and the almost complete rejection of Fe, Mn and Cr to the same solution when stripping the resin with 5% sulphuric acid solution after 5 stages of contact. Cu and Zn were fully extracted by the resin and fully stripped during acid stripping.

#### **Solution recovery:**

9. Over 99% Ni and Co recovery was achieved from the solution phase of the HPAL discharge pulp from a 5-stage RIP circuit, based on the following exit solution concentration:
  - Solution in feed pulp from stage 5: 5.9 ppm Ni, 1.3ppm Co
10. A 1-2% higher metal recovery appears to have been achieved than the typical 97-98% expected from a conventional 6-7 stage CCD wash circuit, depending on clay adsorption, underflow solids density etc

#### ***Solids recovery:***

11. Complete recovery of the nickel and cobalt that precipitated from solution during pH 4 pulp neutralisation prior to resin-in-pulp contact was achieved on the resin, as indicated by the following results:
  - Solids in leach discharge pulp: 1140 - 1160 ppm Ni, 52 -70 ppm Co
  - Solids in RIP stage 1 pulp: 1215 ppm Ni, 65 ppm Co
  - Solids in RIP stage 2 pulp: 1011 ppm Ni, 60 ppm Co
12. The resin appeared to provide better performance for recovery of Ni and Co lost to any solid precipitate than would occur in a conventional neutralisation and iron removal circuit where Ni/Co losses to the iron precipitate would generally be lost from the circuit
13. 2-4% additional nickel and cobalt extraction was achieved by the resin from the HPAL feed solids as a result of the 40-60 % additional leaching of solids in the RIP circuit due to "sorption leaching" effect claimed by Clean TeQ resins on the leach residue solids:
  - Solids in leach discharge pulp: 1140-1150 ppm Ni, 52-70 ppm Co
  - Solids in stage 5 contact pulp at pH 5.9 approx.: 694 ppm Ni, 25 ppm Co
14. Possibly 1-2 % higher extraction was achieved by the resins from the leach solids than in traditional CCD circuits where in some cases, such as copper and uranium leaching, only small amounts (say 1%) of additional metal might be leached from the solids during CCD washing

#### ***Combination solids and solution recovery:***

15. The combination of the high soluble recovery from solution and solids during RIP contact suggests that there might be up to 3 – 7% higher combined Ni and Co recovery from pulp being treated in an RIP circuit, than from a conventional CCD wash circuit -at least for Jervois High Grade Young ore.

#### ***Additional comments on RIP results:***

16. Resin-in-pulp performance was apparently independent of constraints that might conceivably limit CCD wash circuit performance for more difficult ores such as possible metal adsorption on clays during CCD washing or unduly low underflow solids concentration due to poor settling characteristics
17. The resin showed no apparently adverse effects on Ni and Co extraction of over-neutralisation of the RIP pulp feed when using magnesite for pH adjustment ahead of RIP at 90C. pH 4 was the input pH into the first stage of RIP contact and pH 5.9 was achieved after the last stage of RIP contact. This compares with a potentially significant loss of Ni and Co if inadvertent over-neutralisation of solution occurs ahead of a normal CCD wash circuit



18. Strip pregnant liquor composition produced from desorption of the resin from the first stage of resin-in-pulp contact using 5% H<sub>2</sub>SO<sub>4</sub> as the strip liquor was relatively high-grade, being:
  - upgraded in valuable metals by over three fold relative to HPAL discharge liquor to 13,000 ppm Ni, 1,000 ppm Co and 155 ppm Cu respectively
  - reduced in impurities relative to HPAL discharge liquor by over 25 to 100 fold to 20ppm Fe, 11 ppm Mn, 0.8 ppm Cr and 45 ppm Zn
  - a good candidate for either further upgrading by SX/EW or for high-grade intermediate sulphide, hydroxide or oxide production for refining or sale
19. The Clean TeQ resin is reported to be capable of being stripped by most common stripping reagents including HCl, H<sub>2</sub>SO<sub>4</sub>, Na<sub>2</sub>CO<sub>3</sub> and (NH<sub>4</sub>)<sub>2</sub>CO<sub>3</sub>.
20. Recognising the high Clean TeQ resin selectivity for Ni and Co over Mn, Zn and Cr, a number of interesting possibilities could arise for producing one or more of the following products via RIP processing or by RIP + SX processing:
  - a high purity chloride leading to high purity nickel/cobalt oxide intermediate production by amine SX to remove Co and then pyrolysis to recover Ni, somewhat similar to Inco's proposed flowsheet for Goro, except using RIP for the initial purity step rather than SX
  - a high purity sulphide for export
  - a high purity hydroxide for domestic sale
  - metal via an improved Bulong route such as IX for Cu, Zn, Fe and Mn removal, Co metal by SX/EW and Ni metal by SX/EW
  - other options involving chloride or sulphate circuit variants as may be appropriate

#### Issues Requiring Follow Up:

There are a number of issues that need to be addressed before a decision might be made to proceed with further resin-in-pulp testwork. These issues include:

1. **Solids assay discrepancy:** There remains an unresolved major discrepancy between two sets of solid assays between two different labs, whereby one lab was producing assays of approximately five times the other laboratory. To date it has not been possible to resolve the difference. To give the existing preliminary data credibility, this difference must be resolved, if necessary by independent third party audit.
2. **Iron oxidation:** It is puzzling as to how iron oxidation occurred without any apparent efforts being made to oxidise the pulp. As ferrous oxidation to ferric is usually slow, it is important to know how this oxidation occurred, in case there is a scale up problem.
3. **Resin supply:** At this stage, it is understood that the Clean TeQ resin would be supplied on the basis of local manufacture using an existing resin manufacturer in Australia. Issues such as resin cost, quality, inventory and plant design basis would need to be addressed if future test results are positive.:
4. **Scale of testwork:** Clean TeQ have advised that they are able to carry out laboratory testing to a limited pilot scale tests using their own facilities in Dandenong. It is understood that testing on any subsequent larger scale, if required, would require facilities that would need to be constructed by the mine owner or others. The scale of large-scale testwork, independent verification of the results and plant design criteria development would need to be reviewed.

5. **Plant design:** Resin-in-pulp processing has been used successfully on uranium in the USA, on a gold plant in South Africa and on large-scale gold and uranium plants in the CIS. Reagent suppliers such as Cognis are looking at resins. Key issues in successful application of resin-in-pulp technology are likely to include:

- cost, quality and reliability of supply of resins,
- kinetics and selectivity of the resins,
- size of the resin inventory,
- mechanical and chemical stability of the resin
- adequacy of plant design for successful countercurrent resin-in-pulp treatment in a commercial plant.

These issues will need to be addressed well as part of the process of deciding whether resin-in-pulp processing might become one of the preferred processing routes for Young.

**Turnkey RIP plant supply:** The resin supplier has expressed interest in supplying a RIP plant on a turnkey design and construct basis, if continuing testwork should be favourable and the project owner should decide to proceed to a commercial plant. The capabilities and interests of a resin supplier in the design and construction of such a plant need careful consideration, in case project financing requirements down the track might possibly require the use of an alternative contractor for plant design and construction.

Fig 5: Clean Teq Resin-In-Pulp Solution Treatment Profile: Young High Grade

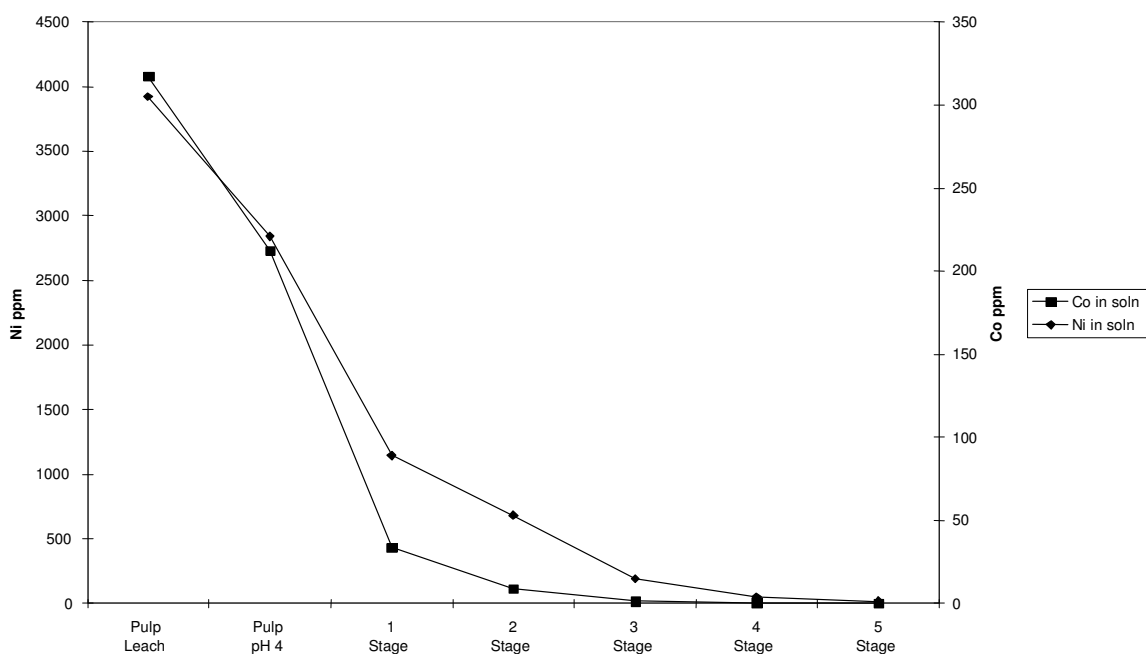


Fig 6: Clean TeQ Resin-In-Pulp Leached Solids Assay Profile: Young "High Grade"

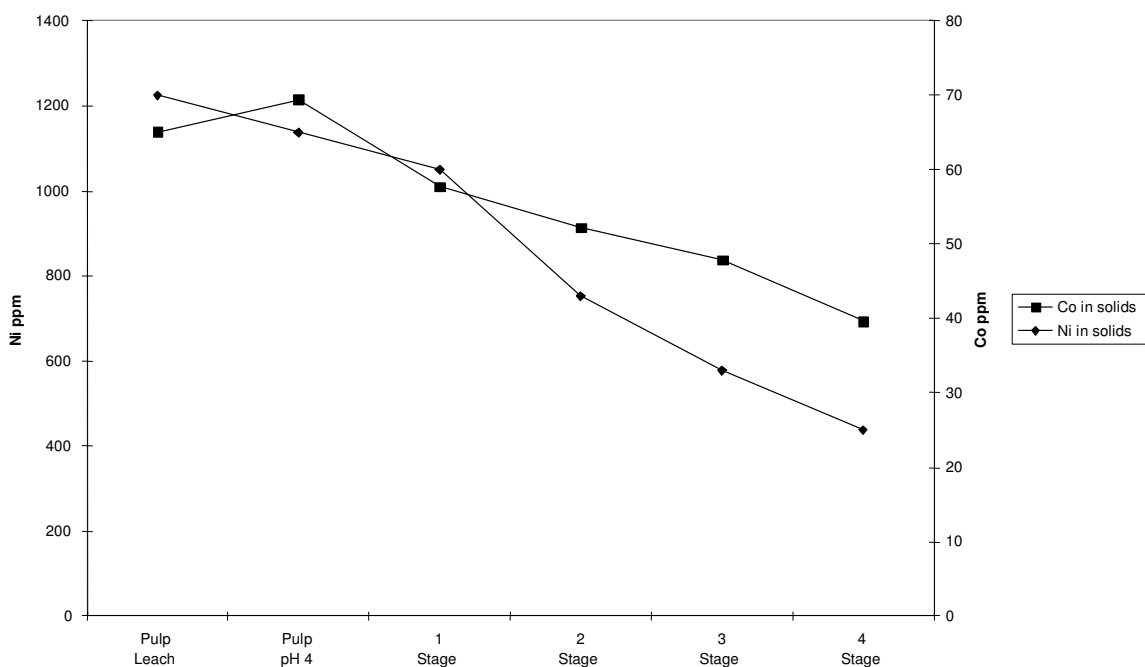
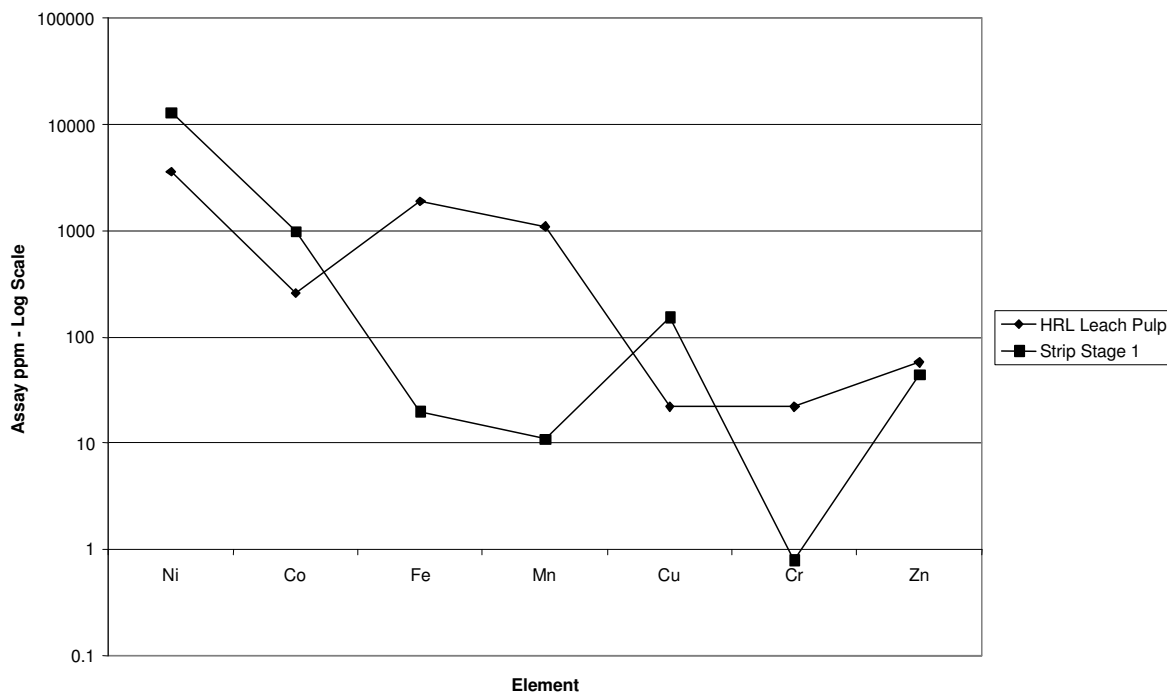


Fig 7: Clean Teq Leach Feed Soln versus RIP Stage 1 Strip Solution Assay Profile



## 14. HG AND ROM CONCLUSIONS

### HG and ROM Composite Preparation:

1. The composites appeared to fall into two main assay groups:
  - High Grade and Ardnaree composites with the higher nickel grades (0.78 – 0.99%Ni, 0.066-0.069% Co, 17.9-18.9% Fe, 10.5-11.1% MgO, 6.1-7.0% Al<sub>2</sub>O<sub>3</sub> and 26-27 gpt Sc)
  - East and West Thuddungra with up to 20% lower nickel grades, 20% higher cobalt grades and more extreme iron and magnesium grades (0.61-0.73% Ni, 0.081-0.083%Co, 16.6 - 25.5%Fe, 5.5-14.1% MgO, 7.4-9.9% Al<sub>2</sub>O<sub>3</sub> and 28-39 gpt Sc)
2. In all composites, the sum of the assays for iron (Fe) and magnesium (MgO) was essentially constant at 29.0-31.0%, despite a 50% increase in Fe from 16.6 to 25.5%Fe and a 250% increase in Mg from 5.5 to 14.1% MgO.
3. SG measurements of the composite materials using an air pycnometer were 2.95-3.15.
4. The material tended to agglomerate during sample preparation, with the finer samples Ardnaree, East Thuddungra and West Thuddungra being particularly problematic.
5. The general range of ore types included within the four composites included limonite (18-58%), saprolite (21-48%), weathered serpentinite (10-42%), fresh serpentinite (0-3%), hematite (0-10%) and some limited quantities of unidentified clays, with the compositions varying between composites.
6. Particle size decreased from coarse for Ardnaree (p80 of 420 micron) to moderately fine for East Thuddungra (p80 of 135 micron), with the other composites being of intermediate size (p80 of 180-190 micron).
7. Metal distributions with size were generally similar for the four composites including:
  - Ni relatively constant up to 212-600 microns, then decreasing with increasing size
  - Co broad concentration peak between 100-600 microns
  - Magnesium generally elevated in plus 2mm material
  - Plus 2mm material, representing 7-16% of the ore mass, was removed to provide a slight upgrading of the ore for subsequent testwork.

### Feed Thickening:

13. Thickening behaviour ranged from favourable for Ardnaree to reasonable for High Grade and East Thuddungra and difficult for West Thuddungra. Key results after prior dilution to 3% solids of minus 212 micron ore, use of 10L raked cylinders and flocculent addition included:

- Maximum final settled solids densities ranging from 28% for West Thuddungra to 33-36% for High Grade, 39% for East Thuddungra (after coagulant addition) and 40-44% solids for Ardnaree
- Depth corrected thickener unit areas ranging from to 0.31 m<sup>2</sup>/tpd for Ardnaree (35% solids using 82 gpt Magnafloc 358) to 0.38 m<sup>2</sup>/tpd for High Grade (at 30% solids and 125g/t Magnafloc 358)
- No improvement in settling rates or underflow density with use of highly saline water

- Greatly improved thickening rates and lower flocculent dosage with East and West Thuddungra composites when using coagulant prior to flocculent addition

#### Pressure Acid Leaching:

14 Leach tests conducted on ore ground to 100% passing 212 microns, and leached for 90 minutes under batch leaching conditions at an operating temperature of 250-265C and an initial solids density of 30-33% yielded the following key results:

- Leach extractions of up to:
  - 94-95% Co for all ores (0.074-0.085% Co head grade)
  - 93-95% Ni for High Grade, Ardnaree and East Thuddungra ores (0.71-1.06% Ni head grades)
  - 86% Ni for West Thuddungra (0.75% Ni head grade)
  - 75-82% MgO for High Grade, East and West Thuddungra (5.7-13.8 % MgO head grade )
  - 52% MgO for Ardnaree (10.4% MgO head grade)
- 20-50% refractory magnesia level in feed, especially Ardnaree
- No additional nickel extraction by post leach conditioning
- Variable acid additions ranging from a maximum of 0.42 – 0.48 t/t for High Grade, Ardnaree and West Thuddungra to a low of 0.33 t/t for East Thuddungra
- Variable leach kinetics ranging from fast for Ardnaree to slow for High Grade and West Thuddungra where 80% was easily leached and the remainder more slowly leached
- Nickel extractions generally increasing with higher free acid concentration at the end of the leach, higher reaction temperature and possibly finer grind
- Reaction mechanism suspected to be chemical reaction control rather than diffusion control
- Favourably low levels of leach impurities including iron (1-1.3 gpl Fe), aluminium (0.7-2.5 gpl Al) and chrome in the leach liquor (<100 ppm) and no soluble chromium
- Poor predictions of acid soluble magnesium extraction by an HRL developed atmospheric diagnostic leaching test

#### Leach Residue Thickening:

15. Key results included:

- Final solids densities decreasing from 36-37% for High Grade, to 30% solids for Ardnaree and 28-31% for East and West Thuddungra
- Flocculent dosages of 90-130 gpt Magnafloc 800HP except for at least 130 gpt for West Thuddungra
- Thickener unit settling areas ranging from a high of 2.4 m<sup>2</sup>/tpd for West Thuddungra to 2.0 m<sup>2</sup>/tpd for Ardnaree, 0.8 m<sup>2</sup>/tpd for East Thuddungra and 0.4 m<sup>2</sup>/tpd for High Grade (all at 28 wt% solids)

### Resin-In-Pulp:

16. Higher than expected solution recoveries were achieved with Clean TeQ resin-in-pulp treatment of a feed pulp, after pH 4 neutralisation of the high acid feed pulp with approx. 200% stoichiometric Young magnesite at 90C
17. Higher than expected additional solids leaching appeared to occur during the resin-in-pulp contacting at 90C, suggesting that actual leach recoveries via resin-in-pulp may be significantly higher than laboratory recoveries achieved by batch leaching and CCD treatment.
18. Over-neutralisation of the feed pulp with magnesite had no detrimental effect on subsequent Ni and Co recovery, despite an initial precipitation of some of the aqueous Ni and Co in the solids
19. Strip solution grades after resin-in-pulp treatment were over three-fold enriched in nickel and cobalt from the acid leach and very low in iron, manganese and chromium impurities, suggesting more flexibility in production of a very high-grade intermediate product direct from leaching than from more conventional processes.

### Mineralogy:

20. The mineralogy was extremely complex and mineral phases generally impure due to intricate mixing of phases:
  - smectite was present as co-dominant or dominant mineral in all size fractions, but particularly in the -20 micron fraction
  - lizardite, chlorite, goethite, talc, chromite were present as a minor to co-dominant depending on the size fraction
  - quartz was present as trace to co-dominant depending on the size fraction
  - separate specific Ni-bearing phases were not detectable, but Ni was most considered most likely to occur in smectite and possibly goethite.

## 15. HG AND ROM RECOMMENDATIONS

To offset a lower than average resource grade compared with other competing projects, it is recommended that future high grade testwork should address issues such as:

- metallurgical treatment of higher resource grades for early years of operations, preferably above 1% Ni grade, if available
- characteristics of the major ore types and blends expected during the mine life
- economic optimisation of Ni and Co leach recoveries versus acid consumption and neutralisation costs
- use of resin-in-pulp processing rather than conventional neutralisation and CCD wash thickener processing, if feasible
- relative costs of neutralisation using site-based resources, including low-grade ore, Causmag low-grade and high-grade magnesite and/or other low cost materials such as limestone, where available
- possibility of using finely ground magnesite as a lower cost alternative to magnesia for complete Ni/Co hydroxide precipitation from neutralised HPAL solutions.
- grades of sulphide or oxide intermediate products produced by conventional CCD wash, iron precipitation and intermediate product precipitation flowsheets versus resin-in-pulp flowsheets
- conditions to achieve maximum thickener underflow solids.
- use of site water for critical testwork

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