# APPENDIX 21

## Geochemical Assessment

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For:

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**ON BEHALF OF MANGOOLA COAL OPERATIONS PTY LIMITED**

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# Geochemical Assessment of the Mangoola Coal Continued Operations Project

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# List of Abbreviations

## Abbreviations Used in Geochemical Assessment

| | |
|---|---|
| ARD | Acid Rock Drainage |
| AMD | Acid, Metalliferous and Saline Drainage |
| NMD | Neutral and Metalliferous Drainage |
| ABA | Acid Base Account |
| $pH_{1:2}$ | pH of a sample slurry with a solid to water ratio of 1:2 (by weight) |
| $EC_{1:2}$ | Electrical Conductivity of a sample slurry with a solid to water ratio of 1:2 (by weight) |
| ESP | Exchangeable Sodium Percentage |
| ECEC | Effective Cation Exchange Capacity |
| S | Sulphur |
| CRS | Chromium Reducible Sulphur |
| KCl | Potassium Chloride |
| HCl | Hydrochloric Acid |
| $H_2SO_4$ | Sulphuric Acid |
| $SO_4$ | Sulphate |
| $CaCO_3$ | Calcium Carbonate |
| ANC | Acid Neutralising Capacity in kg $H_2SO_4$/t |
| $ANC_{ABCC}$ | Acid Neutralising Capacity in kg H2SO4/t estimated from ABCC testing |
| CNV | Carbonate Neutralising Value in kg H2SO4/t |
| MPA | Maximum Potential Acidity, calculated from total S in kg $H_2SO_4$/t |
| NAPP | Net Acid Producing Potential, calculated from ANC and total S (or MPA) in kg $H_2SO_4$/t. |
| NAG | Net Acid Generation (test) |
| NAGpH | pH of NAG solution before titration |
| $NAG_{(pH4.5)}$ | NAG acidity titrated to pH 4.5 in kg $H_2SO_4$/t |
| $NAG_{(pH7.0)}$ | NAG acidity titrated to pH 7.0 in kg $H_2SO_4$/t |
| ABCC | Acid Buffering Characteristic Curve |
| GAI | Geochemical Abundance Index based on multi-elements of solids |
| PAF | Potentially Acid Forming |
| PAF-LC | Potentially Acid Forming - Low Capacity |
| NAF | Non Acid Forming |
| UC | Uncertain |
| AC | Acid Consuming |

## Units of Measurement

| | |
|---|---|
| % | Percentage |
| °C | Degrees Celsius |
| dS | Deci Siemen |
| $\mu$m | Micrometre |
| mm | Millimetre |
| m | Metre |
| mg | Milligram |
| g | Gram |
| mg | Milligram |
| kg | Kilogram |
| t | Tonne |
| L | Litre |
| ml | Millilitre |

## Other Abbreviations

| | |
|---|---|
| ALS | Australian Laboratory Services |
| EGi | Environmental Geochemistry International Pty Ltd |
| ROM | Run-of-Mine |

# Executive Summary

Environmental Geochemistry International Pty Ltd (EGi) was commissioned by Umwelt (Australia) Pty Limited on behalf of Mangoola Coal Operations Pty Limited (Mangoola) to carry out a geochemical assessment of the Mangoola Coal Continued Operations (MCCO) Project, located 20km west of Muswellbrook in NSW. The MCCO Project would involve continuation of the existing open cut Mangoola Coal Mine. This report will contribute to an environmental impact statement (EIS) for the MCCO Project.

The objectives of the work were to assess the potential for acid rock drainage (ARD), salinity, metal/metalloid leaching (including neutral mine drainage, NMD) of the proposed mine materials, identify any geochemical issues, and provide recommendations for materials management and any follow up test work required. The assessment was based on visual inspection of the site and mine materials, review of a raw coal sulphur (S) database, and testing of 66 tailings samples from current operations, 5 coarse rejects samples from current operations, 9 overburden/interburden samples from current operations, and 20 representative overburden/interburden samples from the MCCO Project Additional Mining Area.

Visual inspection and testing of materials from the current operations and results of testing of MCCO Project core indicate that overburden/interburden and coarse rejects materials placed in pit back fill and ex-pit emplacement constructed as part of the MCCO Project are likely to be non acid forming (NAF) and non saline. They are also expected to be alkalinity producing, providing an additional factor of safety.

Results of testing tailings from the previously utilised Tailings Dam (TD) 1 and TD2 as a proxy for future tailings geochemical compositions also indicate that the bulk of the tailings from the MCCO Project are likely to be NAF with associated elevated chloride salinities from residual process water, and moderate alkalinities. Although there are likely to be higher S PAF tailings in localised zones close to spigot locations (due to gravity segregation effects), these would not be expected to be a significant source of acid seepage due to the background alkalinities from surrounding NAF tailings, and produce only a minor contribution of sulphate salinity relative to the inherent chloride salinities present in the process water.

The local background water quality is circum-neutral and saline with elevated alkalinity, and results to date indicate the proposed MCCO Project is not likely to have a significant impact on pit water quality, or require modification of the current saline water management.

However, the tailings may produce local effects on final cover layers from both acid and salinity, potentially compromising rehabilitation success without appropriate controls in place. The current conceptual cover for the tailings involves placement of a minimum 3m cover of clean overburden. More detailed design considerations will be required prior to capping to confirm the acid and salinity in the tailings does not migrate into the growth horizon and affect rehabilitation, with recommendations to address this issue provided in this report.

The following is recommended for the MCCO Project during operations:

- Carry out visual inspection of any further core drilling in the MCCO Project area for evidence of pyrite occurrence to confirm the strong dominance of NAF overburden/interburden across the deposit.
- Include routine monthly monitoring of pH, EC and acidity/alkalinity from tailings dam discharges and/or decant ponds, with analysis of key ARD indicators including dissolved SO4, Ca, Mg, K, Na, Cl, Al, As, Co, Cr, Cu, Fe, Mn, Ni and Zn. Results should be reviewed after 12 months and checked for any evidence of ARD development, and compared to the quality of representative receiving surface water and groundwater to identify any potential impacts and determine if further ongoing monitoring is required.
- Arrange for total S testing to be carried out during regular testing of washery waste samples to check for consistency with results to date. Preserve selected samples and review in 6 months to determine if more detailed work is required and assess the need for continuation of S testing.
- Prior capping of the tailings dams prepare a detailed plan for the cover design system. The design should provide adequate control of upward migration of salts and acid so that rehabilitation efforts are not compromised. The detailed cover design should consider:
    - Physical and hydrological characterisation of the existing tailings (TD1 and TD2) and potential cover materials, and water flux modelling of cover system options to assess performance. The proposed 3m clean overburden cover may be sufficient, but should be confirmed.
    - The need for incorporation of crushed (agricultural) limestone into the final tailings surface in the zone radiating approximately 100m from the spigot point to ensure neutralisation of any existing acidity.

# 1.0 Introduction

Environmental Geochemistry International Pty Ltd (EGi) was commissioned by Umwelt (Australia) Pty Limited (Umwelt) on behalf of Mangoola Coal Operations Pty Limited (Mangoola) to complete a geochemical assessment of the Mangoola Coal Continued Operations Project (MCCO Project). The purpose of the assessment is to form part of an Environmental Impact Statement being prepared by Umwelt to support an application for development consent under Divisions 4.1 and 4.7 of Part 4 of the Environmental Planning and Assessment Act 1979 (EP&A Act) for the MCCO Project.

## 1.1 Project Overview

Mangoola Coal Mine is an open cut coal mine located approximately 20 kilometres (km) west of Muswellbrook and 10 km north of Denman in the Upper Hunter Valley of NSW. Mangoola has operated the Mangoola Coal Mine in accordance with Project Approval (PA) 06_0014 since mining commenced at the site in September 2010.

The MCCO Project will allow for the continuation of mining at Mangoola Coal Mine into a new mining area to the immediate north of the existing operations. The MCCO Project will extend the life of the existing operation providing for ongoing employment opportunities for the Mangoola workforce. The MCCO Project Area includes the existing approved Project Area for Mangoola Coal Mine.

The MCCO Project generally comprises:

- open cut mining peaking at up to the same rate as that currently approved (13.5 Million tonnes per annum (Mtpa) of run of mine (ROM) coal) using truck and excavator mining methods
- continued operations within the existing Mangoola Coal Mine
- mining operations in a new mining area located north of the existing Mangoola Coal Mine, Wybong Road, south of Ridgelands Road and east of the 500 kV Electricity Transmission Line (ETL)
- construction of a haul road overpass over Big Flat Creek and Wybong Road to provide access from the existing mine to the proposed Additional Mining Area
- establishment of an out-of-pit overburden emplacement area
- distribution of overburden between the proposed Additional Mining Area and the existing mine in order to optimise the final landform design of the integrated operation
- realignment of a portion of Wybong Post Office Road
- the use of all existing or approved infrastructure and equipment for the Mangoola Coal Mine with some minor additions to the existing mobile equipment fleet
- construction of a water management system to manage sediment laden water runoff, divert clean water catchment, provide flood protection from Big Flat Creek and provide for reticulation of mine water. The water management system will be connected to that of the existing mine
- continued ability to discharge excess water in accordance with the Hunter River Salinity Trading Scheme (HRSTS)

- establishment of a final landform in line with current design standards at Mangoola Coal Mine including use of natural landform design principles consistent with the existing site
- rehabilitation of the proposed Additional Mining Area using the same revegetation techniques as at the existing mine
- a likely construction workforce of approximately 145 persons. No change to the existing approved operational workforce
- Continued use of the mine access for the existing operational mine and access to/from Wybong Road, Wybong Post Office Road and Ridgelands Road to the MCCO Project Area for construction, emergency services, ongoing operational environmental monitoring and property maintenance.

## 1.2 Scope of Work

The objectives of the work were to assess the acid rock drainage (ARD), salinity, metal/metalloid leaching (including neutral and metalliferous drainage, NMD) of the proposed mine materials, identify any geochemical issues and opportunities, and provide recommendations for materials management and any follow up test work required.

The scope of work comprised the following:

- An initial scoping phase involving liaison with relevant project personnel, compilation of background project data, including project mine planning information, geological data, and relevant groundwater and surface water quality monitoring.
- Site visits in July 2017 and April 2018 to:
    - examine current operations, including inspection of pits, overburden spoil piles, and deposited washery waste materials;
    - examine representative core through the proposed mine stratigraphic sequence, and check for evidence of pyrite and neutralising carbonate occurrence, and obtain a better understanding of the continuity and variation of the major rock types;
    - coordinate sampling programmes of mine materials from existing operations as an indication of geochemical properties for future materials, including emplaced overburden/interburden, emplaced coarse rejects, and spigotted fine rejects (tailings) in disposal facilities; and
    - coordinate collection of samples from core holes drilled in the MCCO Project additional mining area to represent the main overburden/interburden units.
- Sample preparation and geochemical testing of samples collected during the site visits; and
- Assessment of results and water quality data, and reporting.

# 2.0 Background and Geology

It is understood that the MCCO Project will be an extension of the existing Mangoola Coal Mine operations, with overlapping development of the MCCO Project Additional Mining Area in the last few years of the existing Mangoola Coal Mine to maintain ROM coal production rates of up to 13.5 Mt/annum ROM. The MCCO Project would utilise the existing infrastructure, mining and processing methods to produce a bituminous rank thermal coal with low sulphur and phosphorus from the same target seams:

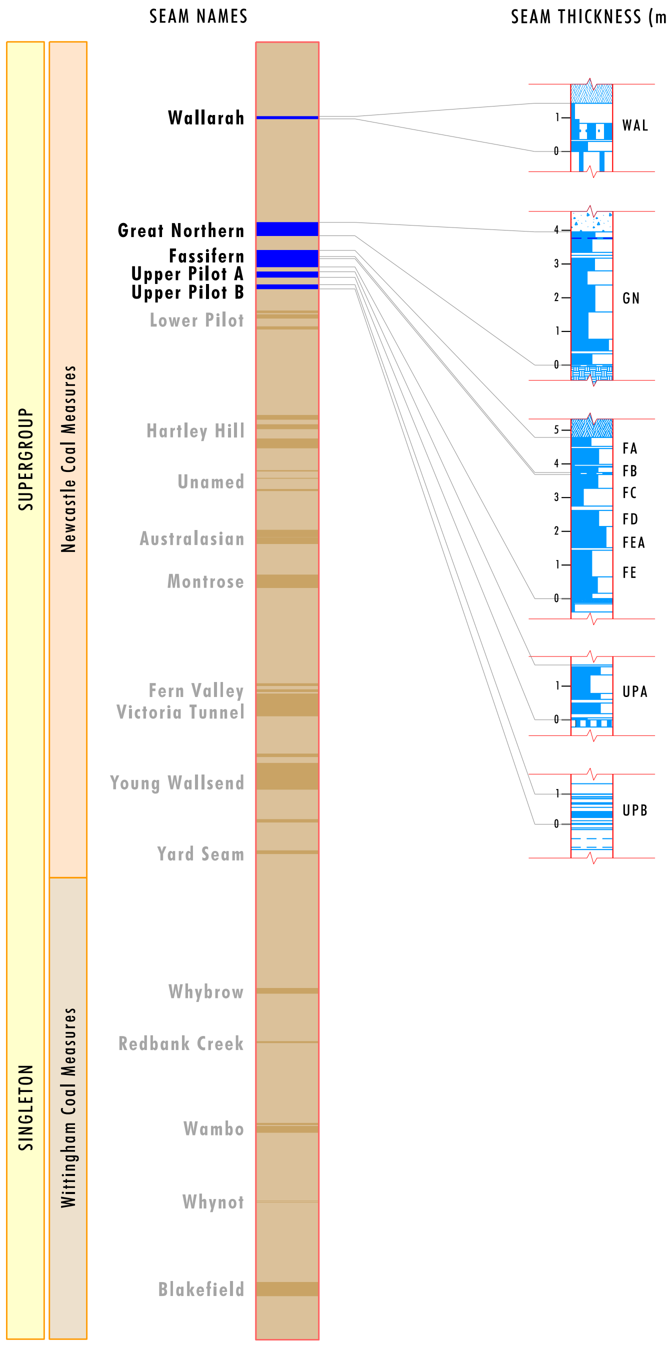
- Wallarah (<1m thick)
- Great Northern (2-4m thick)
- Fassifern (4-6m thick)
- Upper Pilot A (~1m thick).

The target seams are hosted within the upper section of the late Permian Newcastle Coal Measures (Figure 1), with the Wallarah Seam conformably overlain by Triassic Narrabeen Group. The seams dip westerly at a shallow angle of approximately 2°. The depth of oxidation ranges from approximately 10m to 30m. It is understood the stratigraphy is consistent and laterally continuous across the existing approved Mangoola operational area and MCCO Project Additional Mining Area deposits. Wallarah Seam overburden is dominated by conglomerates, with minor sandstone and claystone. The floor of the Wallarah Seam comprises a carbonaceous mudstone (approximately 0.5m), underlain by a sandstone (approximately 1m), with conglomerate (Teralba Conglomerate) making up the vast bulk of the interburden between the Wallarah and Great Northern Seams. The Great Northern and Fassifern Seams are separated by a unit called the Awaba Tuff, with carbonaceous shale on the floor of the Great Northern Seam and roof of the Fassifern Seam. A unit referred to as the Zebra Stripes separates the Fassifern and Upper Pilot A Seams, and comprises interlayered bands of carbonaceous shale and tuff. The Fassifern Seam floor is carbonaceous shale, the Upper Pilot A Seam roof is tuff, and the Upper Pilot A Seam floor is carbonaceous shale.

Mangoola Coal Mine uses the approved on-site tailings storage facilities, which consists of a series of tailings dams for emplacement of fine tailings material.

Water drained from tailings emplacement facilities is captured within decant dams or sumps, where possible, for re-use in the CHPP or for dust suppression. As mining progresses the tailings dams and decant facilities will be decommissioned or mined through with the Tailings Decant Dam being the primary facility being utilised going forward to capture excess water from TD4. Coarse rejects from coal preparation is transported by truck to the open cut overburden emplacement areas for emplacement and subsequent covering by overburden material.

The MCCO Project deposit would be mined as an open cut using conventional truck and shovel methods as per the existing Mangoola Coal Mine, with the pit likely to reach a maximum depth of around 90m. Coal would be washed at the existing Mangoola process plant, with coarse rejects co-disposed with overburden/interburden, and tailings placed in the existing tailings storage facilities within TD4, consistent with current practice. Initially, MCCO Project overburden would be either placed within the existing Mangoola Coal Mine mining area, or in an out of pit emplacement area adjacent to the MCCO Project pit. Overburden/interburden would be progressively backfilled into the MCCO Project pit as the pit progresses.



Data Source: Glencore (2015)

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**Figure 1: Typical stratigraphic profile.**

The original depositional environment largely controls the distribution and abundance of pyrite in coal bearing sedimentary sequences, with influences such as seawater incursions and presence of organic matter key to pyrite formation. Pyrite formation requires a source of sulphate, with seawater a key source, and reducing conditions, typical of swampy environments conducive to peat accumulation, to convert the sulphate to sulphide. Because of these controls, pyrite is usually preferentially distributed in particular lithologies (such as carbonaceous mudstones) and stratigraphic horizons, with coal seam roof and floor and washery rejects materials commonly having higher ARD potential than other mine materials. It is understood that the coal seam sequence for the MCCO Project is largely fluvial, resulting in low S coal and a generally low pyrite content in mine materials.

# 3.0 Visual Assessment of Overburden/Interburden

Drill holes MN0001, MN0003 and MN0006 were examined during the site visit, with locations of holes viewed shown in Figure 2 relative to the proposed pit. Hole MN0001 was collared west of the deepest part of the pit and, although outside of the proposed pit, included the full equivalent stratigraphy to be mined. Hole MN0003, was collared close to the western high wall, and represents the unweathered portion of the materials to be mined in that zone. Hole MN0006 was drilled in a shallow portion of the pit and includes typical weathered Great Northern Seam overburden. The core was inspected to check for evidence of pyrite and neutralising carbonate occurrence, and obtain a better understanding of the continuity and variation of the major rock types through the planned mine stratigraphic sequence.

The holes examined were drilled in 2014, and after a few years of storage any pyritic zones should be readily apparent due to partial oxidation and development of distinctive yellow and white jarosite and sulphate salt crustings. Pyrite was rarely observed in the core, with most core devoid of any evidence of pyrite (Plate 1 and 2). Where present it was evident as iron staining and/or yellow colouration of isolated clasts and coaly wisps (Plates 3, 4 and 5). Inspection of in situ rock in the existing Mangoola Coal Mine operations was generally consistent with this, with exposed coal seams, roof, and interburden in two pits viewed showing no evidence of pyrite oxidation or acid seepage (Plates 6 and 7). The only exception was an area of exposed shale in the northeast corner of Tailings Dam 2 (TD2) that showed a yellow colouration, most likely due to jarosite and sulphate salts formation from partial pyrite oxidation (Plate 8). Site personnel advised that this shale unit occurs above the Fassifern Seam, and is an isolated unit. A sample of this material was collected during the site visit to help determine any implications for overburden/interburden management.

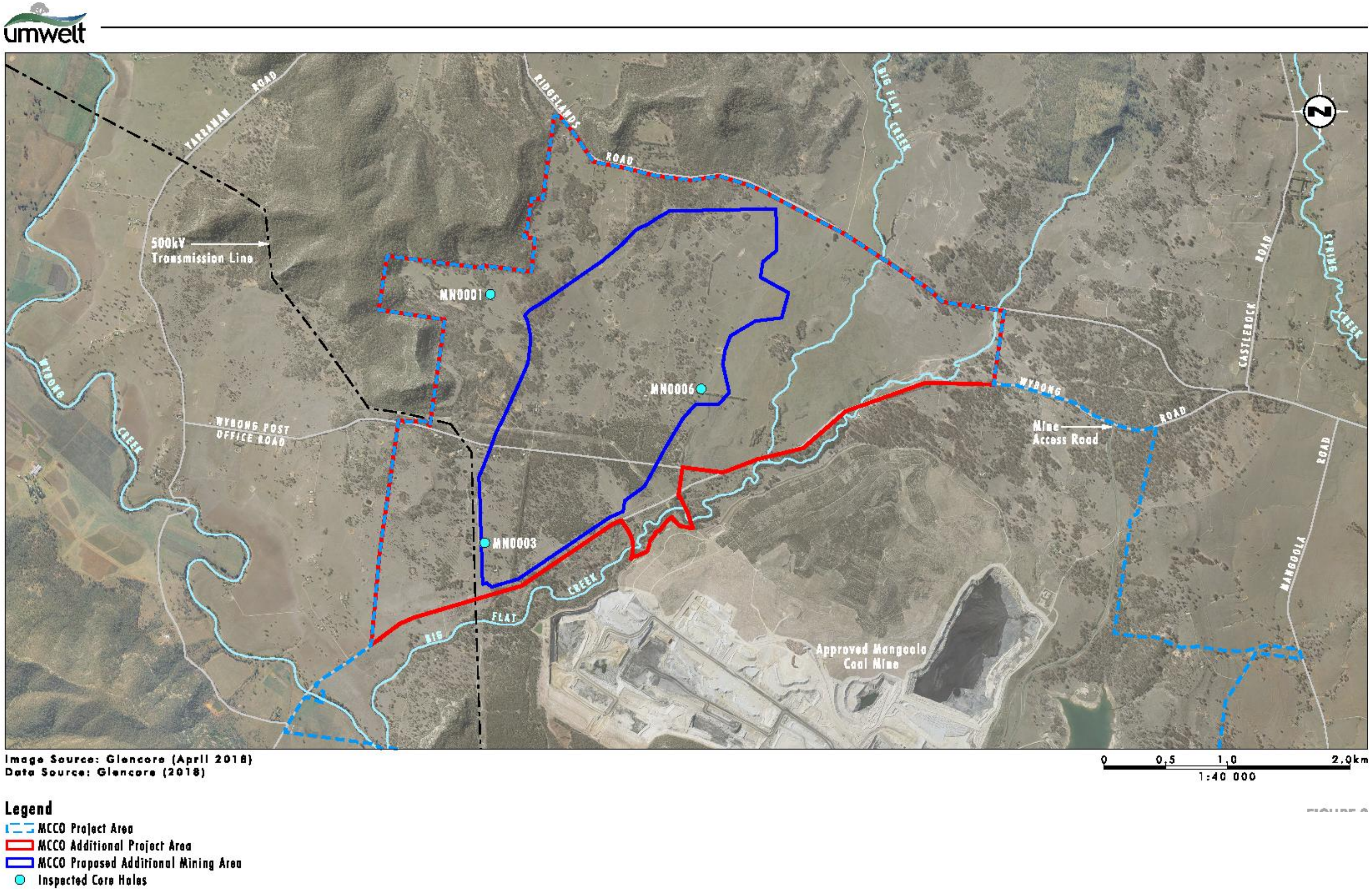

Image Source: Glencore (April 2018)
Data Source: Glencore (2018)

Figure 2: Plan showing location of core holes inspected relative to proposed MCCO pit shell.

**Plate 1:** Typical conglomerate with no evidence of pyrite. Hole MN0003, between 39.0m and 41.4m.

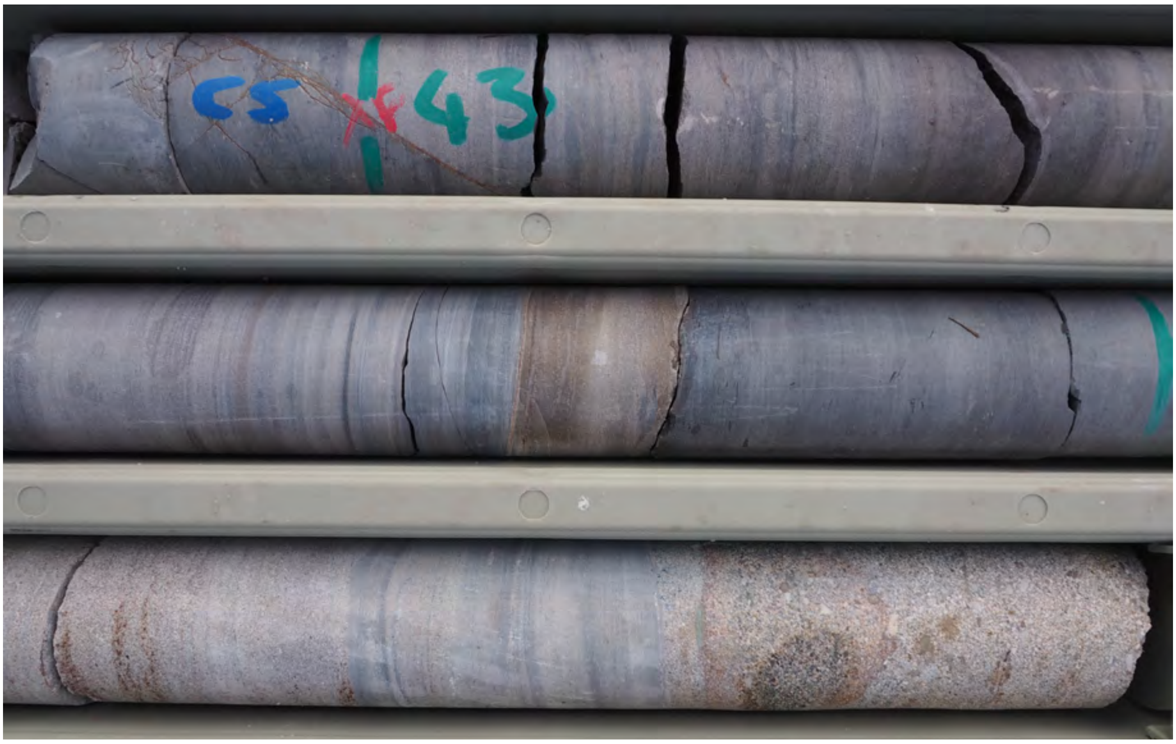

**Plate 2:** Typical sandstone and siltstone with no evidence of pyrite. Hole MN0003, between 42.9m and 44.9m.

**Plate 3: Iron staining and jarosite/sulphate salts after minor pyrite in conglomerate clast. Hole MN0001, 99.2m.**

**Plate 4: Jarosite/sulphate salts after minor pyrite in conglomerate clast. Hole MN0001, 98.5m.**

**Plate 5: Iron staining after minor pyrite from coaly wisp in sandstone. Hole MN0003, 24.3m.**

**Plate 6: Exposed Great Northern and Fassifern Seam in Main Pit West Strip 8, showing no evidence of pyrite, pyrite oxidation or acid seepage from overburden/interburden or coal.**

**Plate 7: Exposed Great Northern Seam and Fassifern Seam in the original pit, showing no evidence of pyrite, pyrite oxidation or acid seepage from overburden/interburden or coal.**

**Plate 8: Shale exposed in the northeast corner of TD2, with yellow colouration most likely due to pyrite oxidation producing jarosite and sulphate salts.**

Application of 10% HCl to rock containing significant amounts of reactive acid neutralising carbonates (such as calcite and dolomite) results in vigorous fizzing. During inspection of the core, 10% HCl was applied intermittently to provide an indication of the presence of reactive carbonate. The overburden/interburden showed only very weak fizzing, indicating low abundance of reactive carbonate.

# 4.0 Visual Assessment of Coal Seams and Review of Site Sulphur Results

The immediate roof and floor, and partings of coal seams are commonly the most pyritic portions in a coal seam sequence due to the depositional controls discussed above. These units were not present in the core inspected as they were sampled for coal quality test work. Available S results were provided for seam roof, partings and floor for the target seams, which are presented as a box plot in Figure 3, grouped by seam. Results show that total S is relatively low overall, with medians generally below 0.2%S and with 75% of S values generally below 0.3%S. The floors of the Great Northern, Fassifern and Upper Pilot A Seams have the highest median S values of close to 0.2%S. There are some samples with elevated S values of 0.5%S or greater, but overall these represent less than 5% of samples tested, and are likely to be insignificant relative to the volume of overburden/interburden mined. The total S results suggest a relatively low ARD potential.

The Zebra Stripes between the Fassifern and Upper Pilot A Seams were also removed from the core, but some typical materials were observed alongside a haul road in the Main Pit East (Plate 9) and in other dump locations, and no evidence of pyrite or pyrite oxidation was observed. One sample of a Zebra Stripes intersection from Hole MN0001 was tested as part of coal quality testing, which returned low total S of 0.17%S. Site observations and previous testing indicate the Zebra Stripes unit would have low ARD potential, and samples of Zebra Stripes materials were collected to confirm these observations.

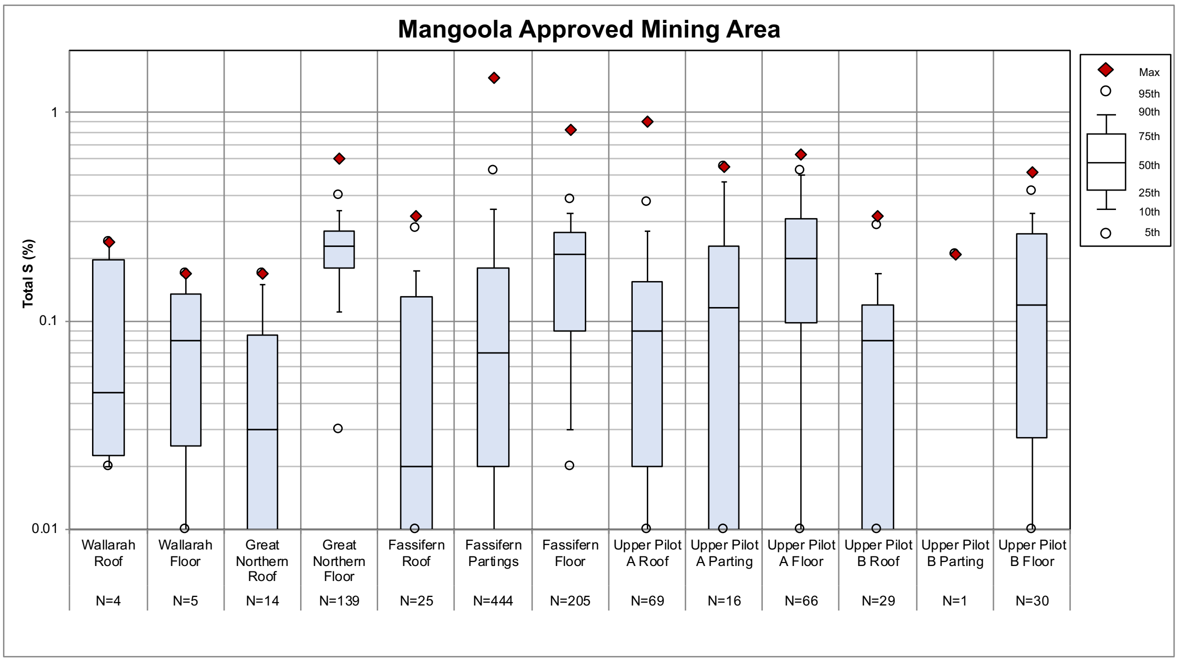

**Figure 3: Box plot showing the distribution of S in roof, partings and floor grouped by seam. Box plots have 5th, 10th, 25th, 50th (median), 75th, 90th and 95th percentiles marked, and maximum also shown.**

**Plate 9: Zebra Stripes materials in Main Pit East, showing no evidence of pyrite or pyrite oxidation.**

# 5.0 Visual Assessment of Rejects and Tailings

As with coal seam roof and floor, coarse rejects and tailings materials commonly have greater ARD potential than other coal mine materials.

Coarse rejects are co-disposed with overburden/interburden in the existing Mangoola mining area (Plate 10), which will continue for the MCCO Project. Inspection of coarse reject piles at a number of locations around the existing Mangoola Coal Mine showed a distinct lack of any evidence of pyrite or pyrite oxidation products (Plate 11). Samples of coarse reject materials were collected to confirm these observations.

Tailings (fine rejects) placed in Tailings Dam 1 (TD1) and Tailings Dam 2 (TD2) were also inspected, and most of the tailings showed no evidence of pyrite or pyrite oxidation products (Plates 12 and 13). However, localised zones close to the spigot discharge points in TD1 and TD2 showed distinct yellow and white salts typical of partially oxidised pyritic materials (Plates 14 to 17). It was expected that these zones represent preferential deposition and concentration of pyrite due to deposition of the heavier pyrite fraction as the discharge energy decreases away from the spigot, and the bulk of the tailings would therefore be benign. Samples of tailings materials were collected during the site visits for testing to help determine implications for tailings management.

**Plate 10: Coarse rejects co-disposed with overburden/interburden in South Pit.**

**Plate 11: Coarse rejects co-disposed with overburden/interburden in South Pit showing no evidence of pyrite or pyrite oxidation.**

**Plate 12: Tailings in the W corner of TD1 showing no evidence of pyrite or pyrite oxidation.**

**Plate 13: Tailings in the E corner of TD1 showing no evidence of pyrite or pyrite oxidation.**

**Plate 14: Tailings in the S corner of TD2 showing white sulphate salts typical of pyrite oxidation.**

**Plate 15: Tailings in the E corner of TD2 showing yellow and white salts typical of pyrite oxidation.**

**Plate 16: Tailings in the E corner of TD2 showing yellow and white salts typical of pyrite oxidation.**

**Plate 17: Tailings in the E corner of TD2 showing yellow and white salts typical of pyrite oxidation.**

# 6.0 Geochemical Testing Results

## 6.1 Sample Collection and Descriptions

### *6.1.1 Tailings and Rejects Samples*

A total of 66 tailings samples were collected from TD1 and TD2, which comprise materials placed as part of current operations, and are representative of tailings materials that would be placed as part of the MCCO Project. Locations are shown in Figure 4.

Samples of 10 to 15kg were collected by a combination of long reach excavation and hand shovel (Plate 18), with samples placed in plastic bags, and cable tied for sample transport. The long reach samples were collected close to tailings spigot points to obtain information on vertical variation of ARD potential at the key pyrite accumulation points, and the hand shovel samples were collected to map lateral variation in ARD potential with distance from the spigot. Layers of pyritic materials such as those shown in Plates 15 and 16 were observed close to the spigot points, but generally absent 20m away.

Five 10 to 15kg grab samples of typical coarse rejects were collected by hand shovel with the assistance of the operations geologist from a number of locations across the current operations, with approximate locations shown in Figure 5. All samples showed a distinct lack of pyrite or pyrite oxidation products, consistent with Plate 10 and 11.

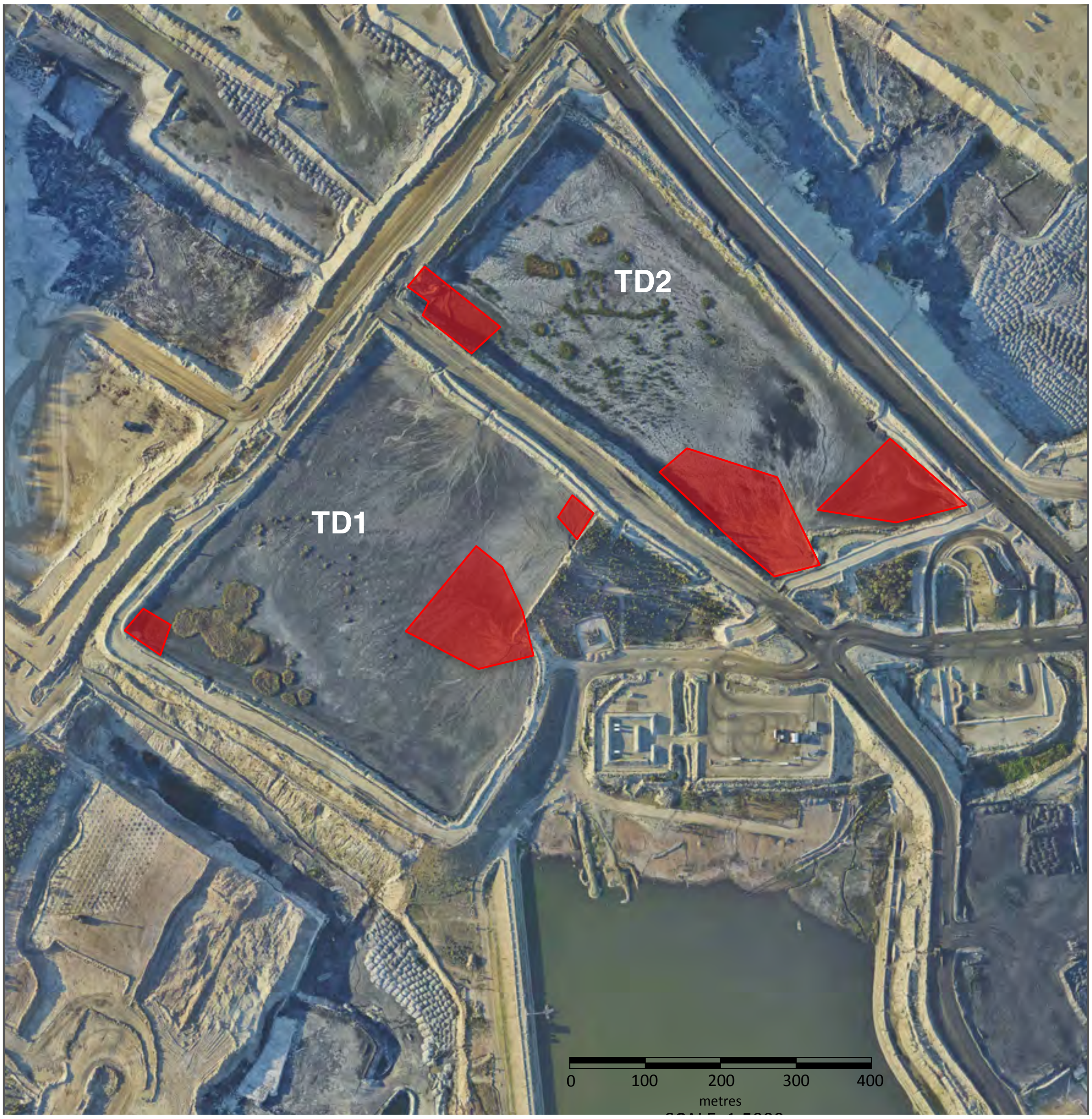

**Figure 4: Plan showing areas of tailings sampling (as red shaded outlines) from TD1 and TD2.**

**Plate 18: Long reach and hand shovel sampling at TD2 south corner.**

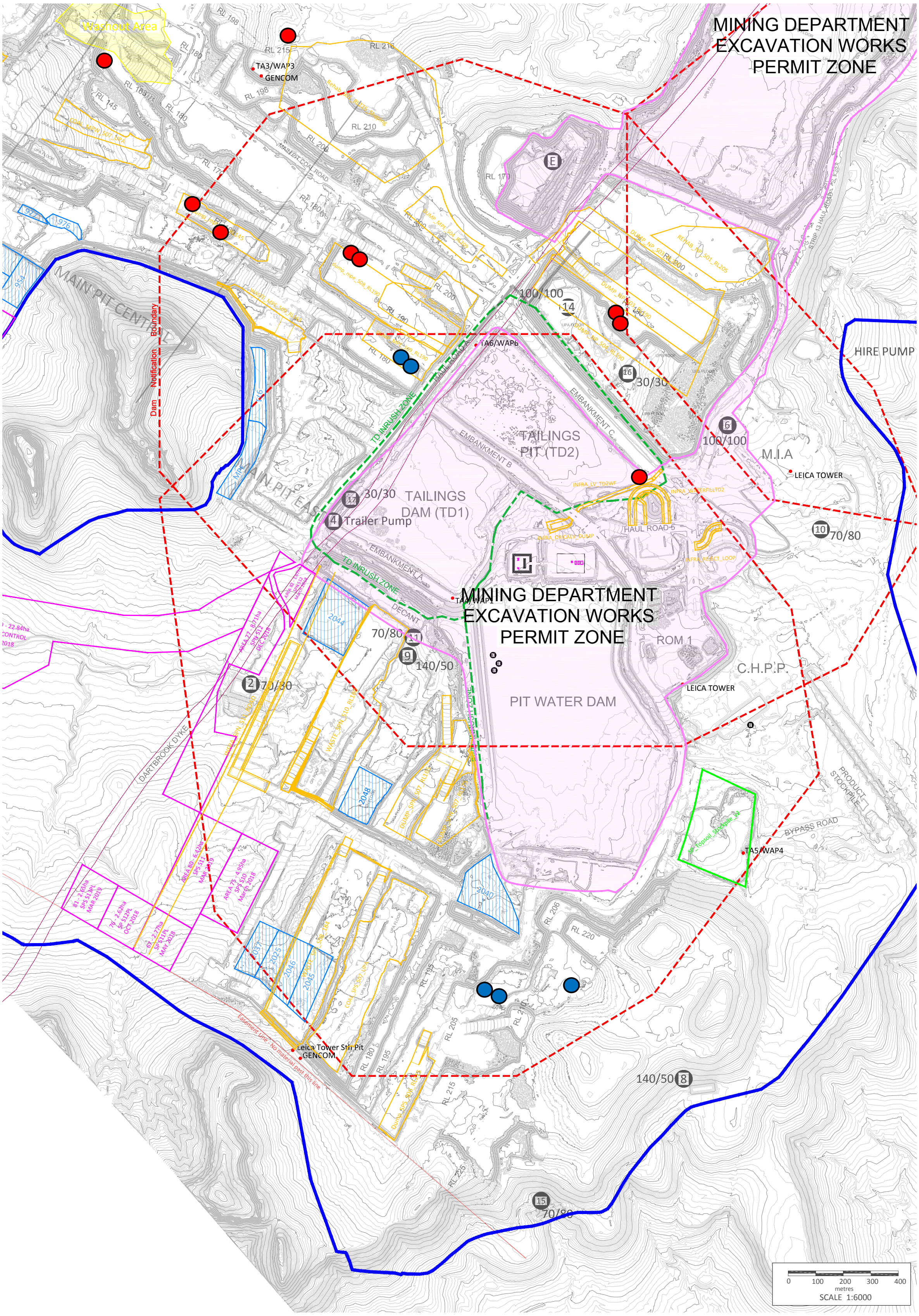

**Figure 5: Plan showing approximate locations of coarse reject (blue circles) and overburden/interburden (red circles) samples collected from the current operations.**

### *6.1.2 Overburden/Interburden Samples*

The site geologist assisted with the collection of typical overburden/interburden materials of various deposition ages from the current operations. A total of nine grab samples of 10 to 15kg each were collected by hand shovel and placed in plastic bags, and cable tied for sample transport. Approximate locations are shown in Figure 5. These materials are generally expected to be typical of overburden/interburden that will be produced in the MCCO Additional Project Area. The only exception was a sample of the partly pyritic (jarositic) shale unit within Great Northern Seam and Fassifern Interburden shown in Plate 8, which is an isolated unit.

Twenty samples of core from holes MN0001, MN0003 and MN0006 in the MCCO Additional Project Area were also collected to supplement the operational materials, and to allow better stratigraphic control on overburden/interburden types. These were the holes visually assessed as described in Section 3, with hole locations shown in Figure 2. Representative samples of the main overburden/interburden units were collected as individual intervals of approximately 0.5m to 3m length, and composites of 2 to 3 individual samples of approximately 1m length for massive and uniform conglomerate units. Whole continuous core sections were collected for each selected interval.

## 6.2 Methodology

A total of 100 samples were collected for geochemical testing.

Sample preparation was carried out by International Resource Laboratories (IRL) (Brisbane), which involved drying (as required), crushing to a nominal -4mm, splitting, pulverising a 500g split to -75$\mu$m, and dispatch of 500g of -75$\mu$m pulverised samples and 500g of -4mm crushed samples to EGi for testing.

All samples were analysed for the following:

- pH and electrical conductivity (EC) of deionised water extracts at a ratio of 1 part solid to 2 parts water ($pH_{1:2}$ and $EC_{1:2}$);
- Leco total sulphur (S);
- acid neutralising capacity (ANC);
- net acid producing potential (NAPP) - calculated from total S and ANC; and
- single addition net acid generation (NAG) test.

Further testing was carried out on selected samples to help resolve uncertainties in the above test results, as follows:

- extended boil and calculated NAG testing to account for high organic carbon contents (19 samples);
- sulphur speciation to obtain a guide to the proportion of pyritic S (14 samples); and
- acid buffering characteristic curve (ABCC) testing to define the relative availability of the ANC measured (17 samples).

A general description of ARD test methods and calculations used is provided in Appendix B.

In addition, selected samples were assayed for the following to identify any potential elemental concerns and to provide initial elemental solubility data:

- multi-element testing of solids (23 samples); and
- multi-element testing of deionised water extracts at a ratio of 1 part solid to 2 parts water (23 samples).

Water extractions for $pH_{1:2}$ and $EC_{1:2}$ and multi-element testing were carried out on -4mm crushed samples. Pulverised samples were used for all other tests.

The sulphur speciation procedure involved Leco total S, chromium reducible sulphur (CRS) and KCl digestion to help differentiate pyritic S, acid forming sulphate, non-acid forming sulphate and other S forms (including organic S, jarosite S and elemental S).

Total sulphur assays and multi-element testing of solids were carried out by IRL (Brisbane). CRS analyses of sample solids were carried out by ALS Laboratory Group (Brisbane). Multi-element analyses of water extracts were carried out by ALS Laboratory Group (Sydney). Analyses of NAG solutions and S analysis of KCl digest solutions were carried out by Levay & Co. Environmental Services (Adelaide). All other analyses were carried out by EGi.

## 6.3 Standard Geochemical Test Results

Acid forming characteristics of the grab samples from current operations and core samples from the proposed Additional Mining Area are presented in Appendix A - Table A1, comprising results of standard geochemical tests pH and EC of water extracts, total S, maximum potential acidity (MPA), ANC, NAPP, ANC/MPA ratio and single addition NAG.

### *6.3.1 pH and EC*

The $pH_{1:2}$ and $EC_{1:2}$ results were determined by equilibrating the sample in deionised water for approximately 16 hours at a solid to water ratio of 1:2 (w/w). Results give an indication of the inherent acidity and salinity of the waste material when exposed in a waste emplacement area.

The $pH_{1:2}$ values ranged from 2.1 to 8.8, with most samples (85%) having a pH greater than 6, showing no inherent acidity. Five tailings samples with distinct pyrite, jarosite and sulphate salts had an acidic pH of less than 4. All coarse rejects samples had circum-neutral pH. The in situ jarositic shale had an acidic pH of 3.5, with the remaining overburden/interburden samples circum-neutral.

$EC_{1:2}$ values ranged from 0.2 to 15.6 dS/m. Approximately half the tailings samples were saline (EC greater than 1.6 dS/m) or strongly saline (EC greater than 3.2 dS/m), with the remainder moderately saline. All of the coarse rejects samples were non saline, ranging from 0.25 to 0.35 dS/m. The in situ shale sample was saline at 1.8 dS/m, but the remaining overburden/interburden samples were non saline to slightly saline with an EC less than 0.8 dS/m.

Figure 6 is a plot of $pH_{1:2}$ and $EC_{1:2}$ versus total S, split by material type. The tailings show variable pH and EC with S content, but the lower pH (<4) samples had elevated S greater than 0.4%S, and the associated elevated salinities of greater than 1.6 dS/m are likely to be controlled by pyrite oxidation reactions. The high EC in the lower S (<0.4%S) tailings indicate another source of salinity apart from pyrite oxidation. Note that not all tailings samples showing distinct pyrite and jarosite and high S were acidic, which reflects the amount of ANC present. Figure 7 is a plot of $pH_{1:2}$ versus ANC split by material type, which shows that the lower pH values are restricted to samples with low ANC less than 10 kg $H_2SO_4$/t. Results show that the ANC in the tailings has been effective in buffering the pyritic materials at least in the short term, with acid formation occurring only after the ANC is depleted.

The coarse rejects had low S, low EC and circum-neutral pH.

The in situ shale sample has elevated S of 0.4%S, acid pH of 3.5 and saline EC of 1.8 dS/m, suggesting acid salt generation due to partial pyrite oxidation effects, consistent with site observations of (Plate 8). The remaining overburden/interburden samples had low S, low EC and circum-neutral pH.

Overall, results indicate some influence from ARD on sample salinity, but also suggest additional sources contributing to sample salinity, most likely related to the process water and/or inflowing groundwater.

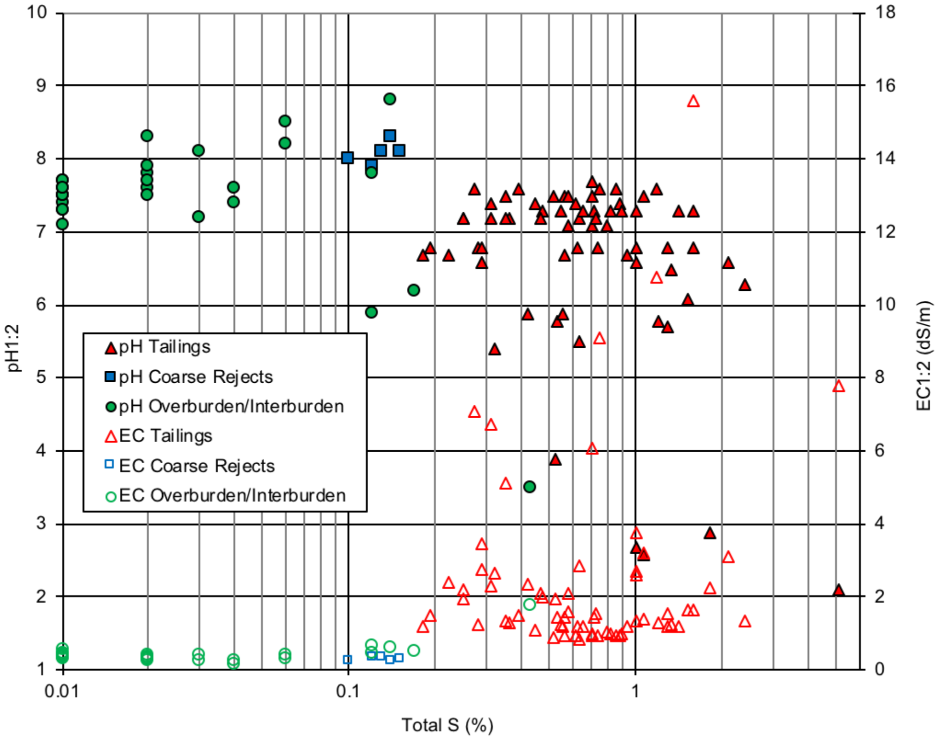

**Figure 6: Plot showing $pH_{1:2}$ and $EC_{1:2}$ versus total S, split by mine material type.**

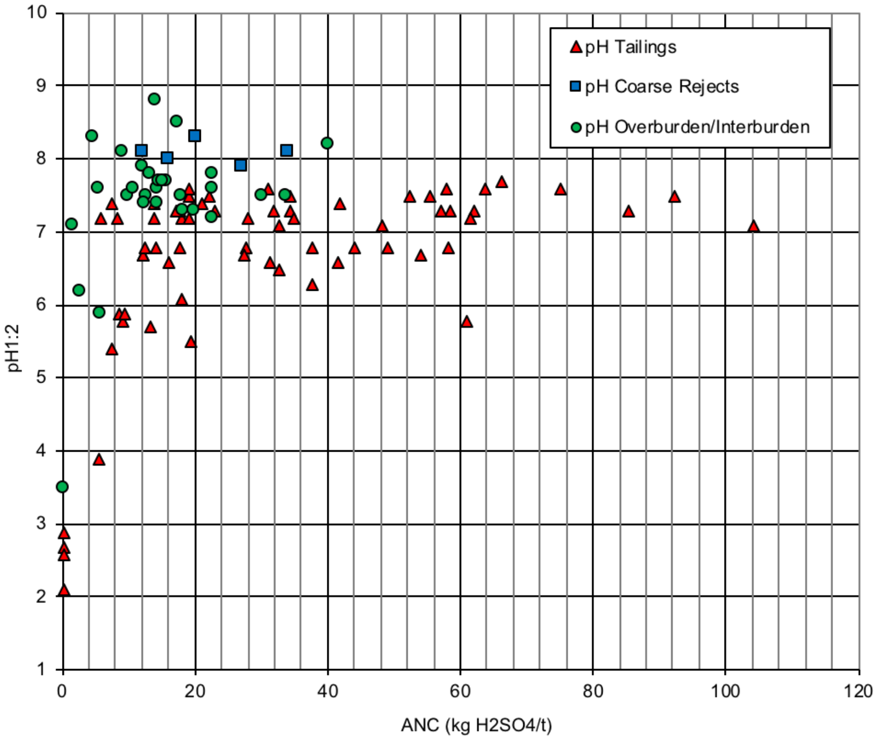

**Figure 7: Plot showing $pH_{1:2}$ versus ANC, split by mine material type.**

### *6.3.2 Acid Base (NAPP) Results*

Total S showed a broad range from below detection to 5.0%S. Figure 8 is a box plot of the distribution of S split by mine material type. The plot shows that overburden/interburden and coarse reject samples have low S below 0.2%S, except for the partly pyritic shale unit described above (Plate 8), with a moderate S value of 0.4%S. The tailings samples show distinctly higher S than other materials types, with a median of 0.6%S, and 25% of samples having S greater than 1%S. Note that a significant portion of total S in coal washery tailings is typically in organic form and non acid generating.

ANC ranged up to 100 kg $H_2SO_4$/t. Figure 9 is a box plot of the distribution of ANC split by mine material type. Results show that although the tailings have the highest S, they also have the highest ANC, which will help offset any acid generated by pyrite oxidation. The relatively low ANC in the overburden/interburden is consistent with the low abundance of reactive carbonate indicated by visual assessment (Section 3).

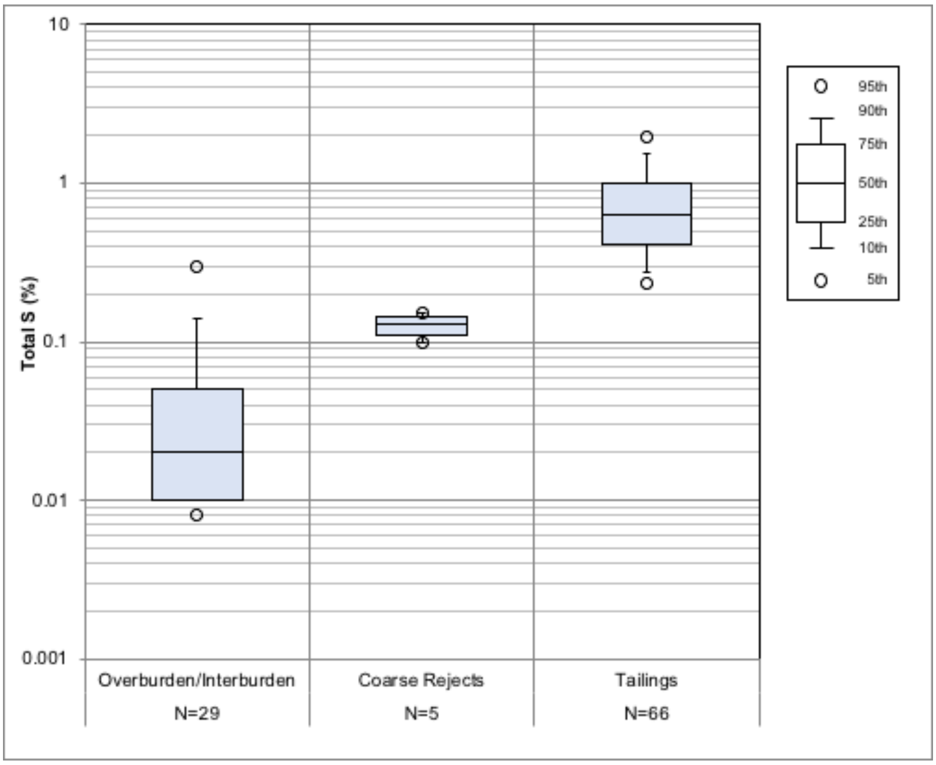

Figure 8: Box plot showing the distribution of S split by mine material type. Box plots have 5th, 10th, 25th, 50th (median), 75th, 90th and 95th percentiles marked.

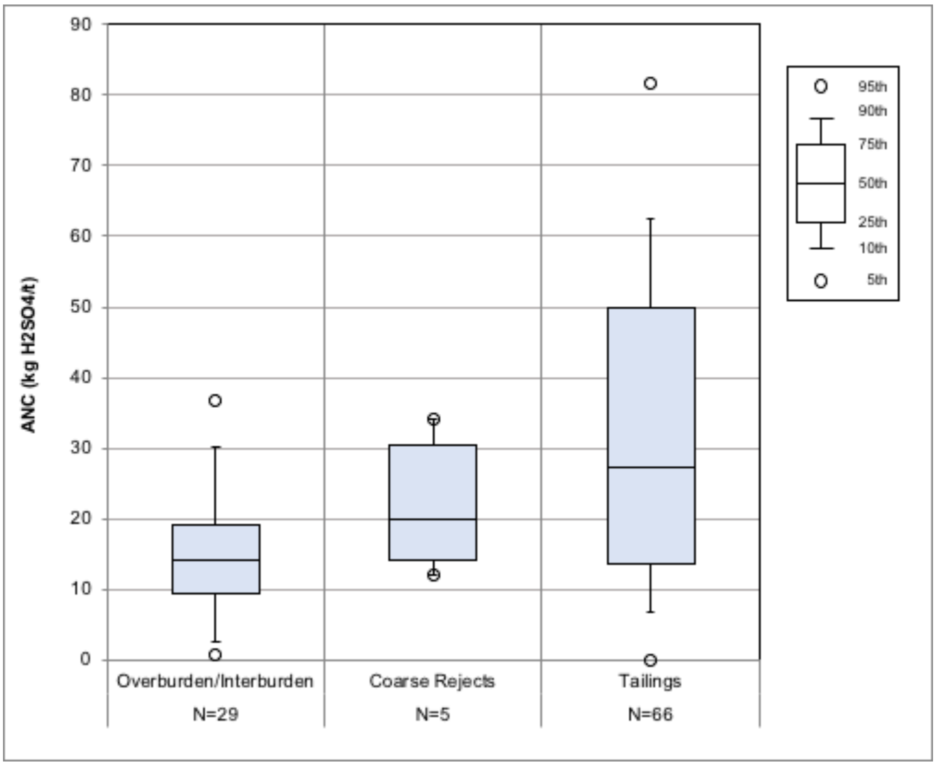

Figure 9: Box plot showing the distribution of ANC split by mine material type. Box plots have 5th, 10th, 25th, 50th (median), 75th, 90th and 95th percentiles marked.

The NAPP value is an acid-base account calculation using measured total S and ANC values. It represents the balance between the MPA and ANC. A negative NAPP value indicates that the sample may have sufficient ANC to prevent acid generation. Conversely, a positive NAPP value indicates that the material may be acid generating. Note that sulphidic materials with sufficient ANC to prevent acid generation may still produce saline and metalliferous drainage under circum-neutral pH conditions.

Figure 10 is an acid-base account plot of ANC verses total S split by rock unit. Figure 11 is the same as Figure 10, but re-scaled to better represent S below 2%S. The NAPP zero line is shown which defines the NAPP positive and NAPP negative domains, and the line representing an ANC/MPA ratio value of 2 is also plotted. Note that the NAPP = 0 line is equivalent to an ANC/MPA ratio of 1. The ANC/MPA ratio is used as an indication of the relative factor of safety within the NAPP negative domain. Usually a ratio of 2 or more signifies a high probability that the material will remain circum-neutral in pH and thereby should not be problematic with respect to acid drainage.

The plots show that all coarse rejects and all but two overburden/interburden samples plot in the NAPP negative domain, with a narrow S range and generally with ANC/MPA ratios or 2 or more. One of the two NAPP positive overburden/interburden samples is the partly pyritic shale unit, which only has moderate S, but no ANC.

The tailings samples show a broad distribution of S and ANC combinations, with 65% of tailings samples plotting in the NAPP negative domain, and 35% having ANC/MPA ratios of 2 or greater.

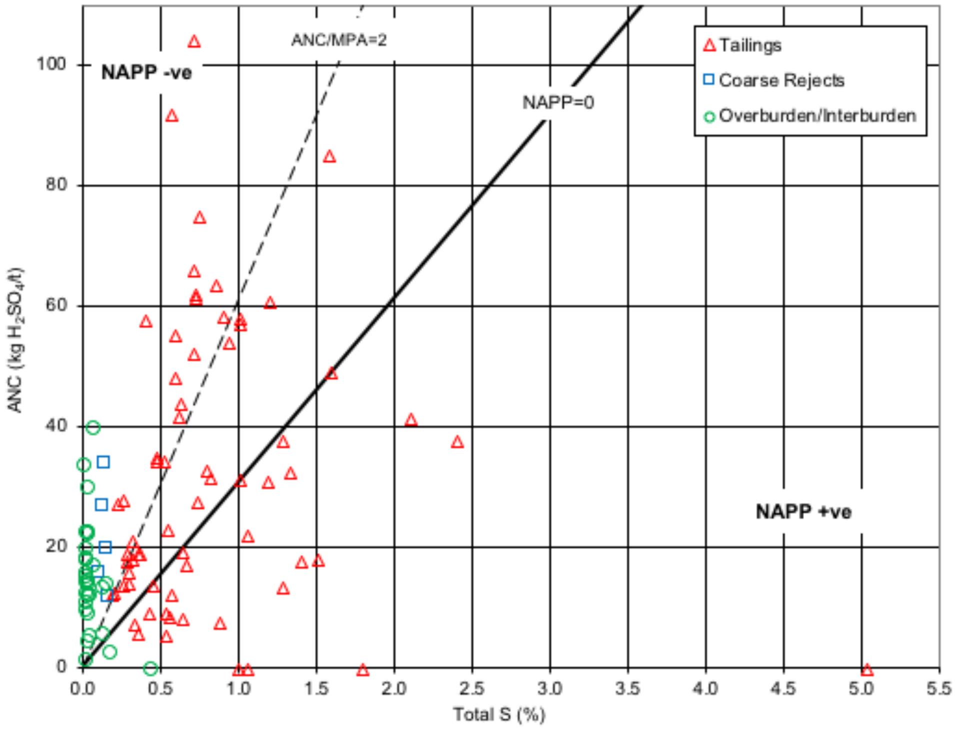

**Figure 10: Acid-base account (ABA) plot split by mine material type.**

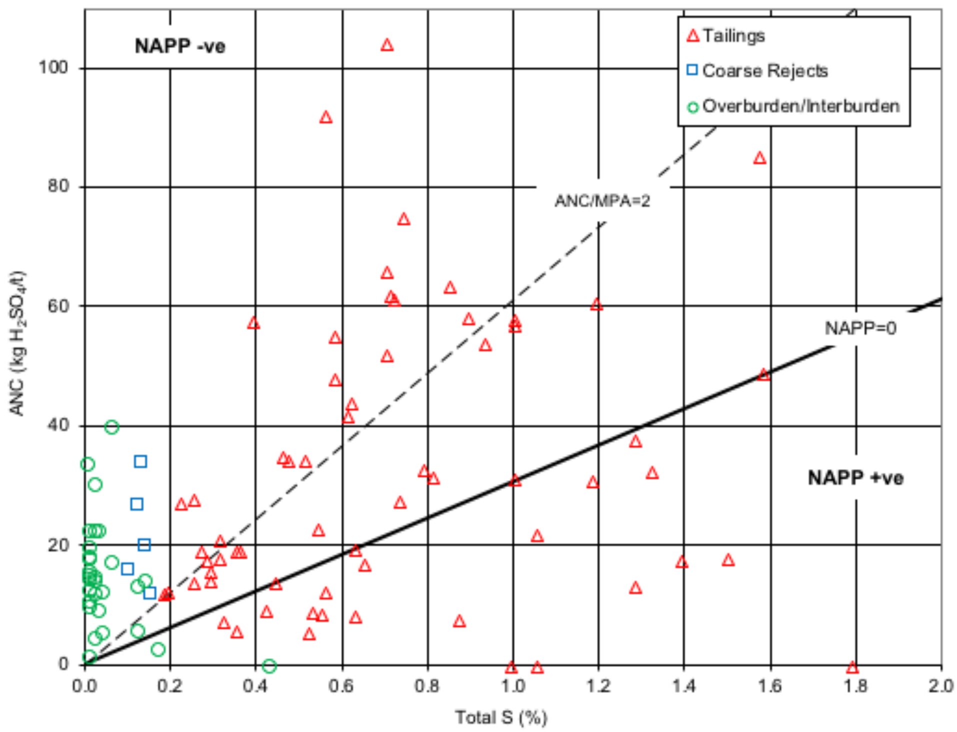

**Figure 11: As for Figure 10 with expanded axes.**

### *6.3.3 Single Addition NAG Results*

Generally, a NAGpH value less than 4.5 indicates a sample may be acid forming. However, samples with high organic carbon contents (such as coal, carbonaceous sedimentary and washery waste materials) can cause interference with standard NAG tests due to partial oxidation of carbonaceous materials. This can lead to low NAGpH values and high acidities in standard single addition NAG tests unrelated to acid generation from sulphides.

Half of the samples tested had NAGpH values of 4.5 and greater, indicating they are likely to be non-acid forming (NAF). Although the remaining samples had a NAGpH less than 4.5, these were all associated with carbonaceous horizons and carbonaceous washery wastes, and results are inconclusive in isolation due to potential organic acid effects that may contribute acidity to the sample liquor in addition to that released from sulphide oxidation. Organic acid effects are indicated by a large difference between the $NAG_{(pH4.5)}$ and $NAG_{(pH7.0)}$ values, and/or $NAG_{(pH4.5)}$ values very close to or exceeding that of MPA. In these samples the NAG results can overestimate the acid potential. Samples showing organic acid effects are highlighted yellow in Table A1 (Appendix A).

NAG test results are used in conjunction with NAPP values to classify samples according to acid forming potential. Figure 12 is an ARD classification plot showing NAGpH versus NAPP value. Figure 13 is the same as Figure 12, but with an expanded NAPP axis to better represent the range -60 to 60 kg $H_2SO_4$/t. Potentially acid forming (PAF), non acid forming (NAF) and uncertain (UC) classification domains are indicated. A sample is classified PAF when it has a positive NAPP and NAGpH < 4.5, and NAF when it has a negative NAPP and NAGpH ≥ 4.5. Samples are classified uncertain when there is an apparent conflict between the NAPP and NAG results, i.e.

when the NAPP is positive and NAGpH ≥ 4.5, or when the NAPP is negative and NAGpH < 4.5.

Forty six samples plot in the NAF domain. Thirty seven of these have ANC/MPA ratios of 2 or more, indicating a high factor of safety, and these samples are expected to be NAF. Note that some of these are tailings samples with elevated S greater than 0.5%S, but also show elevated ANC. Nine tailings samples plotting in NAF domain have ANC/MPA ratios of less than 2, and further testing was carried out to confirm that buffering was sufficient to account for acid generated from these samples.

There are 21 samples that plot in the PAF domain. Samples 12430 and 14496 did not show evidence of organic acid effects, and were confirmed PAF. Nineteen samples had inconclusive NAG results due to organic acid effects. Of these, 6 samples have $pH_{1:2}$ values lower than 4.5, and are confirmed acid forming. Specialised testing was carried out to help to confirm the classification of these samples.

A total of 29 samples plot in the lower left uncertain domain, all of which are carbonaceous and showed organic acid effects in the NAG tests. Follow up tests to check for organic acid effects and availability and nature of the acid neutralising capacity were carried out to resolve the classification of these samples.

Four tailings samples plot in the upper right uncertain domain. These have elevated total S of 0.65 to 2.1%S, moderate ANC values of 17 to 39 kg $H_2SO_4$/t, and NAGpH values greater than 4.5. The NAG test would not necessarily account for all the pyritic S in these samples, and they are conservatively assumed to be PAF, consistent with NAPP results, where specialised testing was not carried out.

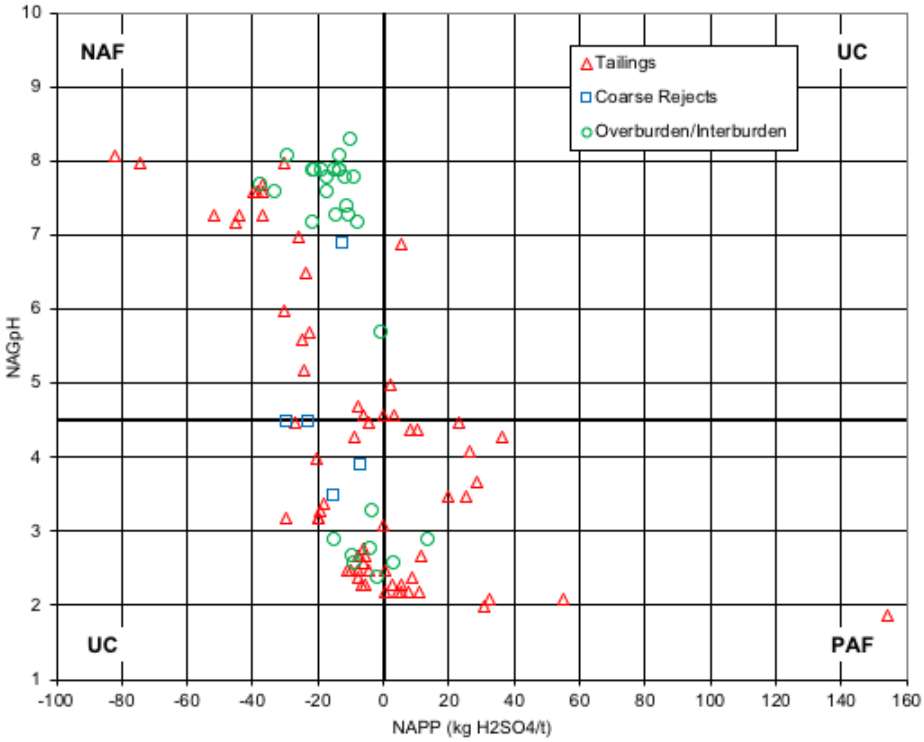

**Figure 12: ARD classification plot showing NAGpH versus NAPP split by mine material type, with ARD classification domains included for reference.**

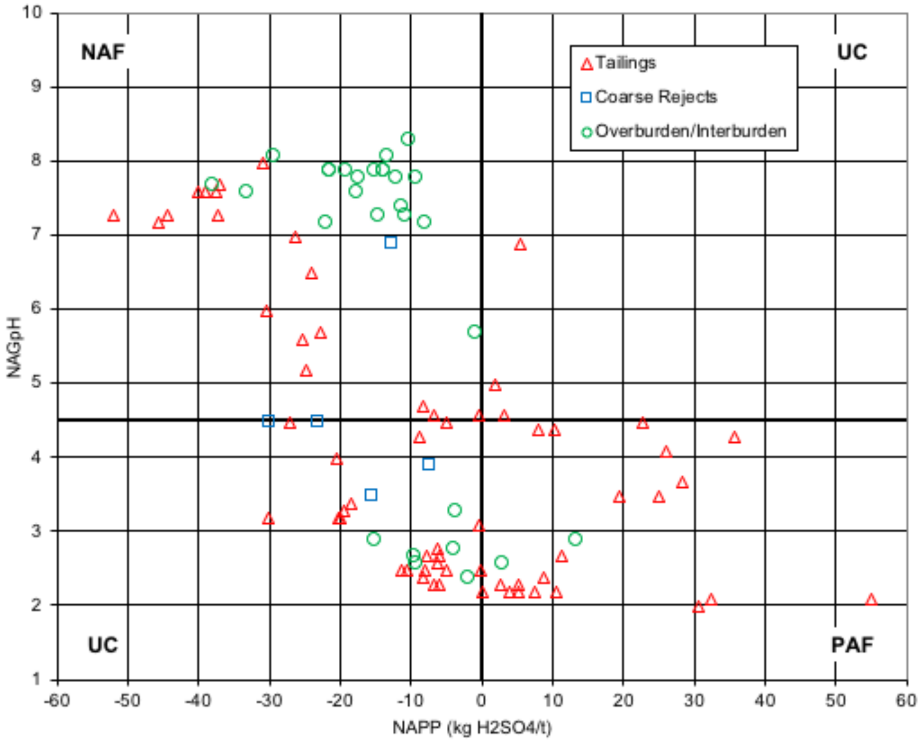

**Figure 13: As for Figure 12 with an expanded NAPP axis.**

## 6.4 Extended Boil and Calculated NAG Results

Extended boil and calculated NAG testing were carried out on 19 selected samples to help resolve uncertainties in ARD classification based on standard NAG test results, as discussed in the previous section. Results are shown in Appendix A, Table A2.

Results show that the NAGpH value for most samples increases 2 to 4 pH units after the extended boiling step. The increase in NAGpH confirms the effects of organic acids. The extended boil NAGpH of samples 12431, 12432, 14479, 14483, 14484 and 14487 remained less than 4.5, indicating these samples are likely to be acid producing.

Note that the extended boil NAGpH value can be used to confirm samples are PAF, but an extended boil NAGpH value greater than 4.5 does not necessarily mean that samples are NAF, due to some loss of free acid during the extended boiling procedure. To address this issue, a calculated NAG value is determined from assays of anions and cations released to the NAG solution. A calculated NAG value of less than or equal to 0 kg $H_2SO_4$/t indicates the sample is likely to be NAF, and a value of more than 0 kg $H_2SO_4$/t indicates the sample may be PAF.

The calculated NAG values for 11 of the samples (12426, 12427, 12435, 12436, 12438, 12439, 14492, 14498, 14503, 14518 and 14519) were negative or equal to zero, indicating that all acid generated in the standard NAG test for these samples is organic, and that materials represented by these samples are unlikely to be acid producing under field conditions. These samples also had NAPP values of 0 kg $H_2SO_4$/t or less and S values less than 1%S.

The remaining eight samples (12429, 12431, 12432, 14479, 14483, 14484, 14487 and 14489) had positive calculated NAG values, indicating these samples are likely to be acid producing. All except for sample 12429 had positive NAPP values, with sample 12429 having a marginally negative NAPP value of -1 kg $H_2SO_4$/t. Results suggest that lower S (<1%S) NAPP negative samples with organic acid effects evident in the NAG test can be assumed to be NAF, and NAPP positive samples can be assumed to be PAF in the absence of direct calculated NAG test work.

## 6.5 Acid Buffering Characteristic Curve (ABCC) Testing

Acid buffering characteristic curve (ABCC) testing was carried out on 6 selected samples to evaluate the availability of the ANC measured.

Acid buffering characteristic curve (ABCC) testing was carried out on 17 selected samples to evaluate the availability of the ANC measured. The ABCC test involves slow titration of a sample with acid while measuring the solution pH. The acid buffering of a sample to pH 4 can be used as an estimate of the proportion of readily available ANC. Results are shown in Appendix A, Figures A1 to A7, with calcite, dolomite, ferroan dolomite and siderite standard curves as reference. Calcite and dolomite readily dissolve in acid and exhibit strongly buffered pH curves in the ABCC test, rapidly dropping once the ANC value is reached. The siderite standard provides very poor acid buffering, exhibiting a very steep pH curve in the ABCC test. Ferroan dolomite is between siderite and dolomite in acid buffering availability.

The ABCC curves for three of the samples 12427, 12435 and 14463 (Figure A2) show strong buffering similar to the dolomite standard curve, with 100% of the total ANC readily available.

Six samples 12433 (Figure A7), 12440 (Figure A4), 14443 (Figure A1), 14452 (Figure A4), 14454 (Figure A2) and 14459 (Figure A2) have profiles that plot between dolomite and ferroan dolomite standard curves. The readily available ANC portion ranges from 60% to 100% of the total ANC, and the reaction rates are likely to be slower than dolomite.

Samples 12438 (Figure A3), 14449 (Figure A2), 14451 (Figure A5), 14460 (Figure A1) and 14465 (Figure A1) have profiles that plot close to the ferroan dolomite standard curve, indicating slow reactivity, with a readily available ANC portion of 50% to 80%.

The remaining samples 12429 (Figure A6), 14439 (Figure A3) and 14448 (Figure A3) have sideritic profiles, with a low portion of readily available buffering of 20% to 40%.

Overall results show variable ANC reactivity, with ferroan carbonate influence, but indicate that at least 50% of the ANC will be readily available and effective.

## 6.6 Sulphur Speciation

Sulphur speciation analysis was carried out on 14 selected samples, with results presented in Appendix A, Table A3. The chromium reducible sulphur (CRS) method provides an indication of pyritic S content, but will not recover sulphates (gypsum/anhydrite), jarosite or alunite. Note that the pyritic S value should only be treated as a guide to the pyrite content in the sample due to issues with repeatability in the chromium reducible sulphur (CRS) method[1].

Given the salts observed in the tailings samples and high EC values, it was expected that a significant portion of the total S in these materials would be in sulphate form, with a portion of acid sulphate. Sulphur speciation results for the 7 tailings samples confirm a high proportion of non-acid sulphate in the tailings samples, with a minor proportion of acid sulphate in samples 12431 and 12432. The total acid generating S proportion for the tailings and rejects accounts for 15% to 46%, showing that the total S overestimates the acid potential of these samples.

The results for the in situ jarositic shale sample (12430) confirms that most (70%) of the total S is present in acid generating forms. The acid generating S content of the other four overburden/interburden samples was minor, accounting for only 20-30% of the total S.

Table A3 includes a recalculated NAPP based on acid generating S and readily available ANC from ABCC testing, which indicates a NAF classification for tailings and coarse rejects samples 12427, 12433, 12438 and 12440, a NAF classification for overburden/interburden sample 14459, and a confirmed PAF classification for tailings samples 12429, 12431 and 12432, and the in situ jarositic shale sample 12430.

In addition, the acid generating S proportion in samples 12428 and 12434 was less than 0.5%S. For low S samples the NAG test would normally account for most of the pyritic S at these values, confirming a NAF classification consistent with NAG test results.

## 6.7 Multi-Element Analysis of Solids and Water Extracts

Results of multi-element scans of solids from 23 selected samples were compared to the median soil abundance (from Bowen, 1979[2]) to highlight enriched elements. The extent of enrichment is reported as the Geochemical Abundance Index (GAI), which relates the actual concentration with an average or median abundance on a log 2 scale. The GAI is expressed in integer increments where a GAI of 0 indicates the element is present at a concentration similar to, or less than, median soil abundance; and a GAI of 6 indicates approximately a 100-fold enrichment above median soil abundance. As a general rule, a GAI of 3 or greater signifies enrichment that warrants further examination.

Results of multi-element analysis of solids are presented in Appendix A, Table A4, and the corresponding GAI values are presented in Table A5.

Most of the tailings samples showed enrichment to slight enrichment in S (already discussed above in regard to acid forming potential) and Tl. Many of the tailings samples also show slight enrichment in As, Be and Se. The elevated Tl and As are likely to be associated with pyrite. Although Be is slightly enriched relative to soils, Be contents are within the typical range for coal

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[1] Environmental Geochemistry International, Levay and Co. and ACeSSS, 2008. *ACARP Project C15034: Development of ARD Assessment for Coal Process Wastes*, EGi Document No. 3207/817, July 2008. www.acarp.com.au.

[2] Bowen, H.J.M. (1979) Environmental Chemistry of the Elements. Academic Press, New York, p 36-37.

and carbonaceous materials. One sample is enriched and another slightly enriched in Ba, but Ba has low solubility in sulphate solutions and is not expected to be of environmental concern.

The two coarse rejects samples show slight enrichment in Be and Se.

Overburden/interburden samples show slight enrichment in Be, with the jarositic shale sample also showing slight enrichment in S, Se and Tl.

The same sample solids were subjected to water extraction at a solids:liquor ratio of 1:2. Results are shown in Appendix A, Table A6.

Tailings samples 12431 and 12432 showed distinctly yellow and white sulphate salts typical of pyrite oxidation products, and produced acid extracts with pH less than 3, strong salinities greater than 3 dS/m, and acidities of 700 and 2700 mg $CaCO_3$/L, confirming the effects of ARD on these materials. The bulk chemistry is dominated by acid, Mg and $SO_4$, with lesser Na and Cl. The low pH has associated elevated Al, Fe, Mn, Ni and Zn, and slightly elevated As, Co, Cr and Cu.

The remaining six tailings samples produced circum-neutral pH leachates with moderate alkalinities of 30 to 70 mg $CaCO_3$/L, which would help offset acidity generated by PAF tailings.

Tailings samples 12428, 12429, 12433 and 12434 had white salts but no yellow salts, and although these materials did not produce acid leachates, the leachates are strongly saline at greater than 3 dS/m, with salinity dominated by $SO_4$ salts and hence primarily influenced by pyrite oxidation reactions. Some elevated Mn was measured in samples 12429 and 12433.

There was no salt accumulation observed for tailings samples 12427 and 12435, but these leachates are also strongly saline at greater than 3 dS/m, but in this case dominated by Cl salts, reflecting a process water and/or ground water influence. No significant metal/metalloid mobilisation was evident for these samples.

The two coarse rejects samples 12438 and 12440 produced circum-neutral leachate with slightly saline EC of less than 0.8 dS/m, and again moderate alkalinities of 30 to 70 mg $CaCO_3$/L. Bulk chemistry was dominated by Cl salts, and no significant metal/metalloid mobilisation was evident.

The in situ jarositic shale also produced acid leachate, but with lower acidities than the acidic tailings samples. Bulk chemistry was again dominated by $SO_4$ salts, as expected, with elevated concentrations of Al, Fe, Mn and Zn.

The remaining overburden/interburden samples produced circum-neutral to slightly alkaline pH, and non saline to slightly saline EC of less than 0.8 dS/m. The Fassifern Seam and Upper Pilot A Interburden Sample 14459 produced a high alkalinity of 245 mg $CaCO_3$/L, but the remaining samples had low to moderate alkalinities of 10 to 80 mg $CaCO_3$/L. No significant metal/metalloid mobilisation was evident for these samples, except for slightly elevated Mn concentrations of 0.7 to 1.9 mg/L from Wallarah Seam and Great Northern Seam Interburden samples 14460, 14464 and 14465.

Results confirm the acid and metal/metalloid leaching potential of the tailings observed close to spigots in TD2 due to pyrite oxidation effects, with white salts observed elsewhere producing circum-neutral leachates but acting as a potential source of salinity and possibly dissolved Mn. Results also show that most of the mine materials are likely to produce alkaline leachate, helping to offset the expected localised acidity produced by PAF tailings.

Although the in situ shale was confirmed as a potential source of acid and dissolved metals/metalloids, geologists advise this unit is isolated, and the bulk of the overburden/interburden mined is expected to be similar to the core and dump grab samples collected, which are benign and slightly to moderately alkaline leaching.

## 6.8 Distribution of Geochemical Rock Types and Potential Effects on the Receiving Environment

The results and discussions presented above were used to resolve uncertainties in results and classify samples as NAF and PAF in Table A1.

Figure 14 is a plot of the spatial distribution of S for tailings collected from TD1 and TD2, which shows a clear clustering of higher S values close to the spigot points, confirming the preferential deposition and concentration of pyrite due to gravity separation. Results suggest the vast majority of the tailings deposited in TD1 and TD2 will have low S less than 0.7%S. Figure 15 is a plot of the ARD classification distribution for the tailings, which shows that tailings materials in TD2 tend to be classified NAF away from the spigot, with the high S PAF materials restricted to within 50m of the spigot. There are some PAF tailings away from the spigot in TD2, but they have relatively low S. Note that although the TD1 samples are all NAF, they have elevated S close to the spigot, and would be a local source of sulphate salinity. These higher S materials are restricted in areal extent, and will act as only a localised source of acid and sulphate salinity. Water extracts show that lower S tailings materials will be a source of salinity, but this is chloride dominated, and appears to reflect the presence of residual process water. Water extracts also show that NAF tailings are a source of moderate alkalinity.

The long reach excavations close to spigot sites confirmed elevated pyrite persist at depth, but depletion of ANC is common at surface, which would be expected from acid generation effects as the oxidation front progresses from surface.

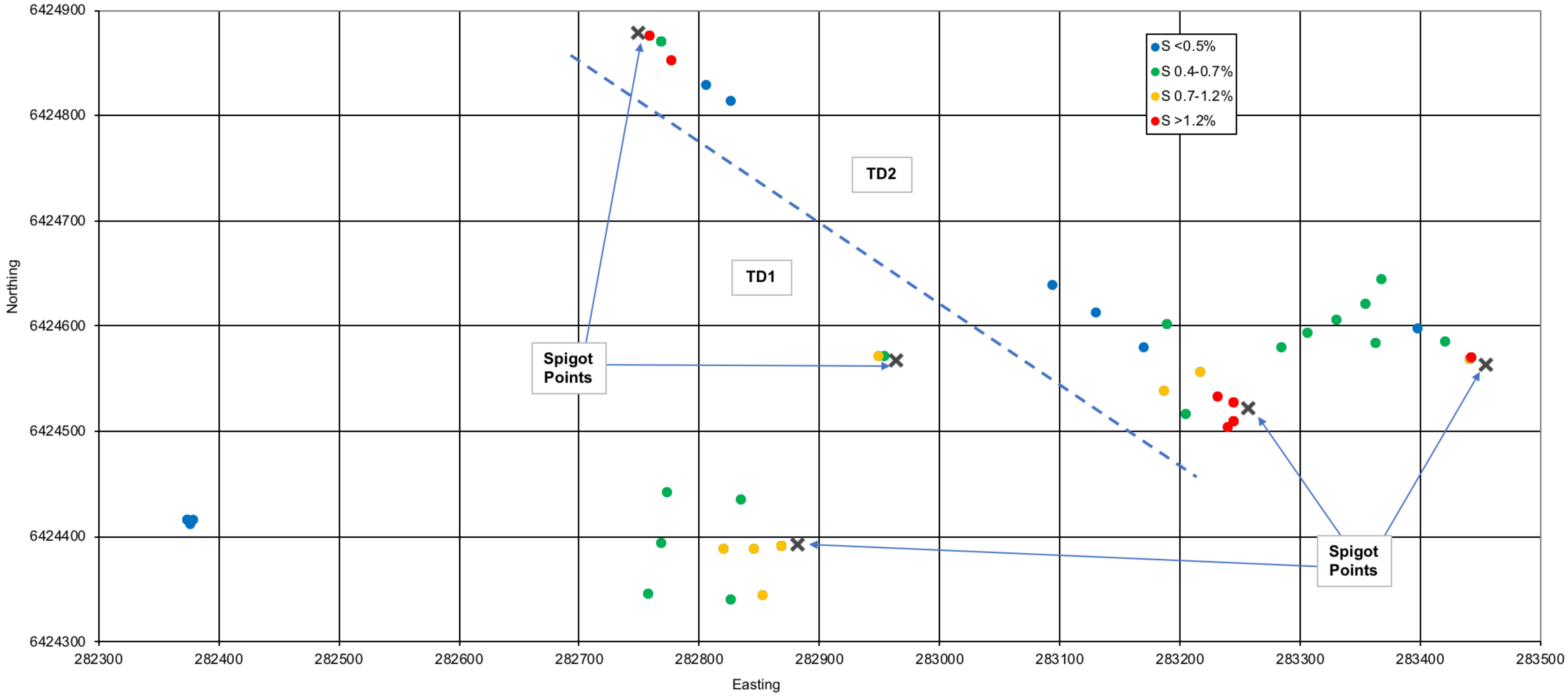

Figure 14: Spatial distribution of S for tailings collected from TD1 and TD2.

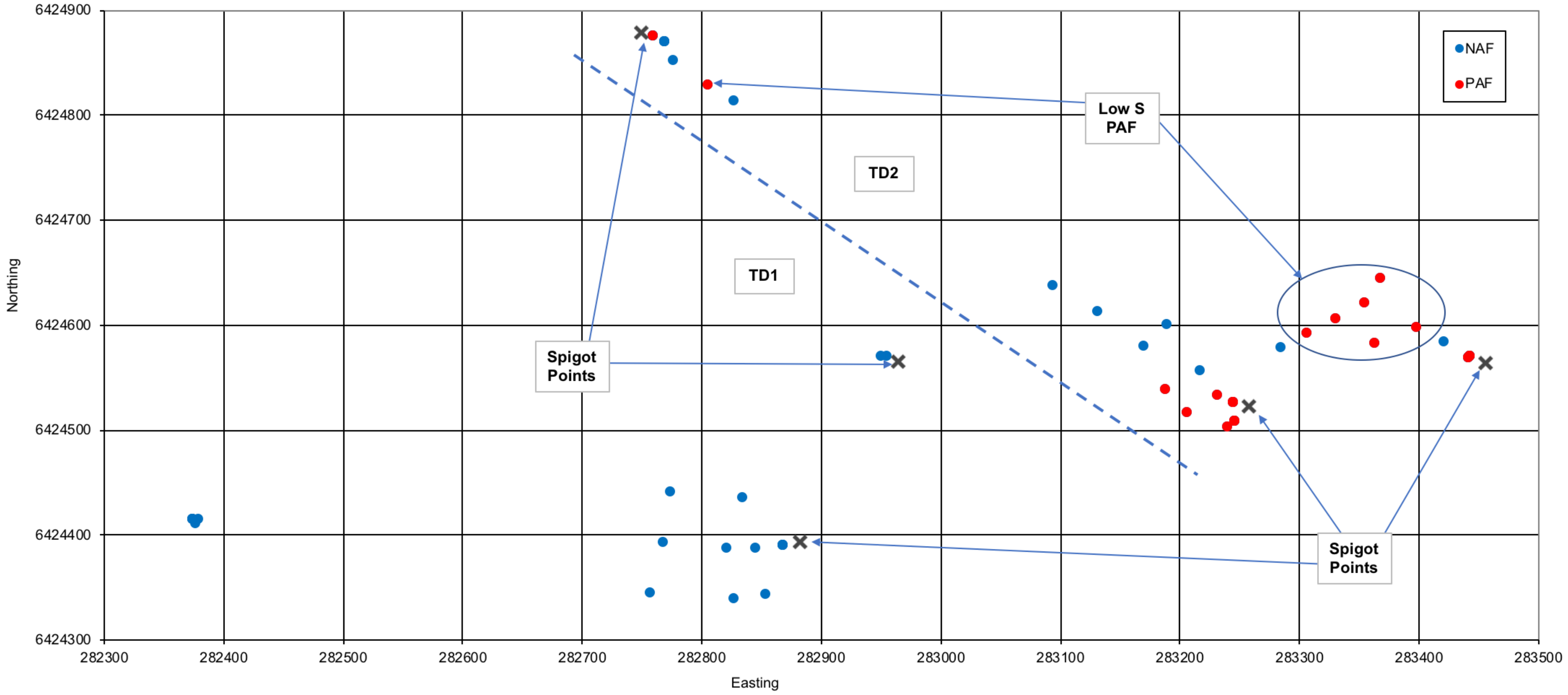

Figure 15: Spatial distribution of ARD classification for tailings collected from TD1 and TD2.

Results confirm that the bulk of the tailings from TD1 and TD2 are likely to be NAF with associated elevated chloride salinities from residual process water, and moderate alkalinities. Higher S PAF tailings are restricted to localised zones close to spigot locations due to gravity segregation effects, and would not be expected to be a significant source of acid seepage due to the background alkalinities from surrounding NAF tailings. Overall, the tailings at TD1 and TD2 are primarily a chloride salinity source, with only minor influence from ARD effects. A similar spatial distribution of tailings geochemistry can be expected from coal washing for the MCCO Project, which will involve mining and processing of the same seams.

All 5 rejects samples and all except one of the overburden/interburden samples were classified NAF, with low S, none to slight salinity, and low to moderate alkalinity. Results confirmed the in situ jarositic shale sample was PAF, but this unit is understood to represent a spatially isolated unit above the Fassifern Seam, and would not be expected to have any significicate effect on seepage water quality from overburden/interburden dumps due to limited occurrence. Results indicate that overburden/interburden and coarse rejects materials represented by the materials tested are benign, and hence pit back fill and ex-pit emplacement constructed as part of the MCCO Project are not expected to be a source of salinity or ARD. Conversely, these materials will produce excess alkalinity, providing an additional factor of safety.

Water quality investigations for the current and previous (Anvil Hill Coal Project) approvals showed that background groundwater quality was saline with a circum-neutral pH[3, 4]. The coal seam aquifers were identified as the key salinity source, causing high salinities in drainage upstream of the mine and also in pit drainage. Mine pit seepage quality for the current operations was predicted to be saline at an average of 5 dS/m with a dominance by NaCl, primarily due groundwater originating from the coal seams[4].

Figures A8 to A12, Appendix A, show water quality trends (pH, alkalinity, EC, Cl and $SO_4$) for key pits and tailings storages across site, comprising:

- decant pond draining TD1 and TD2 – 3 sampling events late 2017
- decant pond draining TD3 and TD4 – 4 sampling events late 2017
- Main Pit North Dam – 16 sampling events 2014 to 2015
- Main Pit West Dam – 42 sampling events 2015 to 2017
- Pit Water Dam – 94 sampling events 2011 to 2017

Results are consistent with expectations of a strong groundwater influence as mentioned above, with the pH circum-neutral and high EC dominated by Cl salinity with lesser $SO_4$ salinity. In addition, results show a high excess alkalinity typically greater than 200 mg $CaCO_3$/L. Results indicate that minor inputs of ARD from pyritic tailings are unlikely to significantly affect the downstream surface water pH given the background alkalinity and the lack of any current pH

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[3] Mackie Environmental Research, 2015. *Mangoola Coal, Mangoola North Pre-Feasibility Study, Regional Groundwater Model*, June 2015.

[4] Mackie Environmental Research, 2006. *Centennial Hunter Pty Limited, Anvil Hill Project: Groundwater Management Studies*, May 2006.

effects on the tailings decant ponds. The salinity of the tailings is likewise unlikely to affect the overall water quality, given the already high salinity inputs from the coal seam aquifers. The current concentrations of dissolved metals/metalloids in the tailings decant sites are low, indicating that although significant concentrations of metals and metalloids were released in water extracts from the strongly pyritic tailings (Section 6.7), this has not had a significant effect after dilution and mixing with background water.

Overall, background water quality is poor, and the localised pyritic tailings identified to date are not expected to have a significant impact on final water quality, or require modification of the current saline water management.

Although the tailings do not appear to represent a concern in terms of pit water quality impacts, there could be local effects from both acid and salinity, particularly in regard to success of final rehabilitation of the tailings dams. It is understood that the final cover system for the tailings had not been finalised, but conceptually a minimum 3m cover of clean overburden is currently planned. More detailed design considerations will be required to confirm the acid and salinity in the tailings does not migrate into the growth horizon and affect rehabilitation. An understanding of the physical and hydrological properties of the tailings and potential cover materials would be required, followed by water flux modelling of cover system options.

# 7.0 Conclusions and Recommendations

Visual inspection and testing of materials from current operations and results of testing MCCO Project core indicate that overburden/interburden and coarse rejects materials placed in pit back fill and ex-pit emplacement constructed as part of the MCCO Project are likely to be NAF and non saline. They are also expected to be alkalinity producing, providing an additional factor of safety.

Results of testing tailings from TD1 and TD2 as a proxy for future tailings geochemical compositions also indicate that the bulk of the tailings from the MCCO Project are likely to be NAF with associated elevated chloride salinities from residual process water, and moderate alkalinities. Although there are likely to be higher S PAF tailings in localised zones close to spigot locations (due to gravity segregation effects), these would not be expected to be a significant source of acid seepage due to the background alkalinities from surrounding NAF tailings, and produce only a minor contribution of sulphate salinity relative to the inherent chloride salinities present in the process water.

The local background water quality is circum-neutral and saline with elevated alkalinity, and results to date indicate the proposed MCCO Project is not likely to have a significant impact on pit water quality, or require modification of the current saline water management.

However, the tailings may produce local effects on final cover layers from both acid and salinity, potentially compromising rehabilitation success without appropriate controls in place. The current conceptual cover for the tailings involves placement of a minimum 3m cover of clean overburden. More detailed design considerations will be required to confirm the acid and salinity in the tailings does not migrate into the growth horizon and affect rehabilitation.

The following is recommended for the MCCO Project during operations:

- Carry out visual inspection of any further core drilling in the MCCO Project area for evidence of pyrite occurrence to confirm the strong dominance of NAF overburden/interburden across the deposit.
- Include routine monthly monitoring of pH, EC and acidity/alkalinity from tailings dam discharges and/or decant ponds, with analysis of key ARD indicators including dissolved SO4, Ca, Mg, K, Na, Cl, Al, As, Co, Cr, Cu, Fe, Mn, Ni and Zn. Results should be reviewed after 12 months and checked for any evidence of ARD development, and compared to the quality of representative receiving surface water and groundwater to identify any potential impacts and determine if further ongoing monitoring is required.
- Arrange for total S testing to be carried out during regular testing of washery waste samples to check for consistency with results to date. Preserve samples and review in 6 months to determine if more detailed work is required and assess the need for continuation of S testing.
- Prior to capping of the tailings dams prepare a detailed plan for the cover design system. The design should provide for adequate control of upward migration of salts and acid so that rehabilitation efforts are not compromised. The detailed cover design should consider:
    - Physical and hydrological characterisation of the existing tailings (TD1 and TD2) and potential cover materials, and water flux modelling of cover system options to assess performance. The proposed 3m clean overburden cover may be sufficient, but this should be confirmed.
    - The need for incorporation of crushed (agricultural) limestone into the final tailings surface in the zone radiating approximately 100m from the spigot point to ensure neutralisation of any existing acidity.

# APPENDIX A

## Geochemical Testwork Results

Table A1: Acid forming characteristics of grab samples collected during the site visit.

| Sample ID | Sample Type | Site/Hole ID | Location and Description | Comments | Depth | EGi Sample Number | pH$_{1:2}$ | EC$_{1:2}$ | ACID-BASE ANALYSIS | | | | | SINGLE ADDITION NAG | | | ARD Classification | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | Total %S | MPA | ANC | NAPP | ANC/MPA | NAGpH | NAG$_{(pH4.5)}$ | NAG$_{(pH7.0)}$ | | |
| Sample 1 | Tailings | TD2 | TD2W - No salt | | | 12426 | 7.2 | 6.78 | 0.31 | 9 | 18 | -9 | 1.90 | 2.4 | 49 | 88 | NAF | 🟦 |
| Sample 2 | Tailings | TD2 | TD2W - No salt | | | 12427 | 7.5 | 5.13 | 0.35 | 11 | 19 | -8 | 1.77 | 2.5 | 45 | 80 | NAF | 🟦 |
| Sample 3 | Tailings | TD2 | TD2S - White salts | | | 12428 | 7.6 | 10.76 | 1.18 | 36 | 31 | 5 | 0.86 | 6.9 | 0 | 0 | NAF | 🟦 |
| Sample 4 | Tailings | TD2 | TD2S - White salts | | | 12429 | 6.8 | 15.61 | 1.58 | 48 | 49 | -1 | 1.01 | 3.1 | 8 | 39 | PAF | 🟥 |
| Sample 6 | Tailings | TD2 | TD2E - Yellow and white salts. Profile sample. | | | 12431 | 2.9 | 2.29 | 1.79 | 55 | 0 | 55 | 0.00 | 2.1 | 69 | 113 | PAF | 🟥 |
| Sample 7 | Tailings | TD2 | TD2E - Yellow and white salts. Selective sample of high salt layers | | | 12432 | 2.1 | 7.83 | 5.03 | 154 | 0 | 154 | 0.00 | 1.9 | 88 | 109 | PAF | 🟥 |
| Sample 8 | Tailings | TD1 | TD1E - Minor white salt | | | 12433 | 7.7 | 6.11 | 0.70 | 21 | 66 | -45 | 3.08 | 7.3 | 0 | 0 | NAF | 🟦 |
| Sample 9 | Tailings | TD1 | TD1E - Minor white salt | | | 12434 | 7.6 | 9.12 | 0.74 | 23 | 75 | -52 | 3.31 | 7.3 | 0 | 0 | NAF | 🟦 |
| Sample 10 | Tailings | TD1 | TD1W - No salt | | | 12435 | 7.4 | 2.34 | 0.31 | 9 | 21 | -12 | 2.21 | 2.5 | 48 | 82 | NAF | 🟦 |
| Sample 11 | Tailings | TD1 | TD1W - No salt | | | 12436 | 7.6 | 7.09 | 0.27 | 8 | 19 | -11 | 2.30 | 2.5 | 50 | 83 | NAF | 🟦 |
| 1 | Tailings | TD2 | TD2E Excavation Site 1: 3m Deep | Layers of pyrite evident throughout | 0.0-0.5m | 14466 | 2.7 | 2.65 | 0.99 | 30 | 0 | 30 | 0.00 | 2.0 | 113 | 170 | PAF | 🟥 |
| 2 | Tailings | TD2 | TD2E Excavation Site 1: 3m Deep | Layers of pyrite evident throughout | 0.5-1.0m | 14467 | 6.8 | 0.98 | 0.73 | 22 | 28 | -5 | 1.24 | 4.5 | 0 | 35 | UC(NAF) | 🟦 |
| 3 | Tailings | TD2 | TD2E Excavation Site 1: 3m Deep | Layers of pyrite evident throughout | 1.0-2.0m | 14468 | 6.7 | 1.21 | 0.93 | 28 | 54 | -26 | 1.90 | 5.6 | 0 | 4 | UC(NAF) | 🟦 |
| 4 | Tailings | TD2 | TD2E Excavation Site 1: 3m Deep | Layers of pyrite evident throughout. Base hit at 3m | 2.0-3.0m | 14469 | 7.1 | 1.07 | 0.79 | 24 | 33 | -9 | 1.35 | 4.7 | 0 | 25 | UC(NAF) | 🟦 |
| 5 | Tailings | TD2 | TD2E Excavation Site 2: 3m Deep | Less pyritic layers than site 1 | 0.0-0.5m | 14470 | 2.6 | 3.22 | 1.05 | 32 | 0 | 32 | 0.00 | 2.1 | 95 | 143 | PAF | 🟥 |
| 6 | Tailings | TD2 | TD2E Excavation Site 2: 3m Deep | Less pyritic layers than site 1 | 0.5-1.0m | 14471 | 3.9 | 1.96 | 0.52 | 16 | 6 | 10 | 0.35 | 2.2 | 91 | 140 | PAF | 🟥 |
| 7 | Tailings | TD2 | TD2E Excavation Site 2: 3m Deep | Less pyritic layers than site 1 | 1.0-2.0m | 14472 | 6.8 | 1.21 | 0.62 | 19 | 44 | -25 | 2.32 | 5.2 | 0 | 9 | NAF | 🟦 |
| 8 | Tailings | TD2 | TD2E Excavation Site 2: 3m Deep | Less pyritic layers than site 1 | 2.0-3.0m | 14473 | 7.1 | 1.04 | 0.70 | 21 | 104 | -83 | 4.86 | 8.1 | 0 | 0 | NAF | 🟦 |
| 9 | Tailings | TD2 | TD2E Surface Grab Sample | 25m from first 2 trenches. | 0.0-0.5m | 14474 | 7.3 | 1.26 | 0.54 | 17 | 23 | -6 | 1.38 | 2.7 | 38 | 76 | UC(NAF) | 🟦 |
| 10 | Tailings | TD2 | TD2E Surface Grab Sample | 50m from first 2 trenches. | 0.0-0.5m | 14475 | 5.4 | 2.66 | 0.32 | 10 | 7 | 2 | 0.76 | 2.3 | 78 | 126 | UC(PAF) | 🟥 |
| 11 | Tailings | TD2 | TD2E Surface Grab Sample | 100m from first 2 trenches. | 0.0-0.5m | 14476 | 5.8 | 1.45 | 0.53 | 16 | 9 | 7 | 0.55 | 2.2 | 68 | 114 | UC(PAF) | 🟥 |
| 12 | Tailings | TD2 | TD2E Surface Grab Sample | 25mE at right angles to Bag 11 | 0.0-0.5m | 14477 | 5.9 | 1.23 | 0.55 | 17 | 8 | 8 | 0.50 | 2.4 | 27 | 62 | UC(PAF) | 🟥 |
| 13 | Tailings | TD2 | TD2E Surface Grab Sample | 25mW at right angles to Bag 11 | 0.0-0.5m | 14478 | 6.7 | 0.98 | 0.56 | 17 | 12 | 5 | 0.71 | 2.2 | 75 | 120 | UC(PAF) | 🟥 |
| 14 | Tailings | TD2 | TD2E Surface Grab Sample | 50mW at right angles to Bag 11 | 0.0-0.5m | 14479 | 7.2 | 0.89 | 0.63 | 19 | 8 | 11 | 0.43 | 2.7 | 12 | 35 | PAF | 🟥 |
| 15 | Tailings | TD2 | TD2E Surface Grab Sample | 75m at right angles to Bag 11, White salts photo. | 0.0-0.5m | 14480 | 5.5 | 2.88 | 0.63 | 19 | 19 | 0 | 1.00 | 2.2 | 92 | 151 | UC(NAF) | 🟦 |
| 16 | Tailings | TD2 | TD2E Surface Grab Sample | | 0.0-0.5m | 14481 | 5.9 | 2.36 | 0.42 | 13 | 9 | 4 | 0.72 | 2.2 | 87 | 137 | UC(PAF) | 🟥 |
| 17 | Tailings | TD2 | TD2S Excavation Site 1: 2.5m Deep | White salts and iron staining nearby. | 0.0-0.5m | 14482 | 5.8 | 1.32 | 1.19 | 36 | 61 | -24 | 1.67 | 6.5 | 0 | 1 | UC(NAF) | 🟦 |
| 18 | Tailings | TD2 | TD2S Excavation Site 1: 2.5m Deep | Pyrite layer | 0.5-1.0m | 14483 | 5.7 | 1.56 | 1.28 | 39 | 13 | 26 | 0.34 | 4.1 | 2 | 29 | PAF | 🟥 |
| 19 | Tailings | TD2 | TD2S Excavation Site 1: 2.5m Deep | Pyrite Layer | 1.0-2.0m | 14484 | 6.3 | 1.36 | 2.39 | 73 | 38 | 36 | 0.51 | 4.3 | 3 | 37 | PAF | 🟥 |
| 20 | Tailings | TD2 | TD2S Excavation Site 1: 2.5m Deep | Base hit at approximately 2.5m. | 2.0-2.5m | 14485 | 7.3 | 1.21 | 1.39 | 43 | 18 | 25 | 0.42 | 3.5 | 2 | 29 | UC(PAF) | 🟥 |
| 21 | Tailings | TD2 | TD2S Excavation Site 2: 3m Deep | Some white salts, some yellow | 0.0-0.5m | 14486 | 6.8 | 1.24 | 1.28 | 39 | 38 | 2 | 0.96 | 5.0 | 0 | 11 | UC(PAF) | 🟥 |
| 22 | Tailings | TD2 | TD2S Excavation Site 2: 3m Deep | No salts. | 0.5-1.0m | 14487 | 7.4 | 0.98 | 0.87 | 27 | 7 | 19 | 0.28 | 3.5 | 3 | 33 | PAF | 🟥 |
| 23 | Tailings | TD2 | TD2S Excavation Site 2: 3m Deep | No salts. | 1.0-2.0m | 14488 | 6.6 | 1.35 | 1.00 | 31 | 31 | -1 | 1.02 | 4.6 | 0 | 28 | UC(NAF) | 🟦 |
| 24 | Tailings | TD2 | TD2S Excavation Site 2: 3m Deep | | 2.0-3.0m | 14489 | 6.5 | 1.27 | 1.32 | 40 | 33 | 8 | 0.81 | 4.4 | 1 | 27 | PAF | 🟥 |
| 25 | Tailings | TD2 | TD2S Surface Grab Sample | Yellow, sandy crust along side in drainage line. | 0.0-0.5m | 14490 | 6.1 | 1.69 | 1.50 | 46 | 18 | 28 | 0.39 | 3.7 | 1 | 32 | UC(PAF) | 🟥 |
| 26 | Tailings | TD2 | TD2S Surface Grab Sample | | 0.0-0.5m | 14491 | 7.3 | 1.02 | 0.81 | 25 | 32 | -7 | 1.28 | 4.6 | 0 | 23 | UC(NAF) | 🟦 |
| 27 | Tailings | TD2 | TD2S Surface Grab Sample | | 0.0-0.5m | 14492 | 7.4 | 1.11 | 0.44 | 13 | 14 | 0 | 1.02 | 2.5 | 58 | 106 | NAF | 🟦 |
| 28 | Tailings | TD2 | TD2S Surface Grab Sample | | 0.0-0.5m | 14493 | 6.8 | 1.26 | 0.28 | 9 | 18 | -9 | 2.07 | 4.3 | 1 | 21 | UC(NAF) | 🟦 |
| 29 | Tailings | TD2 | TD2S Surface Grab Sample | | 0.0-0.5m | 14494 | 6.7 | 1.23 | 0.18 | 6 | 12 | -7 | 2.18 | 2.8 | 27 | 60 | UC(NAF) | 🟦 |
| 30 | Tailings | TD2 | TD2S Surface Grab Sample | Damp, soft, very fine. | 0.0-0.5m | 14495 | 7.2 | 2.22 | 0.25 | 8 | 14 | -6 | 1.81 | 2.3 | 68 | 106 | UC(NAF) | 🟦 |
| 31 | Tailings | TD2 | TD2S Surface Grab Sample | Dry, salts nearby | 0.0-0.5m | 14496 | 7.5 | 1.42 | 1.05 | 32 | 22 | 10 | 0.69 | 4.4 | 1 | 16 | UC(PAF) | 🟥 |
| 32 | Tailings | TD2 | TD2S Surface Grab Sample | Slightly damp, white salts | 0.0-0.5m | 14497 | 7.3 | 1.22 | 0.65 | 20 | 17 | 3 | 0.85 | 4.6 | 0 | 23 | UC(PAF) | 🟥 |
| 33 | Tailings | TD2 | TD2W Excavation Site 1: 2m Deep | No salts. | 0.0-0.5m | 14498 | 6.6 | 2.76 | 0.29 | 9 | 16 | -7 | 1.78 | 2.3 | 118 | 179 | NAF | 🟦 |
| 34 | Tailings | TD2 | TD2W Excavation Site 1: 2m Deep | No salts - Damp | 0.5-1.0m | 14499 | 7.6 | 1.51 | 0.39 | 12 | 58 | -46 | 4.84 | 7.2 | 0 | 0 | NAF | 🟦 |
| 35 | Tailings | TD2 | TD2W Excavation Site 1: 2m Deep | No salts - wet. | 1.0-2.0m | 14500 | 7.5 | 1.46 | 0.56 | 17 | 92 | -75 | 5.38 | 8.0 | 0 | 0 | NAF | 🟦 |
| 36 | Tailings | TD2 | TD2W Excavation Site 1: 1m Deep | Coarse Zone with Jarosite layers. | 0.0-0.5m | 14501 | 6.6 | 3.11 | 2.09 | 64 | 42 | 22 | 0.65 | 4.5 | 0 | 24 | UC(PAF) | 🟥 |
| 37 | Tailings | TD2 | TD2W Excavation Site 1: 1m Deep | Coarse Zone with some jarosite - fine and wet at base. | 0.5-1.0m | 14502 | 7.3 | 1.00 | 0.89 | 27 | 58 | -31 | 2.14 | 8.0 | 0 | 0 | NAF | 🟦 |
| 38 | Tailings | TD2 | TD2W Surface Grab Sample | Deep cracks, no salts. | 0.0-0.5m | 14503 | 7.2 | 1.32 | 0.36 | 11 | 19 | -8 | 1.73 | 2.7 | 49 | 93 | NAF | 🟦 |
| 39 | Tailings | TD2 | TD2W Surface Grab Sample | No salts. | 0.0-0.5m | 14504 | 7.2 | 1.36 | 0.35 | 11 | 6 | 5 | 0.53 | 2.3 | 60 | 100 | UC(PAF) | 🟥 |
| 40 | Tailings | TD2 | TD2W Surface Grab Sample | Fine. | 0.0-0.5m | 14505 | 7.3 | 1.65 | 1.57 | 48 | 85 | -37 | 1.78 | 7.7 | 0 | 0 | UC(NAF) | 🟦 |
| 41 | Tailings | TD1 | TD1W Excavation Site 1: 3m Deep | Decant end, no salts, lots of desiccation | 0.0-0.5m | 14506 | 6.8 | 3.46 | 0.29 | 9 | 14 | -5 | 1.58 | 2.5 | 57 | 93 | UC(NAF) | 🟦 |
| 42 | Tailings | TD1 | TD1W Excavation Site 1: 3m Deep | Decant end, no salts, lots of desiccation | 0.5-1.0m | 14507 | 6.7 | 2.45 | 0.22 | 7 | 27 | -20 | 4.04 | 3.2 | 32 | 66 | UC(NAF) | 🟦 |
| 43 | Tailings | TD1 | TD1W Excavation Site 1: 3m Deep | Decant end, no salts, lots of desiccation | 1.0-2.0m | 14508 | 7.2 | 1.96 | 0.25 | 8 | 28 | -20 | 3.65 | 3.2 | 34 | 70 | UC(NAF) | 🟦 |
| 44 | Tailings | TD1 | TD1W Excavation Site 1: 3m Deep | Decant end, no salts, lots of desiccation | 2.0-3.0m | 14509 | 6.8 | 1.52 | 0.19 | 6 | 12 | -7 | 2.13 | 2.6 | 58 | 103 | UC(NAF) | 🟦 |
| 45 | Tailings | TD1 | TD1E Excavation Site 1: 3m Deep | No salts | 0.0-0.5m | 14510 | 7.3 | 3.78 | 1.00 | 31 | 57 | -26 | 1.87 | 7.0 | 0 | 0 | UC(NAF) | 🟦 |
| 46 | Tailings | TD1 | TD1E Excavation Site 1: 3m Deep | No salts. | 0.5-1.0m | 14511 | 7.5 | 0.99 | 0.70 | 21 | 52 | -31 | 2.43 | 6.0 | 0 | 1 | NAF | 🟦 |
| 47 | Tailings | TD1 | TD1E Excavation Site 1: 3m Deep | No salts. | 1.0-2.0m | 14512 | 7.4 | 0.98 | 0.61 | 19 | 42 | -23 | 2.24 | 5.7 | 0 | 2 | NAF | 🟦 |
| 48 | Tailings | TD1 | TD1E Excavation Site 1: 3m Deep | No salts. Hit bottom. | 2.0-3.0m | 14513 | 6.8 | 2.71 | 1.00 | 31 | 58 | -27 | 1.90 | 4.5 | 0 | 16 | UC(NAF) | 🟦 |
| 49 | Tailings | TD1 | TD1E Surface Grab Sample | No salts. 25m | 0.0-0.5m | 14514 | 7.3 | 1.46 | 0.71 | 22 | 62 | -40 | 2.85 | 7.6 | 0 | 0 | NAF | 🟦 |
| 50 | Tailings | TD1 | TD1E Surface Grab Sample | 50m | 0.0-0.5m | 14515 | 7.2 | 1.55 | 0.72 | 22 | 61 | -39 | 2.78 | 7.6 | 0 | 0 | NAF | 🟦 |
| 51 | Tailings | TD1 | TD1E Surface Grab Sample | 100m | 0.0-0.5m | 14516 | 7.5 | 1.63 | 0.58 | 18 | 55 | -38 | 3.12 | 7.3 | 0 | 0 | NAF | 🟦 |
| 52 | Tailings | TD1 | TD1E Surface Grab Sample | | 0.0-0.5m | 14517 | 7.3 | 2.03 | 0.47 | 14 | 34 | -20 | 2.38 | 3.3 | 21 | 54 | UC(NAF) | 🟦 |
| 53 | Tailings | TD1 | TD1E Surface Grab Sample | | 0.0-0.5m | 14518 | 7.2 | 2.14 | 0.46 | 14 | 35 | -21 | 2.48 | 4.0 | 4 | 26 | NAF | 🟦 |
| 54 | Tailings | TD1 | TD1E Surface Grab Sample | | 0.0-0.5m | 14519 | 7.1 | 2.11 | 0.58 | 18 | 48 | -30 | 2.71 | 3.2 | 24 | 64 | NAF | 🟦 |
| 55 | Tailings | TD1 | TD1E Surface Grab Sample | | 0.0-0.5m | 14520 | 7.5 | 0.91 | 0.51 | 16 | 34 | -19 | 2.20 | 3.4 | 19 | 54 | UC(NAF) | 🟦 |
| 56 | Tailings | TD1 | TD1E Surface Grab Sample | | 0.0-0.5m | 14521 | 7.6 | 0.99 | 0.85 | 26 | 64 | -38 | 2.45 | 7.6 | 0 | 0 | NAF | 🟦 |
| Sample 12 | Coarse Rejects | South Pit | Dumped Rejects Grab Sample | | | 12437 | 8.0 | 0.28 | 0.10 | 3 | 16 | -13 | 5.23 | 6.9 | 0 | 0 | NAF | 🟦 |
| Sample 13 | Coarse Rejects | South Pit | Dumped Rejects Grab Sample | | | 12438 | 8.3 | 0.25 | 0.14 | 4 | 20 | -16 | 4.67 | 3.5 | 7 | 22 | NAF | 🟦 |
| Sample 14 | Coarse Rejects | South Pit | Dumped Rejects Grab Sample | | | 12439 | 8.1 | 0.29 | 0.15 | 5 | 12 | -7 | 2.61 | 3.9 | 4 | 26 | NAF | 🟦 |
| Sample 15 | Coarse Rejects | Main Pit East | Dumped Rejects Grab Sample | | | 12440 | 7.9 | 0.35 | 0.12 | 4 | 27 | -23 | 7.35 | 4.5 | 0 | 9 | NAF | 🟦 |
| Sample 16 | Coarse Rejects | Main Pit East | Dumped Rejects Grab Sample | | | 12441 | 8.1 | 0.34 | 0.13 | 4 | 34 | -30 | 8.55 | 4.5 | 0 | 10 | NAF | 🟦 |
| Sample 5 | Overburden/Interburden | TD2 Outcrop | Grab Sample of In Situ Jarositic Shale TD 2 NE corner | Great Northern Seam and Fassifern Interburden | | 12430 | 3.5 | 1.79 | 0.43 | 13 | 0 | 13 | 0.00 | 2.9 | 5 | 11 | PAF | 🟥 |

Table A1: Acid forming characteristics of grab samples collected during the site visit.

| Sample ID | Sample Type | Site/Hole ID | Location and Description | Comments | Depth | EGi Sample Number | pH$_{1:2}$ | EC$_{1:2}$ | ACID-BASE ANALYSIS | | | | | SINGLE ADDITION NAG | | | ARD Classification | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | Total %S | MPA | ANC | NAPP | ANC/MPA | NAGpH | NAG$_{(pH4.5)}$ | NAG$_{(pH7.0)}$ | | |
| 57 | Overburden/Interburden | MN0006 | Wallarah Seam and Great Northern Seam Interburden Core | Weathered Conglomerate | 9.30-12.35m | 14438 | 7.1 | 0.58 | 0.01 | 0 | 1 | -1 | 4.67 | 5.7 | 0 | 4 | NAF | 🟦 |
| 58 | Overburden/Interburden | MN0003 | Wallarah Seam Overburden Core | Composite sample. Fresh conglomerate | 16.00-17.00m, 20.00-20.73m | 14439 | 7.3 | 0.42 | 0.01 | 0 | 20 | -20 | 64.77 | 7.9 | 0 | 0 | NAF | 🟦 |
| 59 | Overburden/Interburden | MN0003 | Wallarah Seam Overburden Core | Fresh Claystone | 20.73-22.45m | 14440 | 7.5 | 0.39 | 0.01 | 0 | 13 | -12 | 41.10 | 7.8 | 0 | 0 | NAF | 🟦 |
| 60 | Overburden/Interburden | MN0003 | Wallarah Seam Overburden Core | Fresh Sandstone | 22.45-23.76m | 14441 | 7.4 | 0.30 | 0.01 | 0 | 14 | -14 | 46.82 | 7.9 | 0 | 0 | NAF | 🟦 |
| 61 | Overburden/Interburden | MN0003 | Wallarah Seam Overburden Core | Composite sample. Fresh conglomerate | 23.76-25.00m, 32.00-33.00m, 39.00-40.00m | 14442 | 7.2 | 0.40 | 0.03 | 1 | 23 | -22 | 24.69 | 7.9 | 0 | 0 | NAF | 🟦 |
| 62 | Overburden/Interburden | MN0001 | Wallarah Seam Overburden Core | Composite sample. Fresh conglomerate | 49.00-50.00m, 61.00-62.00m, 73.00-74.00m | 14443 | 7.5 | 0.38 | 0.01 | 0 | 10 | -9 | 31.78 | 7.8 | 0 | 0 | NAF | 🟦 |
| 63 | Overburden/Interburden | MN0001 | Wallarah Seam Overburden Core | Fresh Claystone | 75.12-75.54m | 14444 | 7.7 | 0.37 | 0.01 | 0 | 16 | -15 | 51.49 | 7.9 | 0 | 0 | NAF | 🟦 |
| 64 | Overburden/Interburden | MN0001 | Wallarah Seam Overburden Core | Fresh Sandstone | 75.54-76.00m | 14445 | 7.5 | 0.42 | 0.01 | 0 | 18 | -18 | 58.68 | 7.8 | 0 | 0 | NAF | 🟦 |
| 65 | Overburden/Interburden | MN0001 | Wallarah Seam Overburden Core | Composite sample. Fresh conglomerate | 76.00-77.00m, 80.00-81.00m, 91.00-92.00m | 14446 | 7.6 | 0.29 | 0.02 | 1 | 14 | -14 | 23.15 | 8.1 | 0 | 0 | NAF | 🟦 |
| 66 | Overburden/Interburden | MN0001 | Wallarah Seam and Great Northern Seam Interburden Core | Fresh Carbonaceous MS | 95.30-95.91m | 14447 | 8.3 | 0.24 | 0.02 | 1 | 4 | -4 | 7.33 | 3.3 | 5 | 21 | NAF | 🟦 |
| 67 | Overburden/Interburden | MN0001 | Wallarah Seam and Great Northern Seam Interburden Core | Fresh Sandstone | 95.91-98.22m | 14448 | 7.8 | 0.32 | 0.02 | 1 | 22 | -22 | 36.68 | 7.9 | 0 | 0 | NAF | 🟦 |
| 68 | Overburden/Interburden | MN0001 | Wallarah Seam and Great Northern Seam Interburden Core | Composite sample. Fresh conglomerate | 99.00-100.00m, 110.00-111.00m, 123.00-123.87m | 14449 | 7.7 | 0.34 | 0.02 | 1 | 15 | -14 | 23.96 | 7.9 | 0 | 0 | NAF | 🟦 |
| 69 | Overburden/Interburden | MN0003 | Wallarah Seam and Great Northern Seam Interburden Core | Fresh Claystone | 42.65-44.06m | 14450 | 7.9 | 0.33 | 0.02 | 1 | 12 | -11 | 19.75 | 7.4 | 0 | 0 | NAF | 🟦 |
| 70 | Overburden/Interburden | MN0003 | Wallarah Seam and Great Northern Seam Interburden Core | Fresh Sandstone | 44.06-45.32m | 14451 | 8.2 | 0.42 | 0.06 | 2 | 40 | -38 | 21.79 | 7.7 | 0 | 0 | NAF | 🟦 |
| 71 | Overburden/Interburden | MN0003 | Wallarah Seam and Great Northern Seam Interburden Core | Composite sample. Fresh conglomerate | 46.00-47.00m, 55.00-56.00m, 62.00-63.00m | 14452 | 7.5 | 0.40 | 0.02 | 1 | 30 | -30 | 49.25 | 8.1 | 0 | 0 | NAF | 🟦 |
| 72 | Overburden/Interburden | MN0006 | Great Northern Seam and Fassifern Seam Interburden | Awaba Tuff, Red Oxidised. | 16.62-19.48m | 14453 | 7.6 | 0.37 | 0.01 | 0 | 11 | -10 | 35.25 | 8.3 | 0 | 0 | NAF | 🟦 |
| 73 | Overburden/Interburden | MN0003 | Great Northern Seam and Fassifern Seam Interburden | Awaba Tuff, all there | 67.33-69.75m | 14454 | 7.7 | 0.44 | 0.01 | 0 | 15 | -15 | 49.55 | 7.3 | 0 | 0 | NAF | 🟦 |
| 74 | Overburden/Interburden | MN0001 | Great Northern Seam and Fassifern Seam Interburden | Awaba Tuff, Carbonceous interval missing, end portion missing | 127.63-130.30m | 14455 | 7.5 | 0.47 | 0.01 | 0 | 34 | -33 | 219.55 | 7.6 | 0 | 0 | NAF | 🟦 |
| 75 | Overburden/Interburden | MN0006 | Upper Pilot A Floor | Part missing | 28.95-29.25m | 14456 | 6.2 | 0.50 | 0.17 | 5 | 3 | 3 | 0.49 | 2.6 | 36 | 58 | NAF | 🟦 |
| 76 | Overburden/Interburden | MN0003 | Upper Pilot A Floor | | 139.96-140.40m | 14457 | 7.6 | 0.17 | 0.04 | 1 | 6 | -4 | 4.53 | 2.8 | 15 | 32 | NAF | 🟦 |
| 77 | Overburden/Interburden | Main Pit | Great Northern Seam and Fassifern Interburden - Dump Grab Sample 1 | Awaba Tuff - 6 months old with Carbonaceous materials | | 14458 | 8.5 | 0.33 | 0.06 | 2 | 17 | -15 | 9.37 | 2.9 | 15 | 35 | UC(NAF) | 🟦 |
| 78 | Overburden/Interburden | Main Pit | Fassifern Seam and Upper Pilot A Interburden - Dump Grab Sample 2 | Zebra Stripes - Recent | | 14459 | 8.8 | 0.61 | 0.14 | 4 | 14 | -10 | 3.28 | 2.7 | 23 | 47 | NAF | 🟦 |
| 79 | Overburden/Interburden | Main Pit | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample 3 | Fresh Conglom Taralba - 3 months | | 14460 | 7.4 | 0.26 | 0.04 | 1 | 12 | -11 | 10.01 | 7.3 | 0 | 0 | NAF | 🟦 |
| 80 | Overburden/Interburden | Main Pit | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample 4 | Weathered Conglom Taralba - 3 months | | 14461 | 7.3 | 0.38 | 0.01 | 0 | 18 | -18 | 59.55 | 7.6 | 0 | 0 | NAF | 🟦 |
| 81 | Overburden/Interburden | Main Pit | Fassifern Seam and Upper Pilot A Interburden - Dump Grab Sample 5 | Zebra - 1 year old? | | 14462 | 7.8 | 0.65 | 0.12 | 4 | 13 | -10 | 3.61 | 2.6 | 31 | 59 | NAF | 🟦 |
| 82 | Overburden/Interburden | Main Pit | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample 6 | Conglom - 18 months. Mixed fresh with oxide in joints (typical). | | 14463 | 7.6 | 0.41 | 0.01 | 0 | 23 | -22 | 73.71 | 7.2 | 0 | 0 | NAF | 🟦 |
| 83 | Overburden/Interburden | Main Pit | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample 7 | Conglom with GN Seam in it - lots of Carbonaceous material | | 14464 | 5.9 | 0.46 | 0.12 | 4 | 6 | -2 | 1.56 | 2.4 | 51 | 87 | NAF | 🟦 |
| 84 | Overburden/Interburden | Main Pit | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample 8 | Fresh Conglom, new (1 week?), No Coal | | 14465 | 8.1 | 0.24 | 0.03 | 1 | 9 | -8 | 9.91 | 7.2 | 0 | 0 | NAF | 🟦 |

**KEY**

pH$_{1:2}$ = pH of 1:2 extract

EC$_{1:2}$ = Electrical Conductivity of 1:2 extract (dS/m)

MPA = Maximum Potential Acidity (kg$H_2SO_4$/t)

ANC = Acid Neutralising Capacity (kg$H_2SO_4$/t)

NAPP = Net Acid Producing Potential (kg$H_2SO_4$/t)

🟨 Standard NAG results overestimate acid potential due to organic acid effects

NAGpH = pH of NAG liquor

NAG$_{(pH4.5)}$ = Net Acid Generation capacity to pH 4.5 (kg$H_2SO_4$/t)

NAG$_{(pH7.0)}$ = Net Acid Generation capacity to pH 7.0 (kg$H_2SO_4$/t)

🟦 NAF = Non-Acid Forming

🟥 PAF = Potentially Acid Forming

Table A2: Extended boil and calculated NAG test results for selected samples.

| Sample ID | Sample Type | Location and Description | EGi Sample Number | ACID-BASE ANALYSIS | | | | | STANDARD NAG TEST | | | Extended Boil NAG | Calculated NAG | ARD Classification |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Total %S | MPA | ANC | NAPP | ANC/MPA | NAGpH | $NAG_{(pH4.5)}$ | $NAG_{(pH7.0)}$ | | | |
| Sample 1 | Tailings | TD2W - No salt | 12426 | 0.31 | 9 | 18 | -9 | 1.90 | 2.4 | 49 | 88 | 6.1 | -1 | NAF |
| Sample 2 | Tailings | TD2W - No salt | 12427 | 0.35 | 11 | 19 | -8 | 1.77 | 2.5 | 45 | 80 | 5.4 | -7 | NAF |
| Sample 4 | Tailings | TD2S - White salts | 12429 | 1.58 | 48 | 49 | -1 | 1.01 | 3.1 | 8 | 39 | 5.2 | 11 | PAF |
| Sample 6 | Tailings | TD2E - Yellow and white salts. Profile sample. | 12431 | 1.79 | 55 | 0 | 55 | 0.00 | 2.1 | 69 | 113 | 2.7 | 36 | PAF |
| Sample 7 | Tailings | TD2E - Yellow and white salts. Selective sample of high salt layers | 12432 | 5.03 | 154 | 0 | 154 | 0.00 | 1.9 | 88 | 109 | 2.4 | 103 | PAF |
| Sample 10 | Tailings | TD1W - No salt | 12435 | 0.31 | 9 | 21 | -12 | 2.21 | 2.5 | 48 | 82 | 5.8 | -2 | NAF |
| Sample 11 | Tailings | TD1W - No salt | 12436 | 0.27 | 8 | 19 | -11 | 2.30 | 2.5 | 50 | 83 | 6.5 | -10 | NAF |
| Sample 13 | Coarse Rejects | Dumped Rejects Grab Sample | 12438 | 0.14 | 4 | 20 | -16 | 4.67 | 3.5 | 7 | 22 | 6.9 | -10 | NAF |
| Sample 14 | Coarse Rejects | Dumped Rejects Grab Sample | 12439 | 0.15 | 5 | 12 | -7 | 2.61 | 3.9 | 4 | 26 | 5.5 | -7 | NAF |
| 14 | Tailings | TD2E Surface Grab Sample | 14479 | 0.63 | 19 | 8 | 11 | 0.43 | 2.7 | 12 | 35 | 3.4 | 5 | PAF |
| 18 | Tailings | TD2S Excavation Site 1: 2.5m Deep | 14483 | 1.28 | 39 | 13 | 26 | 0.34 | 4.1 | 2 | 29 | 4.1 | 13 | PAF |
| 19 | Tailings | TD2S Excavation Site 1: 2.5m Deep | 14484 | 2.39 | 73 | 38 | 36 | 0.51 | 4.3 | 3 | 37 | 4.3 | 32 | PAF |
| 22 | Tailings | TD2S Excavation Site 2: 3m Deep | 14487 | 0.87 | 27 | 7 | 19 | 0.28 | 3.5 | 3 | 33 | 3.4 | 12 | PAF |
| 24 | Tailings | TD2S Excavation Site 2: 3m Deep | 14489 | 1.32 | 40 | 33 | 8 | 0.81 | 4.4 | 1 | 27 | 4.6 | 2 | PAF |
| 27 | Tailings | TD2S Surface Grab Sample | 14492 | 0.44 | 13 | 14 | 0 | 1.02 | 2.5 | 58 | 106 | 4.8 | -9 | NAF |
| 33 | Tailings | TD2W Excavation Site 1: 2m Deep | 14498 | 0.29 | 9 | 16 | -7 | 1.78 | 2.3 | 118 | 179 | 7.3 | -19 | NAF |
| 38 | Tailings | TD2W Surface Grab Sample | 14503 | 0.36 | 11 | 19 | -8 | 1.73 | 2.7 | 49 | 93 | 5.5 | -14 | NAF |
| 53 | Tailings | TD1E Surface Grab Sample | 14518 | 0.46 | 14 | 35 | -21 | 2.48 | 4.0 | 4 | 26 | 4.5 | -27 | NAF |
| 54 | Tailings | TD1E Surface Grab Sample | 14519 | 0.58 | 18 | 48 | -30 | 2.71 | 3.2 | 24 | 64 | 6.9 | -29 | NAF |

**KEY**

MPA = Maximum Potential Acidity ($kgH_2SO_4/t$)

ANC = Acid Neutralising Capacity ($kgH_2SO_4/t$)

NAPP = Net Acid Producing Potential ($kgH_2SO_4/t$)

NAGpH = pH of NAG liquor

$NAG_{(pH4.5)}$ = Net Acid Generation capacity to pH 4.5 ($kgH_2SO_4/t$)

$NAG_{(pH7.0)}$ = Net Acid Generation capacity to pH 7.0 ($kgH_2SO_4/t$)

Extended Boil NAGpH = pH of NAG liquor after extended heating

Calculated NAG = The net acid potential based on assay of anions and cations released to the NAG solution ($kgH_2SO_4/t$)

Table A3: Sulphur speciation results for selected samples.

| EGi Sample Number | Sample Type | Description | Total %S | Pyritic S (%) | Acid Sulphate %S | Total Acid Generating S (%) | Non-Acid Sulphate %S | Other S Forms (%) | Proportion Total Acid Generating to Total S | Readily Available ANC* (kg $H_2SO_4$/t) | Re-calculated NAPP** (kg $H_2SO_4$/t) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 12427 | Tailings | TD2W - No salt | 0.38 | 0.14 | 0.00 | 0.14 | 0.11 | 0.00 | 37% | 19 | -15 |
| 12428 | Tailings | TD2S - White salts | 1.27 | 0.41 | 0.00 | 0.41 | 0.86 | 0.00 | 32% | | |
| 12429 | Tailings | TD2S - White salts | 1.64 | 0.40 | 0.00 | 0.40 | 1.24 | 0.00 | 24% | 10 | 2 |
| 12431 | Tailings | TD2E - Yellow and white salts. Profile sample. | 1.79 | 0.20 | 0.08 | 0.28 | 1.51 | 0.00 | 15% | 0 | 8 |
| 12432 | Tailings | TD2E - Yellow and white salts. Selective sample of high salt layers | 5.53 | 2.30 | 0.22 | 2.52 | 2.37 | 0.64 | 46% | 0 | 77 |
| 12433 | Tailings | TD1E - Minor white salt | 0.84 | 0.21 | 0.00 | 0.21 | 0.63 | 0.00 | 25% | 69 | -63 |
| 12434 | Tailings | TD1E - Minor white salt | 0.79 | 0.27 | 0.00 | 0.27 | 0.52 | 0.00 | 34% | | |
| 12438 | Rejects | Dumped Rejects Grab Sample | 0.16 | 0.04 | 0.00 | 0.04 | 0.07 | 0.00 | 25% | 10 | -9 |
| 12440 | Rejects | Dumped Rejects Grab Sample | 0.13 | 0.04 | 0.00 | 0.04 | 0.03 | 0.00 | 31% | 19 | -18 |
| 12430 | Overburden/Interburden | Great Northern Seam and Fassifern Interburden In Situ Jarositic Shale TD 2 NE corner | 0.37 | 0.19 | 0.08 | 0.27 | 0.10 | 0.00 | 73% | 0 | 8 |
| 14456 | Overburden/Interburden | Upper Pilot A Floor | 0.18 | 0.05 | 0.00 | 0.05 | 0.04 | 0.09 | 28% | | |
| 14459 | Overburden/Interburden | Fassifern Seam and Upper Pilot A Interburden - Dump Grab Sample 2 | 0.15 | 0.04 | 0.00 | 0.04 | 0.01 | 0.10 | 27% | 7 | -6 |
| 14462 | Overburden/Interburden | Fassifern Seam and Upper Pilot A Interburden - Dump Grab Sample 5 | 0.14 | 0.03 | 0.00 | 0.03 | 0.01 | 0.10 | 21% | | |
| 14464 | Overburden/Interburden | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample 7 | 0.15 | 0.03 | 0.00 | 0.03 | 0.03 | 0.09 | 20% | | |

Pyritic S (%) = CRS (%)

Acid Sulphate S = KCl Acid Sulphate S

Total Acid Generating S = Pyritic S + Acid Sulphate S

Non-Acid Sulphate S = KCl S – KCl Acid Sulphate S

Other S Forms = Total S - (CRS + KCl S)

* estimated from ABCC testing, or if the original ANC was 0 kg $H_2SO_4$/t then it is also 0 kg $H_2SO_4$/t by default

**based on acid generating S (pyrite and acid sulphate S) and readily available ANC

Table A4: Multi-element composition of selected sample solids (mg/kg except where shown).

| Element | Detection Limit | Sample Code/Description | | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Coarse Rejects | Coarse Rejects | Wallarah Seam Overburden | | Wallarah Seam and Great Northern Seam Interburden | | | | Great Northern Seam and Fassifern Seam Interburden - In-situ Shale | Great Northern Seam and Fassifern Seam Interburden - Core | Fassifern Seam and Upper Pilot A Interburden - Dump Grab Sample | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample | | | |
| | | 12427 | 12428 | 12429 | 12431 | 12432 | 12433 | 12434 | 12435 | 12438 | 12440 | 14439 | 14443 | 14448 | 14449 | 14451 | 14452 | 12430 | 14454 | 14459 | 14460 | 14463 | 14464 | 14465 |
| Ag | 0.01 | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 | 0.1 | 0.04 | 0.04 | 0.05 | 0.04 | 0.05 | 0.03 | <0.1 | 0.08 | 0.11 | 0.04 | 0.03 | 0.04 | 0.03 |
| Al | 0.01% | 7.68% | 8.39% | 5.36% | 6.31% | 5.95% | 6.72% | 6.72% | 7.98% | 8.48% | 10.28% | 5.02% | 4.74% | 7.81% | 5.62% | 6.93% | 5.48% | 6.75% | 6.93% | 9.07% | 6.03% | 5.68% | 6.18% | 5.60% |
| As | 0.2 | 25 | 25 | 27 | 28 | 37 | 32 | 27 | 21 | 14 | 23 | 2.1 | 2.7 | 12.9 | 8.2 | 14 | 13.1 | 10 | 1.4 | 5.9 | 13.4 | 6.4 | 10.6 | 10.8 |
| Ba | 10 | 296 | **>5000** | 2617 | 236 | 929 | 507 | 507 | 174 | 255 | 229 | 270 | 310 | 630 | 520 | 530 | 430 | 649 | 220 | 250 | 450 | 600 | 300 | 440 |
| Be | 0.05 | 1.4 | 1.2 | 1.6 | 1.6 | 1.1 | 1.3 | 1.2 | 1.3 | 1.8 | 1.2 | 0.9 | 0.89 | 1.48 | 1.05 | 1.55 | 1.18 | 2.4 | 1.79 | 2.16 | 1.27 | 1.43 | 2.93 | 1.31 |
| Bi | 0.01 | 0.62 | 1.07 | 0.7 | 0.7 | 1.2 | 0.78 | 0.79 | 0.72 | 0.68 | 0.62 | 0.13 | 0.13 | 0.23 | 0.15 | 0.27 | 0.15 | 0.43 | 0.72 | 0.98 | 0.16 | 0.15 | 0.47 | 0.16 |
| Ca | 0.01% | 0.64% | 1.09% | 0.48% | 0.32% | 0.25% | 2.34% | 2.32% | 0.59% | 0.42% | 0.81% | 0.25% | 0.19% | 0.40% | 0.36% | 0.80% | 0.61% | 0.06% | 0.55% | 0.29% | 0.27% | 0.76% | 0.11% | 0.16% |
| Cd | 0.02 | 0.13 | 0.13 | 0.14 | 0.54 | 0.17 | 0.14 | 0.13 | 0.13 | 0.2 | 0.25 | 0.06 | 0.05 | 0.07 | 0.05 | 0.09 | 0.05 | <0.05 | 0.06 | 0.12 | 0.05 | 0.04 | 0.07 | 0.04 |
| Co | 0.1 | 8.7 | 4.2 | 13.7 | 5.3 | 3.9 | 2.4 | 2.1 | 4.3 | 3.6 | 5 | 8.2 | 7.4 | 10.9 | 7.7 | 17.1 | 8.5 | 4.3 | 4.6 | 2.6 | 7.4 | 7 | 5.6 | 6.7 |
| Cr | 1 | 13 | 14 | 44 | 22 | 24 | 9 | 9 | 13 | 10 | 13 | 71 | 80 | 89 | 78 | 82 | 61 | 24 | 7 | 14 | 49 | 70 | 52 | 60 |
| Cs | 0.05 | 11.7 | 5.2 | 2.4 | 3.2 | 4 | 8.5 | 7.4 | 9.5 | 9.6 | 4.7 | 3.16 | 3.22 | 6.75 | 3.05 | 6.25 | 2.88 | 10.6 | 6.22 | 7.26 | 3.26 | 3 | 2.27 | 2.98 |
| Cu | 0.2 | 20 | 10 | 15 | 12 | 12 | 11 | 11 | 13 | 13 | 15 | 19.1 | 19.5 | 23.5 | 16.2 | 24.4 | 15.2 | 18 | 4.5 | 5.8 | 15.7 | 15.4 | 11.5 | 15 |
| Fe | 0.01% | 1.41% | 5.42% | 2.66% | 3.28% | 9.51% | 4.33% | 4.25% | 1.21% | 2.87% | 1.67% | 2.62% | 2.20% | 2.37% | 2.17% | 1.98% | 1.95% | 2.49% | 1.57% | 2.56% | 2.02% | 2.25% | 1.15% | 1.87% |
| Ga | 0.05 | 17.4 | 16.7 | 12.3 | 14.2 | 13.1 | 13.9 | 13.4 | 18.1 | 16.7 | 21.4 | 11.2 | 10.95 | 17.2 | 12.35 | 17.85 | 13.4 | 18.5 | 15.4 | 22.1 | 13.75 | 12.7 | 14.35 | 13.15 |
| Ge | 0.05 | 1.6 | 0.6 | 3.6 | 0.5 | 0.9 | 0.4 | 0.2 | 0.9 | 0.3 | 0.6 | 0.11 | 0.11 | 0.15 | 0.16 | 0.15 | 0.17 | 1.4 | 0.15 | 0.18 | 0.17 | 0.2 | 0.16 | 0.17 |
| Hf | 0.01 | 5.1 | 4.6 | 3.3 | 3.8 | 3.4 | 4.2 | 4.3 | 5.5 | 5.5 | 4.9 | 2.8 | 2.6 | 4.4 | 3.4 | 4.7 | 3.5 | 4.1 | 3.5 | 6.9 | 3.6 | 3.6 | 3.6 | 3.6 |
| Hg | 0.005 | 0.12 | 0.15 | 0.11 | 0.17 | 0.25 | 0.12 | 0.14 | 0.16 | 0.08 | 0.17 | 0.021 | 0.036 | 0.061 | 0.046 | 0.06 | 0.043 | <0.05 | 0.068 | 0.066 | 0.057 | 0.03 | 0.069 | 0.042 |
| In | 0.005 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | 0.044 | 0.05 | 0.06 | 0.054 | 0.059 | 0.05 | <0.05 | 0.037 | 0.082 | 0.056 | 0.056 | 0.056 | 0.055 |
| K | 0.01% | 1.07% | 0.52% | 0.48% | 0.44% | 0.46% | 0.90% | 0.76% | 0.70% | 1.82% | 0.88% | 1.25% | 1.24% | 2.25% | 1.83% | 2.17% | 1.82% | 1.85% | 0.35% | 0.46% | 1.98% | 2.13% | 1.28% | 1.98% |
| Li | 0.2 | 26 | 25.4 | 17.8 | 21.9 | 17.2 | 20.8 | 19.6 | 29.7 | 37.8 | 28.9 | 16.4 | 15.9 | 17.8 | 12.8 | 18.4 | 13.5 | 15.9 | 11.4 | 15.3 | 14.7 | 17.8 | 23 | 13.4 |
| Mg | 0.01% | 0.45% | 0.73% | 0.74% | 0.44% | 0.35% | 0.86% | 1.08% | 0.59% | 0.29% | 0.21% | 0.49% | 0.43% | 0.55% | 0.42% | 0.58% | 0.52% | 0.27% | 1.07% | 0.51% | 0.42% | 0.29% | 0.17% | 0.30% |
| Mn | 5 | 173 | 1308 | 868 | 3349 | 1198 | 1018 | 897 | 128 | 458 | 623 | 291 | 393 | 294 | 548 | 254 | 423 | 164 | 184 | 185 | 460 | 415 | 284 | 440 |
| Mo | 0.05 | 1.4 | 2.6 | 3.6 | 3.7 | 7.5 | 3.4 | 4 | 2.1 | 1.2 | 1.9 | 0.29 | 0.3 | 0.4 | 2.21 | 0.46 | 1.08 | 0.9 | 2.14 | 1.36 | 0.72 | 0.48 | 2.32 | 1.28 |
| Na | 0.01% | 0.22% | 0.38% | 0.67% | 0.40% | 0.90% | 0.25% | 0.36% | 0.59% | 0.13% | 0.14% | 0.11% | 0.40% | 0.72% | 1.25% | 0.34% | 0.86% | 0.20% | 0.25% | 0.14% | 0.88% | 1.00% | 0.38% | 0.81% |
| Nb | 0.1 | 6.4 | 8.7 | 12.2 | 8.5 | 8.5 | 6.1 | 5.9 | 7.7 | 7 | 9.4 | 4.3 | 4.3 | 7.1 | 5.1 | 7.6 | 5.3 | 9.2 | 8.2 | 10.5 | 5.6 | 5.6 | 6.7 | 5.5 |
| Ni | 0.2 | 13 | 5 | 37 | 13 | 7 | 4 | 3 | 9 | 4 | 8 | 23.2 | 22.6 | 33.8 | 23.5 | 27.4 | 20.4 | 5 | 2.7 | 4.1 | 19.2 | 18.7 | 13.5 | 16.1 |
| P | 10 | 95 | 138 | 299 | 149 | 138 | 165 | 185 | 117 | 127 | 136 | 150 | 150 | 480 | 410 | 310 | 380 | 78 | 60 | 110 | 420 | 430 | 240 | 400 |
| Pb | 0.5 | 25 | 41 | 21 | 24 | 28 | 30 | 32 | 30 | 27 | 20 | 8.7 | 8.2 | 18.3 | 10.5 | 19.4 | 11.1 | 17 | 33.3 | 50.7 | 12.9 | 11.1 | 18.5 | 11.4 |
| Rb | 0.1 | 80.1 | 33.8 | 24.4 | 24.9 | 27.4 | 56.5 | 47.5 | 50.7 | 99.3 | 53.1 | 65.1 | 63.8 | 111 | 83.7 | 92.5 | 80.8 | 78 | 22 | 40.4 | 95.3 | 96 | 61.6 | 95.4 |
| Re | 0.002 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.002 | <0.002 | <0.002 | <0.002 | <0.002 | <0.002 | <0.05 | <0.002 | <0.002 | <0.002 | <0.002 | <0.002 | <0.002 |
| S | 0.01% | 0.34% | 1.17% | 1.49% | 1.62% | 4.84% | 0.75% | 0.77% | 0.33% | 0.14% | 0.12% | 0.02% | 0.02% | 0.02% | 0.03% | 0.02% | 0.04% | 0.41% | 0.01% | 0.16% | 0.06% | 0.01% | 0.11% | 0.04% |
| Sb | 0.05 | 0.7 | 0.7 | 1 | 0.6 | 1.3 | 0.6 | 0.5 | 0.6 | 0.4 | 0.5 | 0.63 | 0.67 | 1.18 | 0.87 | 1.35 | 0.92 | 0.8 | 0.68 | 1.16 | 0.94 | 0.82 | 2 | 0.81 |
| Sc | 0.1 | 10 | 8 | 8 | 10 | 7 | 7 | 8 | 8 | 7 | 8 | 11.5 | 10.6 | 15.4 | 11.6 | 11.6 | 11.4 | 10 | 3.2 | 6.7 | 11.9 | 12.2 | 8.5 | 11.4 |
| Se | 1 | <2 | <2 | <2 | <2 | <2 | <2 | <2 | <2 | <2 | <2 | <1 | <1 | <1 | <1 | <1 | <1 | <2 | <1 | <1 | <1 | <1 | 1 | <1 |
| Sn | 0.2 | 4.2 | 5.7 | 3 | 3.5 | 4.7 | 4.2 | 4.1 | 5.4 | 4.3 | 4.6 | 1.4 | 1.5 | 2.6 | 1.8 | 2.7 | 1.9 | 3.2 | 4.6 | 6.6 | 2.1 | 1.8 | 2.6 | 1.9 |
| Sr | 0.2 | 103 | 124.3 | 107.1 | 77.6 | 98.1 | 239.6 | 218.5 | 122.7 | 84.3 | 75.8 | 79.5 | 72.7 | 159 | 111.5 | 97.7 | 92.7 | 50.5 | 178.5 | 124 | 122 | 97.4 | 52.7 | 131 |
| Ta | 0.05 | 0.75 | 1.09 | 1.05 | 0.81 | 0.83 | 0.79 | 0.77 | 0.9 | 1.02 | 1.06 | 0.32 | 0.32 | 0.55 | 0.38 | 0.58 | 0.39 | 0.6 | 0.91 | 1.07 | 0.42 | 0.43 | 0.55 | 0.41 |
| Te | 0.05 | <0.1 | <0.1 | <0.1 | <0.1 | 0.1 | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 | <0.05 | <0.05 | 0.07 | 0.07 | 0.06 | <0.05 | <0.1 | <0.05 | <0.05 | <0.05 | <0.05 | 0.07 | <0.05 |
| Th | 0.2 | 18.83 | 31.09 | 13.58 | 21.86 | 17.36 | 21.42 | 21.69 | 23.5 | 19.39 | 17.91 | 5.79 | 5.67 | 13.2 | 8.44 | 12.35 | 7.81 | 12.19 | 26.1 | 31.8 | 9.35 | 9.26 | 14 | 8.81 |
| Ti | 0.005% | 0.24% | 0.20% | 0.27% | 0.18% | 0.17% | 0.18% | 0.18% | 0.22% | 0.22% | 0.34% | 0.27% | 0.27% | 0.47% | 0.29% | 0.49% | 0.29% | 0.27% | 0.19% | 0.24% | 0.29% | 0.30% | 0.23% | 0.28% |
| Tl | 0.02 | 1.11 | 2.77 | 2.2 | 2.67 | 7.71 | 1.81 | 1.94 | 0.71 | 0.58 | 0.57 | 0.29 | 0.29 | 0.56 | 0.48 | 0.61 | 0.49 | 1.03 | 0.26 | 0.7 | 0.56 | 0.5 | 0.52 | 0.53 |
| U | 0.1 | 4.76 | 7.81 | 7.21 | 5.87 | 4.81 | 5.39 | 5.4 | 6.5 | 4.43 | 5.52 | 1.4 | 1.3 | 3.1 | 1.8 | 4.2 | 1.8 | 3.54 | 7.4 | 7.1 | 2.1 | 1.8 | 3.1 | 2 |
| V | 1 | 43 | 23 | 41 | 20 | 22 | 23 | 19 | 32 | 31 | 27 | 67 | 60 | 99 | 61 | 80 | 59 | 83 | 25 | 22 | 62 | 61 | 42 | 55 |
| W | 0.1 | 1.6 | 2.4 | 1.9 | 2.1 | 2.2 | 1.8 | 1.6 | 1.6 | 1.1 | 2.7 | 0.9 | 0.9 | 1.8 | 1.2 | 2.4 | 1.2 | 5.6 | 3 | 3 | 1.4 | 1.2 | 1.8 | 1.4 |
| Y | 0.1 | 21.9 | 28.3 | 16.6 | 27.4 | 16.6 | 22.7 | 21.2 | 24.5 | 28.5 | 22.6 | 19.7 | 20 | 20.2 | 19.8 | 13.6 | 19.4 | 26.5 | 15.8 | 34.7 | 20.4 | 21.9 | 22.9 | 20.4 |
| Zn | 2 | 41 | 61 | 67 | 105 | 64 | 33 | 38 | 38 | 54 | 71 | 66 | 57 | 106 | 51 | 99 | 57 | 38 | 40 | 49 | 55 | 58 | 46 | 48 |
| Zr | 0.5 | 138.8 | 117.7 | 98.2 | 108.5 | 94.1 | 108.7 | 109.5 | 147.2 | 157.8 | 124.4 | 102 | 100 | 158 | 126 | 165 | 126 | 127.3 | 100.5 | 212 | 130.5 | 131 | 124.5 | 129.5 |

< element at or below analytical detection limit.

Table A5: Geochemical abundance indices (GAI) of selected sample solids. Values 3 and over are highlighted in yellow .

| Element | Median Soil Abundance * | Sample Code/Description | | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Coarse Rejects | Coarse Rejects | Wallarah Seam Overburden | | Wallarah Seam and Great Northern Seam Interburden | | | | Great Northern Seam and Fassifern Seam Interburden - In-situ Shale | Great Northern Seam and Fassifern Seam Interburden - Core | Fassifern Seam and Upper Pilot A Interburden - Dump Grab Sample | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample | | | |
| | | 12427 | 12428 | 12429 | 12431 | 12432 | 12433 | 12434 | 12435 | 12438 | 12440 | 14439 | 14443 | 14448 | 14449 | 14451 | 14452 | 12430 | 14454 | 14459 | 14460 | 14463 | 14464 | 14465 |
| Ag | 0.05 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 1 | - | - | - | - |
| Al | 7.1% | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| As | 6 | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | - | - | 1 | - | 1 | 1 | - | - | - | 1 | - | - | - |
| Ba | 500 | - | **>3** | 2 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Be | 0.3 | 2 | 1 | 2 | 2 | 1 | 2 | 1 | 2 | 2 | 1 | 1 | 1 | 2 | 1 | 2 | 1 | 2 | 2 | 2 | 1 | 2 | 3 | 2 |
| Bi | 0.2 | 1 | 2 | 1 | 1 | 2 | 1 | 1 | 1 | 1 | 1 | - | - | - | - | - | - | 1 | 1 | 2 | - | - | 1 | - |
| Ca | 1.5% | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Cd | 0.35 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Co | 8 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 1 | - | - | - | - | - | - | - | - |
| Cr | 70 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Cs | 4 | 1 | - | - | - | - | 1 | - | 1 | 1 | - | - | - | - | - | - | - | 1 | - | - | - | - | - | - |
| Cu | 30 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Fe | 4.0% | - | - | - | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Ga | 20 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Ge | 1 | - | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Hf | 6 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Hg | 0.06 | - | 1 | - | 1 | 1 | - | 1 | 1 | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - |
| In | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| K | 1.4% | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Li | 25 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Mg | 0.5% | - | - | - | - | - | - | 1 | - | - | - | - | - | - | - | - | - | - | 1 | - | - | - | - | - |
| Mn | 1000 | - | - | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Mo | 1.2 | - | 1 | 1 | 1 | 2 | 1 | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Na | 0.5% | - | - | - | - | - | - | - | - | - | - | - | - | - | 1 | - | - | - | - | - | - | - | - | - |
| Nb | 10 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Ni | 50 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| P | 800 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Pb | 35 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Rb | 150 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Re | 0 | | | | | | | | | | | | | | | | | | | | | | | |
| S | 0.07% | 2 | 3 | 4 | 4 | 6 | 3 | 3 | 2 | - | - | - | - | - | - | - | - | 2 | - | 1 | - | - | - | - |
| Sb | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Sc | 7 | - | - | - | - | - | - | - | - | - | - | - | - | 1 | - | - | - | - | - | - | - | - | - | - |
| Se | 0.4 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 | 1 | 2 | 1 | 1 | 1 | 1 | 1 | 1 |
| Sn | 4 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Sr | 250 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Ta | 2 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Te | 0 | | | | | | | | | | | | | | | | | | | | | | | |
| Th | 9 | - | 1 | - | 1 | - | 1 | 1 | 1 | 1 | - | - | - | - | - | - | - | - | 1 | 1 | - | - | - | - |
| Ti | 0.50% | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Tl | 0.2 | 2 | 3 | 3 | 3 | 5 | 3 | 3 | 1 | 1 | 1 | - | - | 1 | 1 | 1 | 1 | 2 | - | 1 | 1 | 1 | 1 | 1 |
| U | 2 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | - | - | - | - | - | - | - | 1 | 1 | - | - | - | - |
| V | 90 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| W | 1.5 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 1 | - | - | - | - | - | - |
| Y | 40 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Zn | 90 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Zr | 400 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |

*Bowen H.J.M.(1979) Environmental Chemistry of the Elements.

Table A6: Chemical composition of water extracts

| Parameter | | Detection Limit | Sample Code/Description | | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Tailings | Coarse Rejects | Coarse Rejects | Wallarah Seam Overburden | | Wallarah Seam and Great Northern Seam Interburden | | | | Great Northern Seam and Fassifern Seam Interburden - In-situ Shale | Great Northern Seam and Fassifern Seam Interburden - Core | Fassifern Seam and Upper Pilot A Interburden - Dump Grab Sample | Wallarah Seam and Great Northern Seam Interburden - Dump Grab Sample | | | |
| | | | 12427 | 12428 | 12429 | 12431 | 12432 | 12433 | 12434 | 12435 | 12438 | 12440 | 14439 | 14443 | 14448 | 14449 | 14451 | 14452 | 12430 | 14454 | 14459 | 14460 | 14463 | 14464 | 14465 |
| | | | 0.35%S | 1.18%S | 1.58%S | 1.79%S | 5.03%S | 0.7%S | 0.74%S | 0.31%S | 0.14%S | 0.12%S | 0.01%S | 0.01%S | 0.02%S | 0.02%S | 0.06%S | 0.02%S | 0.43%S | 0.01%S | 0.14%S | 0.04%S | 0.01%S | 0.12%S | 0.03%S |
| pH | | 0.1 | 7.6 | 7.7 | 7.1 | 2.8 | 2.1 | 6.8 | 7.3 | 7.5 | 8.5 | 8.3 | 7.9 | 8.2 | 8.4 | 8.1 | 8.4 | 7.9 | 3.5 | 8.4 | 8.9 | 8.0 | 8.4 | 7.2 | 8.0 |
| EC | dS/m | 0.001 | 5.36 | 12.02 | 10.79 | 3.21 | 8.92 | 6.91 | 9.97 | 5.32 | 0.421 | 0.349 | 0.441 | 0.389 | 0.375 | 0.521 | 0.312 | 0.416 | 1.66 | 0.534 | 0.629 | 0.231 | 0.491 | 0.475 | 0.322 |
| Alkalinity ($CaCO_3$) | mg/l | 1 | 73 | 48 | 39 | - | - | 32 | 45 | 61 | 86 | 78 | 25 | 37 | 48 | 33 | 45 | 48 | - | 80 | 245 | 35 | 28 | 13 | 16 |
| Acidity ($CaCO_3$) | mg/l | 1 | - | - | - | 697 | 2662 | - | - | - | - | - | - | - | - | - | - | - | 71 | - | - | - | - | - | - |
| Ag | mg/l | 0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| Al | mg/l | 0.01 | <0.01 | <0.01 | <0.01 | 80.7 | 98.6 | 0.18 | 0.02 | <0.01 | 0.01 | 0.04 | <0.01 | 0.03 | 0.05 | <0.01 | <0.01 | <0.01 | 4.02 | 0.54 | <0.01 | <0.01 | <0.01 | <0.01 | <0.01 |
| As | mg/l | 0.001 | 0.001 | <0.001 | <0.001 | 0.092 | 0.111 | <0.001 | <0.001 | 0.001 | <0.001 | 0.016 | <0.001 | 0.002 | 0.019 | 0.003 | 0.014 | 0.004 | 0.004 | 0.002 | <0.001 | 0.002 | <0.001 | <0.001 | <0.001 |
| B | mg/l | 0.05 | 0.18 | 0.29 | 0.48 | 0.99 | 0.36 | 0.33 | 0.21 | 0.25 | <0.05 | 0.08 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | 0.12 | 0.1 | <0.05 | 0.08 | 0.06 | 0.09 | 0.06 |
| Ba | mg/l | 0.001 | 0.624 | 0.078 | 0.077 | 0.028 | 0.052 | 0.068 | 0.08 | 0.608 | <0.001 | 0.53 | 0.434 | 0.422 | 0.392 | 0.173 | 0.15 | 0.127 | 0.07 | 0.022 | 0.002 | 0.071 | 0.419 | 0.082 | 0.094 |
| Be | mg/l | 0.001 | <0.001 | <0.001 | <0.001 | 0.113 | 0.063 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | 0.039 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| Ca | mg/l | 1 | 177 | 323 | 456 | 354 | 346 | 519 | 407 | 327 | 11 | 10 | 3 | 5 | 5 | 11 | 6 | 11 | 52 | <1 | <1 | 32 | 19 | 15 | 14 |
| Cd | mg/l | 0.0001 | <0.0001 | <0.0001 | 0.0005 | 0.0795 | 0.019 | 0.0002 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | 0.0009 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | 0.0001 | <0.0001 |
| Cl | mg/l | 1 | 1130 | 1390 | 2550 | 697 | 2000 | 787 | 1690 | 3740 | 63 | 30 | 103 | 63 | 42 | 33 | 53 | 44 | 45 | 65 | 36 | 116 | 205 | 52 | 38 |
| Co | mg/l | 0.001 | 0.004 | 0.002 | 0.038 | 0.963 | 0.373 | 0.003 | <0.001 | 0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | 0.003 | 0.096 | <0.001 | <0.001 | 0.013 | <0.001 | 0.129 | 0.075 |
| Cr | mg/l | 0.001 | <0.001 | <0.001 | <0.001 | 0.035 | 0.107 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | 0.014 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| Cu | mg/l | 0.001 | 0.006 | 0.006 | 0.006 | 0.027 | 0.102 | 0.004 | 0.003 | 0.006 | <0.001 | 0.001 | 0.001 | <0.001 | 0.002 | 0.002 | 0.002 | 0.004 | 0.081 | <0.001 | <0.001 | 0.001 | <0.001 | 0.002 | <0.001 |
| F | mg/l | 0.1 | 0.9 | 1.2 | 0.8 | 0.1 | <0.1 | 1 | 1.7 | 1.2 | 1 | 1.4 | 0.9 | 0.4 | 0.6 | 0.3 | 1.2 | 0.6 | 0.2 | 2.2 | 2.1 | 0.5 | 0.8 | 0.2 | 0.2 |
| Fe | mg/l | 0.05 | <0.05 | <0.05 | <0.05 | 35.1 | 516 | 0.54 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 | 3.12 | 0.07 | <0.05 | <0.05 | <0.05 | <0.05 | <0.05 |
| Hg | mg/l | 0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 | <0.0001 |
| K | mg/l | 1 | 28 | 48 | 39 | 4 | 1 | 37 | 37 | 32 | 5 | 6 | 8 | 4 | 4 | 5 | 8 | 8 | 9 | <1 | <1 | 10 | 7 | 8 | 8 |
| Mg | mg/l | 1 | 134 | 932 | 1700 | 1210 | 912 | 464 | 634 | 382 | 7 | 5 | 6 | 4 | 4 | 9 | 9 | 14 | 97 | <1 | <1 | 62 | 23 | 27 | 25 |
| Mn | mg/l | 0.001 | 0.098 | 0.81 | 10.9 | 1070 | 261 | 2.42 | 0.16 | 0.249 | 0.012 | 0.008 | 0.016 | 0.005 | 0.009 | 0.118 | 0.02 | 0.107 | 10.2 | <0.001 | <0.001 | 0.713 | 0.026 | 1.88 | 1.47 |
| Mo | mg/l | 0.001 | 0.052 | 0.022 | 0.002 | <0.001 | 0.001 | 0.004 | 0.016 | 0.071 | <0.001 | 0.078 | 0.001 | 0.001 | 0.01 | 0.075 | 0.011 | 0.024 | <0.001 | 0.016 | 0.005 | 0.005 | 0.003 | <0.001 | <0.001 |
| Na | mg/l | 1 | 492 | 1140 | 2080 | 1020 | 1730 | 440 | 866 | 1540 | 69 | 42 | 68 | 59 | 62 | 44 | 77 | 49 | 84 | 12 | 4 | 69 | 90 | 56 | 30 |
| Ni | mg/l | 0.001 | 0.006 | 0.004 | 0.053 | 1.81 | 0.647 | 0.007 | 0.004 | 0.004 | <0.001 | 0.002 | <0.001 | <0.001 | <0.001 | 0.003 | 0.001 | 0.005 | 0.083 | <0.001 | <0.001 | 0.025 | <0.001 | 0.345 | 0.16 |
| P | mg/l | 1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 | <1 |
| Pb | mg/l | 0.001 | <0.001 | <0.001 | <0.001 | <0.010 | 0.003 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| Sb | mg/l | 0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | 0.002 | <0.001 | 0.002 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| Se | mg/l | 0.01 | 0.06 | 0.13 | 0.38 | 0.86 | 0.51 | <0.01 | 0.02 | 0.05 | <0.01 | 0.02 | 0.01 | 0.01 | 0.01 | <0.01 | 0.01 | <0.01 | 0.04 | <0.01 | <0.01 | <0.01 | <0.01 | <0.01 | <0.01 |
| Si | mg/l | 0.1 | 4.6 | 2.9 | 1.5 | 7.8 | 5.5 | 4.0 | 4.9 | 4.9 | 4.6 | 4.4 | 1.5 | 1.6 | 2.5 | 1.5 | 2.0 | 1.45 | 8.7 | 39.45 | 25.38 | 1.589 | 2.34 | 2.8 | 2.06 |
| Sn | mg/l | 0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| SO4 | mg/l | 1 | 400 | 4570 | 8080 | 8850 | 7290 | 2760 | 2610 | 875 | 41 | 24 | 38 | 29 | 61 | 91 | 97 | 105 | 621 | 187 | 29 | 300 | 49 | 171 | 145 |
| Sr | mg/l | 0.001 | 5.15 | 6.93 | 9.94 | 5.23 | 2.94 | 9.68 | 9.74 | 10.3 | 0.355 | 0.204 | 0.042 | 0.233 | 0.316 | 0.59 | 0.08 | 0.103 | 0.292 | 0.011 | 0.003 | 0.306 | 0.154 | 0.168 | 0.184 |
| Th | mg/l | 0.001 | <0.001 | <0.001 | <0.001 | 0.039 | 1.2 | 0.002 | 0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | 0.002 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| Tl | mg/l | 0.001 | <0.001 | <0.001 | <0.001 | 0.013 | 0.008 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| U | mg/l | 0.001 | 0.004 | 0.003 | <0.001 | 0.062 | 0.229 | 0.002 | 0.001 | 0.006 | 0.003 | 0.002 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 | 0.017 | 0.001 | <0.001 | <0.001 | <0.001 | <0.001 | <0.001 |
| Zn | mg/l | 0.005 | 0.043 | 0.062 | 0.087 | 10.9 | 5.18 | 0.093 | 0.027 | 0.043 | <0.005 | <0.005 | 0.007 | <0.005 | 0.009 | 0.016 | 0.009 | 0.011 | 0.627 | 0.007 | <0.005 | 0.035 | 0.005 | 0.082 | 0.04 |

< element at or below analytical detection limit.

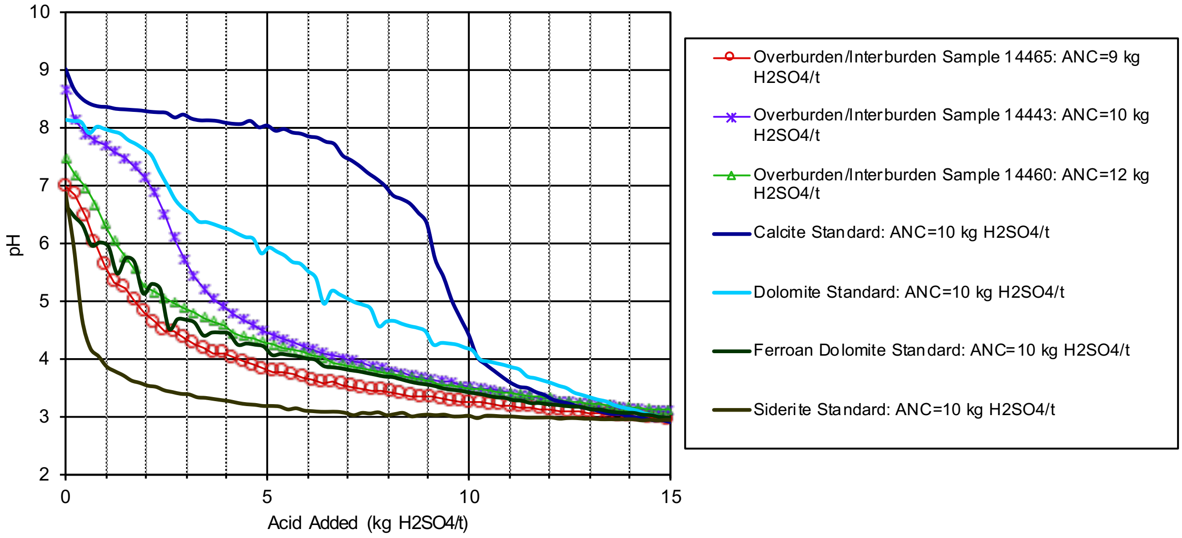

Figure A1: ABCC profile for samples with an ANC value close to 10 kg $H_2SO_4$/t. Carbonate standard curves are included for reference.

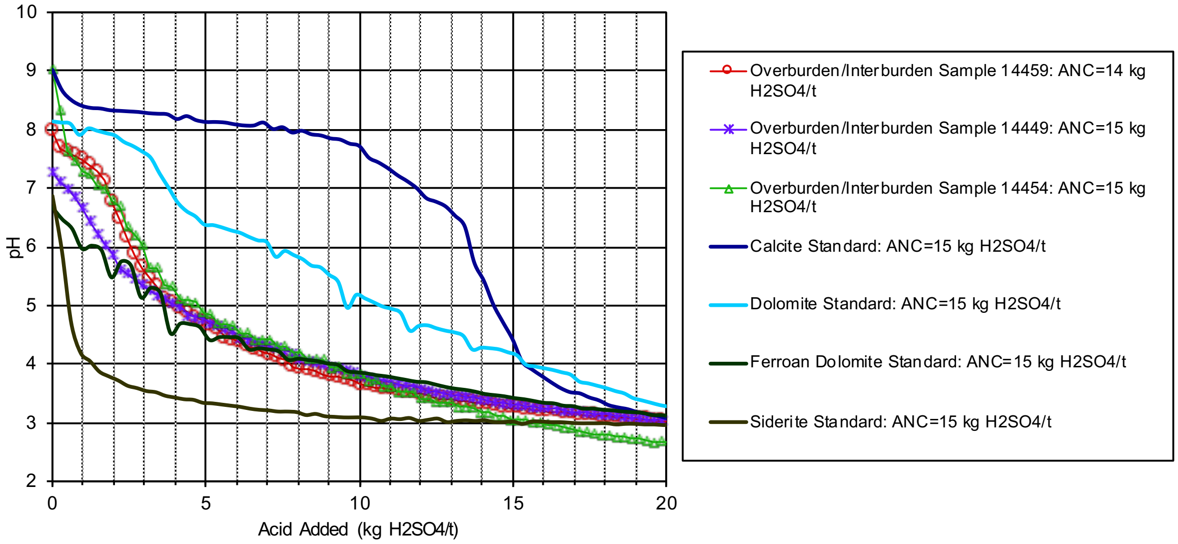

Figure A2: ABCC profile for sample 33051 with an ANC value of 15 kg $H_2SO_4$/t. Carbonate standard curves are included for reference.

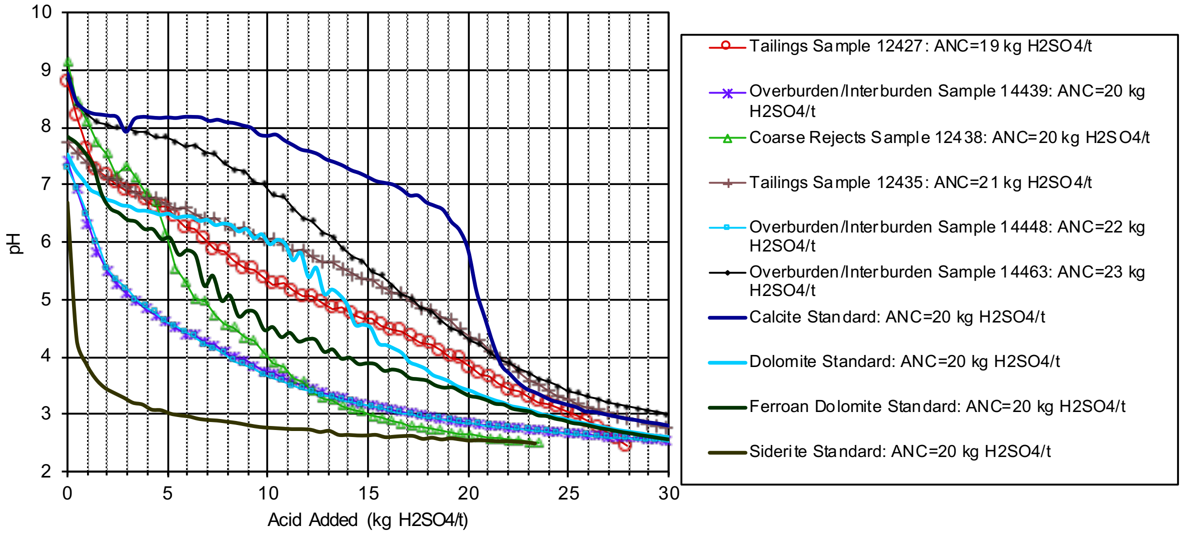

Figure A3: ABCC profile for samples with an ANC value close to 20 kg $H_2SO_4$/t. Carbonate standard curves are included for reference.

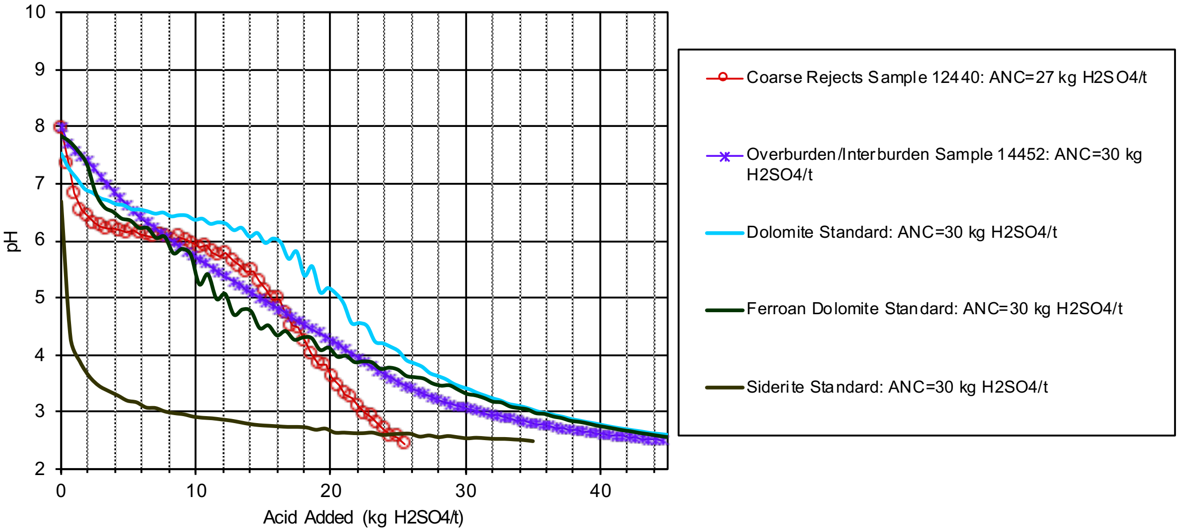

Figure A4: ABCC profile for samples with an ANC value close to 30 kg $H_2SO_4$/t. Carbonate standard curves are included for reference.

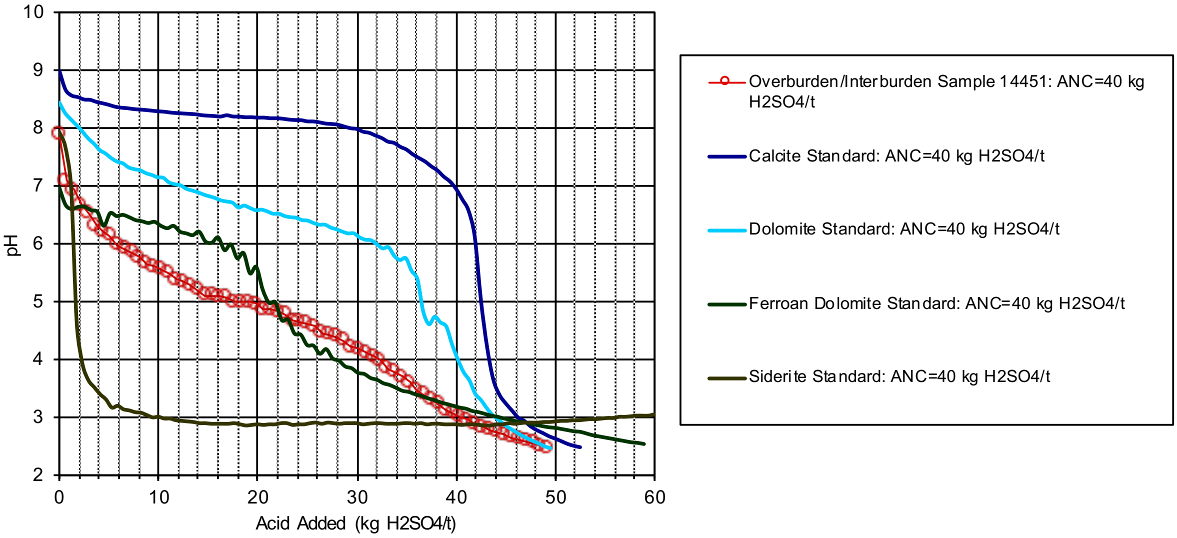

Figure A5: ABCC profile for sample 14451 with an ANC value of 40 kg $H_2SO_4$/t. Carbonate standard curves are included for reference.

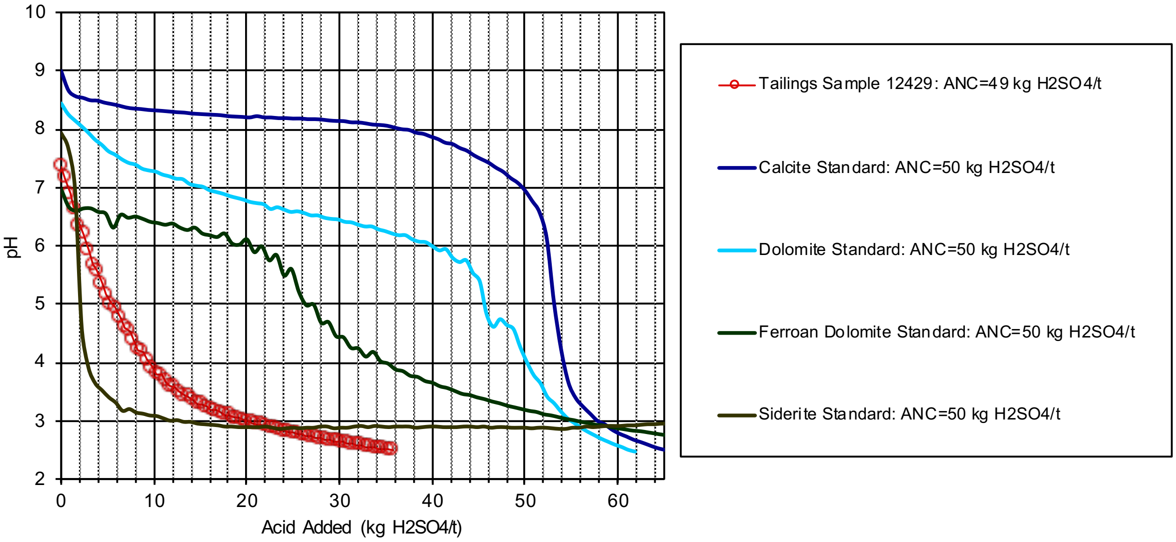

Figure A6: ABCC profile for sample 12429 with an ANC value close to 50 kg $H_2SO_4$/t. Carbonate standard curves are included for reference.

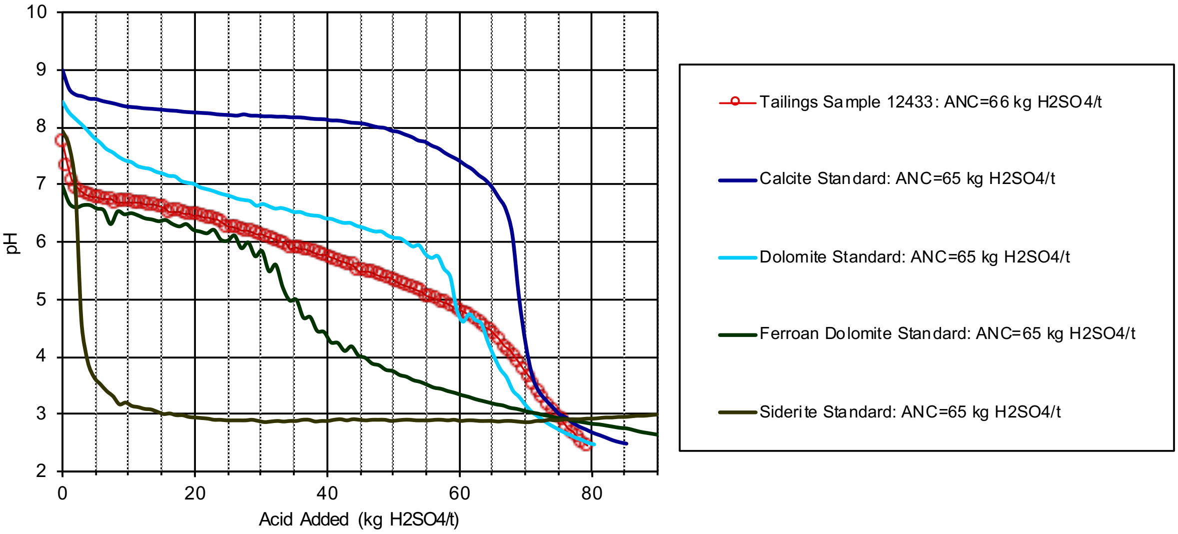

Figure A7: ABCC profile for sample 12433 with an ANC value close to 65 kg $H_2SO_4$/t. Carbonate standard curves are included for reference.

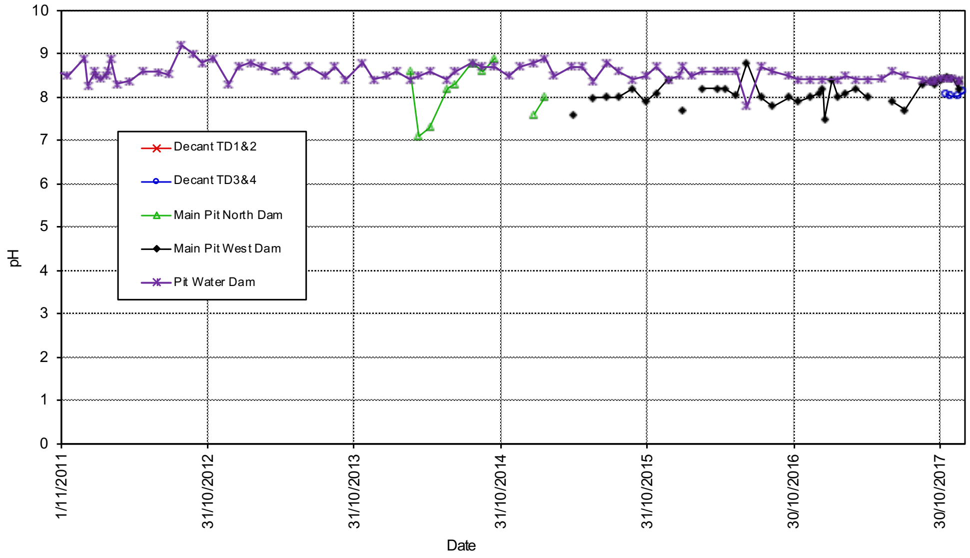

**Figure A8: pH trends for key pits and tailings storages.**

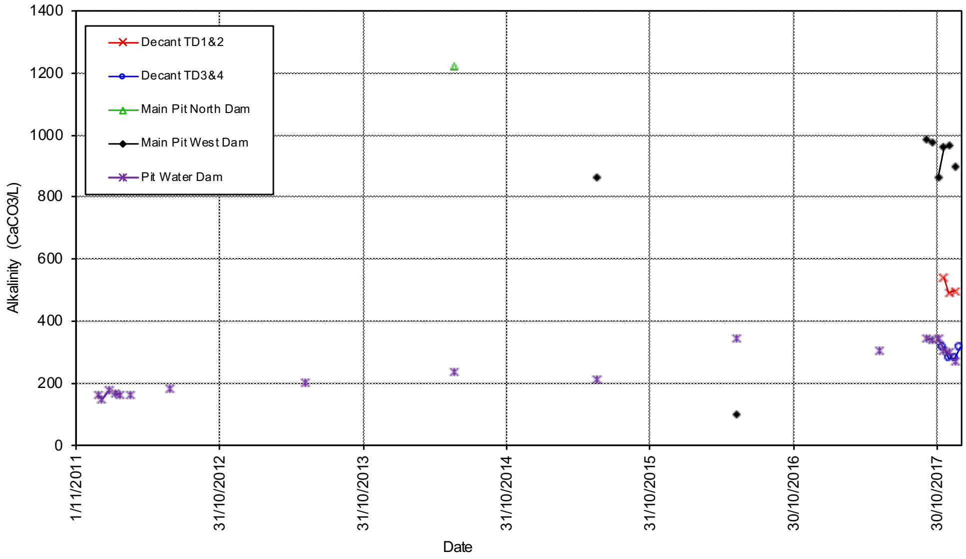

**Figure A9: Alkalinity concentration trends for key pits and tailings storages.**

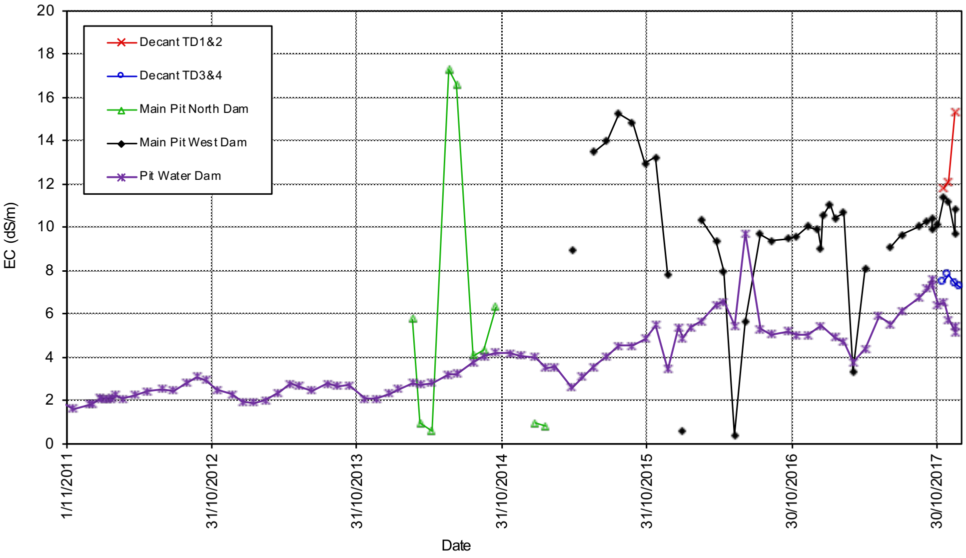

**Figure A10: Electrical conductivity trends for key pits and tailings storages.**

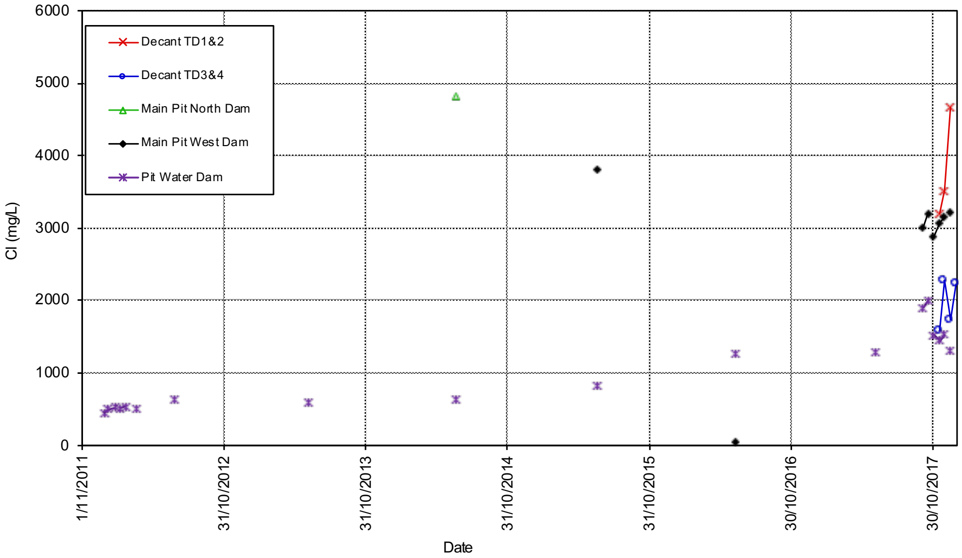

Figure A11: Cl concentration trends for key pits and tailings storages.

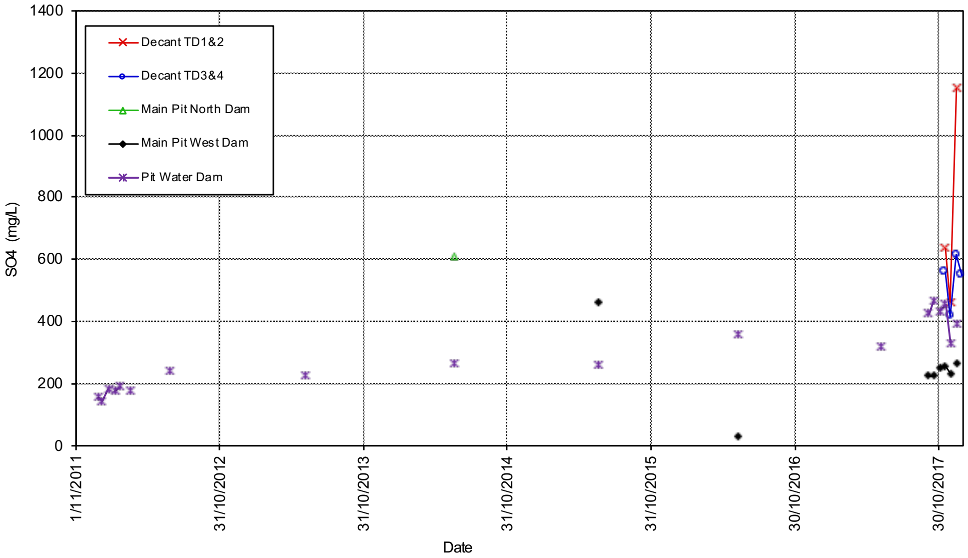

Figure A12: SO4 concentration trends for key pits and tailings storages.

# APPENDIX B

## Assessment of Acid Forming Characteristics

# Assessment of Acid Forming Characteristics

## Introduction

Acid rock drainage (ARD) is produced by the exposure of sulphide minerals such as pyrite to atmospheric oxygen and water. The ability to identify in advance any mine materials that could potentially produce ARD is essential for timely implementation of mine waste management strategies.

A number of procedures have been developed to assess the acid forming characteristics of mine waste materials. The most widely used methods are the Acid-Base Account (ABA) and the Net Acid Generation (NAG) test. These methods are referred to as static procedures because each involves a single measurement in time.

## Acid-Base Account

The acid-base account involves static laboratory procedures that evaluate the balance between acid generation processes (oxidation of sulphide minerals) and acid neutralising processes (dissolution of alkaline carbonates, displacement of exchangeable bases, and weathering of silicates).

The values arising from the acid-base account are referred to as the potential acidity and the acid neutralising capacity, respectively. The difference between the potential acidity and the acid neutralising capacity value is referred to as the net acid producing potential (NAPP).

The chemical and theoretical basis of the ABA are discussed below.

### *Potential Acidity*

The potential acidity that can be generated by a sample is calculated from an estimate of the pyrite ($FeS_2$) content and assumes that the pyrite reacts under oxidising conditions to generate acid according to the following reaction:

$$FeS_2 + 15/4\ O_2 + 7/2\ H_2O \Rightarrow Fe(OH)_3 + 2\ H_2SO_4$$

Based on the above reaction, the potential acidity of a sample containing 1 %S as pyrite would be 30.6 kilograms of $H_2SO_4$ per tonne of material (i.e. kg $H_2SO_4$/t). The pyrite content estimate can be based on total S and the potential acidity determined from total S is referred to as the maximum potential acidity (MPA), and is calculated as follows:

$$\text{MPA (kg } H_2SO_4/\text{t)} = (\text{Total \%S}) \times 30.6$$

The use of an MPA calculated from total sulphur is a conservative approach because some sulphur may occur in forms other than pyrite. Sulphate-sulphur, organic sulphur and native sulphur, for example, are non-acid generating sulphur forms. Also, some sulphur may occur as other metal sulphides (e.g. covellite, chalcocite, sphalerite, galena) which yield less acidity than pyrite when oxidised or, in some cases, may be non-acid generating.

The total sulphur content is commonly used to assess potential acidity because of the difficulty, costs and uncertainty involved in routinely determining the speciation of sulphur forms within samples, and determining reactive sulphide-sulphur contents. However, if the sulphide mineral forms are known then allowance can be made for non- and lesser acid generating forms to provide a better estimate of the potential acidity.

### *Acid Neutralising Capacity (ANC)*

The acid formed from pyrite oxidation will to some extent react with acid neutralising minerals contained within the sample. This inherent acid buffering is quantified in terms of the ANC.

The ANC is commonly determined by the Modified Sobek method. This method involves the addition of a known amount of standardised hydrochloric acid (HCl) to an accurately weighed sample, allowing the sample time to react (with heating), then back-titrating the mixture with standardised sodium hydroxide (NaOH) to determine the amount of unreacted HCl. The amount of acid consumed by reaction with the sample is then calculated and expressed in the same units as the MPA (kg $H_2SO_4$/t).

### *Net Acid Producing Potential (NAPP)*

The NAPP is a theoretical calculation commonly used to indicate if a material has potential to produce acidic drainage. It represents the balance between the capacity of a sample to generate acid (MPA) and its capacity to neutralise acid (ANC). The NAPP is also expressed in units of kg $H_2SO_4$/t and is calculated as follows:

$$\text{NAPP} = \text{MPA} - \text{ANC}$$

If the MPA is less than the ANC then the NAPP is negative, which indicates that the sample may have sufficient ANC to prevent acid generation. Conversely, if the MPA exceeds the ANC then the NAPP is positive, which indicates that the material may be acid generating.

### *ANC/MPA Ratio*

The ANC/MPA ratio is frequently used as a means of assessing the risk of acid generation from mine waste materials. The ANC/MPA ratio is another way of looking at the acid base account. A positive NAPP is equivalent to an ANC/MPA ratio less than 1, and a negative NAPP is equivalent to an ANC/MPA ratio greater than 1. A NAPP of zero is equivalent to an ANC/MPA ratio of 1.

The purpose of the ANC/MPA ratio is to provide an indication of the relative margin of safety (or lack thereof) within a material. Various ANC/MPA values are reported in the literature for indicating safe values for prevention of acid generation. These values typically range from 1 to 3. As a general rule, an ANC/MPA ratio of 2 or more signifies that there is a high probability that the material will remain circum-neutral in pH and thereby should not be problematic with respect to acid rock drainage.

*Acid-Base Account Plot*

Sulphur and ANC data are often presented graphically in a format similar to that shown in Figure A-1. This figure includes a line indicating the division between NAPP positive samples from NAPP negative samples. Also shown are lines corresponding to ANC/MPA ratios of 2 and 3.

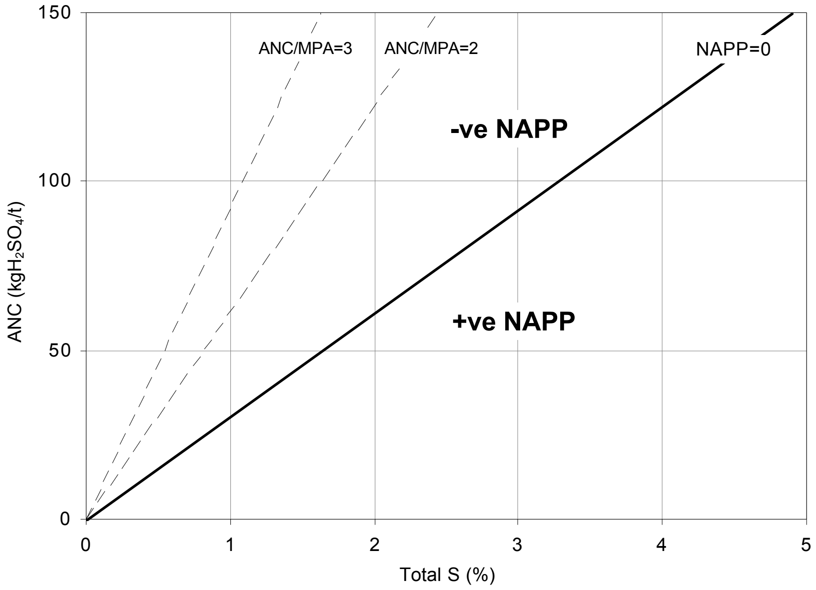

Figure A-1: Acid-base account (ABA) plot

## Net Acid Generation (NAG) Test

The NAG test is used in association with the NAPP to classify the acid generating potential of a sample. The NAG test involves reaction of a sample with hydrogen peroxide to rapidly oxidise any sulphide minerals contained within a sample. During the NAG test both acid generation and acid neutralisation reactions can occur simultaneously. The end result represents a direct measurement of the net amount of acid generated by the sample. The final pH is referred to as the NAGpH and the amount of acid produced is commonly referred to as the NAG capacity, and is expressed in the same units as the NAPP (kg $H_2SO_4$/t).

Several variations of the NAG test have been developed to accommodate the wide geochemical variability of mine waste materials. The four main NAG test procedures currently used by EGi are the single addition NAG test, the sequential NAG test, the kinetic NAG test, and the extended boil and calculated NAG test.

*Single Addition NAG Test*

The single addition NAG test involves the addition of 250 ml of 15% hydrogen peroxide to 2.5 g of sample. The peroxide is allowed to react with the sample overnight and the following day the sample is gently heated to accelerate the oxidation of any remaining sulphides, then vigorously boiled for several minutes to decompose residual peroxide. When cool, the NAGpH and NAG capacity are measured.

An indication of the form of the acidity is provided by initially titrating the NAG liquor to pH 4.5, then continuing the titration up to pH 7. The titration value at pH 4.5 includes acidity due to free acid (i.e. $H_2SO_4$) as well as soluble iron and aluminium. The titration value at pH 7 also includes metallic ions that precipitate as hydroxides at between pH 4.5 and 7.

### *Sequential NAG Test*

When testing samples with high sulphide contents it is not uncommon for oxidation to be incomplete in the single addition NAG test. This can sometimes occur when there is catalytic breakdown of the hydrogen peroxide before it has had a chance to oxidise all of the sulphides in a sample. To overcome this limitation, a sequential NAG test is often carried out. This test may also be used to assess the relative geochemical lag of PAF samples with high ANC.

The sequential NAG test is a multi-stage procedure involving a series of single addition NAG tests on the one sample (i.e. 2.5 g of sample is reacted two or more times with 250 ml aliquots of 15% hydrogen peroxide). At the end of each stage, the sample is filtered and the solution is used for measurement of NAGpH and NAG capacity. The NAG test is then repeated on the solid residue. The cycle is repeated until such time that there is no further catalytic decomposition of the peroxide, or when the NAGpH is greater than pH 4.5. The overall NAG capacity of the sample is then determined by summing the individual acid capacities from each stage.

### *Kinetic NAG Test*

The kinetic NAG test is the same as the single addition NAG test except that the temperature and pH of the liquor are recorded. Variations in these parameters during the test provide an indication of the kinetics of sulphide oxidation and acid generation. This, in turn, can provide an insight into the behaviour of the material under field conditions. For example, the pH trend gives an estimate of relative reactivity and may be related to prediction of lag times and oxidation rates similar to those measured in leach columns. Also, sulphidic samples commonly produce a temperature excursion during the NAG test due to the decomposition of the peroxide solution, catalysed by sulphide surfaces and/or oxidation products.

### *Extended Boil and Calculated NAG Test*

Organic acids may be generated in NAG tests due to partial oxidation of carbonaceous materials[1] such as coal washery wastes. This can lead to low NAGpH values and high acidities in standard single addition NAG tests unrelated to acid generation from sulphides. Organic acid effects can therefore result in misleading NAG values and misclassification of the acid forming potential of a sample.

The extended boil and calculated NAG tests can be used to account for the relative proportions of pyrite derived acidity and organic acidity in a given NAG solution, thus providing a more reliable

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[1] Stewart, W., Miller, S., Thomas, J.E., and Smart R. (2003), 'Evaluation of the Effects of Organic Matter on the Net Acid Generation (NAG) Test', in *Proceedings of the Sixth International Conference on Acid Rock drainage (ICARD), Cairns, 12-18th July 2003*, 211-222.

measure of the acid forming potential of a sample. The procedure involves two steps to differentiating pyritic acid from organic derived acid:

| | |
|---|---|
| Extended Boil NAG | decompose the organic acids and hence remove the influence of non-pyritic acidity on the NAG solution. |
| Calculated NAG | calculate the net acid potential based on the balance of cations and anions in the NAG solution, which will not be affected by organic acid. |

The extended boiling test is carried out on the filtered liquor of a standard NAG test, and involves vigorous boiling of the solution on a hot plate for 3-4 hours. After the boiling step the solution is cooled and the pH measured. An extended boil NAGpH less than 4.5 confirms the sample is potentially acid forming (PAF), but a pH value greater than 4.5 does not necessarily mean that the sample is non acid forming (NAF), due to some loss of free acid during the extended boiling procedure. To address this issue, a split of the same filtered NAG solution is assayed for concentrations of S, Ca, Mg, Na, K and Cl, from which a calculated NAG value is determined[2].

The concentration of dissolved S is used to calculate the amount of acid (as $H_2SO_4$) generated by the sample and the concentrations of Ca, Mg, Na and K are used to estimate the amount of acid neutralised (as $H_2SO_4$). The concentration of Cl is used to correct for soluble cations associated with Cl salts, which may be present in the sample and unrelated to acid generating and acid neutralising reactions.

The calculated NAG value is the amount of acid neutralised subtracted from the amount of acid generated. A positive value indicates that the sample has excess acid generation and is likely to be PAF, and a zero or negative value indicates that the sample has excess neutralising capacity and is likely to be NAF.

## Sample Classification

The acid forming potential of a sample is classified on the basis of the acid-base and NAG test results into one of the following categories:

- Barren;
- Non-acid forming (NAF);
- Potentially acid forming (PAF); and
- Uncertain (UC).

*Barren*

A sample classified as barren essentially has no acid generating capacity and no acid buffering capacity. This category is most likely to apply to highly weathered materials. In essence, it represents an ‘inert’ material with respect to acid generation. The criteria used to classify a sample

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[2] Environmental Geochemistry International, Levay and Co. and ACeSSS, 2008. *ACARP Project C15034: Development of ARD Assessment for Coal Process Wastes*, EGi Document No. 3207/817, July 2008.

as barren may vary between sites, but for hard rock mines it generally applies to materials with a total sulphur content ≤ 0.1 %S and an ANC ≤ 5 kg $H_2SO_4$/t.

*Non-acid forming (NAF)*

A sample classified as NAF may, or may not, have a significant sulphur content but the availability of ANC within the sample is more than adequate to neutralise all the acid that theoretically could be produced by any contained sulphide minerals. As such, material classified as NAF is considered unlikely to be a source of acidic drainage. A sample is usually defined as NAF when it has a negative NAPP and the final NAG pH ≥ 4.5.

*Potentially acid forming (PAF)*

A sample classified as PAF always has a significant sulphur content, the acid generating potential of which exceeds the inherent acid neutralising capacity of the material. This means there is a high risk that such a material, even if pH circum-neutral when freshly mined or processed, could oxidise and generate acidic drainage if exposed to atmospheric conditions. A sample is usually defined as PAF when it has a positive NAPP and a final NAGpH < 4.5.

*Uncertain (UC)*

An uncertain classification is used when there is an apparent conflict between the NAPP and NAG results (i.e. when the NAPP is positive and NAGpH > 4.5, or when the NAPP is negative and NAGpH ≤ 4.5). Uncertain samples are generally given a tentative classification that is shown in brackets e.g. UC(NAF).

Figure A-2 shows the format of the classification plot that is typically used for presentation of NAPP and NAG data. Marked on this plot are the quadrats representing the NAF, PAF and UC classifications.

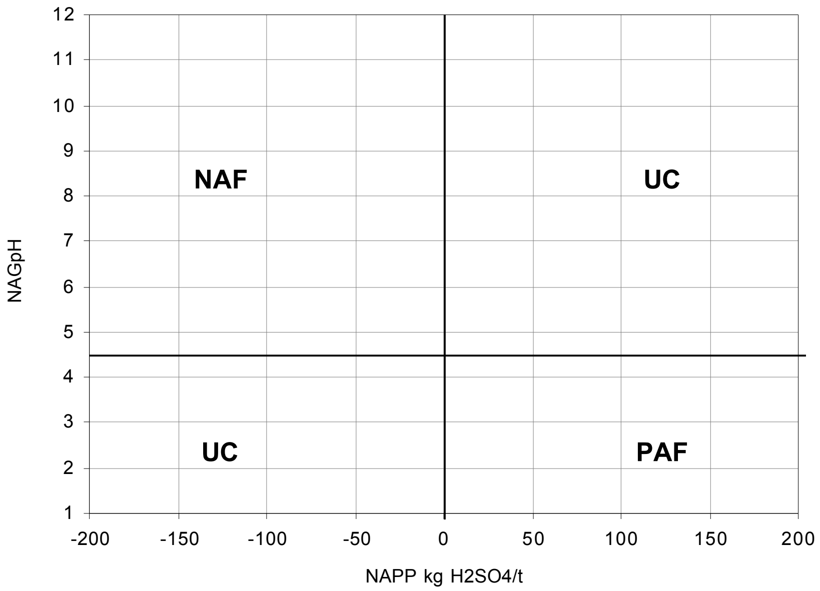

Figure A-2 ARD classification plot

## Other Methods

Other test procedures may be used to define the acid forming characteristics of a sample.

### *pH and Electrical Conductivity*

The pH and electrical conductivity (EC) of a sample is determined by equilibrating the sample in deionised water for a minimum of 12 hours (or overnight), typically at a solid to water ratio of 1:2 (w/w). This gives an indication of the inherent acidity and salinity of the waste material when initially exposed in a waste emplacement area.

### *Acid Buffering Characteristic Curve (ABCC) Test*

The ABCC test involves slow titration of a sample with acid while continuously monitoring pH. These data provides an indication of the portion of ANC within a sample that is readily available for acid neutralisation.