Geochemical Characterisation- Waste Rock Solute Source Terms Stage 2 and Main Pit Tailings

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Acknowledgements

SUMMARY

Background to Report

The primary objective of the Rum Jungle Rehabilitation Project from an environmental geochemistry perspective is to relocate and appropriately contain mine waste materials such that surface water quality objectives will be maintained for the indefinite future. This report addresses outstanding issues related to minimising the future geochemical risk posed by sulfidic waste (tailings and waste rock) on site.

Issues related to tailings management are addressed in the first part of the report whilst those focussed on waste rock handling are addressed in the second part. The work on tailings is new and the reasons for doing this are summarised below. The additional work on waste rock addresses specific recommendations to confirm and enhance the reliability of neutralant demand (limestone addition to neutralise existing contained acidic sulfide oxidation products) estimates made in the original geochemical characterisation report (RGC and DJEE 2016).

The original preferred option (Option 6) in the Stage 2a plan involved backfilling the Main Pit with the highest sulfide-containing potentially acid forming (PAF) waste rock (PAF I and PAF II categories), and placing the remaining residual PAF-II, and all of the lowest sulfide (PAF-III and non acid forming, NAF) rock, in a new purpose built waste rock storage facility (WSF). However, being able to safely and effectively backfill the Main Pit with rock is complicated by the occurrence of a substantial (approximately 50m deep) layer of deposited tailings in the base of the pit.

It was originally concluded that placing backfill rock over tailings left *in situ* posed an unknown level of geotechnical risk for two reasons: (1) the load bearing characteristics of the tailings were not known and could pose a substantial physical and safety risk to the successful placement of the rock and (2) ongoing consolidation of the tailings following placement of the waste rock and ultimate dry cover could have compromised the long term stability of the landform necessitating ongoing post rehabilitation management.

As a result of then unknown properties of the tailings the costed preferred option submitted to the Commonwealth in mid-2016 included the conservative provision that the tailings would be removed, treated, dewatered and encapsulated in the new WSF. Subsequent to the submission of this plan the issue was revisited as it became apparent that it could be possible, based on experience elsewhere with capping in-pit tailings, to dewater the pit and progressively place the capping layer. Whichever of the two options (leave or remove tailings) is finally selected it was essential that the properties of the tailings profile be determined to fully inform the decision process.

The highest priority was for geotechnical characterisation of the tailings profile. However, it was also important to establish the geochemical properties. In the event of the tailings being removed, the neutralant demand needed to be determined for the tailings reclaim treatment plant and the residual sulfide content measured to inform future AMD potential in the above grade waste storage facility. If the tailings were to be left in place then there would be some degree of mixing of tailings and associated pore water (induced by vertical expression by the overlying pressure of the capping rock) with the inflowing groundwater, with excess water needing to be treated by the site's water treatment plant. Thus an assessment of the potential extent of this pathway was also required.

Tailings Characterisation

A tailings sampling program to 28m below the tailings water interface was undertaken in the Main Pit in January 2018 by SRK Consulting, using a sampling rig mounted on a floating barge. The geochemical characterisation results for selected profile samples are reported here. The original purpose of this characterisation work was to understand requirements for safe storage of the tailings material within the WSF as proposed in the Stage 2 design. However, it will also inform the potential for tailings to impact the overlying water column in the event that the status quo condition of the Main Pit is maintained.

The results obtained from the tailings profile samples show that there is a general trend of increasing pH (pH 4 to 6) with depth within the tailings bed, which is attributed to self-neutralisation by the chlorite-schist matrix of the ore having occurred during the 50-60 y since the initially strongly acidic (pH 1.5) tailings were deposited.

The tailings have a moderate potential (corresponding to the PAF-II classification for waste rock) to produce AMD if the contained sulfide–S was to be completely oxidised. This will not be an issue if they are left in situ. However, if the tailings were to be removed from the pit they would need to be appropriately encapsulated in the new waste storage facility (WSF) to prevent future oxidation. In contrast to the waste rock there is minimal content of jarosite so this source of slowly reacting contained acidity does not need be be specifically addressed by the amount of neutralant added.

A probable worst case estimate of composition of tailings porewater needing to be treated as a result of the mixing interaction of the topmost tailings with waste rock is provided by the 5:1 water leachate from sample 1002. To this estimated solute source from the tailings needs to be added solutes and acidity coming from the added waste rock. If limestone was to be added to the rock in advance of deposition, then the combined solute source terms would be greatly reduced.

A substantial reduction in the concentration of most leachable metals can be achieved by raising the pH of the tailings to 7. However, for Co, Mn, and Ni a target pH of 8 may be needed, depending on the how the treated water stream is to be managed on site. It is noted that approximately 50% more neutralant will be needed in the event that the target pH for neutralisation of the tailings sludge is 8 rather than 7 (see section 2.7.4 for details). The high concentrations of extractable As, especially at pH 8 and 9, for samples in the lower part of the profile, are of concern for the supernatant water that could be produced as a product of neutralisation of the tailings (for the remove tailings option). This would be a risk factor for treatment only if the tailings were to be removed, with the occurrence of extracted As needing to be addressed by the treatment process.

The neutralant demand was found to decrease very steeply over the top 5m and then much more gradually thereafter. The demand for the top 5m will be most relevant if the tailings are left in situ, whereas the much lower demand at less than 5m would be applicable if the tailings needed to be removed prior to backfill. For comparison, the neutralant demand for the top 5m is similar to the average neutralant demand for PAF-II waste rock (6.8 kg $CaCO_3$/t dry weight).

The data produced by this program of testwork will enable estimates to be made of the amounts of neutralant required to treat the lens of contaminated water overlying the tailings, tailings porewater, and tailings sludge, thus enabling the addressing of many possible rehabilitation scenarios for the pit.

Waste Rock Characterisation

The second major part of this report contains the data and assessment thereof, for the additional inverse leach work that was recommended to be done for waste rock in the geochemical characterisation report originally submitted in June 2016 (RGC and DJEE 2016). Given that the focus of the work contained in this report was on PAF-I and PAF-II material from the Main and Intermediate WRDs it was recommended that further work be done to confirm the estimated neutralant demand for the lower end of the existing acidity range, since this this material will comprise a large fraction of the volume of the WSF. In particular, it was noted that some additional inverse leach work should be done for the PAF-II and PAF-III material from Main WRD and PAF-III material from the Dysons WRD, since this latter material was not subject to more detailed leach testing assessment in the original work program.

The most recent program of testwork has confirmed the methods used for estimating neutralant demand of the waste rock samples. However, for two of the samples tested the neutralant demand was found to be substantially higher than when fresh material was tested in late 2014. This is suspected to be the result of oxidation that has occurred during storage over the 3.5 years since the samples were collected and is a reflection of the storage conditions for at least some of the library of samples. When the samples were retrieved for the current work it was noted that some of the containers were starting to deteriorate and identifying marker labels fading as a result of UV exposure. Accordingly, it is recommended that an inspection be undertaken as a matter of urgency of the sample inventory. Material in physically suspect containers should be repackaged and containers with fading labels should be re-labelled.

For all of the samples bar one achieving the target pH range of 6.5-7.5 resulted in the removal of all metals apart from Mn to below, and in most cases well below, 1mg/L. The one exception was the high PAF-I sample from the Intermediate WRD. This most recent testwork indicates that achieving substantive reductions in high leachable concentrations of Co and Ni by addition of agricultural lime ($CaCO_3$) will be more problematic for material from the Intermediate Pit. This finding highlights and further reinforces that all of the material from the Intermediate WRD needs to placed first in the pit so that it is covered by material from the Main WRD, the material from which generally contains lower leachable concentrations of Mn, Co and Ni.

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1 INTRODUCTION

Figure 1.1 shows the locations of the key site features discussed in this report.
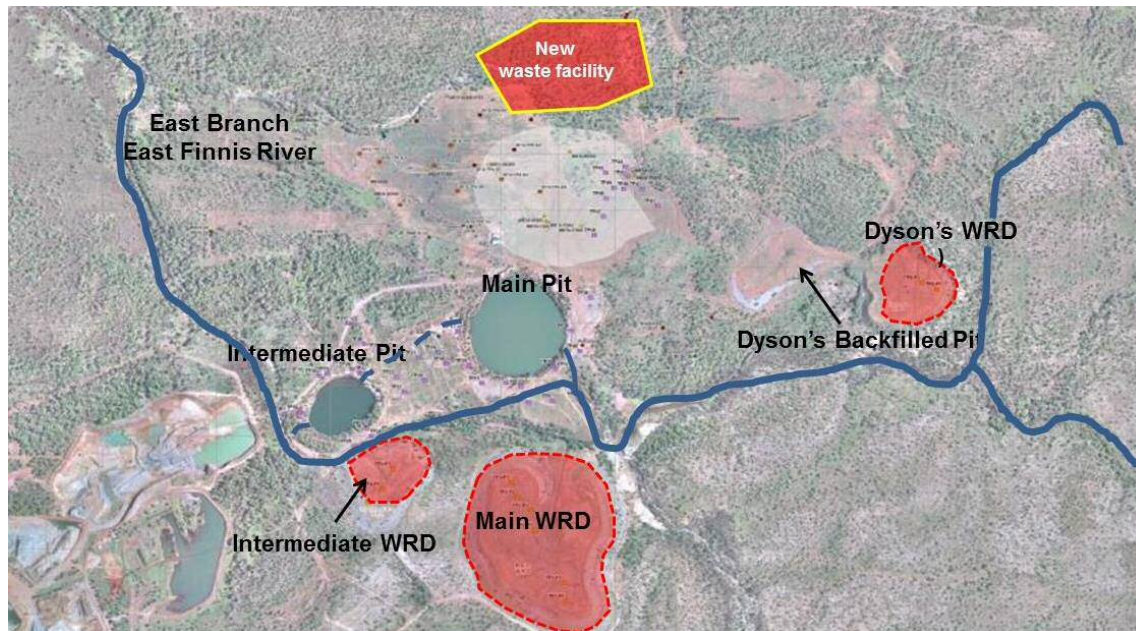
Figure 1.1 Locations of existing features and the proposed new WSF at Rum Jungle

The primary objective of the Rum Jungle Rehabilitation Project from an environmental geochemistry perspective is to relocate and appropriately contain the waste rock such that surface water quality objectives will be maintained for the indefinite future. To achieve this objective, the preferred option involves backfilling the Main Pit with the highest sulfide-containing potentially acid forming (PAF) material (PAF I and PAF II categories), and placing the remaining residual PAF-II, and all of the lowest sulfide (PAF-III and non-acid forming, NAF) rock, in a new purpose built waste rock storage facility (WSF). However, being able to safely and effectively backfill the Main Pit is complicated by the occurrence of a substantial (approximately 50m deep) layer of deposited tailings. There are several geotechnical-related factors that need to be addressed in this context.

- The load bearing characteristics of the tailings which could seriously complicate the placement of overlying waste rock.
- The safety issues associated with (1) in the context of surface instability.
- The ongoing vertical consolidation of the capped tailings that could lead to the need for long term engineering maintenance of the surface cover system.

As a result of the then unknown nature of the tailings layers the costed preferred option submitted to the Commonwealth in mid-2016 included the conservative provision that the tailings would be removed, treated, dewatered and encapsulated in the new waste storage facility. The cost of this process was a substantial proportion of total estimated project cost. Subsequently, the issue has been revisited and it is now considered that it could be possible, based on experience elsewhere with capping in-pit tailings, to dewater the pit and progressively place the capping layer. Whichever of the two options is finally selected (leave or remove tailings) it was essential that the properties of the tailings profile be determined to fully inform the decision process.

The highest priority was for geotechnical characterisation of the tailings profile. However, it was also important to establish the geochemical properties. In the event of the tailings being removed, the neutralant demand needed to be determined for the tailings reclaim treatment plant and the residual sulfide content measured to inform future AMD potential in the above grade waste storage facility. If the tailings were to be left in place then there would be some degree of mixing of tailings and associated pore water (induced by vertical expression by the overlying pressure of the capping rock) with the inflowing groundwater, with excess water needing to be treated by the site's water treatment plant. Thus an assessment of the potential extent of this pathway was also required.

A tailings sampling program was undertaken in the Main Pit in January 2018, using a floating barge bearing a sampling rig. The geochemical characterisation results for selected profile samples are reported here (Section 2). The results of the geotechnical characterisation will be reported separately by SRK Consulting.

The second major part of this report (Section 3) contains the data and assessment thereof, for the additional inverse leach work that was recommended to be done for waste rock in the geochemical characterisation report originally submitted in June 2016.

The previous work had focused on those materials that are at the higher end of the range of existing acidity, so as to confirm the initial estimates of neutralant demand and to investigate how fast the jarosite (a very low solubility iron hydroxyl sulfate secondary mineral) component is consumed following the addition of neutralant (RGC and DJEE 2016). It was recommended that further work be done on the lower end of the existing acidity range since this this material will comprise the bulk of the WSF. In particular, that some additional inverse leach work should be done on the PAF-II and PAF-III material from Main WRD and PAF-III material from the Dysons WRD.

2   TAILINGS CHARACTERISATION

2.1   Overview

Tailings profile samples from the Main Pit were collected from a floating barge during January/February 2018 by a team from SRK Consulting. A combination of piston cores and grab samples were obtained. Selected samples from the most intensive of these profiles were used for the geochemical characterisation work reported here.

This section of the report is structured as follows:

- History of tailings deposition at Rum Jungle;
- Description of the field sampling program and selection of samples;
- Description of methods used for subsampling of bulk samples
- Summary of geochemical characterisation methods; and
- Reporting and assessment of geochemical data

2.2   History of tailings production at Rum Jungle

The uranium ore was ground to about 51% <74μm prior to extraction. The uranium was then extracted using a conventional sulfuric acid leach process, with pyrolusite (MnO_2) used as the oxidant (Davy 1975). The tailings were discharged as a 55% (by weight) un-neutralised slurry at the rate of about 1ML/d at a pH of about 1.5. Reference to the Table 2.1 below shows that the tailings from the Rum Jungle South ore deposit comprise the majority of tailings in the Main Pit. Published records indicate that no other significant amounts of non-uranium tailings (eg from Cu ores) were deposited in the Pit.

Table 2.1 Summary of Tailings Disposal History at Rum Jungle

Year	Source material	Destination
1954-1961	Whites, Dysons	Surface disposal – poorly constrained-tailings "dam" area
1961-1965	Whites, Mt Burton, Dysons, Intermediate, toll processing from other locations	Dysons Pit. 600,000t
1965-1971	Rum Jungle South	Main Pit –Subaqueous disposal, 700,000t

Compiled from Mudd (2005) and AAEC (Davy 1975)

Chloritic schist was the dominant mineral phase in the material extracted from the Rum Jungle South deposit. Most of the U mineralisation was contained in greyish green pyritic chlorite/biotite schist, with a minor amount in adjacent black slate. (Berkman 1968).

Coupling of current bathymetric survey data with that of the original pit shell indicates a maximum depth of tailings of about 50 m in the centre of the pit, with about 25% of the original pit volume being filled with tailings (RGC 2016). Figure 2.1 reproduced from RGC 2016 shows the current cross-sectional configuration of water and solids in the pit
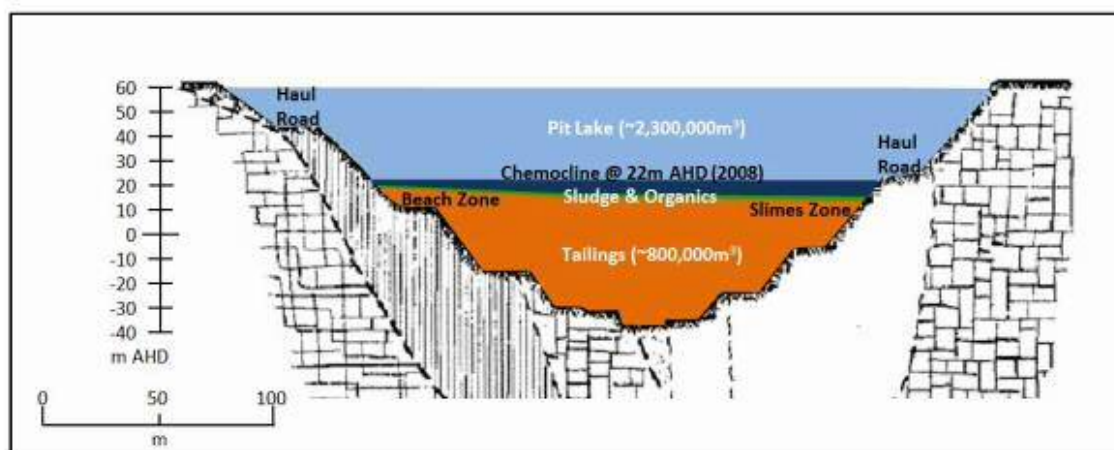
Figure 2.1 Schematic cross section of Main Pit (from RGC 2016)

2.3 Water Quality Profile in Main Pit

At the end of mineral processing especially the Main Pit and to a lesser extent Intermediate Pit had been the repository for the poorest water quality on site. Subsequently these pits also received the acidic and metal rich leachate emanating from the waste rock dumps. It was clearly recognised as part of the initial planning for the remediation program conducted in the 1980s that the discharge quality from the site would not be improved unless the water quality in the pits was substantially improved. Accordingly, a very large proportion of the total site rehabilitation was ultimately spent on water treatment. The strategy involved withdrawing the highly acidic and metal-rich water from depth and returning the lower density (lower TDS) to the surface. By this means a vertically stratified system was created with the cleaner water on top. The original post treatment thickness of the lens of still highly contaminated water in the Main Pit was about 22m above the tailings water interface. Through time this lens has been progressively reducing in thickness as a result of annual hydrodynamic stripping flushing occurring during the wet season through flow. The most recent profile measurement was made by the EMU group in mid-2014. At that time the lens was 3-4m thick,

The composition of the bottom lens of water is provided for reference in Table 2.2 below as it is of direct relevance for the current work. This composition is very similar to that reported several decades ago for the composition of untreated Main pit water (Lawton 1996). Calculations of the ion balance coupled with the negligible concentrations of dissolved oxygen (DO) and acidic pH indicate that the majority of dissolved iron is likely to be present in the ferrous (Fe^{2+}) form.

Table 2.2 Composition of bottom water in Main Pit (Boland 2008)

Parameter	Units	Value
pH		3.8
EC	µS/cm	8300
DO	mg/L	<0.3
Ca	mg/L	475
Mg	mg/L	900
SO4	mg/L	7800
Fe	mg/L	1000
Al	mg/L	150
Mn	mg/L	220
Cu	mg/L	32
Co	mg/L	13.5
Ni	mg/L	11
Zn	mg/L	5.8
Cd	mg/L	0.015

This water will inevitably need to be treated, irrespective of whether the tailings remain in place during pit backfill or are removed prior to pit backfill. This water is also likely to be closely reflective of the composition of the tailings pore water in the near surface tailings, and hence be manifest in the pH, EC and leachable metals present in this material. Of significance is that the bottom water is anoxic/anerobic indicating that strongly reducing conditions prevail. This is a major contrast to the variably oxic surficial environment of the waste rock dumps and will have an influence on the processes controlling metal solubility.

2.4   Field Sampling of Tailings & Sample Selection

The locations of samples collected during January and February 2018 are shown in Fig 2.2. They were obtained by a third party for geotechnical testwork. The author of this report was not involved with the sample collection so had no control over maintenance of geochemical integrity from collection to when final sub-sampling was done for geochemical characterisation.

The description below of the sampling process was provided by Andy Thomas from SRK.

"Piston samples were collected by extruding the sample from the sonic core into the piston tube (the ends of which were then sealed with molten wax plugs – words added by author of this report). Bulk samples were collected following logging of the core trays. These were exposed to oxygen for varying amounts of time. The samples collected in the slimes zone (18CPT10) were sampled immediately. However, other samples were exposed to the atmosphere in the core trays for a substantially longer period of time. The bulk samples were collected by hand wearing nitrile/latex gloves and placed into Ziploc bags and double-wrapped. Unlike the piston samples, the pore-water of the bulk samples may have been mixed with rain water if there was precipitation (which could affect pH). No pH or EC measurements were collected in the field."
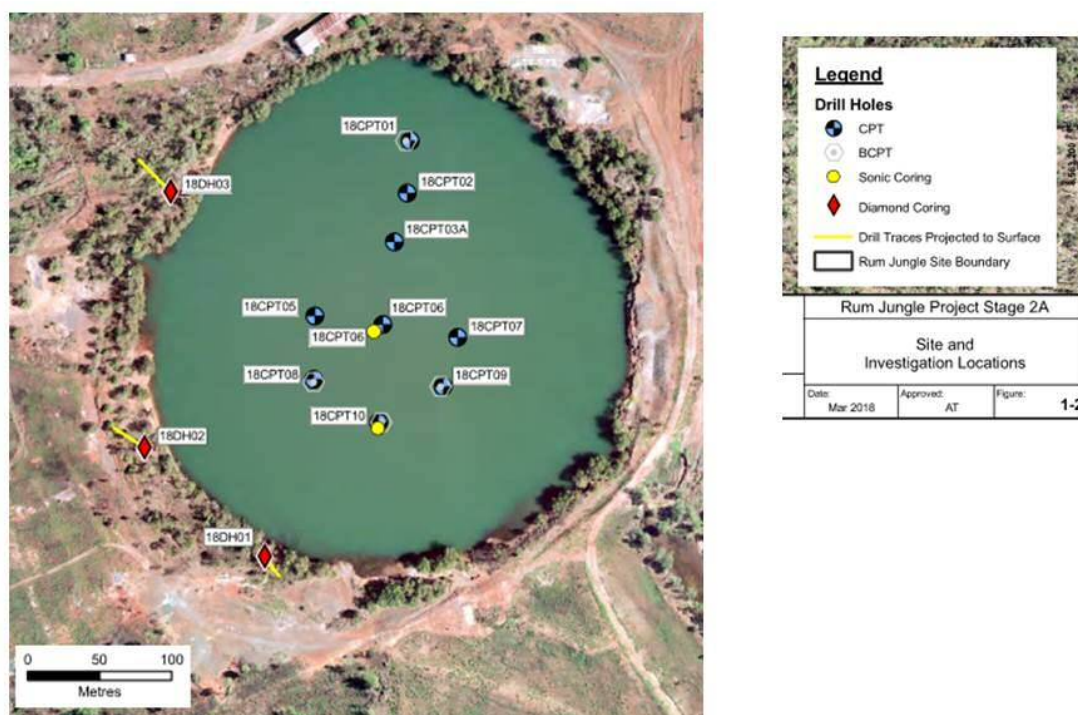
Figure 2.2 Tailings sampling locations in Main Pit (Andy Thomas SRK, pers comm)

Given both the depth of profile able to be sampled and the likely better extent of preservation of geochemical integrity of the samples, preference was given to those profiles from which sonic cores had been collected. Only two locations, 18CPT06 and 18CPT10 met these conditions. Note, that since no measurements of EC or pH were made in-the field it is not possible to compare these high-level indicator parameters between locations to provide an indication of how representative these two sonic core profiles might be of the tailings mass as a whole. Given the original method of tailings deposition was from around the perimeter of the pit, the only thing that can be said is that since the locations of the sonic cores are towards the centre of the pit they are more likely to contain a higher proportion of fines than would be the case for those samples obtained from closer to the pit walls.

Following consultation with M Fogg from the DPIR regarding the primary requirement of sufficient samples for geotechnical testing, and the emphasis on piston core samples for geochemical testing it was decided to focus on the 18CPT10 profile from which many more piston core samples were available for selection.

2.5   Subsampling of Tailings for Characterisation (May 2018)

The bulk samples were contained in ZipLock plastic bags and the piston core samples were contained in metal tubes. All samples had been further contained in plastic storage bins with lockable lids.

A major concern was that the samples had been collected with geotechnical characterisation in mind rather than geochemical characterisation. In this respect ingress of oxygen and/or drying out of a sample could have comprised its integrity, noting that a period of approximately 5 months had elapsed

since the samples had been collected to when they were sub-sampled for geochemical testing. The "bulk" samples were of especial concern in this regard.

Some weeks prior to undertaking this work Michael Fogg from DPIR had undertaken an inspection of the samples and noted that some of the available bulk samples had leaked and/or dried out. These were not used. Michael also double-bagged the remaining bulk samples and wrapped cling wrap around the ends of the piston cores, for some of which the wax seals did not appear to be fully competent (viz leakage).

The 18CPT10 profile samples selected for geochemical testing are listed in Table 2.3 below. It should be noted that the deepest sample able to be collected was only just over half of the maximum depth of tailings in that part of the pit.

In order to protect the samples from oxidative geochemical change during sub-sampling, the bags and cores were opened under an atmosphere of welding grade argon. The bulk samples were manipulated in a large plastic bag purged with argon. The piston core tubes were opened and subsampled in an improvised glove box purged with argon. This process is documented in a series of images in Appendix 1.

The extracted samples were inspected for evidence of oxidation. The small volume of excess water associated with the bulk samples did display evidence of ferrolysis (ie oxidation of ferrous iron). However, given the very fined-grained (heavy plastic clay) fully saturated nature and greyish colour of the bulk mass of the material the oxidation front may have not penetrated significantly (see later assessment of this issue).The several kg bulk mass of the material from each bulk sample was hand mixed (by kneading) prior to extraction of the sub-samples for analysis.

In the case of the piston cores, samples were collected from each end of the intact vertical profile section to determine if there was any evidence of short-range vertical variation in properties.

For each sample two splits were taken for submission to the analysis laboratory (ALS Brisbane). One split for drying and ABA suite, and the second split to be maintained wet for water leaching and titration. Three replicate samples were also submitted to provide a check on the reproducibility of the sub-sampling procedure.

An additional aliquot of the typically plastic sludge was mixed at approximately a 2:1 ratio with demineralised water to measure slurry pH and EC. A limited subset of samples was also titrated in the EMU lab with sodium hydroxide solution to measure neutralant demand and metal removal as a function of pH.

2.6   Tailings Characterisation

2.6.1   Darwin Laboratory Measurements

2.6.1.1   Physicochemical Parameters

As noted above measurements of 2:1 slurry pH and EC were made immediately after sub-sampling. The results are compiled in Table 2.3, together with the solids content (for reference) of the materials subsequently measured by ALS.

It is clear that the deeper tailings are better consolidated as evidenced by the trend of increasing wt% solids with depth. Indeed, only the top metre or so is below the notional 55wt% figure used for the initial tailings deposition. The relevance of these wt% values to the geotechnical properties of the

tailings are the subject of a separate report to be produced by SRK Consulting. They will be used here to normalise metal contents, leachable metals and neutralant demand as a function of depth.

Table 2.3 Physicochemical characteristics tailings profile samples

Sample ID [1]	Depth (m)	EC µs/cm	pH	Solids wt%
1002	0.5	3820	3.4	47.6
1005	2	3330	3.78	63.6
1007	3	2760	3.67	66.7
P3T	6	3480	5.02	70.6
P3B	6.5	3350	5.57	70.6
1019	9	1622	3.89	64.3
1025	12	1485	4.01	69.6
1029	14	1747	4.26	74.3
P7T	18	1468	5.59	71.4
P7B	18.5	1956	5.3	72.2
1037	21	1504	4.85	73.5
P8T	25	1313	4.49	77.8
P8B	25.5	943	5.5	73.3
P10T	27.5	1002	6.06	73.3
P10B	28	985	6.05	81.3
1047 Rep	27.5	1025	4.6	77.1

[1] P designates piston core samples, bulk samples have a numeric code. The suffix T refers to the top of the piston sample, and B designates the bottom. Piston cores were approximately 0.5m in length.

The EC profile exhibits a step change below about 6.5 m where it halves before declining steadily further with depth (Figure 2.3). The EC values do not correlate in any systematic way with pH.
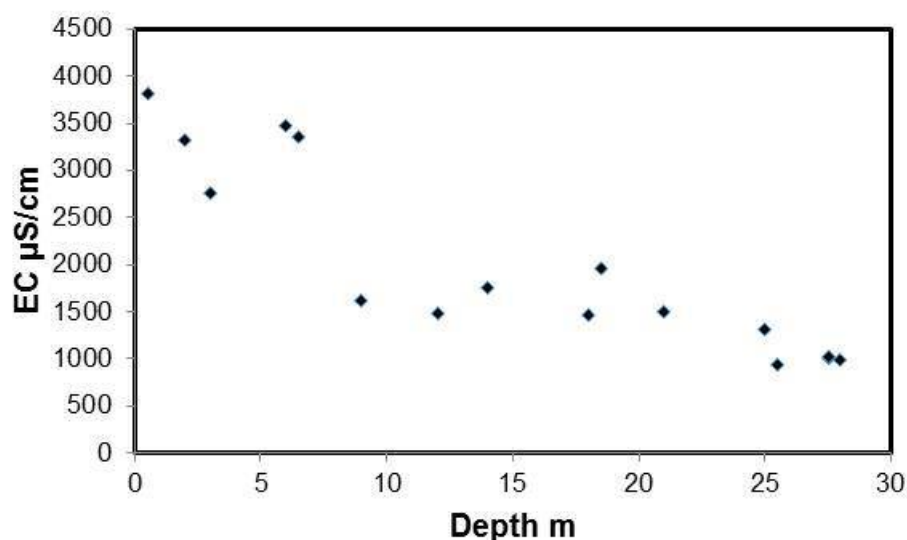
Figure 2.3 Electrical Conductivity Profile 2:1 water extract

Figure 2.4 shows all of the pH data from the bulk and piston core samples. Whilst there is a visual trend of increasing pH from the top (close to water/solids interface) and bottom of the profile, the plot is very "noisy", with the Piston Core samples appearing to have a systematically higher pH (by

about 1 to 1.5 units) than the bulk samples (Figure 2.5). This offset suggests that some oxidation has occurred in the bulk samples leading to a reduction in pH. Notwithstanding this offset, the trend of increasing pH with depth is apparent for both types of samples. Unfortunately, piston core samples were not available for geochemical testing for depths less than 6m owing to the requirement for the shallower piston cores to be used for geotechnical testwork. Thus it is not known if the observed offset would have been maintained for shallower depths.
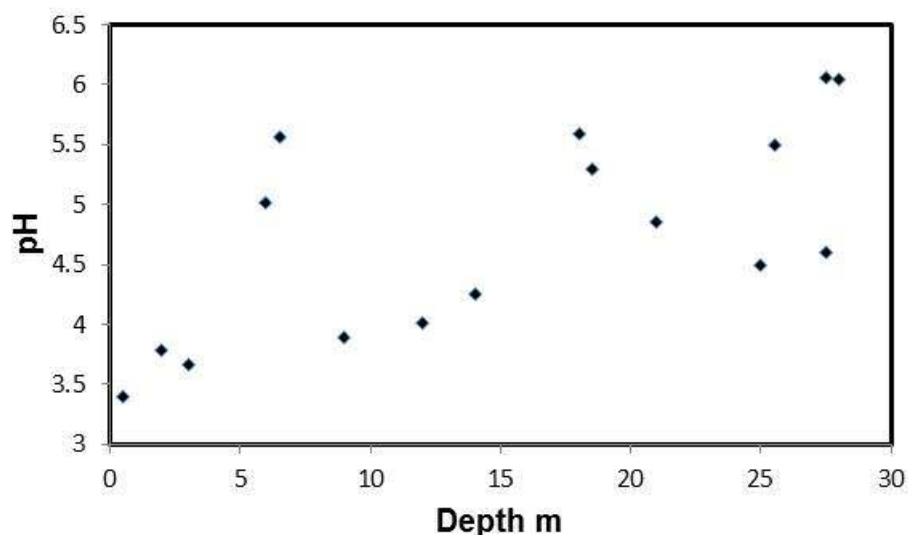
Figure 2.4 pH Profile for 2:1 water extracts
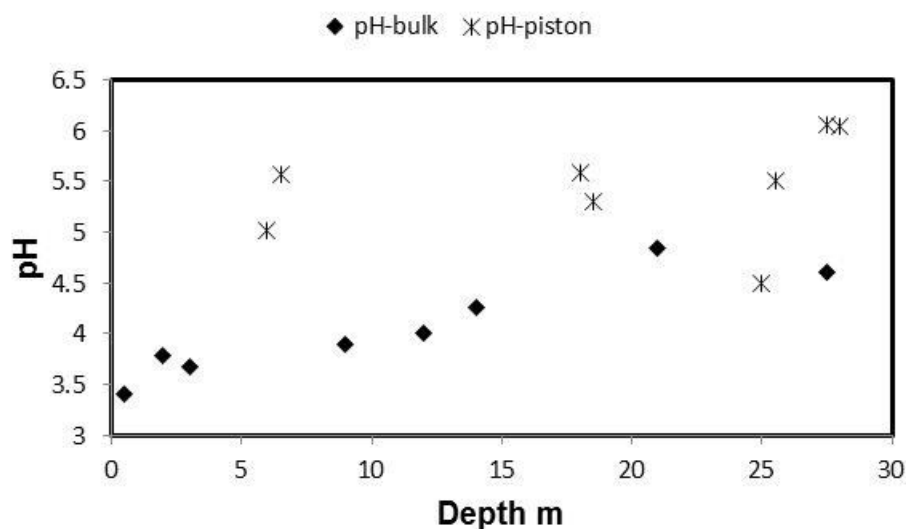
Figure 2.5 Comparison between pH data for bulk and piston core samples

Noting the initial pH of the deposited tailings was about 1.5, there has been a substantial increase in pH over the 50-60 y since this material was deposited. An increase in pH through time is to be expected as a result of the neutralisation of the initial acidity by reaction with the chlorite-schist matrix of the ore. Given that the deeper tailings were deposited some years earlier than the shallower tailings it is logical to expect a higher pH at depth, noting that the mineralogical composition of the feed ore from RJCS remained similar over the years of processing.

A substantial amount of work (Miller et al, 2010; Ciccarelli 2013) has been done over the past decade on the capacity of silicate minerals to neutralise AMD in waste rock, under conditions where the initial carbonate neutralising capacity has been exceeded or for waste that has an intrinsically low or zero initial carbonate content. The conclusion is that the neutralising capacity supplied by reactive gangue silicate minerals (e.g. chlorite) can buffer low-order acid generation to pH 4.5-7 in mine waste samples. This range of pH values is consistent with those observed for the bulk of the Main Pit tailings profile samples, noting that this is a "closed" system with a non-renewing initial source of acidity in contrast to a waste rock dump with ongoing production of acid from sulfide oxidation.

2.6.1.2 Leachable Solutes and Neutralant Demand

A limited number of samples (5) were characterised for leachable metals and neutralant demand in the EMU laboratory. This involved titrating up to 500mL of an approximately 5:1 L/S slurry of tailings material with sodium hydroxide to target pH values of 7, 8, and 9. The objective of this work was to inform the treatment requirements needed for tailings for either the leave-in-place or removal-prior-to-backfill options. This larger scale work is complemented by the small-scale titration (to pH 7) work done on all samples by ALS. The results from this latter work are reported in a subsequent section.

The data in Table 2.4 indicate that the highest neutralant demand occurs down to at least 10m in depth, with substantively lower neutralant demand at the bottom of the sampled profile. The pH 7 titration data to be reported later for all of the samples will define the interval between 10m and 21m.

The metals concentration data are compiled in Table 2.5. It is clear that although the neutralant demand is a strong function of the starting pH, the optimum pH for removal of most metals is around pH 8 for all samples. Any higher than this results in a substantial increase in the concentrations of amphoteric metals (Al) and those metals/metalloids that are present as oxyanions (As, Se) or form oxyanionic species (U) at higher pH values. There is also a substantive increase in neutralant demand above pH 8 as a result of precipitation of Ca and Mg, which is not desirable from a neutralant efficiency perspective.

The high concentrations of extractable As, especially at pH 8 and 9, for samples in the lower part of the profile, are of concern for the supernatant water that could be produced as a product of neutralisation of the tailings (remove tailings option). This would be a risk factor for water treatment, with the occurrence of extracted As needing to be addressed by the treatment process.

A limited investigation was also done of neutralant demand for the supernatant only versus the complete tailings slurry using sample 1005. The smaller volume 2:1 L/S ratio samples prepared for the paste pH and EC measurements were used for this assessment, and the results are shown in Table 2.6. It is the ratio of the titres for the supernatant and slurry that is of relevance here, noting that the absolute volumes of the titres cannot be compared directly with the data in Table 2.4.

Table 2.4 Neutralant demand for selected tailings profile samples

Sample ID	Depth m	pH	CaCO3 kg/t wet [1]	CaCO3 kg/t dry [2]
1005	2	3.73		
		7	2.18	3.42
		8	2.80	4.40
		9	4.00	6.29
1019	9	3.79		
		7	2.23	3.46
		8	3.32	5.17
		9	4.02	6.26
P8T	25	4.04		
		7	1.07	1.37
		8	1.46	1.88
		9	2.10	2.70
P8B	25.5	6.46		
		7	0.12	0.16
		8	0.37	0.50
		9	0.85	1.15
1047	27.5	4.56		
		7	0.50	0.65
		8	0.80	1.04
		9	1.08	1.39

[1] per kg wet weight of tailings; [2] per kg dry weight of tailings

It is clear that the majority of the neutralant demand is contributed by the solids component in the slurry. This finding is of practical relevance if only the settled supernatant from an initial slurry suspension was to be treated. This could happen if an initially solids-rich slurry suspension produced by dumping of waste rock into the tailings mass was allowed to settle before pumping off the supernatant to a treatment plant.

The starting pH metals concentration data for sample 1005 can be used to explore the processes controlling solubility of the solutes in the tailings, by comparing the scaled (by dilution) concentrations of the solutes by the contaminated water overlying the tailings. Sample 1005 is 63.6% wt solids which means that the remainder is water that can contain dissolved solutes. Since 100g of the wet material was mixed with 500 mL of water for this work this equates to a 1/12 dilution of the volume of contained water (46 mL in a total of 546 mL). The measured concentrations of dissolved solutes were multiplied by 12 and are compared with the composition of the overlying water in Table 2.7. The excellent agreement (to well within a factor of 2 in most cases) for the majority of metals of interest (Al, Co, Cu, Mn, Ni and Zn) suggests that the composition of pore water for these metals within these near surface tailings is likely to be very similar to that in the immediately overlying water, and unconstrained by phase solubility and adsorption limitations. Ca and Fe are notable exceptions.

Table 2.5 Concentrations of soluble metals as a function of pH

Sample ID	As µg/L	Al µg/L	Co µg/L	Cu µg/L	Fe µg/L	Mn µg/L	Ni µg/L	Se µg/L	U µg/L	Zn µg/L	Ca mg/L	Mg mg/L	SO₄ mg/L
1005 pH 3.73	1.5	6910	1230	2130	826	14600	1180	1.6	161	682	266	166	1330
1005 pH 7	1.9	10.2	101	3.47	<10	6430	146	4	1	7.2	223	138	1440
1005 pH 8	3.1	28.3	11.7	2.87	<10	1970	19.6	6.4	2.93	2.5	206	123	1460
1005 pH 9	7.5	23	0.97	3.94	<10	91.3	2.32	9.2	40.5	3.3	167	76.7	1460
1019 pH 3.79	1.4	1120	840	976	40	4720	805	1.2	29.1	175	40.8	89.5	464
1019 pH 7	3.1	6.9	26.3	1.78	<2	825	58.9	7.4	0.552	1.3	19.2	45.3	519
1019 pH 8	10	58.5	1.45	2.6	<2	92.1	3.84	20.6	9.8	0.9	9.9	24.4	536
1019 pH 9	25	150	0.51	3.7	<2	13.6	1.6	32.6	46.3	2.7	5.3	12.4	544
P8 Top pH 4.04	243	633	4620	16100	170	4440	6100	5	34.3	161	35.4	49.5	325
P8 Top pH7	405	2.6	542	18	<2	1400	769	10.2	2.44	3.7	26.6	37	399
P8 Top pH8	904	38.5	48	10	<2	274	74.6	9.8	14.6	17.9	19.3	27.2	407
P8 Top pH9	3150	29.2	2.03	9.72	<2	10.7	5.6	8.8	71.3	0.2	8.6	10.3	435
P8 Bottom pH 6.46	852	8	501	21.3	4	1760	946	8.4	1.74	2.2	44.1	47.1	297
P8 Bottom pH7	816	1.1	456	17.2	<2	1710	847	8.4	1.79	1.4	45	47.5	305
P8 Bottom pH8	1150	3	103	8.3	<2	696	200	12.2	58.4	0.4	48	47	355
P8 Bottom pH9	3300	9.6	5.28	9.9	<2	65.9	10.9	16.8	913	0.1	42.1	34.2	403
1047 pH 4.56	5.9	313	2180	2270	2980	3470	2900	4	100	167	31	43.9	264
1047 pH 7	85	1.9	594	11.8	<2	2000	1100	9.6	4.85	6.1	26.7	37.5	300
1047 pH 8	224	6.5	60.9	4.67	<2	552	138	17.8	14.2	1.5	22.1	30.5	321
1047 pH 9	673	25.7	6.77	4.52	<2	101	16.4	21.6	40.6	0.2	16	21	325

Table 2.6 Comparison of neutralant (0.5M NaOH) titre volumes for supernatant and slurry plus supernatant for sample 1005

	Titre mL 0.5M NaOH		
pH	Supernatant	Slurry	Ratio
3.73	0	0	
7	0.6	3.2	5.33
8	0.7	4	5.71
9	0.95	5.4	5.68

In the case of Ca this behaviour was expected based on the constraints of gypsum solubility (400-500 mg/L Ca). In the case of Fe it is likely that the solubility has been constrained by redox factors associated with anoxic bottom conditions, notably sulfate reduction and formation of Fe sulfide phases.

Table 2.7 Comparison of scaled leachable solute concentrations for sample 1005 (2m) with the composition of overlying water (bottom lens).

Solute		1005x12	Bottom Lens
Al	µg/L	82920	150000
Co	µg/L	14760	13500
Cu	µg/L	25560	32000
Fe	µg/L	9912	1000000
Mn	µg/L	175200	220000
Ni	µg/L	14160	11000
Zn	µg/L	8184	5800
Ca	mg/L	3192	475
Mg	mg/L	1992	900
SO4	mg/L	15960	7800

2.7 Tailings Characterisation-External Laboratory Data

2.7.1 Testwork Program

The tailings samples were characterised using the testwork program summarised below:

1) Standard Acid Base Accounting (ABA) suite to determine residual acidification and neutralisation potential
2) Alkaline carbonate leach to infer jarosite content
3) Water leach to determine water extractable metals to inform water treatment requirement
4) Base titration to measure neutralant demand and determine effectiveness of soluble metal removal at different target pH values
5) Quantitative X-ray diffraction on selected samples to determine mineralogy

2.7.2 Mineralogy

Four profile samples were submitted to the Central Analytical Research Facility at the University of Queensland for quantitative mineralogy using the technique of powder X-ray diffraction. The full

report has been provided in Appendix 2, with a summary of the findings and additional interpretation in a geochemical context below (Table 2.8).

As noted above, chlorite is the major (non quartz) mineral comprising the RJCS orebody. However, the data in Table 2.8 shows that illite mica, smectite and kaolinite together comprise the major component of the tailings. This assemblage is consistent with the weathering transformation sequence of chlorite (chlorite to vermiculite to kaolinite), the rate of which would have been greatly accelerated by the initially low pH deposition environment. Goethite is also often found as an associated weathering product of chlorite.

Table 2.8 Table 2.8 Normalised mineral phase abundance (%)

Phase	Sample ID			
	1005 (2m)	P3T (6m)	P7T(18m)	P8T (25m)
Quartz	29	30.6	21.7	18.9
Pyrite		0.7	1.2	2.5
Calcite			1.3	0.9
Magnesite		1.2		
Goethite	3.6	1.8		
K-Feldspar			5.4	5.3
lanthinite/Lanthinite	1	0.4	0.2	0.1
Kaolinite	6.2	8	1.6	1.9
Chlorite/clinochlore	13.4	20.3	13	10.4
Total illite/mica	23	10.8	40.1	42.9
Smectite	8.2	10.7	3.1	3.2
Non-diffracting (amorphous)	15.7	15.6	12.6	13.9

Trace levels of magnesite and calcite were indicated to be present. However, given the prevailing pH conditions it is considered unlikely that these mineral phases are present. Trace amounts of the U oxide mineral lanthinite were also indicated to be present based on the fact that the line spectrum of lanthanite gave a better residual fit than gypsum. However, it is considered that it is much more likely that gypsum will be present (rather than lanthanite) given that the U had been extracted from the ore.

Pyrite was detected in 3 of the samples. This is considered to be a real occurrence given that pyrite was present (about 1.2%) in the U ore from RJCS (Beckman 1968). This aspect will be further checked using the ABA data set. Notably no jarosite was detected. This is in marked contrast to the situation with waste rock, where the presence of this secondary mineral contributed substantially to the neutralant demand.

2.7.3 *Acid Base Accounting and Acid Extractable Metals*

All of the tailings samples were subjected to a standard battery of static ABA testwork to determine residual acid producing potential. These data are of specific relevance to the option of placing neutralised tailing material into the new WSF.

Table 2.9 summarises the ABA data set. Table 2.9 also includes carbonate extractable S (to check for presence of jarosite) and includes acid extractable S to provide a cross check on Total-S measured by the standard Leco furnace combustion method.

Table 2.9 Acid Base Accounting Suite and Sulfur Speciation (dry weight basis)

Sample	Depth	Total S-Leco	Total S-Acid [1]	MPA [2]	ANC	NAPP [3]	Total SO4-S [4]	Pyrite-S [5]	AP [6]	AP-ANC	ANC/AP	Water Sol SO4-S	Jarosite-S
	m	wt%	wt%	kg H2SO4/t	kg H2SO4 equiv./t	kg H2SO4/t	%	wt%	kg H2SO4/t	kg H2SO4/t	wt%	wt%	Wt% [7]
1002A	0.5	4.9	4.78	150	4.3	146	4.39	0.51	15.7	11.4	0.27	0.45	NA
1005A	2	0.78	0.78	23.9	13.7	10.2	0.48	0.3	9.21	-4.49	1.49	0.42	0.06
1005A-R	2	0.82	0.83	25.2	18.4	6.7	0.54	0.28	8.60	-9.80	2.14	0.44	0.10
1007A	3	0.86	0.94	26.4	15	11.3	0.41	0.45	13.8	-1.19	1.09	0.34	0.07
P3-TA	6	0.9	1	27.6	19	8.5	0.4	0.5	15.4	-3.65	1.24	0.38	0.02
P3-BA	6.5	0.86	0.97	26.4	22.9	3.4	0.38	0.48	14.7	-8.16	1.55	0.34	0.04
1019A	9	0.64	0.67	19.6	16.6	3	0.23	0.41	12.6	-4.01	1.32	0.14	0.09
1025A	12	0.76	0.74	23.3	17.9	5.4	0.17	0.59	18.1	0.21	0.99	0.13	0.04
1025A-R	12	0.73	0.74	22.4	18.2	4.1	0.14	0.59	18.1	-0.09	1.00	0.13	0.01
1029A	14	0.89	0.96	27.3	16.2	11	0.15	0.74	22.7	6.52	0.71	0.12	0.03
P7-TA	18	1	1.12	30.7	22.5	8.1	0.21	0.79	24.3	1.75	0.93	0.12	0.09
P7-BA	18.5	1.1	1.24	33.8	23.4	10.3	0.2	0.9	27.6	4.23	0.85	0.17	0.03
1037A	21	1.69	1.87	51.9	16	35.7	0.29	1.4	43.0	27.0	0.37	0.12	0.17
P8-TA	25	1.82	2.01	55.9	24.3	31.4	0.16	1.66	51.0	26.7	0.48	0.12	0.04
P8-BA	25.5	1.58	1.73	48.5	24.2	24.1	0.18	1.4	43.0	18.8	0.56	0.12	0.06
P10-TA	27.5	1.42	1.6	43.6	23.4	20	0.1	1.32	40.5	17.1	0.58	0.08	0.02
P10-BA	28	1.57	1.75	48.2	23.4	24.6	0.12	1.45	44.5	21.1	0.53	0.07	0.05
1047A	27.5	1.44	1.58	44.2	9.4	34.7	0.1	1.34	41.1	31.7	0.23	0.09	0.01
1047A-R	27.5	1.43	1.53	43.9	17.9	25.8	0.13	1.3	39.9	22.0	0.45	0.09	0.04

[1] strong oxidising acid extractable S; [2] MPA = Total S% x 30.6; [3] NAPP = MPA-ANC; [4] Total SO4 obtained by alkaline carbonate extraction; [5] Pyrite S = Total-S minus SO4-S;
[6] Pyrite-S = Total S minus SO4-S; [7] Jarosite-S = SO4-S minus water extractable S

In contrast to the titration work that was done on the still wet samples to preserve geochemical integrity of the leachable solutes and existing acidity, the ABA suite was run on a second subset of samples that had been dried at 60°C. This pre-drying step was necessary as the samples had to be ground to <85 μm prior to subsampling for analysis.

AMD is produced if the amount of acidity generated by sulfidic waste rock exceeds the acid-neutralizing capacity of the sample. To predict this occurrence, the maximum amount of acidity that could be produced from a sample as a result of pyrite oxidation is compared to maximum amount of acid-consuming species that are available. The tests that are used to estimate these quantities are referred to as 'static tests' because they cannot predict the rate or timing of AMD generation. The comparison of static tests of a sample's acid-generating potential and acid-neutralizing capacity is the basis for Acid Base Accounting (ABA).

MPA is the 'Maximum Potential Acidity' of a sample. It is calculated by multiplying S_{total} (in wt. % S) by a conversion factor of 30.6 (see Section 2.1.1), assuming that sulfur in the sample is present only in the form of pyrite (i.e. $S_{total} = S_{sulfide}$). MPA is a conservative estimate of potential acidity content because it does not account for:

- Non-acid-generating forms of sulfur, including sulfate sulfur (S_{SO4}) and native sulfur (S^o) in a sample.
- Metal sulfides that yield less acidity than pyrite during oxidation, such as sphalerite (ZnS) and galena (PbS).
- Secondary acid sulfate minerals, such as jarosite and alunite, which are secondary sources of acidity in a sample.

The Acid Generating Potential (AP) of a sample is less conservative than MPA. AP is derived by subtracting non-sulfide sulfur forms, such as jarosite sulfur (S_{jar}) and water-soluble sulfur (S_{SO4}), from S_{total} in order to better approximate the $S_{sulfide}$ content of a sample. $S_{sulfide}$ is then multiplied by 30.6 to calculate AP.

In addition to NAPP, the ratio ANC/AP (or 'Neutralization Potential Ratio', NPR) is often calculated as a supporting ABA parameter. This ratio is another way of comparing the acid-generating and acid-consuming characteristics of a sample. Samples with an NPR greater than two are unlikely to generate AMD.

In summary, the data in Table 2.9 indicate that the tailings have a moderate potential (corresponding to the PAF-II classification for waste rock) to produce AMD if the contained sulfide –S was to be completely oxidised. This will not be an issue if the tailings are left in situ. However, if the tailings were to be removed from the pit they would need to be appropriately encapsulated in the new WSF to prevent future oxidation. In contrast to the waste rock there is minimal content of jarosite so this source of slowly reacting contained acidity does not need be specifically addressed by the amount of neutralant required. That is, the neutralant demand determined on the basis of titratable acidity is sufficient for this material.

The concentrations of strong acid extractable metals were also determined to complete the characterisation data for the tailings to completely inform any future management strategies for this material (Table 2.10).

Table 2.10 Acid Extractable Metals (dry weight basis)

SAMPLE	Al %	As mg/kg	Ca %	Cd mg/kg	Co mg/kg	Cr mg/kg	Cu mg/kg	Fe %	Mg %	Mn mg/kg	Mo mg/kg	Ni mg/kg	Pb mg/kg	S %	U mg/kg	V mg/kg	Zn mg/kg
1002A	3.09	215	4.55	<0.5	76	66	1085	8.35	2.31	406	5	113	552	4.78	180	263	72
1005A	4.7	20	0.34	<0.5	66	65	202	7.61	4.45	318	2	91	251	0.78	100	275	104
1005A-REP	4.86	22	0.39	<0.5	68	66	219	7.74	4.52	328	2	93	258	0.83	100	281	107
1007A	5.06	14	0.29	<0.5	74	63	116	7.54	5.07	308	2	86	325	0.94	140	293	106
P3-TA	4.95	21	0.32	<0.5	82	144	270	8.05	5.29	382	4	135	282	1	150	286	102
P3-BA	4.89	26	0.32	<0.5	82	68	352	8.12	5.39	439	5	105	337	0.97	150	279	103
1019A	5.15	28	0.11	<0.5	78	66	309	7.88	5.24	274	2	98	395	0.67	120	303	102
1025A	4.95	28	0.09	<0.5	85	64	301	7.47	5.08	275	2	105	383	0.74	130	273	111
1025A-REP	4.93	27	0.09	<0.5	84	64	294	7.51	5.12	275	2	105	389	0.74	130	273	111
1029A	4.54	35	0.13	<0.5	98	59	311	7.3	4.83	251	2	114	306	0.96	120	251	113
P7-TA	3.23	663	0.35	<0.5	243	51	3070	4.97	3.09	201	17	313	332	1.12	280	153	60
P7-BA	3.31	609	0.36	<0.5	239	86	3790	5.46	3.3	209	17	311	298	1.24	280	162	64
1037A	2.38	1110	0.44	<0.5	575	36	5730	4.33	2.21	175	22	677	206	1.87	420	110	52
P8-TA	3.13	1280	0.43	<0.5	488	55	3650	5.31	2.83	203	28	612	263	2.01	230	148	54
P8-BA	3.01	1395	0.5	<0.5	400	44	3100	4.89	2.85	201	28	544	273	1.73	250	142	52
P10-TA	2.95	1040	0.41	<0.5	383	118	1355	4.55	2.85	183	23	518	229	1.6	210	137	49
P10-BA	2.89	1130	0.42	<0.5	391	44	1525	4.58	2.79	178	23	507	234	1.75	200	131	47
1047A	2.66	837	0.37	<0.5	409	40	2000	4.43	2.54	150	23	504	213	1.58	220	130	53
1047A-REP	2.51	787	0.35	<0.5	393	39	1915	4.22	2.41	143	22	493	200	1.53	220	123	52

With the exception of sample 1002 the measured range of total-S is consistent with the value of 1.2% reported by Beckman (1968) for ore from the RJCS deposit. Sample 1002 is anomalous given its high total-S value and the fact that this is all extractable by alkaline carbonate. This suggests that the near surface tailings material is rich in gypsum.

The very high acid extractable Ca value of 4.5wt% supports this assertion (Table 2.10). A surface layer rich in gypsum would be consistent with the bulk application of lime in the Main Pit during the initial stages of the water treatment process during the site rehabilitation works in the 1980s.

A plot of total-S versus depth indicates a step increase below about 18m (Figure 2.6). This change is mirrored by a major increase in acid extractable Cu and other chalcophilic elements (C, Ni, As) (Table 2.10), suggesting a change in the ore that was being fed to the processing plant. The decrease in EC shown in Figure 2.3 also occurs around this horizon. Further investigation of this aspect would require review of the mill processing records, if these still exist.
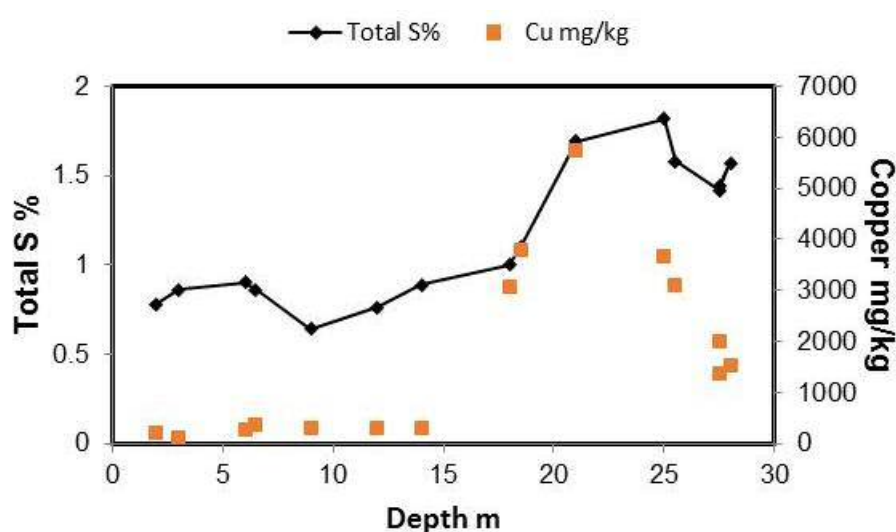
Figure 2.6 Concentrations of Total S and strong acid extractable copper as a function of depth

2.7.4   *Water Extractable Metals and Neutralant Demand*

All of the samples prepared in the Darwin laboratory were submitted to ALS for determination of water extractable metals and neutralant demand to pH 7. The target pH value of 7 specified for the work done by ALS was based on the findings from the previous work on neutralisation of waste rock, where pH 7 was found to provide the optimum performance for that component of the rehabilitation strategy. If the tailings are to be removed from the pit they will need to be slurried with water, and treated with lime prior to dewatering and placement in the WSF.

The ALS procedure involved equilibrating tailings material at a 3.75 L/S ratio based on wet weight (20g in 75mL) with shaker mixing for 5h before (a) taking the water extract sample for analysis, or (b) starting the titration on a second lot of equilibrated material. This L/S (wet weight) ratio was selected to approximate a 5:1 L/S ratio on a dry weight basis. Note that a 5:1 L/S (wet weight) basis was used for the titrations done in the EMU laboratory.

An auto-titrator was used for the ALS work. Intrinsically this will differ from a manual procedure, depending on how the endpoint detection algorithm is implemented. In particular, metals like Co, Mn and Ni will be especially problematic for direct comparison (viz the EMU and ALS data) at a target

value of pH 7 since this is right on the solubility edge for these metals where relatively small differences in the procedure used can make big differences in the amount remaining in solution.

An additional major issue arose with the ALS work since a serious mistake was made when the work was first done. The slurry sub-samples taken for analysis were digested with acid prior to being filtered for analysis, rather than just being filtered. The resultant data were therefore meaningless in the context of the test objectives. The work had to be repeated to correct this error. Unfortunately, it took two months from when the samples were originally submitted to do this. In the meantime there were steadily decreasing amounts of sample available and the original geochemical integrity likely increasingly comprised. The subsamples prepared in Darwin had been protected by storage under argon gas, but repeated opening and sub-sampling of the containers would have dissipated this initial protection.

For the reasons given above it is considered that the results produced in the EMU laboratory using fresh bulk sample are likely to be more reliable and provide a better indication of metal removal by neutralisation than the ALS data set. However, the greater numbers of samples tested by ALS does enable assessment to be made of trends in behaviour with depth.

The concentration data for metals in the 5:1 (dry weight basis) leachates produced at the starting pH values for each sample, and at the target value of pH 7 are compiled in Table 2.11. While the observed behaviour across all samples is consistent with the findings reported above from the limited selection of samples titrated in the EMU lab to pH 7, 8 and 9 (Table 2.5), it should be noted that in many cases much higher concentrations of Mn, Co and Ni were found in solution at pH 7. In accord with the data in Table 2.5 it is apparent that for Co, Mn, and Ni a target pH of 8 may be needed, depending on the how the treated water stream is to be managed on site (for example, interim storage in the Intermediate Pit prior to wet season flushing discharge).

A target pH of 8 would increase the concentration of As in the supernatant water, as shown by the data in Table 2.11. However, this is only likely to be an issue (for the removal of tailings option) for the tailings below 25m where high concentrations of water and alkaline extractable As are found. These high concentrations coincide with the peak of strong oxidising acid extractable As in Table 2.10.

A probable worst case estimate of the composition of porewater to be treated as a result of the mixing interaction of the topmost tailings with waste rock (tailings left in place option) is provided by the 5:1 water leachate from sample 1002. This assumes an approximately 5:1 L/S ratio and the removal of essentially all of the bottom lens of heavily contaminated water in advance of backfill. To this estimated solute source would need to be added solutes and acidity coming from the added waste rock. If limestone was to be added to the rock in advance of deposition, then the combined solute source terms would be greatly reduced.

The neutralant demand data for the tailings samples are compiled in Table 2.12 together with the corresponding EMU-produced data from Table 2.5 for comparison. The agreement between the two data sets is good, apart from sample 1019 (highlighted in Table 2.12) for which the EMU value is substantially higher. Based on the measured lower pH of this sample the EMU-produced value is more likely to be correct.

Table 2.11 Concentrations of metals in 5:1 (dry weight basis) leachates at ambient pH and at pH 7

Sample	Depth m	pH	Ca mg/L	Mg mg/L	SO4 mg/L	Al mg/L	As mg/L	Cd mg/L	Co mg/L	Cu mg/L	Fe mg/L	Mn mg/L	Ni mg/L	U mg/L	Zn mg/L
1002	0.5	3.4	575	235	2570	80.8	0.01	0.0066	4.32	63.1	0.57	34.6	4.04	0.86	1.88
		7.0				<0.01	0.004	0.0002	0.346	0.02	<0.05	9.17	0.329	<0.001	<0.006
1005	2	3.6	478	295	2440	35.2	0.002	0.004	2.86	11.9	2.83	21.1	2.66	1.61	1.41
		7.0				0.04	<0.001	0.0014	0.898	0.07	<0.05	13.6	1.1	0.004	0.097
1005-REP		3.4	482	292	2170	28.1	0.002	0.0038	2.6	11.2	2.17	20.8	2.4	0.943	1.36
		7.0				0.02	<0.001	0.001	0.543	0.028	<0.05	12.5	0.664	0.002	0.042
1007	3	3.4	418	298	2180	36.9	0.002	0.0044	2.94	4.54	3.63	17.2	2.48	2.26	1.02
		7.0				<0.01	<0.001	0.0005	0.251	0.003	<0.05	7.78	0.348	0.002	0.008
P3-T	5	5.2	377	393	2390	0.2	<0.001	0.0133	1.29	0.907	<0.05	14.2	1.02	0.034	0.149
		7.0				<0.01	<0.001	0.0019	0.266	0.008	<0.05	8.02	0.285	<0.001	<0.005
P3-B	6.5	5.4	396	360	2030	0.02	<0.001	0.0044	0.75	0.228	<0.05	11.7	0.584	0.004	0.043
		7.0				<0.01	<0.001	0.0005	0.101	0.004	<0.05	4.83	0.098	0.001	<0.005
1019	9	4.0	114	232	1170	19.7	0.001	0.0042	3.12	16.2	1.6	9.8	2.79	1.51	0.733
		7.0				0.4	<0.001	0.0018	1.26	0.874	<0.05	5.27	1.23	0.012	0.167
1025	12	3.8	89	198	1010	9.51	0.001	0.006	2.99	9.28	2.37	8.48	2.8	2.96	1.57
		7.0				0.27	<0.001	0.0027	1.52	0.56	<0.05	5.2	1.53	0.074	0.421
1025-REP		3.7	83	193	973	7.89	<0.001	0.0048	2.92	8.77	1.71	8.27	2.82	2.5	1.57
		7.0				0.04	<0.001	0.0014	0.992	0.107	<0.05	4.86	1.27	0.015	0.145
1029	14	3.9	104	149	833	2.6	<0.001	0.003	2.12	5.18	0.16	6.55	2.08	1.04	1.25
		7.0				<0.01	0.002	0.0002	0.11	0.003	<0.05	1.9	0.237	0.002	0.007
P7-T	18	4.3	178	156	1160	1.71	0.104	0.0024	9.38	73	0.92	9.75	12.2	0.197	0.334
		7.0				0.93	0.089	<0.0005	1.29	0.198	1.42	3.31	1.57	0.016	<0.026
P7-B	18.5	4.4	336	154	1530	4.25	0.032	0.0034	13.2	127	1.3	10.5	17.3	0.797	0.448
		7.0				0.04	0.039	0.0016	10.3	9.05	<0.05	8.79	13.9	0.005	0.202
1037	21	4.9	110	111	816	2.48	0.108	0.0017	19.1	108	0.09	8.91	20.9	0.024	0.548
		7.0				0.05	0.129	0.0007	15.6	9.52	<0.05	7.56	16.8	0.001	0.27

Table 2.11 Contd: Concentrations of metals in 5:1 (dry weight basis) leachates at ambient pH and at pH 7

Sample	Depth	pH	Ca	Mg	SO4	Al	As	Cd	Co	Cu	Fe	Mn	Ni	U	Zn
	m		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
P8-T	25	4.3	90	117	879	2.9	0.629	0.0029	29.3	93	1.12	9.67	41.4	0.066	0.562
		7.0				0.04	0.238	0.0011	19.6	5.03	<0.05	7.75	27.6	0.001	0.228
P8-B	25.5	5.2	194	128	1060	2.72	0.094	0.0017	19.3	110	1.54	8.62	24.3	0.236	0.227
		7.0				0.05	0.09	0.0009	15.2	10	<0.05	7.53	19.6	0.003	0.12
P10-T	27.5	5.4	108	101	682	0.2	1.01	0.0007	15.6	7.04	<0.05	5.04	20.6	0.005	0.082
		7.0				<0.01	1.21	0.0003	9.24	0.214	<0.05	3.63	14.1	<0.001	0.01
P10-B	28	4.7	89	89	543	0.07	4.9	0.0005	15	4.07	<0.05	4.84	20.8	0.001	0.066
		7.0				<0.01	4.04	<0.0001	2.58	0.018	<0.05	1.58	4.74	<0.001	<0.005
1047	27.5	4.2	85	116	868	5.86	0.051	0.0049	19.3	76.4	1.15	7.27	27.6	0.324	0.677
		7.0				0.14	0.114	0.0028	9.72	7.18	<0.05	4.96	14	0.007	0.308

Table 2.12 Neutralant Demand to pH 7 (kg $CaCO_3$/t <u>wet weight</u>)

Sample	Depth	$CaCO_3$ kg/t ALS	$CaCO_3$ kg/t EMU
1002	0.5	5.82	
1005	2	2.58	2.2
1007	3	2.88	
P3-T	6	0.55	
P3-B	6.5	0.74	
1019	9	1.48	2.2
1025	12	1.66	
1029	14	1.10	
P7-T	18	1.10	
P7-B	18.5	1.02	
1037	21	0.89	
P8-T	25	0.99	1.07
P8-B	25.5	0.60	
P10-T	27.5	0.24	
P10-B	28	0.59	
1047	27.5	0.71	0.5

Figure 2.7 shows a plot of the neutralant demand to pH 7 as a function of depth, together with an overlaid power fitting function. The demand decreases very steeply over the top 5m and then much more gradually thereafter. This is important information to have, especially for the option to remove and treat the tailings prior to containment in the new WSF as it will inform the amount of neutralant that will be required through time as the tailings are progressively removed.
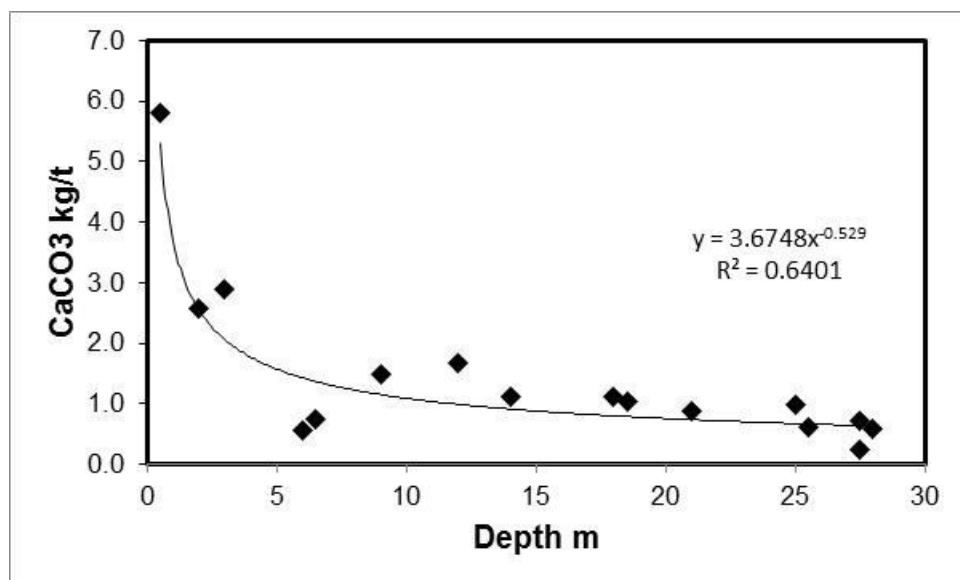
Figure 2.7 Neutralant demand (to pH 7) with depth –wet weight basis

The bulk of the tailings sampled would thus require well below 3kg $CaCO_3$/t wet weight for neutralisation to pH 7. This equates to less than 4.3kg $CaCO_3$/t dry weight, assuming 70 wt% solids. For comparison PAF-II waste rock requires average and 95%ile loadings of 6.8 and 16.3 kg $CaCO_3$/t dry

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weight, respectively to achieve a target pH of 7. The much higher 95%ile for the waste rock is a consequence of the wide distribution of acid content in the rock, reflecting the large heterogeneity of acid production rates through the mass of waste rock. In this context it is likely that the distribution of tailings properties will be much tighter than for waste rock given the consistent amount of acid added for the uranium leaching process and the consistency of mineralogy of the ore from RJCS.

The effect of pH (7, 8 and 9) on removal of metals was specifically investigated by the bulk titration work done at EMU (Table 2.5). From this work it was found that pH 8 would likely be a more effective target for metal removal in a sludge treatment system for tailings. In this context it is noted that the effect of water recycle within the treatment circuit on the build-up of major ions such as magnesium and sulfate was not addressed. **Based on the results from the 4 samples that were titrated to pH 8, an approximately 50% greater amount of neutralant would be required compared with a target pH of 7.**

2.8 Tailings Conclusions

Samples were collected in January 2018 from the top 25m of the tailings profile. No field measurements of pH and EC were made at the time since these samples were primarily intended for geotechnical testwork. If this had been done at the time, we would be in a better position to ascertain how much change there had been in these key indicator parameters during the 5 months that elapsed until geochemical testwork was undertaken on a subset of the samples. There is a general trend of increasing pH with depth, which is attributed to self-neutralisation having occurred during the 50-60 y since the initially strongly acidic (pH 1.5) tailings were deposited. There is a step change, almost halving, of the EC between 6-9m below the surface of the tailings. The reason for this is not known. The fall in EC does not correspond to an increase in pH.

The tailings have a moderate potential (corresponding to the PAF-II classification for waste rock) to produce AMD if the contained sulfide –S was to be completely oxidised. This will not be an issue if they are left in situ. However, if the tailings were to be removed from the pit they would need to be appropriately encapsulated in the new WSF to prevent future oxidation. In contrast to the waste rock there is minimal content of jarosite so this source of slowly reacting contained acidity does not need be specifically addressed by the amount of neutralant required.

A probable worst case estimate of composition of tailings porewater needing to be treated as a result of the mixing interaction of the topmost tailings with waste rock is provided by the 5:1 water leachate from sample 1002. This assumes an approximately 5:1 L/S ratio and the removal of essentially all of the bottom lens of heavily contaminated water in advance of backfill. To this estimated tailings solute source would need to be added solutes and acidity coming from the added waste rock. If limestone was to be added to the rock in advance of deposition, then the combined solute source terms would be greatly reduced.

A substantial reduction in the concentration of most leachable metals can be achieved by raising the pH to 7. However, for Co, Mn, and Ni a target pH of 8 may be needed, depending on the how the treated water stream is to be managed on site. The high concentrations of extractable As, especially at pH 8 and 9, for samples in the lower part of the profile, are of concern for the supernatant water that could be produced as a product of neutralisation of the tailings (remove tailings option). This would be a risk factor for treatment, with the occurrence of extracted As needing to be addressed by the treatment process.

The neutralant demand was found to decrease very steeply over the top 5m and then much more gradually thereafter. The demand for the top 5m will be most relevant if the tailings are left in situ, whereas the much lower demand at less than 5m would be more applicable if the tailings needed to be removed prior to backfill. For comparison, the neutralant demand for the top 5m is similar to the average neutralant demand for PAF-II waste rock (6.8 kg $CaCO_3$/t dry weight).

3 WASTE ROCK CHARACTERISATION STAGE 2

3.1 Introduction

Future oxidation of the residual sulfide contained in the PAF-I and PAF-II waste rock material (incipient acidity in Table 3.1) will be effectively stopped by submergence of this material under the recovered water table in the pit. Future oxidation of the lower sulfide-containing material in the WSF will be minimised by the use of an oxygen diffusion-limiting cover coupled with layers of compacted material between thin (2m) paddock dumped construction lifts (O'Kane Consultants 2016). Future egress of water from the base of the dump will be limited by restricting rainfall infiltration using a combination of a store and release cover and an underlying low permeability layer.

Table 3.1 Acidity Contents of PAF Waste Rock from RGC and DJEE 2016 [1]

Type	AMD Potential	Rinse pH	Jarosite Acidity kg H_2SO_4/t	Titratable Acidity kg H_2SO_4/t	Existing Acidity [2] kg H_2SO_4/t	Incipient Acidity [3] kg H_2SO_4/t
Potentially Acid Forming (PAF) Waste Rock						
PAF-I	High	4.3	16.1	3.0	23.8	136
PAF-II	Medium	4.4	14.4	1.6	15.3	27.5
PAF-III	Low	4.5	6.7	1.4	7.9	4.9

Notes: [1] All values are 80th percentiles; [2] existing acidity = titratable plus jarosite acidity; [3] Incipient acidity = acidity due to residual sulfide S, assuming all is present as pyrite.

Whilst future oxidation of sulfides will be limited by the design features described above, much of the PAF-I to PAF-III materials already contain a substantive load of pre-existing soluble acid and metals as a result of oxidation that has occurred since the material was mined. This existing load is still available to be leached out, even though future oxidation will be largely eliminated. To address this existing environmental load it is proposed to add an excess of finely crushed limestone (agricultural lime), with the dosage dependent on the PAF category.

The two key objectives of the original geochemical characterisation project were to:

1) determine the neutralant demand of waste rock required to reduce leachable metal concentrations as far as practicable, and
2) estimate the concentrations of dissolved ions (the major ions, Mg and SO_4 and metals) within neutralised backfilled material in the Main Pit and in seepage from the neutralised WSF after rehabilitation is complete.

Neutralant demand was determined using a combination of ABA characterisation data and titration methods for all samples, with verification performed on a limited number of samples using batch leach methods with addition of neutralant. The concentrations of ions remaining in pore water (pit) or seepage (WSF) were inferred using a combination of batch leach and inverse batch leach methods, in the presence and absence of neutralant, applied to a limited range of samples.

The neutralization verification work done previously was focused as a matter of priority on those materials that are at the higher end of the range of existing acidity to confirm the initial estimates of neutralant demand and to investigate how fast the jarosite component is consumed following addition of neutralant.

It was therefore recommended (RGC and DJEE 2016) that:

- further verification work be done on the lower end of the range of existing acidities since this material will comprise the bulk of the WSF;
- some additional inverse leach work should be done on the PAF-II and PAF-III material from Main WRD and PAF-III material from the Dysons WRD to confirm the original assessment of likely seepage composition; and
- that limited column testwork, with up to 3 samples of PAF-II and PAF-III material should be initiated to further assess the composition of the seepage likely to be produced from water infiltrating through the WSF.

This report addresses the first two recommendations. The third recommendation was not progressed at this time owing to resourcing limitations for the conduct of this long term type of work.

3.2   Sample Selection

The characteristics of the samples used for the Stage 2 testwork are summarised in Table 3.2. These samples were selected to build on the findings from Stage 1 and, to specifically address the recommendations summarised above. In particular this second stage of the work is to be focussed on PAF-II and PAF-III class materials from the Main and Dysons WRDs. One PAF-I sample from Intermediate Pit (TP7-P1-6) was included to provide the plus neutralant set of data to complement the minus neutralant set of data obtained for this sample in Stage 1 of the geochemical characterisation work.

Table 3.2 Identities and Characteristics of samples selected for Stage 2 testwork

PAF Category & Location	Code	pH	Acidity Jarosite g/kg	Acidity Titrable g/kg	Testwork
PAF-I Int	TP7-P1-6	4	5.4	0.8	Neutralant- INV 4step
PAF-III Main	TP1-P1-3	5.5	7.7	0.2	80th %ile neutralant -INV 4 step
PAF-III Main	TP2-P1-20	4.1	0	0.7	80th %ile neutralant -INV 4 step
PAF-III Main	TP3-P2-2	4.7	14.2	0.9	plus minus neutralant -INV 4 step
PAF-III Dysons	TP6-P1-2	3.6	0	3	plus minus neutralant - INV 4 step
PAF-III Dysons	TP6-P1-3	4.5	0	0.3	80th %ile neutralant -INV 4 step
PAF-II Main	TP2-P2-5	3.3	3.2	1.4	80th %ile neutralant -INV 4 step

The values for jarosite and titratable acidity were taken from the data reported in RGC and DJEE (2016) for the samples collected in October 2014 and characterised over the following 4 months. It was assumed that the archived samples used as the starting point for this most recent work had not reacted (i.e. oxidised) further over the intervening 3.5 y to produce a greater amount of contained existing acidity. This assumption was based on them having been stored in heavy duty ZipLock plastic

bags or in tightly sealed 24L plastic pails. However, it became apparent when some of the bucket samples were retrieved that some photodegradation of the plastic had occurred and the seal compromised. The effect of this will be documented below as it was found in a couple of cases that the titratable acidity was substantially higher than originally reported. This issue is summarised in Table 3.3 which identifies the samples where the pH when neutralant was added was well below the expected target of 6.5-7.5 based on the original derived neutralant demand values. As a result, the initial work done for TP3-P2-2 and TP6-P1-2 needed to be rerun using the (higher) current demand value obtained by titration of the material in the laboratory.

Table 3.3 Conformance of measured pH with original estimate of neutralant demand

PAF Category & Location	Code	Start pH	pH Sample + neutralant
PAF-I Int	TP7-P1-6	4	OK [1]
PAF-III Main	TP1-P1-3	5.5	OK
PAF-III Main	TP2-P1-20	4.1	OK
PAF-III Main	TP3-P2-2	2.9	4.3
PAF-III Dysons	TP6-P1-2	3.6	5.2
PAF-III Dysons	TP6-P1-3	4.5	OK
PAF-II Main	TP2-P2-5	3.3	OK

[1]OK means that pH measured was in the target range of 6.5-7.5 after neutralant was added

This finding provides a cautionary note for careful management of archived samples to ensure long term maintenance of integrity. However, it should be noted that the changes that occurred through time for two of the samples used for this most recent work does not call into question the test methods that were originally used to determine the neutralising capacity of the freshly collected material.

All of the testwork conducted for Stage 2 involved the running of inverse batch leach tests (INV) to better inform composition of leachate that could be produced from neutralised rock to be placed in the new WSF. This method involves (i) mixing water and waste rock to produce a sample of leachate and (ii) mixing that leachate with a fresh batch of solids to produce another leachate. That leachate, in turn, is then mixed with another fresh batch of solids, and so on until four solutions have been produced.

It was found from Stage 1 that the number of contact steps was practically limited to four by the progressively lower volumes of leachate produced at each step. The initial mixture produced 300 mL of leachate from notionally 700 g of solid and 700 mL of water (i.e. a 1:1 L/S ratio by mass). By Step 3 only 90 mL of leachate was available to produce a leachate corresponding to a L/S of 1:4 (i.e. 90 mL leachate, 360 mL solids). Thus, four solutions (INV 4 step in Table 3.2) were produced for analysis corresponding to L/S ratios of 1:1, 1:2, 1:3, and 1:4.

For comparison, a 5:1 L/S ratio is typically used to characterize the leachable (readily soluble) component of mine waste, whereas the L/S ratio in a WRD containing 10% moisture is about 1:10. The reason that a 5:1 ratio is used for a standard extraction test is to minimize the likelihood of solubility limitations by moderately soluble secondary minerals such as gypsum. However, it should

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be noted that a 5:1 water extraction will not dissolve up poorly soluble secondary minerals, such as jarosite or alunite.

In Stage 1 the amount of neutralant added for the inverse batch leach procedure for each sample tested was equal to the existing acidity (jarosite plus titratable acidity) specific to that sample (RGC and DJEE 2016). The reason this was done in Stage 1 was to provide additional verification of the neutralant demand figure that had been derived for that sample. This aspect requires further verification for lower category PAF material (PAF-II and PAF-III) in Stage 2.

However, material that is placed in the pit and in the WSF will have neutralant added based on the 80th percentile of neutralant demand for all samples within a given PAF category (see 80th percentile existing acidity values in Table 3.1). Thus in Stage 2 the testwork program was extended to include a number of samples neutralised with the 80th percentile loading of neutralant. This was done to specifically investigate the effect of surplus neutralant (for the majority of placed material) on the concentrations of major ion species in seepage from the WSF.

3.3 Inverse Leach Testwork

3.3.1 Procedure

As summarised above the inverse leach procedure involves 4 serial contacts of fresh crushed waste rock with a 1:1 w/w ratio of demineralised water. The samples used for this work represent the <2cm size fraction of the originally collected bulk sample, which had been further size reduced to <2mm by passage through a jaw crusher. The <2cm fraction was used for the determination of neutralant demand based on the results from a comprehensive evaluation of the effect of sample particle size on estimated neutralant demand in the Stage 1 report (Chapter 7, RGC and DJEE 2016) Use of this approach results in a neutralant demand that is conservative. This was assessed to be an appropriate regime by two major planning workshops given the potential for a heterogeneous distribution of added neutralant during construction of a WSF.

The <2cm material was then further crushed to <2mm to better ensure representative subsampling from the bulk sample for the leaching testwork, given the masses of sample that were being tested.

The notional sequence of masses of material used for each step is 700g, 350g, 180g, and 80g. These masses were arrived at based on the ability of the standard volume range of Nalgene plastic bottles (125, 250, and 500 mL) to contain the required masses of solids and water for the sequential leach steps, and also to be able to be securely clamped into the mounting brackets of the rotary inversion mixer used for the work. The initial mass of 700g of solid needed to be split 50:50 between two 500 mL bottles for Step 1 of the procedure. It was occasionally necessary to vary (lower) the standard masses and volumes of subsequent steps depending on the amount of liquid that was able to be recovered from the preceding steps.

The requisite volume of water and mass of solid (plus neutralant if required) was weighed out into the bottle which was then tightly capped and mixed by shaking prior to being placed on the rotary inversion mixer. The resulting slurry was then rotated end over end at a frequency of 10rpm for 20-24h. At the end of this time the pH and EC values of the slurry were measured. Solids/liquids separation was done by vacuum suction filtration or by centrifugation at 8000 rpm depending on the nature of the slurry. The mass of extracted liquid was recorded prior to extracting and filtering (0.45um disk filter) the minimum required volume for analysis. The requisite amount of leachate was then

carried forward to the next step, and so on. Several photos of the steps involved in the inverse leach procedure are provided in Appendix 1.

The data obtained from the neutralisation testwork will be discussed below by PAF category – from PAF-I to PAF-III.

3.3.2 PAF-I Waste

As noted above one PAF-I sample (TP7-P1-6) was included in this second stage testwork. Sample TP7-P1-6 had previously (RGC and DJEE 2016) been subjected to the inverse leach procedure with no added neutralant to determine the likely concentrations of metals that would be produced for non-neutralised material. However, given that only one PAF-I sample (TP2-P1-2 from Main WRD) had been subjected to the inverse leach with neutralant, this second stage of testwork provided the opportunity to run TP7-P1-6 with neutralant. TP7-P1-6 comes from the polymetallic Intermediate orebody, the waste from which contains much higher leachable concentrations of metals, especially Co and Ni, than from the Main orebody (Table 3.4). In common with Mn, Co and Ni are both more difficult to remove Cu and Zn. Thus TP7-P1-6 represents a much more challenging sample to assess the effectiveness of added neutralant.

Table 3.4 80th percentile concentrations of metals present in the 5:1 water extracts of samples with pH values <5 collected from test pitting in October 2014

WRD	n	Co mg/L	Cu mg/L	Mn mg/L	Ni mg/L	Zn mg/L
Intermediate	27	9.6	72	6.5	13	45
Main	26	0.7	26	2.4	1.0	0.4
Dysons	6	0.1	1.2	1.1	0.6	0.1

The pH data in the presence of neutralant show that the required target pH was achieved by adding the amount of neutralant needed to exactly account for the existing acidity present in the sample as determined by Stage 1 of the geochemical characterisation program (Table 3.5). Essentially all of the potentially leachable Cu and 90% of the leachable Zn were immobilised. However, the removal performance was not nearly as good for Mn, Co, and Ni for this sample. The issue of Mn has been flagged previously, as a lower removal efficiency for Mn than for Cu and Zn was apparent across all samples. This most recent testwork indicates that achieving substantive reductions in high leachable concentrations of Co and Ni by addition of ag-lime will be more problematic for material from the Intermediate Pit. This finding highlights and further reinforces that all of the material from the Intermediate WRD needs to placed first in the pit so that it is covered by material from the Main Pit that contains much lower leachable Mn, Co and Ni (Table 3.4).

3.3.3 PAF-II Waste

In accord with the recommendations from the Stage 1 report a sample of PAF-II material (TP2-P2-5) from the Main WRD was subjected to the inverse leach procedure with added neutralant. In this case the dosing rate used corresponded to the 80th percentile value (16 kg $CaCO_3$/t) derived for PAF-II material, rather than the measured value of 4.6 kg $CaCO_3$/t for the total existing acidity in this sample. This was done to simulate the effect of the proposed dosing regime on placed PAF-II material.

The results in Table 3.6 show that the target pH was met and that all metals of potential concern have been reduced to sub-mg/L concentrations.

3.3.4 PAF-III waste

3.3.4.1 Overview

The potential future concentration of solutes in leachate from PAF-III material is a major focus of this work since PAF-III waste will be the major component of the new WSF, with the majority coming from the Main WRD, with a lesser amount from Dysons WRD. Accordingly, 3 samples from Main WRD and 2 samples from Dysons WRD were selected (Table 3.2). It should be noted that in contrast to the Main WRD, jarosite (jarosite acidity) was essentially absent in the material from Dysons WRD. The three samples from Main WRD were selected to span a range of total acidities.

As noted in Table 3.3 initial problems were encountered with TP3-P2-2 (Main) and TP6-P1-2 (Dysons) owing to the contained acidity being substantially greater than that determined on fresh material. Thus these samples were initially incompletely neutralised, as evidenced by the pH that was produced in the leachate solutions. When this issue was detected samples of both materials were titrated in the laboratory to determine their current titratable acidity, and the neutralisation treatment was repeated with correspondingly increased amounts of neutralant. Time and personnel resources did not permit the full four step INV procedure to be rerun, but only the first step to check that neutralisation to the target pH would substantially reduce the concentrations of initially leachable metals.

All data are reported in this section because even though TP3-P2-2 and TP6-P1-2 were initially incompletely neutralised the data from this is very useful in the context of assessing the likely effect of localised incomplete neutralisation in the event that intimate mixing with neutralant is not able to be achieved in practice.

3.3.4.2 Main PAF-III

TP3-P2-2 is the highest existing acidity sample that was tested, and was the one for which the target pH was not initially achieved owing to a substantial increase in titratable acidity (from 0.9 to 6.2 g/kg $CaCO_3$ equivalent) that had occurred since the fresh material had been characterised in late 2014. This sample was subjected to the INV-4 leach procedure both with (+N) and without (-N) neutralant (Table 3.7). As noted above this neutralisation was initially incomplete so a repeat single step test was run in the presence of sufficient neutralant to confirm the efficacy of the target pH.

Incomplete neutralisation of TP3-P2-2 resulted in an increase of pH from about 3 to only 4.4. Nevertheless, across the board substantive reductions in the concentrations of metals of most concern occurred. The most probable reason for this is the absorption/co-precipitation of Co, Cu, Ni and Zn) with the precipitating Fe and Al hydroxide phases. Ferric hydroxide precipitates between pH 3 and 3.5, whilst aluminium hydroxide precipitates between pH 4 and 4.5.

When the correct amount of neutralant was added to achieve a pH of 6.8 the concentrations of all metals, with the exception of Mn, were reduced to well below 1 mg/L.

The other important point to note from the major ion data contained in Table 3.7 is that the concentration of Mg in solution decreases substantially compared to the no-added neutralant condition. This behaviour was also found for TP2-P1-2 (RGC and DRJEE 2016) and the importance of this effect for reducing the concentration of Mg well below the solubility of $MgSO_4$ in leachate from the new WSF discussed.

The remaining two samples from the Main WRD were TP1-P1-3 and TP2-P1-20. Both of these were subjected to the INV-4 procedure with neutralant added to the 80th percentile level that had between previously determined (RGC and DJEE 2016) for PAF-III material. The target pH was achieved in both cases with the concentrations of metals apart from Mn all being reduced to well below 1mg/L levels (Table 3.8).

3.3.4.3   Dysons PAF-III

Two samples (TP6-P1-2 and TP6-P1-3) were used for this work. Although located in close proximity in the vertical profile the total acidity (all as titratable acidity) in TP6-P1-2 was originally determined to be ten-fold higher than in TP6-P1-2. These samples were initially subjected to different treatments, with TP6-P1-2 being dosed with sufficient neutralant to just account for the contained acidity whereas TP6-P1-3 had the 80th percentile (for PAF-III material) amount of neutralant added (Table 3.9).

It became apparent by the measured pH that insufficient neutralant had been added to TP6-P1-2, with the measured pH values being 5-6. Notwithstanding this the concentrations of most metals, excepting Mn and Zn, were less than 1mg/L. When the 80th percentile (for PAF-III material) amount of neutralant was added the target pH was achieved with the concentrations of all metals well below 1 mg/L. Subsequent titration of TP6-P1-2 with sodium hydroxide solution showed that it contained 3.75 kg/t $CaCO_3$ titratable acidity which was substantially higher than the originally determined value of 3 kg/t $CaCO_3$ titratable acidity.

The target pH for TP6-P1-3 was met with N80 neutralant addition and the concentrations of all metals were well below 1mg/L.

3.4   Conclusions for Waste Rock

The most recent program of testwork has confirmed the methods used for estimating neutralant demand of the waste rock samples. However, for two of the samples tested the neutralant demand was found to be substantially higher than when fresh material was tested in late 2014. This is suspected to be the result of oxidation that has occurred during storage over the 3.5 years since the samples were collected and is a reflection of the storage conditions for at least some of the library of samples. When the samples were retrieved for the current work it was noted that some of the containers were starting to deteriorate and identifying marker labels fading as a result of UV exposure. Accordingly, it is recommended that an inspection be undertaken as a matter of urgency of the sample inventory. Material in physically suspect containers should be repackaged and containers with fading labels should be re-labelled.

For all of the samples bar one achieving the target pH range of 6.5-7.5 resulted in the removal of all metals apart from Mn to below, and in most cases well below, 1mg/L. The one exception was the high PAF-I sample from the Intermediate WRD. This most recent testwork indicates that achieving substantive reductions in high leachable concentrations of Co and Ni by addition of agricultural lime ($CaCO_3$) will be more problematic for material from the Intermediate Pit. This finding highlights and further reinforces that all of the material from the Intermediate WRD needs to placed first in the pit so that it is covered by material from the Main WRD the material from which generally contains lower leachable concentrations of Mn, Co and Ni.

Table 3.5 Concentration data for TP7-P1-6 in the presence (+N) and absence (-N) of neutralant

	Step	pH	Al mg/L	Co mg/L	Cu mg/L	Fe mg/L	Mn mg/L	Ni mg/L	Se mg/L	U mg/L	Zn mg/L	Ca mg/L	Mg mg/L	SO$_4$ mg/L
TP7-P1-6-N	1	4.02	9.86	31.7	38	19.8	3.45	34			103	82	264	1630
	2	4.15	11.4	53.6	68	16.5	6.15	58.7			150	159	364	2420
	3	4.52	16.5	78.7	116	5.6	8.73	84.6			220	226	525	3480
	4	4.38	23.5	102	141	11.1	11.1	109			284	283	657	4400
TP7-P1-6+N	1	6.85	0.00	15.5	0.40	<10	3.12	10.1	0.09	0.02	7.74	616	388	2630
	2	6.29	0.01	29	0.52	<10	5.77	20.7	0.17	0.03	15.6	569	716	3600
	3	6.66	0.02	25.2	0.68	<20	6.2	17.9	0.13	0.06	15.1	537	1060	4780
	4	6.63	0.12	53	0.97	0.038	8.2	42.7	0.19	0.03	37	547	1310	5850

Table 3.6 Concentration data for TP2-P2-5 in the presence of 80th percentile neutralant (+N80) for PAF-II material

Step	pH	Al mg/L	Co mg/L	Cu mg/L	Fe mg/L	Mn mg/L	Ni mg/L	Se mg/L	U mg/L	Zn mg/L	Ca mg/L	Mg mg/L	SO$_4$ mg/L
1	6.25	0.01	0.26	0.04	<10	0.67	0.37	0.04	0.04	0.21	830	167	1990
2	7.23	0.02	0.27	0.03	<10	0.52	0.47	0.07	0.04	0.00	666	254	1970
3	6.93	0.06	0.38	0.08	<10	1	0.58	0.09	0.06	0.62	690	307	2070
4	7.34	0.03	0.44	0.12	<10	0.98	0.70	0.11	0.05	0.79	687	356	2230

Table 3.7 Concentration data for TP3-P2-2 in the presence (+N) and absence (-N) of neutralant. +N($\downarrow$) indicates insufficient neutralant added

	Step	pH	Al mg/L	Co mg/L	Cu mg/L	Fe mg/L	Mn mg/L	Ni mg/L	Se mg/L	U mg/L	Zn mg/L	Ca mg/L	Mg mg/L	SO$_4$ mg/L
TP3-P2-2-N	1	2.93	169	2.78	4.47	22	8.02	3.38	0.02	7.79	0.49	492	497	4020
	2	2.91	332	5.45	8.66	41	16.2	6.55	0.04	13.8	0.95	470	943	6540
	3	2.91	445	7.81	12.2	64	21.4	9.38	0.06	22.9	1.96	471	1360	9040
	4	2.92	659	11.7	16.5	79	29.8	13.7	0.08	27.4	3.57	468	1770	11400
TP3-P2-2+N($\downarrow$)	1	4.33	12.9	1.71	0.54	0.2	3.88	2.28	0.03	0.36	0.24	530	351	2630
	2	4.36	23.8	2.94	0.90	0.4	6.93	3.8	0.04	0.60	0.32	484	606	3590
	3	4.4	29.9	3.85	1.16	0.6	9.02	5	0.05	0.85	0.80	475	801	4330
	4	4.28	9.96	4.2	0.61	0.2	10.2	5.6	0.07	0.48	1.02	463	896	4560
TP3-P2-2+N	1a	6.8	0.01	0.09	0.02	<2	1.57	0.16	0.04	0.18	0.02	650	274	2530
	1b		0.05	0.04	0.01	<2	0.93	0.06	0.04	0.98	0.00	695	286	2400

Table 3.8 Concentration data for TP1-P1-3 and TP2-P1-20 (Main WRD) in the presence of 80th percentile neutralant (+N80) for PAF-III material

	Step	pH	Al mg/L	Co mg/L	Cu mg/L	Fe mg/L	Mn mg/L	Ni mg/L	Se mg/L	U mg/L	Zn mg/L	Ca mg/L	Mg mg/L	SO$_4$ mg/L
TP1-P1-3+N80	1	7.36	0.00	0.37	0.39	<10	2.59	1.9	0.02	1.56	0.04	605	471	2920
	2	7	0.03	0.56	0.64	<20	3.58	2.67	0.04	2.16	0.03	556	768	3820
	3	7.73	0.04	2.52	0.89	<20	4.89	5.18	0.16	2.45	0.18	560	955	4500
	4	7	0.03	0.80	0.60	<20	4.13	4.08	0.16	2.09	0.04	543	1140	5140
TP2-P1-20+N80	1	7.11	0.03	0.02	0.01	0	0.20	0.01	0.00	0.05	0.01	243	132	826
	2	6.69	0.01	0.05	0.02	<10	0.75	0.03	0.01	0.11	0.00	397	236	1500
	3	7.63	0.10	0.06	0.03	<10	1.03	0.05	0.01	0.16	0.24	528	336	2180
	4	7.76	0.05	0.25	0.05	0	1.47	0.15	0.01	0.17	0.57	623	473	3000

Table 3.9 Concentration data for TP6-P1-2 and TP6-P1-3 (Dysons WRD) in the presence of neutralant. +N($\downarrow$) indicates insufficient neutralant added

	Step	pH	Al mg/L	Co mg/L	Cu mg/L	Fe mg/L	Mn mg/L	Ni mg/L	Se mg/L	U mg/L	Zn mg/L	Ca mg/L	Mg mg/L	SO4 mg/L
TP6-P1-3+N80	1	7.15	0.00	0.00	0.01	<2	0.114	0.01	0.02	0.08	0.01	108	200	624
	2	6.93	0.00	0.00	0.00	0	0.237	0.02	0.05	0.17	0.00	155	369	1220
	3	6.17	0.01	0.01	0.01	<10	0.361	0.03	0.09	0.21	0.01	181	486	1670
	4	7.25	0.06	0.01	0.01	<10	0.406	0.03	0.08	0.27	0.00	213	617	2200
TP6-P1-2+N($\downarrow$)	1	5.24	0.67	0.15	0.04	<10	4.69	1.05	0.03	0.03	0.04	593	491	3230
	2	4.92	1.43	0.16	0.05	<10	8.43	1.1	0.06	0.05	0.03	504	876	4430
	3	5.1	2.61	0.35	0.09	<20	11.4	1.1	0.07	0.07	0.06	495	1230	5660
	4	6.4	0.22	0.37	0.05	<10	14.3	2.16	0.10	0.04	0.04	464	1640	7000
TP6-P1-2+N80	1	7	0.01	0.01	0.01	<10	1.5	0.02	0.03	0.33	0.01	691	467	2900

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APPENDIX 1

SAMPLE PROCESSING

Tailings Sub-Sampling Procedure

Sub-sampling bulk bagged samples

Glove bag for piston cores

Piston core in glove box

Sub-sampling opened piston core

Inverse Leach Procedure for Waste Rock
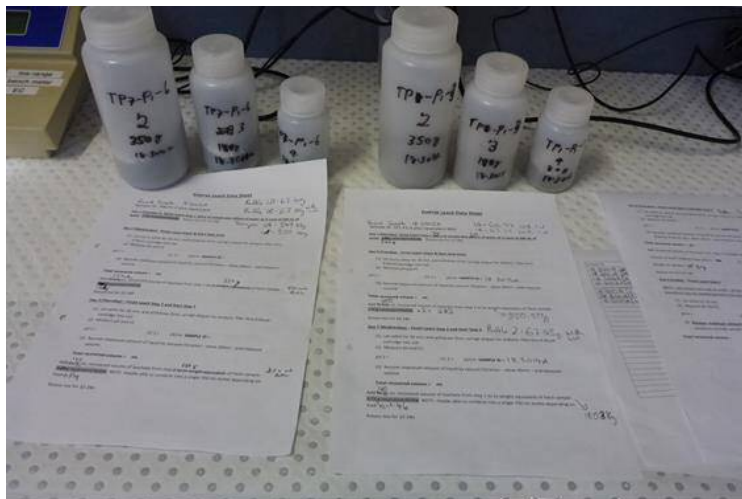
Weighed out samples

Rotary inversion mixers

Vacuum filtration equipment

Filtering sample

APPENDIX 2

X-Ray Diffraction Mineralogy Report for Main Pit Tailings
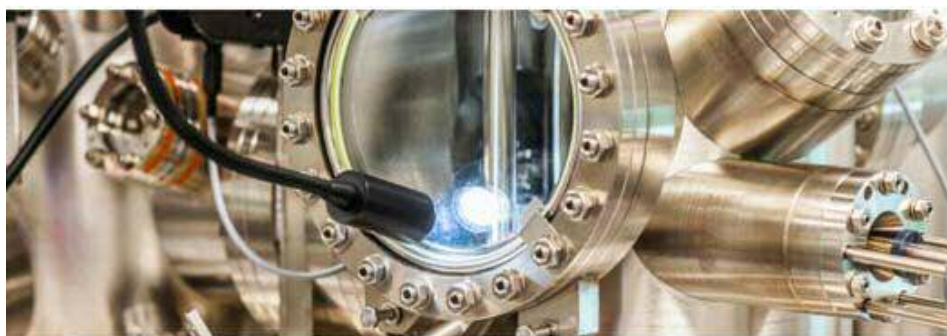
QUT Central Analytical Research Facility
Materials Characterisation Report

CLIENT	ALS Environmental
REPORT DATE	31st July 2018
PREPARED BY	Henry Spratt and David Steele
ANALYSIS REQUESTED	Quantitative XRD
OUR REFERENCE	X18237
YOUR REFERENCE	EB1816475
QUT CONTACTS	Dr David Steele, X-ray Analysis Coordinator Ph: 07 3138 2583 | email: david.steele@qut.edu.au Dr Henry Spratt, Senior Laboratory Technician Ph: 07 3138 9526 | email: henry.spratt@qut.edu.au **X-ray and Particles Laboratory enquiries: xandp@qut.edu.au**

RESULTS

Phase Identification / Quantification

The powder X-ray diffraction patterns show the presence of crystalline phases. Graphics of the collected diffraction patterns along with the phases identified are included at the end of this report. Below are tables of phase abundances. The estimated normalised abundance of the corundum internal standard in the samples is higher than 10 wt%. This means there is an unaccounted for component in the samples (i.e., the samples contain non-diffracting/unidentified material). A peak at 13.48 °2θ which could be confused with gypsum was modelled as Ianthinite / Lanthinite ($UO_2 \cdot 5UO_3 \cdot 10H_2O$), a uranium bearing mineral. A significantly better fit was achieved with Ianthinite when compared to gypsum (or the phosphorus analogue, brushite). Clay analysis is recommended for these samples due to the significant abundance of clay phases in these samples. Smectite and kaolinite were modelled in all samples, but are present at higher abundances in samples 1005A and PS-TA than P7-TA and P8-TA. All samples contain at least 1 chlorite/clinochlore phase.

Table of phase abundances (nominal wt%, absolute)

X18237	1	2	3	4
EB1816475	1005A	PS-TA	P7-TA	P8-TA
Quartz	29.0	30.6	21.7	18.9
Pyrite		0.7	1.2	2.5
Calcite			1.3	0.9
Magnesite		1.2		
Goethite	3.6	1.8		
K-Feldspar			5.4	5.3
Ianthinite/Lanthinite	1.0	0.4	0.2	0.1
Kaolinite	6.2	8.0	1.6	1.9
Chlorite/clinochlore	13.4	20.3	13.0	10.4
Total illite/mica	23.0	10.8	40.1	42.9
Smectite	8.2	10.7	3.1	3.2
Non-diffracting (amorphous)	15.7	15.6	12.6	13.9

APPENDIX 1 – X-RAY DIFFRACTION DATA AND GRAPHICS

Powder X-ray Diffraction Patterns

In the graphics below the coloured line is the collected data, the red line is the Rietveld refinement model, and the grey line is the difference. Note that the phase abundances on the graphics are before taking into account the known addition of corundum at 10 wt%, i.e they are the Rietveld wt% as fitted. Please use the tabulated abundances which require no further manipulation and are wt% in original sample.

X18237-1_1005A.xy

Phase	wt%
Corundum (QUT)	11.65 %
Quartz	30.37 %
Goethite	3.75 %
Ianthinite	1.05 %
Kaolinite	6.50 %
Chlorite IIb	14.02 %
Illite 2M1	8.96 %
Muscovite (2M1)	1.94 %
Phengite (1M)	13.16 %
Ca Montmorillonite	8.60 %

X18237-2_PS-TA.xy

Phase	wt%
Corundum (QUT)	11.63 %
Quartz	32.00 %
Pyrite	0.77 %
Magnesite	1.22 %
Goethite	1.90 %
Ianthinite	0.40 %
Kaolinite	8.31 %
Chlorite IIb	17.44 %
Clinochlore	3.75 %
Phengite (1M)	11.40 %
Ca Montmorillonite	11.17 %

X18237-3_P7-TA.xy

Corundum (QUT)	11.27 %
Quartz	21.99 %
Pyrite	1.18 %
Calcite	1.28 %
Microcline (maximum)	5.42 %
Ianthinite	0.20 %
Kaolinite	1.66 %
Chlorite IIb	9.33 %
Clinochlore	3.85 %
Illite 2M1	18.56 %
Muscovite (2M1)	8.90 %
Phengite (1M)	13.18 %
Ca Montmorillonite	3.17 %

X18237-4_P8-TA.xy

Corundum (QUT)	11.48 %
Quartz	19.55 %
Pyrite	2.53 %
Calcite	0.96 %
Microcline (maximum)	5.29 %
Ianthinite	0.10 %
Kaolinite	1.71 %
Chlorite IIb	7.94 %
Clinochlore	2.67 %
Illite 2M1	20.56 %
Muscovite (2M1)	8.47 %
Phengite (1M)	15.45 %
Ca Montmorillonite	3.30 %

APPENDIX 2 – ANALYTICAL TECHNIQUES

Sample preparation

Sub-samples were accurately weighed and specimens prepared for X-ray diffraction analysis by the addition of a corundum (Al_2O_3) internal standard at 10 wt%. The specimens were micronised in a McCrone mill using zirconia beads and ethanol, then dried in an oven overnight at 40 °C. The resultant homogenous powders were back-pressed into sample holders.

Sample analysis

Step scanned X-ray diffraction patterns were collected for an hour per sample using a PANalytical X'Pert Pro powder diffractometer and cobalt Kα radiation operating in Bragg-Brentano geometry. The collected data was analysed using JADE (V2010, Materials Data Inc.) and X'Pert Highscore Plus (V4, PANalytical) with various reference databases (PDF4+, AMCSD, COD) for phase identification. Rietveld refinement was performed using TOPAS (V5, Bruker) for quantitative phase analysis. The known addition of corundum facilitates reporting of absolute phase abundances for the modelled phases. The sum of the absolute abundances is subtracted from 100 wt% to obtain a residual (called non-diffracting/unidentified, also known as "amorphous"). The residual represents the unexplained portion of the pattern: it may be non-diffracting content but will also contain unidentified phases and the error from poorly modelled phases. It is the least accurate measure as its error is the sum of the errors of the modelled phases. An absorption contrast correction (Brindley) was made on the basis that the average size of the particles in the specimens is approximately 5 μm. The more absorbing phases will be under estimated if their actual average particle size is greater than 5 μm. The estimated uncertainties in the reported phase abundances are 20 wt% relative or better for every modelled phase. Due to propagation of errors the uncertainty in the amorphous (non-diffracting/unidentified) content is higher at approximately 30 wt% relative. The detection limit and limit of quantification using our method is approximately 1 wt% or less depending on the phase in question and sample matrix.

Powder X-ray diffraction is bulk phase analysis, it is not bulk chemical analysis. Phase abundances may be mis-estimated if an incorrect chemical formula is assigned to a phase. Therefore, the closest matches in the reference phase identification databases were used in the Rietveld refinement model, but other members of the identified mineral groups may be present.

APPENDIX 3

Set up Data for Waste Rock Inverse Leach Procedure

Set up Data for Waste Rock Inverse Leach Procedure

TP7-P1-6 + N

LEACH 1		LEACH 2		LEACH 3		LEACH 4	
Mass g (x2)	300	Mass (g)	350	Mass (g)	180	Mass (g)	80
water (mL) (x2)	300	LEACH 1 leachate (mL)	350	LEACH 2 leachate (mL)	180	LEACH 3 leachate (mL)	80
CaCO3 (g)Total	3.799	Mass of CaCO3 (g)	2.201	Mass of CaCO3 (g)	1.141	CaCO3 (g)	0.510
pH	6.85	pH	6.29	pH	6.66	pH	6.33
EC (μS/cm)	3890	EC (μS/cm)	5020	EC (μS/cm)	6470	EC (μS/cm)	5800

TP1 -P1-3 + N80

LEACH 1		LEACH 2		LEACH 3		LEACH 4	
Mass g (x2)	300	Mass (g)	350	Mass (g)	180	Mass (g)	80
water (mL) (x2)	300	LEACH 1 leachate (mL)	350	LEACH 2 leachate (mL)	180	LEACH 3 leachate (mL)	80
CaCO3 (g)t	5.682	CaCO3 (g)	2.820	CaCO3 (g)	1.461	CaCO3 (g)	0.730
pH	7.36	pH	6.51	pH	7.73	pH	6.66
EC (μS/cm)	4100	EC (μS/cm)	3880	EC (μS/cm)	5950	EC (μS/cm)	4970

TP6 -P1-2 -N

LEACH 1		LEACH 2		LEACH 3		LEACH 4	
Mass g (x2)	350	Mass (g)	350	Mass (g)	180	Mass of (g)	80
water (mL) (x2)	350	LEACH 1 leachate (mL)	350	LEACH 2 leachate (mL)	180	LEACH 3 leachate (mL)	80
CaCO3 (g) total	0	CaCO3 (g)	0	CaCO3 (g)	0	CaCO3 (g)	0
pH	3.48	pH	3.42	pH	3.31	pH	3.2
EC (μS/cm)	4230	EC (μS/cm)	4170	EC (μS/cm)	9100	EC (μS/cm)	11900

TP6 -P1-2 +N

LEACH 1		LEACH 2		LEACH 3		LEACH 4	
Mass g (x2)	300	Mass (g)	350	Mass (g)	180	Mass (g)	79
water (mL) (x2)	300	LEACH 1 leachate (mL)	350	LEACH 2 leachate (mL)	180	LEACH 3 leachate (mL)	80
CaCO3 (g) Total	2.201	Mass of CaCO3 (g)	1.086	Mass of CaCO3 (g)	0.562	Mass of CaCO3 (g)	0.251
pH	5.24	pH	4.92	pH	5.1	pH	6.37
EC (μS/cm)	4100	EC (μS/cm)	3660	EC (μS/cm)	6940	EC (μS/cm)	4170

Set up Data for Waste Rock Inverse Leach Procedure (Contd)

Rum Jungle Geochemical Characterisation   DR Jones Environmental Excellence

TP6 -P1-3 +N80

LEACH 1		LEACH 2		LEACH 3		LEACH 4	
Mass g (x2)	350	Mass (g)	350	Mass (g)	180	Mass (g)	80
water (mL) (x2)	350	LEACH 1 leachate (mL)	350	LEACH 2 leachate (mL)	180	LEACH 3 leachate (mL)	80
CaCO3 (g) Total	2.841	CaCO3 (g)	2.8395	CaCO3 (g)	1.462	Mass of CaCO3 (g)	0.651
pH	7.15	pH	6.93	pH	6.17	pH	7.25
EC (µS/cm)	1570	EC (µS/cm)	1879	EC (µS/cm)	3320	EC (µS/cm)	4210

TP2-P1-20 +N80

LEACH 1		LEACH 2		LEACH 3		LEACH 4	
Mass g (x2)	350	Mass (g)	340	Mass (g)	180	Mass (g)	80
water (mL) (x2)	350	LEACH 1 leachate (mL)	340	LEACH 2 leachate (mL)	180	LEACH 3 leachate (mL)	80
CaCO3 (g) Total	5.682	CaCO3 (g)	2.759	CaCO3 (g)	1.461	Mass of CaCO3 (g)	0.730
pH	7.11	pH	6.69	pH	7.63	pH	7.76
EC (µS/cm)	1786	EC (µS/cm)	2880	EC (µS/cm)	3420	EC (µS/cm)	433

TP3-P2-2 -N

LEACH 1		LEACH 2		LEACH 3		LEACH 4	
Mass g (x2)	350	Mass (g)	350	Mass (g)	180	Mass (g)	80
water (mL) (x2)	350	LEACH 1 leachate (mL)	350	LEACH 2 leachate (mL)	180	LEACH 3 leachate (mL)	80
CaCO3 (g) Total	0	CaCO3 (g)	0	CaCO3 (g)	0	Mass of CaCO3 (g)	0
pH	2.93	pH	2.91	pH	2.91	pH	2.92
EC (µS/cm)	5160	EC (µS/cm)	9010	EC (µS/cm)	91010	EC (µS/cm)	11010

TP3-P2-2 +N

LEACH 1		LEACH 2		LEACH 3		LEACH 4	
Mass g (x2)	350	Mass (g)	350	Mass (g)	180	Mass (g)	80
water (mL) (x2)	350	LEACH 1 leachate (mL)	350	LEACH 2 leachate (mL)	180	LEACH 3 leachate (mL)	80
CaCO3 (g)/Total	2.140	CaCO3 (g)	1.081	CaCO3 (g)	0.560	CaCO3 (g)	0.281
pH	4.33	pH	4.36	pH	4.4	pH	4.28
EC (µS/cm)	3760	EC (µS/cm)	4370	EC (µS/cm)	5300	EC (µS/cm)	5480

Set up Data for Waste Rock Inverse Leach Procedure (Contd)

TP2-P2-5 +N80

LEACH 1		LEACH 2		LEACH 3	
Mass g (x2)	350	Mass (g)	350	Mass (g)	
water (mL) (x2)	350	LEACH 1 leachate (mL)	350	LEACH 2 leachate (mL)	
CaCO3 (g) Total	10.922	Mass of CaCO3 (g)	5.461	Mass of CaCO3 (g)	2.8
pH	6.25	pH	7.23	pH	
EC (µS/cm)	3760	EC (µS/cm)	3650	EC (µS/cm)	