G.E.M.
Geo-Enviromental Management

COWAL GOLD MINE E42 MODIFICATION TAILINGS AND WASTE ROCK GEOCHEMICAL ASSESSMENT

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C1 INTRODUCTION

Geo-Environmental Management Pty Ltd (GEM) was commissioned by Barrick Australia Limited (Barrick) to conduct an assessment of the potential geochemical impacts from the proposed modification (E42 Modification) to the approved Cowal Gold Mine (CGM) in New South Wales (NSW), Australia. Changes implemented to the approved CGM by the E42 Modification would result in the modified CGM. This assessment supports an Environmental Assessment of the modified CGM which is being prepared by Resource Strategies Pty Ltd.

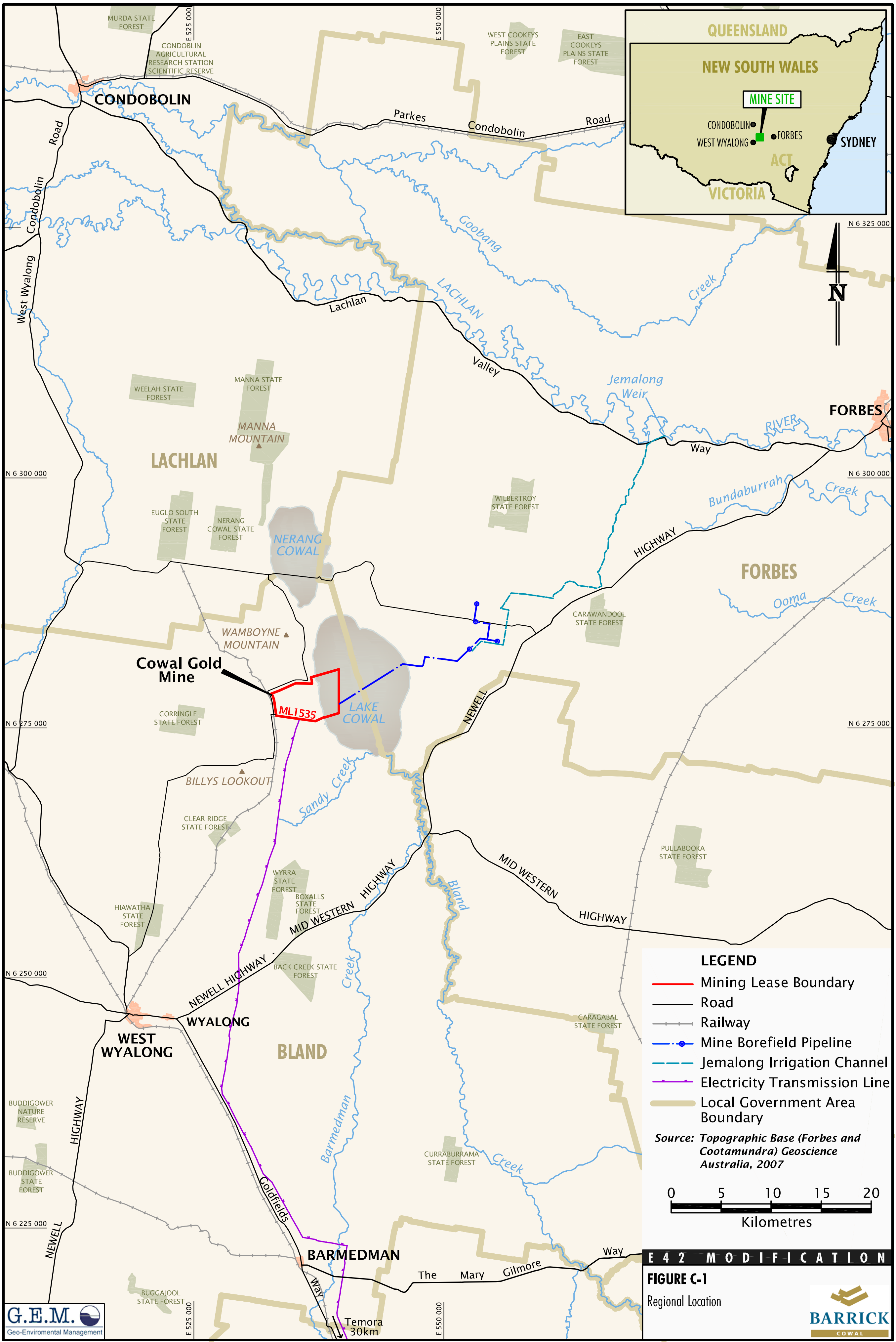
Approved Cowal Gold Mine

The CGM is located approximately 38 kilometres (km) north-east of West Wyalong in central NSW (Figure C-1). It is an open cut mining operation with a pit area of approximately 70 hectares (ha). Mining is currently approved to produce a total of approximately 76 million tones (Mt) of ore, approximately 128 Mt of waste rock and approximately 76 Mt of tailings over the life of the mine. Waste rock emplacements include: Perimeter Waste Emplacement, Northern Waste Emplacement and Southern Waste Emplacement. Tailings storage facilities include: Northern Tailings Storage Facility and Southern Tailings Storage Facility. Low-grade ore is stockpiled on a pad to the west of the pit, which covers an area of approximately 35 ha.

E42 Modification

The modified CGM is scheduled to commence in approximately Year 5 of CGM operations. The main changes to the approved CGM as a result of the E42 Modification would include those presented below:

- An increase to the operational mine life from 13 years to approximately 24 years.
- An increase in total production from approximately 76 million tonnes (Mt) of ore, to approximately 129 Mt of ore.
- An increase in the maximum ore processing rate from approximately 6.9 Mtpa to approximately 7.5 Mtpa.
- An increase in gold production from approximately 2.7 million ounces (Moz) of gold to approximately 3.5 Moz of gold
- An increase in the total surface area of the open pit from approximately 70 ha to approximately 130 ha, with final pit dimensions increased from approximately 1,000 m long, 850 m wide and 325 m deep to approximately 1,250 m long, 1,350 m wide and 440 m deep.
- An increase in the total volume of waste rock to be removed from the open pit from approximately 128 Mt to approximately 184 Mt.
- An increase in the height and area of the northern waste emplacement to an approximate final height of RL 275 m Australian Height Datum (AHD) (increased from RL 243 m AHD) and area of approximately 320 ha (increased from approximately 160 ha).



Source: *Topographic Base (Forbes and Cootamundra) Geoscience Australia, 2007*

E42 MODIFICATION

FIGURE C-1

Regional Location

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- An increase in the height and area of the southern waste emplacement to an approximate final height of RL 255 m AHD (increased from RL 223 m AHD) and area of approximately 140 ha (increased from approximately 120 ha).
- A reduction in the height of the perimeter waste emplacement in places.
- An increase in the total surface area of low grade ore stockpiles from approximately 35 ha to approximately 60 ha.
- An increase in the total volume of tailings produced from approximately 76 Mt to approximately 129 Mt.
- An increase in the heights of the northern and southern tailings storages to a final RL of 252 m (from approximately RL 233.5 m AHD) and 256 m (from approximately RL 241.5 m AHD), respectively.
- Extraction of saline water from a saline groundwater supply borefield located within ML 1535.
- Other associated minor changes to infrastructure, plant, equipment and activities.

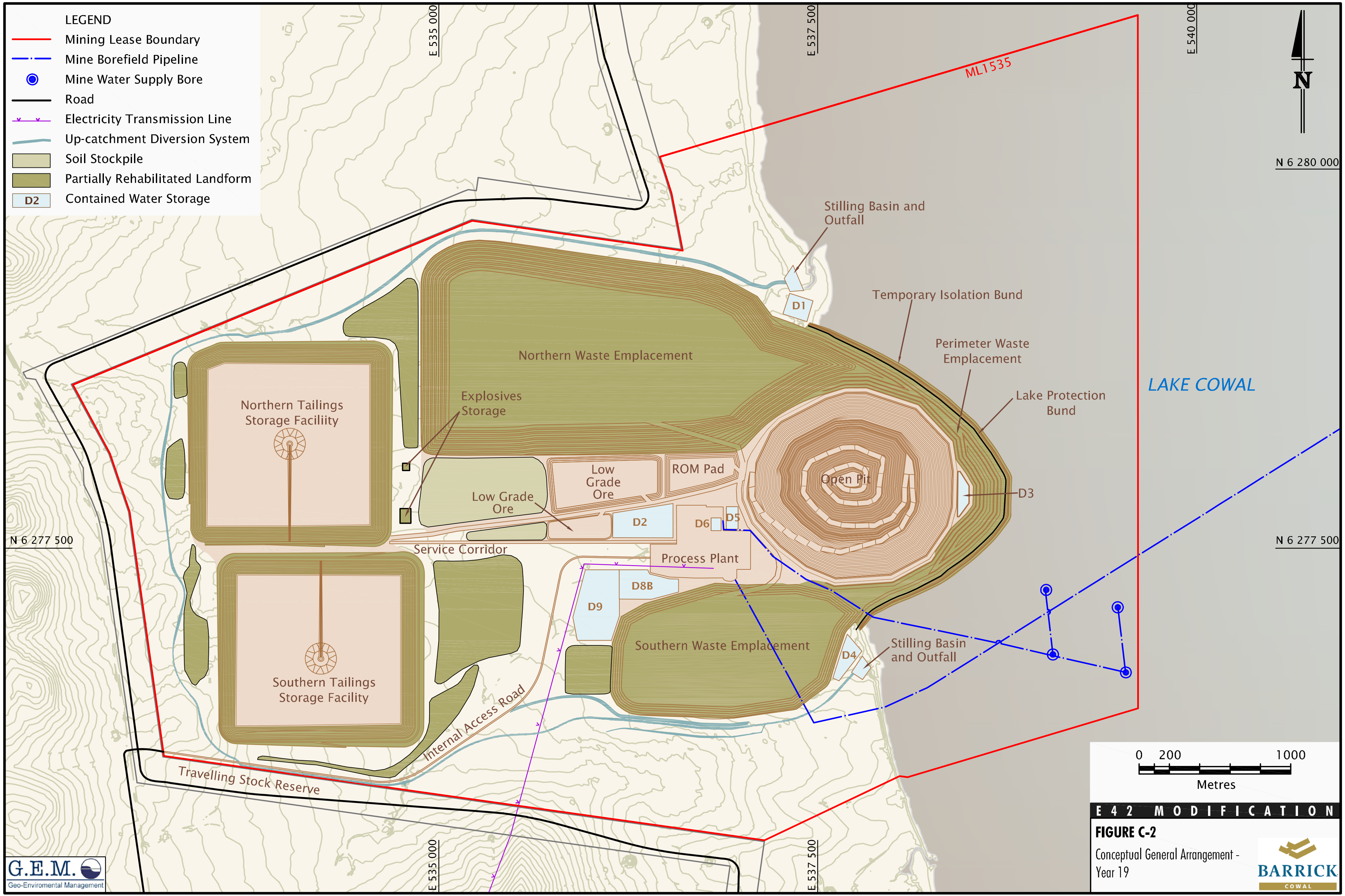
The modified CGM would involve the use of the INCO process as an alternative method for cyanide destruction. Detailed test work commissioned by Barrick to date indicates that this process is capable of cyanide destruction to the approved concentrations. The INCO process for the modified CGM involves the introduction of sulphur dioxide (SO_2) as sodium metabisulphite (SMBS).

The general arrangement of the modified CGM during Year 19 is shown on Figure C-2.

C1.1 Mineral Resource and Processing

The Endeavour 42 (E42) orebody is hosted by a sequence of Ordovician-aged interbedded volcaniclastics and lavas that have been intruded by several diorite and monzodiorite bodies and intersected by a number of mafic to intermediate dykes. The upper zone of these host rocks was weathered during the Tertiary period producing a lateritic profile consisting of saprolite and saprock ranging in depth from 1 to 80 m. Quaternary alluvial and lacustrine sediments cover the Tertiary laterite to depths of 1 to 3 m in the central parts of the deposit, increasing to thicknesses of 20 to 30 m on the flanks of the deposit.

Four major stratigraphic zones are recognised within the E42 orebody: Transported Cover (Quaternary sediments), Soft Oxide (SOX) (Tertiary lateritic saprolite), Hard Oxide (HOX) (Tertiary lateritic saprock) and Primary Rock (Fresh Ordovician sequence and intrusive). Gold mineralisation predominantly occurs within the Primary and Hard Oxide zones. The approximate depth ranges and descriptions of these zones are provided in Table C-1. These zones are used for mining operations and waste rock management at the approved CGM.



E 4 2 MODIFICATION

FIGURE C-2

Conceptual General Arrangement - Year 19

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BARRICK COWAL

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Table C-1: Major stratigraphic zones of the E42 orebody.

Zone	**Depth from Surface***	**Description**
Transported Cover	0 to 40m	Alluvial and lacustrine sediments
SOX	5 to 60 m	Poorly consolidated saprolite consisting of volcaniclastics, lava and diorite
HOX	40 to 80 m	Moderate to well indurated saprock consisting of volcaniclastics, lava and diorite
Primary Rock	>40 m	Fresh rock comprising four dominant stratigraphic units: Upper Volcaniclastics Lava Lower Volcaniclastics Diorite and Monzodiorite

*Overlap in depths due to variability across the orebody.

Gold is recovered from the ore after it is crushed and milled using the Carbon-In-Leach (CIL) process. The milled oxide ore is passed directly through the CIL circuit and, after being treated to destroy the CN to approved concentrations, is discharged to the tailings storage facilities. The primary ore is passed through a flotation circuit to produce a gold bearing concentrate before being leached through the CIL circuit. The tailings from the flotation process and the primary ore residue from the CIL plant, which is treated to destroy CN to approved concentrations, are blended before being discharged to the tailings storage facilities. The location of the tailings storage facilities is shown on Figure C-2.

Cyanide Use and Destruction

Caro's Acid is currently used at the approved CGM to destroy cyanide. Caro's Acid is a mixture of sulphuric acid (H_2SO_4) and hydrogen peroxide (H_2O_2), both of which are separately transported and stored on site. The two cyanide destruction points are served by a Caro's Acid mixing chamber at each point. The main by-product from the Caro's Acid destruction process is cyanate which itself decays through natural processes.

The modified CGM would involve the use of the INCO process as an alternative method for cyanide destruction. The INCO process involves the introduction of SO_2 as SMBS. The reaction is catalysed by the presence of copper, which may have to be added (as copper sulphate). Air is also required, as is lime to maintain the optimal pH range (i.e. pH between 8 and 10) as the reaction proceeds. The two cyanide destruction points would be served by an INCO mixing chamber at each point. Similar to Caro's Acid, the main by-product from the INCO destruction process is cyanate which decays through natural processes. The geochemistry of the tailings resulting from the use of the INCO process is discussed in Section C5.4.2.

The quantity of reagents added to the tailings (for either the Caro's Acid or INCO processes) would continue to be regulated by an on-line free cyanide measurement to ensure cyanide in the tailings is destroyed down to approved concentrations.

C1.2 Study Objectives

The two primary objectives of this study are to:

- Describe the findings from the previous geochemical assessment programs conducted for the Cowal Gold Project Environmental Impact Statement (the EIS) (North Limited, 1998) and subsequent confirmatory geochemical assessment programs.
- Assess and quantify, where relevant, any changes in the potential geochemical impacts to the approved CGM that may result from the E42 Modification. This assessment is based on the results and findings of geochemical characterisation test work carried out on representative waste rock (drill-hole) and tailings samples. The specific objectives of this assessment are to:
    1. Assess the potential changes to waste rock geochemistry including acid forming characteristics, element enrichment and solubility, salinity release characteristics, and leachate chemistry.
    2. Assess the potential changes to tailings geochemistry including acid-forming characteristics, element enrichment, liquor chemistry, nature and fate of cyanide, and leachate chemistry.
    3. Assess the potential changes to the capacity of the sub-soil beneath the tailings storage facilities and waste rock emplacements to attenuate metals due to changes in waste rock and/or tailings geochemistry.
    4. Assess the potential changes to pit void water quality based on the geochemistry of representative drill-hole samples.

C2 PREVIOUS GEOCHEMICAL INVESTIGATIONS

Detailed geochemical investigations were conducted by Environmental Geochemistry International Pty Ltd (EGi) prior to commencing mining operations at the approved CGM. These investigations were commissioned by the previous owners of CGM (North Limited) and were included in the EIS. They provided the required information for mine planning and design and for approval of the mining operation. Subsequent geochemical investigations, commissioned by Barrick, were also conducted by EGi in September 2004 (EGi, 2004) and by O'Kane Consultants Pty Ltd (O'Kane) in January 2008 (O'Kane, in prep.). The objective of these investigations was to provide confirmation of the previous findings and expand the waste rock and tailings geochemical databases required for mining operations, environmental management and closure planning. Table C-2 provides the detail on the geochemical investigations carried out to date.

C2.1 pH, Salinity and Acid Forming Characteristics

The geochemical investigations carried out for the EIS indicated that reactive sulphides occur in the primary waste rock and combined primary tailings (i.e. combined flotation and CIL tailings from primary ores). However, due to the presence of moderate to high acid neutralising capacity (ANC) in these materials they are expected to be non-acid forming (NAF). It was also reported that the oxide waste rock and tailings (i.e. CIL tailings from oxide ores) are expected to be NAF due to low reactive sulphides concentrations. However, these materials were found to be relatively saline. These findings were based on testing a total of 101 samples representing soil, waste rock, ore (including low grade ore) and tailings carried out by EGi in 1995, 1996 and 1997 (see Table C-2) (EGi, 1995; 1996; 1997). Subsequent confirmatory test work carried out in 2004 (100 waste rock and 8 tailings samples) and in 2008 (30 waste rock samples) confirmed the expected salinity and acid forming characteristics of these materials (see Table C-2) (EGi, 2004; O'Kane, in prep.).

C2.2 Metal Enrichment and Solubility

Elemental analyses carried out on selected samples for the EIS indicated that the oxide and primary waste rock is expected to have high concentrations of arsenic (As) (EGi, 1997). The EIS predicted high concentrations of As, boron (B), lead (Pb) and antimony (Sb) in the oxide tailings and silver (Ag), As, cadmium (Cd), molybdenum (Mo), Pb, sulphur (S), Sb and zinc (Zn) in the primary tailings. These elements were also found to be enriched in the waste rock and tailings samples from the confirmatory test work carried out by EGi in 2004 (EGi, 2004).

Table C-2: Geochemical investigation conducted for the approved CGM to date.

Geochemical Investigations	Date of Test Work	Samples Included	Test Work
Environmental Geochemical Assessment of Process Tailings, Mine Rock and Surface Zone Materials (EGi, 1995)	Jun-95	87 Mine Rock, Ore and Surface Materials	Salinity, ABA, NAG, Multi-Elements
		2 Tailings	Salinity, ABA, NAG, Multi-Elements, Leach Columns, CN Attenuation
Environmental Geochemical Assessment of Simulated Tailings (EGi, 1996)	Jan-96	1 Oxide Tailings	ABA, NAG, Multi-Elements, CN Speciation and Decay
Environmental Geochemistry Assessment of Proposed Mining Activities (EGi, 1997) Appendix C in the Cowal Gold Project EIS (North Limited, 1998)	Sep-97	4 Tailings	ABA, NAG, Multi-Elements, CN Decay, Leach Columns
		2 TSF Sub-Soils	ABA, Soil Chemistry, Attenuation Characteristics
		3 Construction Materials	ABA, Soil Chemistry
		2 Waste Rock Composites (Primary and Oxide)	Salinity, ABA, Sequential Batch Extraction
Final Void Water Chemistry (EGi, 1998)	Oct-98	1 Groundwater	Pit Water Chemistry Modelling
Geochemical Assessment of Waste Rock and Process Tailings (EGi, 2004)	Sept-04	100 Mine Rock and Ore	Salinity, ABA, NAG, Multi-Elements
		8 Tailings	ABA, NAG, Multi-Elements
Geochemical Assessment of Waste Rock (O'Kane, in prep.)	Jan-08	30 Mine Rock and Ore	Salinity, ABA, NAG

Note: ABA – Acid-Base Account NAG – Net Acid Generation

The potential for release of environmentally important elements from waste rock and tailings was investigated in the 1995 and 1997 geochemical programs carried out by EGi. The results of single and sequential batch extractions of waste rock samples with deionised water indicated low water solubility for As under natural conditions of neutral or slightly alkaline pH. It was therefore concluded that leaching of environmentally important elements from waste rock at the approved CGM is unlikely to be of concern provided near neutral pH values are maintained.

Column leach tests carried out on the oxide and primary tailings over a 20-week period also indicated an initial flush of soluble Cu and Zn from the oxide tailings and soluble copper (Cu) from the primary tailings (EGi, 1997). EGi concluded that this release is most likely associated with the residual cyanide in the tailings liquor and does not represent a long-term concern (EGi, 1997).

C2.3 Tailings Liquor Chemistry and Nature and Fate of Cyanide in Tailings

The results of test work conducted by EGi in 1996 and 1997 (see Table C-2) to determine the characteristics of CN decay in the tailings liquor indicated that CN would decay rapidly in the ponded decant liquor (i.e. in the tailings storage facility) and in the reclaim water (i.e. in the contained water storage) due primarily to the low metal content and high proportion of free CN (75% of the weak acid dissociable cyanide [CN_{WAD}] is present as free CN) (EGi, 1997). Based on this test work it was reported that on-going CN decay in the tailings storage facilities would result in CN_{WAD} concentrations in the reclaim water from 5 to 10 milligrams per litre (mg/L) when processing oxide ore and 10 to 15 mg/L when processing primary ore (*ibid*.). Once discharge ceases it is expected that within 2 to 3 months the CN_{WAD} complexes in the ponded decant would decay to very low concentrations (*ibid*.).

C2.4 Tailings Storage Facilities Attenuation Characteristics

Sequential batch extraction tests using sub-soil materials from the tailings storage facilities and tailings liquors were carried out by EGi in 1997 to determine the attenuation capacity of the material underlying the tailings storages (i.e. to evaluate the capacity of the tailings storage facilities sub-soils to attenuate CN and soluble elements). The procedure used for these tests involved exposing tailings liquor from the oxide residues and the primary recombined and flotation residues to representative sub-soil material collected from the proposed tailings storage facilities (EGi, 1997). Liquors from the residues were used to investigate the short to medium and longer term chemistry (*ibid*.).

This test work indicated that CN is only poorly attenuated whereas Cu, Zn and As are generally strongly attenuated by the soils underlying the tailings storage facilities (EGi, 1997).

C2.5 Pit Void Water Quality

Final pit wall rocks for the approved CGM would predominantly comprise coarse volcaniclastics and intrusives (diorite/gabbro) (EGi, 1997). Previous testwork for the approved CGM (EGi, 1995) indicates that these rock types are likely to be NAF. From a total 56 coarse volcaniclastics and intrusive samples tested only one was classified as PAF and this sample only had a low capacity to generate acid.

Even though the primary rock is expected to be NAF, the rock contains (on average) about 1 %S and an ANC of about 100 kilograms of H_2SO_4 per tonne (kg H_2SO_4/t) (EGi, 1997). Due to the presence of these reactive sulphides, oxidation of primary rock exposed along the pit walls is likely to generate a moderate load of sulphate salts (*ibid.*). However, EGi considered that, given the regional groundwater has a total dissolved solids (TDS) count of around 40,000 to 45,000 mg/L, the contribution of sulphate salts from the pit walls to the overall TDS of pit water is expected to be negligible (*ibid.*).

C2.6 Previous Recommendations for Management

Based on the findings from the previous geochemical investigations the following recommendations for management were provided (EGi, 2004):

- The results indicate a very low likelihood of acid rock drainage (ARD) generation from waste rock and tailings, therefore no special management requirements would be required for ARD control at the approved CGM. However, operational monitoring and testing should be a carried out on an occasional and as needed basis to confirm the low ARD potential of waste rock with particular focus on any unexpected rock types or alteration types which may be exposed during mining.
- The oxide waste rock has relatively high natural salinity and the primary waste rock has the potential to generate soluble salts (i.e. due to the presence of reactive sulphides, sulphate salts such as gypsum would be generated if these materials are left exposed to surficial weathering processes).
- The waste rock and tailings are expected to be enriched with As and some of these materials are also expected to be enriched with other elements including Cd, Mo, Sb, Pb and Zn. These elements should be included in the site water monitoring program on an occasional basis to confirm the expected low leaching potential of these elements.

C3 GEOCHEMICAL ASSESSMENT PROGRAM

C3.1 Testing Methodology and Program

The geochemical testing program (i.e. sampling and laboratory analysis) for the E42 Modification (i.e. for the proposed modified pit) was implemented by O'Kane and Barrick (for the INCO simulation) in early 2008. The laboratory program for this assessment included the following tests and procedures:

Solid Samples

- pH, electrical conductivity (EC) and alkalinity determination.
- Acid-base analysis (Sulphur forms, ANC, net acid producing potential [NAPP]).
- Single addition NAG test.
- Multi-element scans on solids and water extracts.
- Cyanide (total CN, CN_{WAD}) analysis of water extracts (tailings only).

Water Samples

- pH, EC and alkalinity determination.
- Multi-element scans.
- Cyanide (total CN, CN_{WAD}) analysis.

These analyses were performed by ALS Laboratory Group in Brisbane, Queensland and CSIRO Minerals (Hydrometallurgy), Gold Australian Minerals Research Centre (for the INCO simulation).

Following is an overview of the geochemical testing procedures of this assessment.

pH and Electrical Conductivity Determination

The pH and EC of a sample is determined by equilibrating a solid sample in deionised water for a minimum of 2 hours. Variations to this test include mixing the solids with water at a ratio of 1:2 or 1:5 by weight (w/w), or as a saturated paste. Typically a ratio of 1:2 is used for providing an indication of the inherent acidity and salinity of a material when it is initially exposed. The salinity rankings based on EC values from 1:5 extracts ($EC_{1:5}$), 1:2 extracts ($EC_{1:2}$) and saturation extracts (EC_{sat}) are provided below:

$EC_{1:5}$ (dS/m)	**$EC_{1:2}$ (dS/m)**	**EC_{sat} (dS/m)**	**Salinity**
< 0.2	< 0.5	< 2.0	Non-Saline
0.2 to 0.3	0.5 to 1.5	2 to 4.0	Slightly Saline
0.3 to 0.4	1.5 to 2.5	3 to 8.0	Moderately Saline
> 0.4	> 2.5	> 8.0	Highly Saline

(Rhoades *et al.*, 1999) dS/m = deci-siemens per metre

Acid Forming Characteristic Evaluation

A number of test procedures are used to assess the acid forming characteristics of mine waste rock materials. The most widely used assessment methods are the ABA and the NAG test. These methods are referred to as static procedures because each involves a single measurement in time.

Acid-Base Account (ABA)

The ABA involves 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 ABA are referred to as the maximum potential acidity (MPA) and the ANC, respectively. The difference between the MPA and ANC value is referred to as the NAPP.

The MPA is calculated using the total sulphur content of the sample. This calculation assumes that all of the sulphur measured in the sample occurs as pyrite (FeS_2) and that the pyrite reacts under oxidising conditions to generate acid according to the following reaction:

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

According to this reaction, the MPA of a sample containing 1 % sulphur (%S) as pyrite would be 30.6 kg H_2SO_4/t of material. Hence the MPA of a sample is calculated from the total sulphur content using the following formula:

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

The use of the total sulphur assay to estimate the MPA is a conservative approach because some sulphur may occur in forms other than pyrite. Sulphate-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) that yield less acidity than pyrite when oxidised.

The acid formed from pyrite oxidation will, to some extent, react with acid neutralising minerals contained within the sample. This inherent acid neutralisation is quantified in terms of the ANC and is commonly determined using 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).

The NAPP is a theoretical calculation commonly used to indicate if a material has the potential to produce acid. 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.

The ANC/MPA ratio is used as a means of assessing the risk of acid generation from mine waste materials. 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. Generally, an ANC/MPA ratio of 3 or more signifies that there is a high probability that the material is not acid generating.

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. Therefore, the end result represents a direct measurement of the net amount of acid generated by the sample. This value is commonly referred to as the NAG capacity and is expressed in the same units as NAPP, that is kg H_2SO_4/t.

The standard NAG test involves the addition of 250 millilitres (mL) of 15% hydrogen peroxide to 2.5 grams (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 pH and acidity of the NAG liquor are measured. The acidity of the liquor is then used to estimate the net amount of acidity produced per unit weight of sample.

Metal Enrichment and Solubility

Multi-element scans are carried out to identify any elements that are present in a material at concentrations that may be of environmental concern with respect to water quality and revegetation. The assay results from the solid samples are compared to the average crustal abundance for each element to provide a measure of the extent of element enrichment (Bowen, 1979). The extent of enrichment is reported as the Geochemical Abundance Index (GAI). However, identified element enrichment does not necessarily mean that an element would be a concern for revegetation or water quality and this technique is used to identify any significant element enrichments that warrant further examination.

Multi-element scans are also performed on water extracts from the solid samples (1:2 sample/deionised water) to determine the immediate element solubilities under the existing sample pH conditions of the sample. However, common environmentally important elements that are not identified as significantly enriched or immediately soluble may still present an environmental risk under low pH conditions and where acid forming materials are identified, additional (kinetic) testing is required to develop an understanding of the geochemical behaviour of these materials.

C3.2 Geochemical Classification

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

- Barren.
- NAF.
- Potentially Acid Forming (PAF).
- Acid Forming (AF).
- Uncertain (UC).

Barren

A sample classified as barren essentially has no acid generating capacity and no acid buffering capacity and is considered ‘inert’. This category typically applies to highly weathered materials. The criteria used to classify a sample as barren may vary between sites, but 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 of the acid that could theoretically be produced by any contained sulphide minerals. As such, material classified as NAF is considered unlikely to be a source of acid drainage. A sample is usually defined as NAF when it has a negative NAPP and a final NAGpH ≥ 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 ANC 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.

Acid Forming (AF)
A sample classified as AF has the same characteristics as the PAF samples but also has an existing pH of less than 4.5. This indicates that acid conditions have already been developed, confirming the acid forming nature of the sample.

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).

C3.3 Sample Selection and Preparation

Selection and collection of representative waste rock and tailings (including tailings decant) samples for the E42 Modification geochemical assessment program was co-ordinated by O'Kane and Barrick (for the INCO simulation).

C3.3.1 Waste Rock Samples

A total of 30 drill-hole samples (including 28 waste rock and 2 ore samples) were collected in 1 m intervals from 13 drill-holes representing the different material types that occur within the modified pit area. Table C-3 provides the sample and drill-hole code, depth, lithology, weathering and material type for the drill-hole samples.

The drill-hole samples were prepared for testing by ALS Laboratory Group in Brisbane, Queensland. The samples were crushed to minus 4 millimetres (mm) and a split was then pulverised to minus 75 micrometres (µm) prior to testing.

Table C-3: Drill-hole sample detail for the E42 Modification assessment.

Sample Code	**Drill-Hole**	**Depth (m)**		**Lithology**	**Weathering**	**Material Type**
		from	**to**			
ENV-101	E42D1015	250	251	Diorite	Primary	Waste
ENV-102	E42D1017	455	456	Diorite	Primary	Waste
ENV-103	E42D1007	260	261	Diorite	Primary	Waste
ENV-104	E42D1067	212	213	Diorite	Primary	Waste
ENV-105	E42D1328	375	376	Diorite	Primary	Waste
ENV-106	E42D1136	220	221	Lava	Primary	Waste
ENV-107	E42D1074	195	196	Lava	Primary	Waste
ENV-108	E42D1027	226	227	Lava	Primary	Waste
ENV-109	E42D1062	135	136	Andesitic Volcanics	Primary	Waste
ENV-110	E42D1062	150	151	Andesitic Volcanics	Primary	Waste
ENV-111	E42D1062	170	171	Andesitic Volcanics	Primary	Waste
ENV-112	E42D1027	174	175	Upper Volcaniclastics	Primary	Waste
ENV-113	E42D1029	213	214	Upper Volcaniclastics	Primary	Waste
ENV-114	E42D1029	138	139	Upper Volcaniclastics	Primary	Waste
ENV-115	E42D1029	146	147	Upper Volcaniclastics	Primary	Waste
ENV-116	E42D1136	170	171	Upper Volcaniclastics	Primary	Waste
ENV-117	E42D1312	217	218	Upper Volcaniclastics	Primary	Waste
ENV-118	E42D1312	139	140	Upper Volcaniclastics	Primary	Waste
ENV-119	E42D1312	50	51	Saprolite	Oxidised	Waste
ENV-120	E42D1312	100	101	Saprock	Oxidised	Waste
ENV-121	E42D1346	83	84	Saprock	Oxidised	Waste
ENV-122	E42D1328	320	321	Dyke Material	Primary	Waste
ENV-123	E42D1062	36	37	Saprolite	Oxidised	Waste
ENV-124	E42D1062	192	193	Andesitic Volcanics	Primary	Waste
ENV-125	E42D1054	171	172	Upper Volcaniclastics	Primary	Waste
ENV-126	E42D1054	242	243	Upper Volcaniclastics	Primary	Ore
ENV-127	E42D1062	232	233	Upper Volcaniclastics	Primary	Ore
ENV-128	E42D1074	262	263	Lower Volcaniclastics	Primary	Waste
ENV-129	E42D1346	120	121	Lower Volcaniclastics	Primary	Waste
ENV-130	E42D1346	140	141	Lower Volcaniclastics	Primary	Waste

C3.3.2 Tailings and Tailings Decant Samples

A total of 5 tailings solids and 2 tailings decant samples were collected from the existing Northern and Southern Tailings Storage Facilities. Table C-4 provides the sample detail. The tailings solids and decant samples were prepared for testing by ALS Laboratory Group in Brisbane, Queensland. The solid samples were dried and pulverised to minus 75 µm prior to testing.

Table C-4: Tailings and tailings decant sample detail for the E42 Modification assessment.

Sample Code	Sample Type	Sample Location	Sample Description
NTSF-1	Tailings Solids	Discharge Point for NTSF	Primary Tailings
NTSF-2	Tailings Solids	North Beach of NTSF	Primary Tailings
NTSF-3	Tailings Solids	West Beach of NTSF	Primary Tailings
NTSF-4	Tailings Decant	Decant Recovery Point of NTSF	Decant
STSF-1	Tailings Solids	Western Cell of STSF	Oxide Tailings
STSF-2	Tailings Solids	Western Cell of STSF	Salt Precipitate
STSF-3	Tailings Decant	Decant Recovery Point of STSF	Decant

NTSF - Northern Tailings Storage Facility STSF - Southern Tailings Storage Facility

C3.3.3 INCO Simulation Samples

One litre of tailings was collected from the process plant prior to CN destruction for the INCO simulation test work. The sample was prepared and the simulation test work was conducted by CSIRO Minerals (Hydrometallurgy), Gold Australian Minerals Research Centre. A total of 2.85 g of SMBS (300% stoichiometry) was added to the tailings in three equal portions at 0, 15 and 30 minutes (CSIRO Minerals [Hydrometallurgy], 2008) to confirm that the total CN in the tailings could be sufficiently reduced to approved concentrations or below in the discharged tailings. After 55 minutes the pH of the tailings was 7.4 and was therefore adjusted to 9.0 using NaOH. Samples of the treated tailings were then collected at 1, 2 and 24 hours for CN and selected element analyses (*ibid*.).

C4 WASTE ROCK GEOCHEMISTRY

The geochemical test results for the 28 waste rock samples are provided in Attachment CA and a summary of the pH of saturation extract (pH_{sat}) and EC_{sat}, and acid forming characteristic results is presented in Table C-5.

Table C-5: Summary of the pH and EC, and acid forming characteristics of the waste rock samples for the E42 Modification.

Material Type		**pH_{sat}**	**EC_{sat} (dS/m)**	**Total S (%S)**	**MPA**	**ANC**	**NAPP**
					(kg H_2SO_4/t)		
Oxidised and Primary Waste Rock 28 Samples	min	7.1	0.120	0.02	1	1	-133
	max	9.4	4.530	1.60	49	158	22
	mean	9.1	0.636	0.47	15	69	-54
Oxidised Waste Rock 4 Samples	min	7.1	0.266	0.03	1	1	-15
	max	9.0	4.530	0.07	2	16	0
	mean	8.1	2.724	0.04	1	7	-5
Primary Waste Rock 24 Samples	min	8.7	0.120	0.08	1	27	-133
	max	9.4	0.617	1.60	49	158	22
	mean	9.1	0.288	0.55	17	79	-63

C4.1 pH and Salinity

The pH_{sat} of the waste rock samples ranges from 7.1 to 9.4 with a median value of 9.1. These pH values are relatively consistent for the different material types although the typical pH of the oxidised waste rock is generally 1 pH unit lower (pH_{sat} 8.1) than the primary (fresh) waste rock. These results indicate that the waste rocks represented by the samples tested are likely to have neutral to slightly alkaline pH which is consistent with the results from the previous geochemical investigations with a pH range of 6.6 to 9.7 with a median value of 8.2 (EGi, 1995; 2004).

The EC_{sat} values for the waste rock samples ranges from 0.120 to 4.530 dS/m with a mean value of 0.636 dS/m. As expected, the EC_{sat} values for the oxidised waste rock samples are significantly higher (average 2.724 dS/m) than for the primary (fresh) waste rock (mean 0.288 dS/m). These results indicate that the oxidised waste rocks represented by the samples tested range from non- to moderately saline and the primary waste rock is non-saline. These results are consistent with the results from the previous investigations which reported that the oxidised waste rock was typically moderately saline with an average $EC_{1:2}$ of 1.590 dS/m and the primary waste rock was non-saline with an average $EC_{1:2}$ of 0.334 dS/m (EGi, 1995; 2004).

Seven of the waste rock samples were selected to determine the chemical composition of their water extracts at a ratio of 1 part sample to 2 parts deionised water. The pH, EC, alkalinity and element composition of the water extracts were determined and the results are presented in Attachment CA. These results indicate that the samples have a range in alkalinities from 2 to 186 milligrams of calcium carbonate per litre (mg $CaCO_3$/L) with the oxidised waste rock samples generally having lower alkalinities than the primary waste rock. These results also show that the salinity of the oxidised waste rock is attributed primarily to salts of sodium chloride (NaCl) and, to a lesser degree, calcium sulphate ($CaSO_4$).

C4.2 Acid Forming Characteristics

Figure C-3 is a plot of the total sulphur compared to sulphide sulphur concentrations in all of the drill-hole samples. This plot shows that the majority of the sulphur in these samples (86 to 100%) occurs as sulphide.

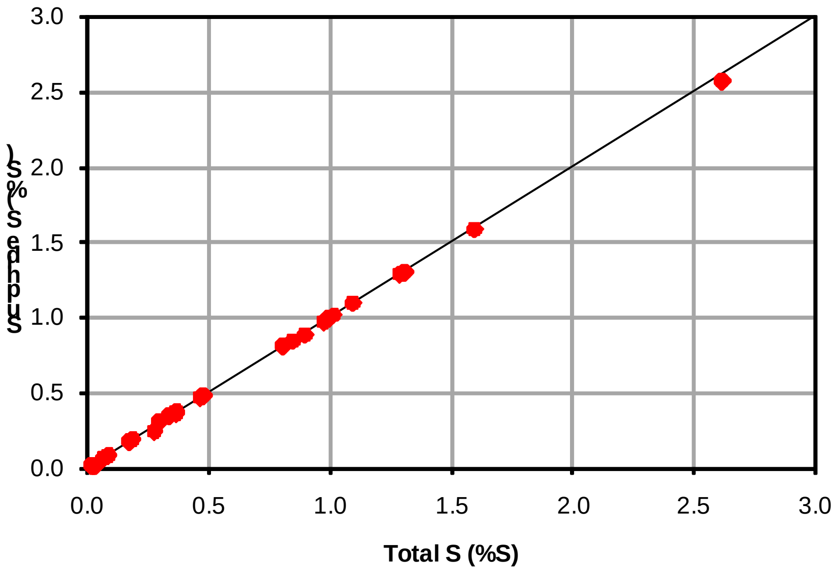

Figure C-3: Total sulphur content compared to sulphide sulphur content in the E42 Modification drill-hole samples.

The total sulphur content of the waste rock samples ranges from 0.02 to 1.60 %S with an average of 0.47 %S. The oxidised waste rock has lower sulphur concentrations than the primary waste rock consistent with the findings from the previous geochemical investigations. The sulphur contents range from 0.03 to 0.07 %S (average 0.04 %S) in the oxidised waste rock samples and from 0.08 to 1.60 %S (average 0.55 %S) in the primary waste rock samples. The sulphur contents in the oxidised waste rock samples from the previous geochemical investigations range from 0.01 to 0.32 %S (average 0.03 %S) and in the primary waste rock range from 0.01 to 4.77 %S (average 0.57 %S), (EGi, 1995; 2004).

The ANC of the waste rock samples ranges from 1 to 158 kg H_2SO_4/t with an average of 69 kg H_2SO_4/t. The primary waste rock samples have relatively high ANC values, ranging from 27 to 158 kg H_2SO_4/t with an average of 79 kg H_2SO_4/t, compared to the oxidised waste rock samples which have relatively low ANC values ranging from 1 to 16 kg H_2SO_4/t with an average of 7 kg H_2SO_4/t. These ANC values are consistent with those from the previous investigations. The primary waste rock samples from the previous investigations have a range in ANC from 6 to 247 kg H_2SO_4/t with an average of 98 kg H_2SO_4/t and the oxide waste rock samples have a range of 1 to 89 kg H_2SO_4/t with an average of 12 kg H_2SO_4/t (EGi, 1995; 2004).

The two ore samples have total sulphur contents of 1.10 and 2.62 %S and relatively high ANC values of 93 and 143 kg H_2SO_4/t. These results are similar to those from the previous geochemical investigations with an average total S content of 2.45 %S and an average ANC of 75 kg H_2SO_4/t (EGi, 1995; 2004).

Figure C-4 is a plot of the total sulphur contents compared to the ANC values for the drill-hole samples from the modified pit for the E42 Modification and from the previous geochemical investigations. Samples that plot above the NAPP = 0 (ANC/MPA = 1) line are NAPP negative indicating an excess in acid buffering capacity over potential acidity. Samples that plot above the ANC/MPA = 2 line have at least a two-fold excess in acid buffering over acid potential and those that plot above the ANC/MPA=3 line have a three-fold excess. This plot shows that, similar to the previous investigations (EGi, 1995; 2004), the majority of the E42 Modification samples are NAPP negative and have a significant excess in acid buffering over the inherent acid potential. Only one of the samples is NAPP positive (22 kg H_2SO_4/t) and this sample represents the Lower Volcaniclastic primary waste rock. The two ore samples are NAPP negative with NAPP values of minus 63 and minus 60 kg H_2SO_4/t.

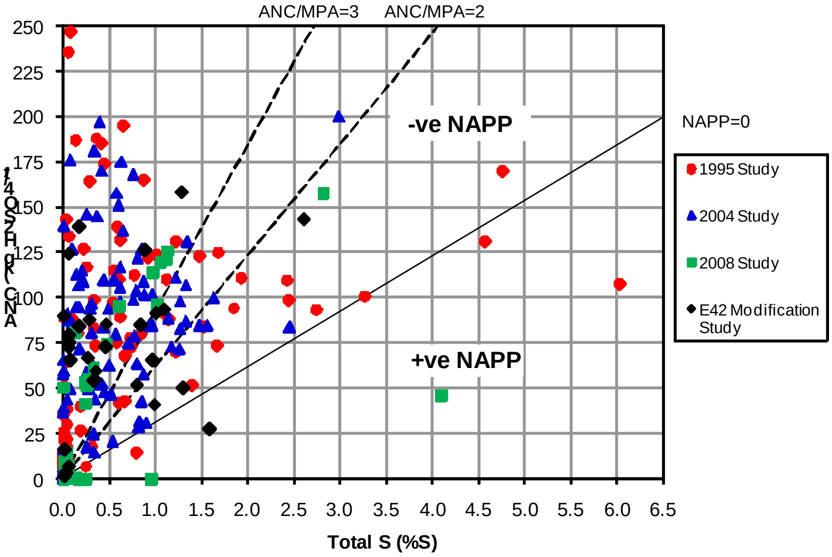

Figure C-4: Acid-base account plot for the previous and E42 Modification drill-hole samples.

Figure C-5 is a plot of the NAPP compared to NAGpH values for the drill-hole samples from the modified pit for the E42 Modification assessment and from the previous geochemical investigations. Samples that plot in the upper left quadrate are NAPP negative with NAGpH values greater than 4.5, and these samples are confirmed as NAF. Samples that plot in the lower right quadrate are NAPP positive with NAGpH values less than 4.5, and these samples are confirmed as PAF. Samples that plot in the lower left quadrate are NAPP negative with NAGpH values below 4.5 and the geochemical classification of these samples is uncertain (UC). This plot shows that the majority of the samples plot on the upper left quadrate and these samples are confirmed as NAF. A number of the samples from the geochemical investigations conducted in 1995 are plotted on the lower right quadrate and these samples are confirmed as PAF. Two samples from the 2008 confirmatory testing program (O'Kane, in prep.) and one E42 Modification sample are plotted on the upper right quadrate and the classification of these samples is uncertain. However, due to the high NAGpH of these samples it is expected that they are NAF and the NAPP positive values are attributed to the presence of non- or low reactivity sulphide forms.

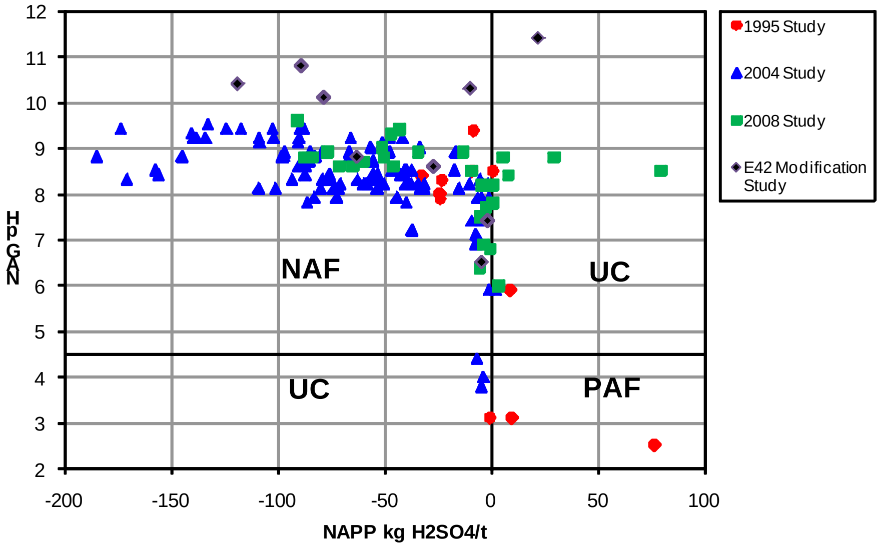

Figure C-5: ARD classification plot for the previous and E42 Modification drill-hole samples from the Cowal Gold Mine.

These results indicate that the waste rock from the modified pit is likely to be NAF, consistent with the waste rock from the approved pit area. Similar to the results from the previous investigations (EGi, 1995; 2004), the oxide waste rock from the modified pit is likely to be moderately saline and relatively inert due to low sulphur concentrations and ANC values. The primary waste rock is likely to have lower salinity, and although this material is expected to be NAF, it is likely to be reactive, containing moderate sulphur and ANC, and has the potential to produce significant salinity if left exposed to atmospheric oxidation.

C4.3 Metal Enrichment and Solubility

Multi-element analyses were performed on all of the drill-hole samples. The results of these analyses and the geochemical abundance indices for these samples are provided in Attachment CA. These results indicate that As is enriched in the majority of the samples and that Cd, Pb and Zn are enriched in some of the samples compared to the average crustal abundance of these elements. These elements have been identified as being significantly enriched in the waste rock samples from the previous geochemical investigations, however, these investigations also identified Sb as being enriched in some of the samples tested (EGi, 1995; 1997; 2004). The concentration ranges and average crustal abundance of As, Cd, Pb and Zn in the waste rock samples from the E42 Modification assessment and from the previous investigations are compared in Table C-6.

Table C-6: Concentration ranges and average crustal abundances for enriched elements in waste rock samples.

Element	Concentration Range (mg/kg)		Average Crustal Abundance (mg/kg) (Bowen, 1979)
	Previous Investigations*	E42 Modification Assessment	
As	6 to 300	< 5 to 110	1.5
Cd	< 0.1 to 15.5	< 1 to 5	0.11
Pb	5 to 504	< 5 to 885	14
Sb	< 0.1 to 0.3	0.1 to 5.5	0.2
Zn	29 to 1850	67 to 1020	75

* Source: EGi, 1995; 2004
mg/kg = milligrams per kilogram

Multi-element scans were performed on 1:2 water extracts (1 part sample/2 parts deionised water) from 8 selected drill-hole samples, including 7 waste rock and one ore sample, in order to provide an indication of relative element solubility. The results from these scans are presented in Attachment CA. These results indicate that none of the identified enriched elements are readily soluble under the neutral to slightly alkaline pH test conditions. The concentration of As in solution ranges from <0.001 to 0.007 mg/L, which is consistent with the reported As solubilities (0.005 mg/L) from the sequential batch extract test work conducted on oxide and primary waste rock from the approved CGM by EGi (EGi, 1997).

C5 TAILINGS GEOCHEMISTRY

The previous geochemical investigations by EGi in 1995, 1996 and 1997 were conducted prior to mining operations commencing and therefore simulated samples of the tailings types and representative samples of the ore types were used to predict the geochemical characteristics of the tailings that would be produced. The E42 Modification assessment has provided the opportunity to assess the geochemical characteristics of the ex-mill and deposited tailings from the approved CGM and also to predict any changes in the geochemical characteristics of the modified CGM tailings based on the geochemistry of two ore samples collected as part of the drill-hole sampling program discussed in Section C3. The test results for the ore samples are provided in Attachment CA and for the tailings samples are provided in Attachment CB.

C5.1 pH and Salinity

The pH of 1:2 extract ($pH_{1:2}$) of the tailings samples ranges from 8.1 to 8.5 indicating that the materials represented by these samples are slightly alkaline. The alkalinity concentrations in 1:2 water extracts from the tailings samples (1 part sample/2 parts water) range from 5 mg $CaCO_3$/L for the oxide tailings to 17 mg $CaCO_3$/L for the primary tailings (Attachment CB).

The $EC_{1:2}$ values for the tailings samples ranges from 1.99 to 6.41 dS/m with the deposited oxide tailings sample having the higher $EC_{1:2}$ value of 6.41 dS/m, the deposited primary (sulphide) tailings samples having $EC_{1:2}$ values of 2.02 and 4.71 dS/m and the discharge primary tailings sample having an $EC_{1:2}$ value of 1.99 dS/m. As expected, the salt precipitate sample collected from the surface of the Southern Tailings Storage Facility sample has a very high $EC_{1:2}$ of 34.0 dS/m (Attachment CB). These results indicate that the deposited oxide tailings are likely to be highly saline and that the deposited primary tailings are likely to be moderately to highly saline. Based on the $EC_{1:2}$ value of the primary tailings sampled from the approved CGM discharge point of the Northern Tailings Storage Facility, the ex-mill primary tailings are moderately saline.

Comparing these results to the results of leach column tests carried out on simulated oxide and primary tailings (EGi, 1997) indicates that the pH and EC predictions from the simulated tailings are consistent with those that occur for the approved CGM discharge (ex-mill) and deposited tailings. The initial collection of leachate (first flush) from the oxide tailings leach column had a pH of 8.8 and an EC of 5.78 dS/m and after 20 weeks of leaching under oxidising conditions the pH was 8.3 and the EC was 5.66 dS/m indicating that the simulated oxide tailings remained slightly alkaline and highly saline. The initial collection of leachate from the primary tailings leach column had a pH of 8.5 and an EC of 0.542 dS/m and after 20 weeks of leaching the pH was 8.2 and the EC was 1.23 dS/m indicating that the simulated primary tailings remained slightly alkaline and after a period of oxidation became moderately saline.

The chemical composition of the 1:2 water extracts presented in Attachment CB indicate that the salinity of the oxide and primary tailings is attributed to salts of NaCl and $CaSO_4$.

C5.2 Acid Forming Characteristics

The total sulphur content of the primary tailings samples ranges from 0.37 to 1.13 %S and the sulphide sulphur content ranges from 0.26 to 1.06 %S indicating that almost all of the sulphur in this material occurs as sulphide. Differing from this, the total sulphur content of the oxide tailings sample is 0.24 %S and the sulphide sulphur content is <0.01 %S indicating that the sulphur in this material occurs as sulphate (Attachment CB).

The ANC of the primary tailings samples is moderate ranging from 64 to 78 kg H_2SO_4/t and of the oxide tailings sample is relatively low being 16 kg H_2SO_4/t (Attachment CB). Figure C-6 is a plot of the sulphide sulphur contents against the ANC values for the oxide and primary tailings samples from the E42 Modification assessment and the previous geochemical investigations. Samples that plot above the NAPP = 0 (ANC/MPA = 1) line are NAPP negative indicating an excess in acid buffering capacity over potential acidity. Samples that plot above the ANC/MPA = 2 line have at least a two-fold excess in acid buffering over acid potential and those that plot above the ANC/MPA=3 line have a three-fold excess. This plot shows that the simulated and deposited oxide tailings from the different studies have similar sulphide contents and ANC values and these samples are all NAPP negative with ANC/MPA ratios greater than three indicating a significant excess in acid buffering over the inherent acid potential. Although the ANC values are consistent, the simulated primary tailings from the previous investigations (EGi, 1997) have significantly higher sulphur concentrations than the discharge (ex-mill) and deposited primary tailings from the current study. This is reflected by slightly NAPP positive simulated primary tailings and NAPP negative primary tailings from the current study with ANC/MPA ratios, at least, greater than 2.

Figure C-7 is a plot of the NAPP values compared to the NAGpH values for the tailings samples from the E42 Modification assessment and from the previous geochemical investigations. This plot shows that the simulated oxide tailings samples from the previous investigations and the E42 Modification oxide tailings are confirmed as NAF. Similarly, the ex-mill and deposited primary tailings are also confirmed as NAF. The simulated primary tailings from the 1997 study have a high NAGpH making the classification of this sample uncertain. This sample only has a low NAPP value (2 kg H_2SO_4/t) and due to the high recorded NAGpH of this sample, it was reported (EGi, 1997) that the positive NAPP value was most likely due to the presence of non reactive sulphur forms and that the materials represented by this sample would most likely be NAF.

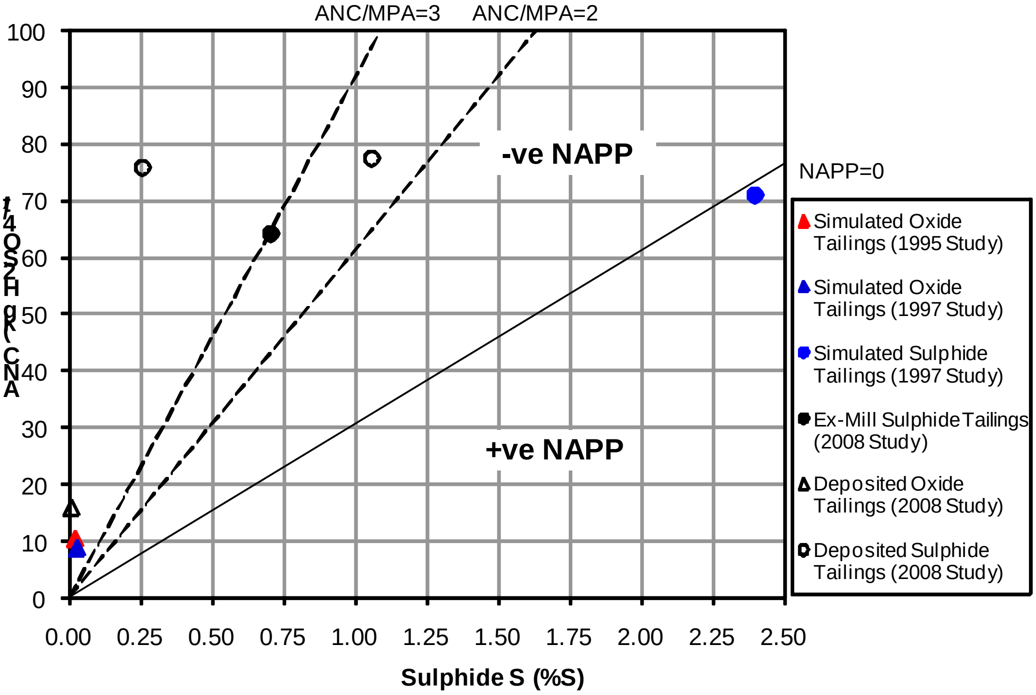

Figure C-6: Acid-base account plot for the previous and E42 Modification tailings samples.

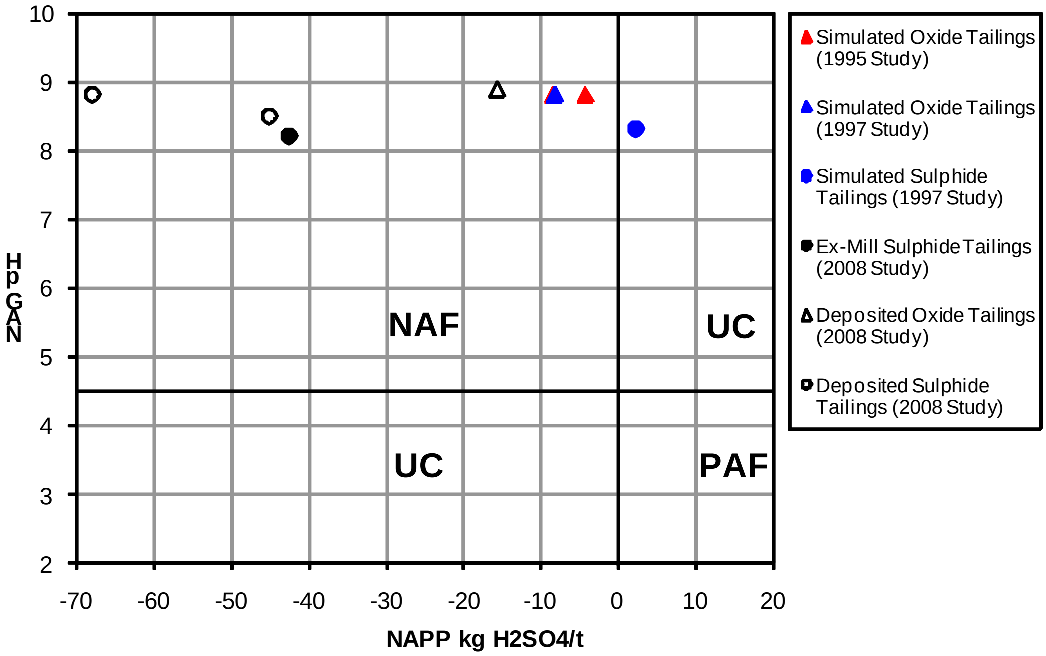

Figure C-7: ARD classification plot for the previous and E42 Modification tailings samples.

These results confirm the findings from the previous geochemical investigations in that the oxide and the primary tailings are likely to be NAF. The oxide tailings are typically low sulphur and low ANC materials and because of this these materials are relatively inert (i.e. non-reactive). However, even though the primary tailings are NAF, these materials contain moderate sulphur and moderate ANC and are likely to be relatively reactive. Because of this, these materials have the potential to produce significant salinity if left exposed to atmospheric oxidation.

In order to determine if the tailings produced from the modified CGM are likely to have similar acid-base characteristics to those from the approved CGM the ore samples from the previous investigations are compared to those for the E42 Modification. The previous ore samples range in sulphur content from 0.37 to 6.04 %S with an average of 2.45 %S and range in ANC from 52 to 188 kg H_2SO_4/t with an average of 106 kg H_2SO_4/t. The resulting NAPP values for these samples range from minus 177 to positive 77 kg H_2SO_4/t with an average of minus 31 kg H_2SO_4/t. The ore samples collected for the E42 Modification assessment have total sulphur contents of 1.10 and 2.62 %S and ANC values of 93 and 143 kg H_2SO_4/t. The resulting NAPP values for these samples are minus 60 and minus 63 kg H_2SO_4/t (Attachment CA). These results indicate that based on the samples tested, the ore samples for the E42 Modification are likely to have similar sulphur contents and ANC values and therefore it is expected that the tailings produced from the modified CGM would have similar acid-base characteristics. It is therefore expected that the tailings for the modified CGM would be NAF.

C5.3 Metal Enrichment and Solubility

The results of multi-element analyses performed on the E42 Modification tailings samples and the geochemical abundance indices for these samples are provided in Attachment CB. These results indicate that As and Cd are enriched in all of the samples and that Mo, Sb, Ni, Pb or Zn are not significantly enriched in any of the samples compared to the average crustal abundance of these elements (Bowen, 1979). The concentration ranges and average crustal abundance of these elements in the tailings samples for the E42 Modification and previous investigations are compared in Table C-7.

Multi-element scans were performed on 1:2 water extracts from the tailings samples in order to provide an indication of any element solubility concerns for these materials. The results from these scans are presented in Attachment CB. Comparing these results to the tailings liquor chemistry and leach column test results conducted by EGi for the EIS (EGi, 1997) indicates that element solubilities in the ex-mill and deposited tailings are likely to be similar to those reported for the simulated oxide and primary tailings samples, and therefore under the slightly alkaline pH test conditions of these samples the solubility of the identified enriched elements is generally low.

Table C-7: Concentration ranges and average crustal abundances for enriched elements in tailings samples.

Element	Concentration Range (mg/kg)		Average Crustal Abundance (mg/kg) (Bowen, 1979)
	Previous Investigations*	E42 Modification Assessment	
As	36 to 195	47 to 82	1.5
Cd	0.6 to 7.4	2 to 4	0.11
Mo	8.5 to 96	< 2	1.5
Ni	43 to 700	3 to 8	80
Pb	10 to 165	29 to 84	14
Sb	0.2 to 8.6	0.2 to 0.5	0.2
Zn	257 to 960	336 to 507	75

* Source: EGi, 1995; 1997; 2004

The concentration of As in solution is low, ranging from <0.001 to 0.003 mg/L, which is consistent with the reported As solubilities (0.004 to 0.01 mg/L) from the simulated tailings leach column tests. Similarly, the concentrations of Cd, ranging from <0.0001 to 0.0009 mg/L, and Sb, ranging from 0.001 to 0.002 mg/L, are low. The concentration of Mo in the extract solutions ranges from 0.029 mg/L in the oxide tailings to 0.064 mg/L in the primary tailings, indicating that this element is moderately soluble under the slightly alkaline pH conditions of these analyses. The concentrations of Mo in these extracts are consistent with those for the simulated oxide (0.046 mg/L) and primary tailings (0.084 mg/L) liquors from the previous test work (EGi, 1997).

C5.4 Tailings Liquor Chemistry

C5.4.1 Tailings Decant Liquor

The chemical composition of the decant samples collected from the Northern and Southern Tailings Storage Facilities are provided in Attachment CB (Table CB-4). These results provide an opportunity to evaluate the predicted tailings liquor chemistry. The geochemical investigations conducted on simulated oxide and primary tailings predicted that although these materials were likely to be NAF the oxide tailings were expected to be highly saline and the primary tailings were expected to be moderately saline. Additionally, it was predicted that, due to the reactive nature (i.e. high sulphur and ANC), the primary tailings were likely to become highly saline if exposed to atmospheric oxidation for extended periods. The EC values of the decant samples confirms that both the oxide tailings (EC of 44.70 dS/m) and the primary tailings (EC of 11.90 dS/m) are highly saline. The high salinity in these waters is due primarily to salts of NaCl and $CaSO_4$.

Due to the enrichment of As, Cd, Mo, and Sb in the simulated tailings samples, these elements were identified as a potential concern for water quality emanating from the tailings. Of these elements As and Cd have been confirmed as being significantly enriched in the deposited oxide and primary tailings (Section C5.3). The chemistry of the tailings decant samples indicates that the solubility of As (<0.01 mg/L), Cd (<0.005 mg/L) and Sb (0.01 mg/L) has remained low and that Mo (0.07 and 0.2 mg/L) is moderately soluble in both the oxide and primary tailings.

Given the results of multi-element analyses on 1:2 water extracts performed on the E42 Modification tailings samples (see Section C5.3), the chemistry of the tailings liquor from the modified CGM is expected to be similar to that of the approved CGM.

C5.4.2 Supernatant from INCO Simulation

The results of the INCO simulation test work (CSIRO Minerals [Hydrometallurgy], 2008) indicate that the tailings sample treated with SMBS to destroy cyanide, as expected, resulted in slightly increased sodium and sulphate concentrations in the tailings supernatant.

Based on the results of this test work the maximum concentration of sodium in the supernatant when SMBS is used is expected to be 4,413 mg/L (CSIRO Minerals [Hydrometallurgy], 2008), which is comparable to the maximum concentration of sodium in the existing tailings (3,618 mg/L) measured between November 2007 and April 2008. Similarly, the maximum concentration of sulphate in the supernatant when SMBS is used is expected to be 6,197 mg/L (CSIRO Minerals [Hydrometallurgy], 2008), which is also comparable to the maximum concentration of sulphate in the existing tailings (5,000 mg/L) measured between November 2007 and April 2008.

The comparable sodium and sulphate concentrations in the supernatant from the existing tailings and after SMBS treatment are attributed to the relatively small quantities of SMBS, hence sodium and sulphur, required to reduce the total cyanide in the tailings to the approved concentration or below, and the dilution of the treated CIL tailings with tailings from the primary ore flotation circuit.

It is also noted that a significant portion of the reported increase in the concentration of sodium in the tailings supernatant from the INCO process simulation attributed to the addition of NaOH required to maintain the optimal pH range as the reaction proceeds (CSIRO Minerals [Hydrometallurgy], 2008).

C5.5 Nature and Fate of Cyanide in Tailings

The total CN and CN_{WAD} concentrations in the tailings decant samples are provided in Attachment CB. These results are consistent with the predicted total CN concentrations in the ponded tailings liquor discussed in Section C2. From the previous work it was predicted that the CN_{WAD} concentrations in the reclaim water when processing oxide ore would range from 5 to 10 mg/L and from 10 to 15 mg/L when processing primary ore. It was also predicted that within 2 to 3 months after discharge ceases the CN_{WAD} complexes in the ponded decants (oxide and primary tailings) would decay to very low concentrations (EGi, 1997).

The concentration of CN_{WAD} in the ponded oxide tailings liquor is 0.034 mg/L and in the ponded primary tailings liquor is 0.034 mg/L (i.e. the CN_{WAD} concentrations in the ponded tailings water at the CGM are well below those predicted in the EIS). The rate of CN decay from the liquor entrained within the tailings and from the ponded decant liquor for the modified CGM is not expected to differ to that reported for the approved CGM tailings storage facilities.

C6 TAILINGS STORAGE FACILITIES ATTENUATION CHARACTERISTICS

As described in Section C5.4, the tailings liquor chemistry for the modified CGM is expected to be similar to the predicted and confirmed tailings liquor chemistry for the approved CGM. Based on the attenuation characteristics of the tailings storage facility basement materials (EGi, 1997) and the similar geochemical characteristics of the tailings and tailings liquor for the approved CGM and the modified CGM, it is anticipated that CN would be only poorly attenuated whereas Cu, Zn and As are expected to be strongly attenuated by the soils underlying the tailings storage facilities.

C7 PIT VOID WATER QUALITY

As described in Section C4.2, consistent with the previous findings, oxidation of primary rock exposed along the pit walls is likely to generate a moderate load of sulphate salts, due to the presence of reactive sulphides. However, given the regional groundwater has a TDS of around 40,000 to 45,000 mg/L, the contribution of sulphate salts from the pit walls to the overall TDS of pit water is expected to be negligible for the modified CGM.

C8 CONCLUSIONS

C8.1 Waste Rock Emplacements

The results of the geochemical characterisation of 28 drill-hole samples representing waste rock from the modified pit indicate that the geochemical characteristics, including pH and salinity, acid forming characteristics and element enrichment and solubility of the oxide and primary waste rock are expected to be similar to the characteristics of the waste rock from the approved CGM. These characteristics include:

- Both waste types would be slightly alkaline, and the oxide waste would be highly saline and primary waste would be moderately saline.
- Both waste types would be NAF with the oxide waste being relatively inert (i.e. low sulphur and low ANC) and the primary waste being relatively reactive (i.e. high sulphur and moderate ANC).
- Elements that have been identified as being enriched in the waste rock (i.e. As, Cd, Pb and Zn) are not expected to be readily soluble.

C8.2 Tailings Storage Facilities

The results of the geochemical characterisation of discharge and deposited primary and oxide tailings confirm the findings from previous investigations. These characteristics include:

- Both tailings types are slightly alkaline and highly saline. High salinity in the deposited primary tailings is due to the oxidation of contained sulphides.
- Both tailings types are NAF with the oxide tailings being relatively inert and the primary tailings being relatively reactive (i.e. high sulphur and moderate ANC).
- The tailings are significantly enriched in As and Cd, however, these elements are not readily soluble.
- The chemistry of water extracts from the tailings and the tailings decant water indicate that Mo is moderately soluble.
- The concentration of CN_{WAD} in the ponded tailings water is very low (i.e. oxide tailings water is 0.034 mg/L CN_{WAD} and primary tailings water is 0.034 mg/L CN_{WAD}).

Comparison of the geochemical characteristics of ore samples from the previous investigations to those from the pit for the modified CGM indicates that the ore samples have similar acid-base characteristics and multi-element composition. Based on the samples tested, it is expected that the tailings generated from the modified CGM would have similar geochemical characteristics to those from the approved CGM.

As is the case for the approved CGM, CN is anticipated to be only poorly attenuated by the soils underlying the tailings storage facilities whereas Cu, Zn and As are expected to be strongly attenuated.

The use of the INCO process (i.e. the introduction of SO_2 as SMBS) as an alternative method for cyanide destruction is expected to result in comparable concentrations of sodium and sulphate in the tailings supernatant. These comparable concentrations are attributed to the relatively small quantity of SMBS required to reduce total CN in the tailings to the approved concentrations or below, and the dilution of the treated tailings with tailings from the primary ore flotation circuit.

C8.3 Water Management

Based on these assessments and the conclusion that the geochemical characteristics of waste and tailings materials would not change as a result of the E42 Modification, there would be no requirement to change the water management practices and principles of the approved CGM.

C8.4 Low Grade Ore Stockpile

Although no samples of low grade ore were included in the E42 Modification assessment, based on the similar geochemical characteristics of the waste rock and ore (i.e. all drill-hole) samples, it is expected that the low-grade ore from the modified CGM would also have similar geochemical characteristics.

C8.5 Pit Void

The results of the geochemical characterisation of 28 drill-hole samples representing waste rock from the modified pit indicate that the geochemical characteristics of the waste rock that would be exposed in the proposed modified pit are expected to be similar to the characteristics of the waste rock exposed in the approved CGM pit.

C9 REFERENCES

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ATTACHMENT CA

Drill-Hole Sample Test Results

Table CA-1: Acid forming characteristics and ARD classification of drill-hole samples.

Drill-Hole	Depth (m)		Interval (m)	Sample Code	Lithology	Weathering	Material Type	Au Grade (ppm)	pH$_{sat}$	EC$_{sat}$	ACID-BASE ANALYSIS						NAG TEST				ARD Class	
	from	to									Total %S	SO4 %S	Sulph %S	MPA$_{(1)}$	ANC	ANC/ MPA$_{(1)}$	NAPP$_{(1)}$	NAGpH	NAG$_{4.5}$	NAG$_{7.0}$		
E42D1015	250	251	1.0	ENV-101	DI	Primary	Waste	0.04	8.9	0.243	0.09	0.02	0.08	2.8	66	24	-63	-	-	-	NAF	🟦
E42D1017	455	456	1.0	ENV-102	DI	Primary	Waste	0.05	9.1	0.322	0.19	0.01	0.18	5.8	84	14	-78	10.1	0	0	NAF	🟦
E42D1007	260	261	1.0	ENV-103	DI	Primary	Waste	0.24	9.1	0.203	0.48	0.01	0.48	14.7	85	5.8	-71	-	-	-	NAF	🟦
E42D1067	212	213	1.0	ENV-104	DI	Primary	Waste	0.03	9.0	0.191	0.85	0.01	0.84	26.0	85	3.3	-59	-	-	-	NAF	🟦
E42D1328	375	376	1.0	ENV-105	DI	Primary	Waste	0.3	9.4	0.292	0.34	0.01	0.34	10.4	54	5.2	-44	-	-	-	NAF	🟦
E42D1136	220	221	1.0	ENV-106	LAVA	Primary	Waste	0.08	9.3	0.284	0.47	0.04	0.46	14.4	73	5.1	-59	-	-	-	NAF	🟦
E42D1074	195	196	1.0	ENV-107	LAVA	Primary	Waste	0.35	9.2	0.161	0.81	0.01	0.81	24.8	52	2.1	-27	8.6	0	0	NAF	🟦
E42D1027	226	227	1.0	ENV-108	LAVA	Primary	Waste	0.28	9.1	0.131	0.37	0.01	0.37	11.3	59	5.2	-48	-	-	-	NAF	🟦
E42D1062	135	136	1.0	ENV-109	UVC(CAN)	Primary	Waste	0.05	9.1	0.389	0.07	0.02	0.06	2.1	79	37	-77	-	-	-	NAF	🟦
E42D1062	150	151	1.0	ENV-110	UVC(CAN)	Primary	Waste	0.25	8.7	0.313	1.29	0.02	1.28	39.5	158	4.0	-119	10.4	0	0	NAF	🟦
E42D1062	170	171	1.0	ENV-111	UVC(CAN)	Primary	Waste	0.02	9.3	0.258	0.07	0.02	0.06	2.1	75	35	-73	-	-	-	NAF	🟦
E42D1027	174	175	1.0	ENV-112	UVC	Primary	Waste	0.07	9.2	0.224	0.30	0.01	0.30	9.2	88	10	-78	-	-	-	NAF	🟦
E42D1029	213	214	1.0	ENV-113	UVC	Primary	Waste	0.36	9.0	0.468	0.98	0.03	0.97	30.0	65	2.2	-35	-	-	-	NAF	🟦
E42D1029	138	139	1.0	ENV-114	UVC	Primary	Waste	0.22	9.2	0.210	0.90	0.05	0.88	27.5	126	4.6	-98	-	-	-	NAF	🟦
E42D1029	146	147	1.0	ENV-115	UVC	Primary	Waste	0.38	9.0	0.537	1.00	0.03	0.99	30.6	41	1.3	-10	-	-	-	NAF	🟦
E42D1136	170	171	1.0	ENV-116	UVC	Primary	Waste	0.13	9.3	0.293	0.37	0.04	0.36	11.3	54	4.8	-43	-	-	-	NAF	🟦
E42D1312	217	218	1.0	ENV-117	UVC	Primary	Waste	0.04	9.2	0.282	0.28	0.12	0.24	8.6	67	7.8	-58	-	-	-	NAF	🟦
E42D1312	139	140	1.0	ENV-118	UVC	Primary	Waste	<0.01	9.1	0.284	0.18	0.03	0.17	5.5	139	25	-133	-	-	-	NAF	🟦
E42D1312	50	51	1.0	ENV-119	SAP	Oxidised	Waste	0.07	7.6	3.740	0.04	0.08	0.01	1.2	3	2.5	-2	7.4	0	0	NAF	🟦
E42D1312	100	101	1.0	ENV-120	SROCK	Oxidised	Waste	0.17	8.5	2.360	0.03	0.06	0.01	0.9	16	18	-15	-	-	-	NAF	🟦
E42D1346	83	84	1.0	ENV-121	SROCK	Oxidised	Waste	0.32	9.0	0.266	0.07	0.03	0.06	2.1	7	3.1	-4	6.5	0	1	NAF	🟦
E42D1328	320	321	1.0	ENV-122	DY	Primary	Waste	0.01	9.3	0.145	0.02	0.04	0.01	0.6	90	146	-89	10.8	0	0	NAF	🟦
E42D1062	36	37	1.0	ENV-123	SAP	Oxidised	Waste	<0.01	7.1	4.530	0.03	0.13	0.01	0.9	1	1.1	0	-	-	-	NAF	🟦
E42D1062	192	193	1.0	ENV-124	UVC(CAN)	Primary	Waste	<0.01	9.3	0.273	0.07	0.02	0.06	2.1	124	58	-122	-	-	-	NAF	🟦
E42D1054	171	172	1.0	ENV-125	UVC	Primary	Waste	0.15	9.1	0.192	1.02	0.03	1.01	31.2	91	2.9	-60	-	-	-	NAF	🟦
E42D1054	242	243	1.0	ENV-126	UVC	Primary	Ore	0.59	9.2	0.228	1.10	0.02	1.09	33.7	93	2.8	-60	-	-	-	NAF	🟦
E42D1062	232	233	1.0	ENV-127	UVC	Primary	Ore	1.03	8.8	0.267	2.62	0.14	2.57	80.2	143	1.8	-63	8.8	0	0	NAF	🟦
E42D1074	262	263	1.0	ENV-128	LVC	Primary	Waste	0.14	9.0	0.120	1.31	0.04	1.30	40.1	50	1.2	-10	10.3	0	0	NAF	🟦
E42D1346	120	121	1.0	ENV-129	LVC	Primary	Waste	0.08	9.0	0.617	1.60	0.08	1.58	49.0	27	0.6	22	11.4	0	0	UC (NAF)	▨
E42D1346	140	141	1.0	ENV-130	LVC	Primary	Waste	0.03	9.3	0.489	0.07	0.03	0.06	2.1	72	34	-70	-	-	-	NAF	🟦

KEY

pH$_{sat}$ = pH of saturation extract
EC$_{sat}$ = Electrical Conductivity of saturation extract (dS/m)
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)

Lithology Code

SAP	Saprolite
SROCK	Saprock
UVC	Upper Volcaniclastics
LVC	Lower Volcaniclastics
UVC (CAN)	Andesitic volcanics (Intrusive)
LAVA	Lava
DI	Diorite
DY	Dyke material

ARD Classification Code

🟦 NAF = Non-Acid Forming
🟥 PAF = Potentially Acid Forming
▨ UC = Uncertain Classification
(expected classification shown in brakets)

NOTES

(1) Calculated using concentration of total S

Table CA-2: Multi-element composition of drill-hole samples.

Element	**Unit**	**Detection Limit**	**Sample Description/Code**														
			Mine Rock														
			Oxide				Primary										
			SAP		SROCK		LAVA			DY	DI					UVC	
			ENV-119	ENV-123	ENV-120	ENV-121	ENV-106	ENV-107	ENV-108	ENV-122	ENV-101	ENV-102	ENV-103	ENV-104	ENV-105	ENV-112	ENV-113
Al	%	0.005%	0.782%	1.650%	2.520%	1.920%	0.439%	2.570%	2.330%	1.590%	2.740%	3.180%	2.680%	1.610%	1.050%	3.370%	2.660%
As	mg/kg	5	50	15	60	22	15	108	35	<	44	<	6	20	36	11	60
Ba	mg/kg	10	50	930	140	30	210	80	70	250	90	90	40	50	60	120	150
Be	mg/kg	1	<	1	<	<	<	<	<	<	<	<	<	<	<	<	<
Ca	%	0.001%	0.205%	0.087%	0.546%	0.358%	2.330%	2.130%	2.360%	3.850%	2.670%	2.650%	3.440%	4.950%	2.130%	3.580%	2.280%
Cd	mg/kg	1	<	<	<	<	<	<	1	<	<	<	<	<	<	<	<
Co	mg/kg	2	8	92	34	20	8	13	13	12	17	20	21	33	12	11	8
Cr	mg/kg	2	15	<	56	6	13	<	15	6	5	21	10	7	<	26	<
Cu	mg/kg	5	171	271	170	363	120	139	160	40	227	163	356	92	32	35	6
Fe	%	0.005%	6.890%	6.260%	7.240%	5.300%	4.450%	5.970%	5.180%	3.510%	3.880%	5.070%	4.940%	6.090%	5.270%	4.450%	4.750%
Hg	mg/kg	0.1	<	<	<	0.5	<	<	<	<	<	<	<	<	<	<	<
K	%	0.001%	0.200%	0.203%	0.185%	0.405%	0.256%	0.423%	0.508%	0.230%	0.446%	0.235%	0.288%	0.629%	0.401%	1.130%	1.080%
Mg	%	0.001%	0.117%	0.529%	1.480%	1.140%	0.442%	1.260%	1.020%	1.060%	1.520%	2.330%	1.860%	2.050%	0.741%	1.200%	1.020%
Mn	mg/kg	5	375	9430	2060	596	1540	2180	1980	1160	899	1390	1570	2120	1850	1900	2260
Mo	mg/kg	2	<	<	<	<	<	<	<	<	<	<	<	<	<	<	<
Na	%	0.001%	0.168%	0.483%	0.378%	0.242%	0.083%	0.216%	0.157%	0.110%	0.476%	0.242%	0.208%	0.067%	0.127%	0.123%	0.372%
Ni	mg/kg	2	5	6	25	9	<	<	<	3	7	14	10	12	2	7	<
P	mg/kg	50	520	270	930	1130	930	1440	1420	1120	970	1010	1050	820	1350	1020	960
Pb	mg/kg	5	30	23	885	9	<	26	14	5	<	<	<	6	14	<	15
Sb	mg/kg	0.1	0.2	<	<	<	<	<	<	<	0.3	<	<	<	<	<	<
Se	mg/kg	1	<	<	<	<	<	<	<	<	<	<	<	<	<	<	<
Sn	mg/kg	5	<	<	<	<	<	<	<	<	<	<	<	<	<	<	<
Sr	mg/kg	2	29	36	45	42	163	153	183	205	157	142	117	124	109	148	128
Ti	mg/kg	10	50	140	70	520	120	480	570	140	890	680	470	190	260	590	400
Tl	mg/kg	5	<	<	<	<	<	<	<	<	<	<	<	<	<	<	<
V	mg/kg	5	57	129	188	137	<	64	52	54	129	162	143	76	19	60	25
Zn	mg/kg	5	202	570	398	119	163	253	375	77	71	134	120	109	139	118	130

< element at or below analytical detection limit.

Table CA-2: Multi-element composition of drill-hole samples. (continued)

Element	**Unit**	**Detection Limit**	**Sample Description/Code**														
			Mine Rock													Ore	
			Primary													Primary	
			UVC						UVC(CAN)				LVC			UVC	
			ENV-114	ENV-115	ENV-116	ENV-117	ENV-118	ENV-125	ENV-109	ENV-110	ENV-111	ENV-124	ENV-128	ENV-129	ENV-130	ENV-126	ENV-127
Al	%	0.005%	2.350%	2.800%	1.940%	1.460%	2.810%	2.890%	2.530%	2.350%	3.570%	1.820%	2.860%	2.060%	4.060%	1.360%	2.800%
As	mg/kg	5	108	192	31	48	12	56	8	67	10	<	110	37	5	135	403
Ba	mg/kg	10	100	200	150	740	220	60	90	70	140	60	80	40	220	70	320
Be	mg/kg	1	<	1	<	<	<	<	<	<	<	<	<	<	<	<	<
Ca	%	0.001%	4.350%	1.080%	1.880%	2.220%	5.270%	3.360%	3.300%	6.330%	2.920%	5.110%	2.340%	1.150%	3.010%	3.060%	4.690%
Cd	mg/kg	1	5	2	1	<	<	<	<	<	<	<	1	<	<	4	12
Co	mg/kg	2	8	14	6	6	6	20	17	18	18	18	18	18	25	8	10
Cr	mg/kg	2	14	12	23	<	8	32	<	8	<	<	8	7	7	19	<
Cu	mg/kg	5	51	213	10	13	8	161	101	250	86	84	82	138	648	46	228
Fe	%	0.005%	4.070%	5.030%	4.360%	4.000%	3.490%	6.040%	4.740%	2.590%	5.170%	4.330%	6.130%	5.230%	6.010%	3.750%	5.500%
Hg	mg/kg	0.1	0.1	<	<	<	<	<	<	<	<	<	<	0.1	<	<	0.2
K	%	0.001%	0.756%	1.280%	0.536%	0.803%	0.770%	0.147%	0.256%	1.020%	0.483%	0.265%	0.221%	0.212%	0.667%	0.619%	1.060%
Mg	%	0.001%	0.671%	0.656%	0.925%	0.910%	0.961%	2.200%	1.700%	0.391%	1.890%	1.030%	2.230%	1.080%	1.730%	0.911%	1.510%
Mn	mg/kg	5	2280	1950	2360	2110	1780	2330	1250	1500	1550	1520	1980	692	1090	2240	2380
Mo	mg/kg	2	<	<	<	4	<	<	<	2	<	<	2	<	<	<	16
Na	%	0.001%	0.135%	0.060%	0.307%	0.031%	0.042%	0.370%	0.120%	0.116%	0.623%	0.039%	0.261%	0.654%	0.096%	0.199%	0.069%
Ni	mg/kg	2	<	7	3	<	<	12	3	5	4	3	4	8	8	<	3
P	mg/kg	50	610	1230	870	750	810	790	700	1550	690	660	1550	1120	1410	700	880
Pb	mg/kg	5	664	155	66	95	5	<	58	72	13	<	56	6	<	125	420
Sb	mg/kg	0.1	<	0.1	<	0.1	<	<	<	<	<	<	<	<	0.1	0.2	0.5
Se	mg/kg	1	<	<	<	<	<	<	<	<	<	<	<	<	<	<	<
Sn	mg/kg	5	<	<	<	<	<	<	<	<	<	<	<	<	<	<	<
Sr	mg/kg	2	140	77	154	163	580	214	90	184	170	221	93	91	57	178	184
Ti	mg/kg	10	320	430	580	220	360	710	190	430	860	70	820	1370	450	410	630
Tl	mg/kg	5	<	<	<	<	<	<	<	<	<	<	<	<	<	<	<
V	mg/kg	5	15	43	20	12	14	179	70	73	113	41	117	132	117	12	31
Zn	mg/kg	5	1020	482	325	246	112	204	126	67	114	70	402	82	136	612	2180

< element at or below analytical detection limit.

Table CA-3: Geochemical abundance indices (GAI) for drill-hole samples.

Element	**Unit**	***Mean Crustal Abundance**	**Sample Description/Code**														
			Mine Rock														
			Oxide				Primary										
			SAP		SROCK		LAVA			DY	DI					UVC	
			ENV-119	ENV-123	ENV-120	ENV-121	ENV-106	ENV-107	ENV-108	ENV-122	ENV-101	ENV-102	ENV-103	ENV-104	ENV-105	ENV-112	ENV-113
Al	%	8.2%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
As	mg/kg	1.5	**4**	**3**	**5**	**3**	**3**	**6**	**4**	1	**4**	1	1	**3**	**4**	2	**5**
Ba	mg/kg	500	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Be	mg/kg	2.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ca	%	4.0%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cd	mg/kg	0.11	<3	<3	<3	<3	<3	<3	**3**	<3	<3	<3	<3	<3	<3	<3	<3
Co	mg/kg	20	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Cr	mg/kg	100	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu	mg/kg	50	1	2	1	2	1	1	1	-	2	1	2	-	-	-	-
Fe	%	4.1%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hg	mg/kg	0.05	-	-	-	**3**	-	-	-	-	-	-	-	-	-	-	-
K	%	2.1%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mg	%	2.3%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mn	mg/kg	950	-	**3**	1	-	-	1	-	-	-	-	-	1	-	-	1
Mo	mg/kg	1.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Na	%	2.3%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ni	mg/kg	80	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
P	mg/kg	1000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb	mg/kg	14	1	-	**5**	-	-	-	-	-	-	-	-	-	-	-	-
Sb	mg/kg	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Se	mg/kg	0.05	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3
Sn	mg/kg	2.2	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1
Sr	mg/kg	370	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ti	mg/kg	5600	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tl	mg/kg	0.6	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2
V	mg/kg	160	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zn	mg/kg	75	1	2	2	-	1	1	2	-	-	-	-	-	-	-	-

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

Table CA-3: Geochemical abundance indices (GAI) for drill-hole samples. (continued)

Element	**Unit**	***Mean Crustal Abundance**	**Sample Description/Code**														
			Mine Rock													Ore	
			Primary													Primary	
			UVC						UVC(CAN)				LVC			UVC	
			ENV-114	ENV-115	ENV-116	ENV-117	ENV-118	ENV-125	ENV-109	ENV-110	ENV-111	ENV-124	ENV-128	ENV-129	ENV-130	ENV-126	ENV-127
Al	%	8.2%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
As	mg/kg	1.5	**6**	**6**	**4**	**4**	2	**5**	2	**5**	2	1	**6**	**4**	1	**6**	**6**
Ba	mg/kg	500	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Be	mg/kg	2.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ca	%	4.0%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cd	mg/kg	0.11	**5**	**4**	**3**	<3	<3	<3	<3	<3	<3	<3	**3**	<3	<3	**5**	**6**
Co	mg/kg	20	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cr	mg/kg	100	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu	mg/kg	50	-	2	-	-	-	1	-	2	-	-	-	1	**3**	-	2
Fe	%	4.1%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hg	mg/kg	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1
K	%	2.1%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mg	%	2.3%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mn	mg/kg	950	1	-	1	1	-	1	-	-	-	-	-	-	-	1	1
Mo	mg/kg	1.5	-	-	-	1	-	-	-	-	-	-	-	-	-	-	**3**
Na	%	2.3%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ni	mg/kg	80	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
P	mg/kg	1000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb	mg/kg	14	**5**	**3**	2	2	-	-	1	2	-	-	-	-	-	**3**	**4**
Sb	mg/kg	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1
Se	mg/kg	0.05	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3	<3
Sn	mg/kg	2.2	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1
Sr	mg/kg	370	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ti	mg/kg	5600	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tl	mg/kg	0.6	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2
V	mg/kg	160	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zn	mg/kg	75	**3**	2	2	1	-	1	-	-	-	-	2	-	-	2	**4**

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

Table CA-4: Chemical composition of water extracts from selected drill-hole samples.

Parameter		**Detection Limit**	**Sample Description Code**							
			Mine Rock							Ore
			Oxide		Primary					Primary
			SAP	SROCK	LAVA	DY	DI	UVC (CAN)	LVC	UVC
			ENV-119	ENV-121	ENV-107	ENV-122	ENV-102	ENV-110	ENV-128	ENV-127
pH		0.1	7.8	8.9	8.9	9.3	9.3	8.7	9.0	8.7
EC	dS/m	0.001	2.610	0.502	0.320	0.307	0.337	0.485	0.274	0.519
Alkalinity ($CaCO_3$)	mg/L	0.5	41.2	2.1	86.0	30.7	75.5	186.0	19.4	152.0
Ag	mg/L	0.001	<	<	<	<	<	<	<	<
Al	mg/L	0.01	<	0.04	0.26	0.05	0.46	0.1	0.28	0.11
As	mg/L	0.001	<	0.03	0.002	0.007	0.006	0.01	0.003	0.002
Ba	mg/L	0.001	0.028	38200	0.059	0.008	0.02	0.025	0.026	0.039
Be	mg/L	0.001	<	0.002	<	<	<	<	<	<
Ca	mg/L	5	60	<	10	5	10	10	10	20
Cd	mg/L	0.0001	0.0002	0.0013	0.0004	<	<	<	<	0.0028
Cl	mg/L	5	1260	160	10	10	30	15	20	40
Co	mg/L	0.001	0.008	0.001	<	<	<	<	<	<
Cr	mg/L	0.001	0.001	<	0.001	<	0.001	<	<	<
Cu	mg/L	0.001	0.007	0.015	0.001	0.001	<	0.002	<	0.003
Fe	mg/L	0.5	<	0.5	<	<	<	<	<	<
Hg	mg/L	0.0001	<	<	<	<	<	<	<	<
K	mg/L	5	60	<	25	30	20	55	10	80
Mg	mg/L	5	55	<	<	<	<	<	<	40
Mn	mg/L	0.001	0.018	0.086	0.023	0.002	0.001	0.013	0.006	0.048
Mo	mg/L	0.001	0.003	0.023	0.025	0.005	0.003	0.288	0.034	0.219
Na	mg/L	5	570	160	25	50	40	40	40	30
Ni	mg/L	0.001	0.007	0.005	<	<	<	<	<	<
P	mg/L	5	<	<	<	<	<	<	<	<
Pb	mg/L	0.001	<	<	<	<	<	<	<	<
Sb	mg/L	0.001	<	<	0.001	0.002	0.004	0.008	0.002	0.002
Se	mg/L	0.01	<	0.02	<	<	<	<	<	<
Sn	mg/L	0.01	<	<	<	<	<	<	<	<
SO4	mg/L	5	250	40	40	30	15	80	30	115
Sr	mg/L	0.5	1	<	0.5	1	1	1	1	2
Ti	mg/L	0.5	<	<	<	<	<	<	<	<
Tl	mg/L	0.001	<	<	<	<	<	<	<	<
V	mg/L	0.01	<	<	<	0.02	0.02	<	<	<
Zn	mg/L	0.005	0.129	6.83	<	<	<	<	<	0.015

< element at or below analytical detection limit.

ATTACHMENT CB

Tailings and Tailings Decant Sample Test Results

Table CB-1: Acid forming characteristics and ARD classification of tailings samples.

Table CB-2: Multi-element composition and geochemical abundance indices (GAI) of tailings samples.

Table CB-3: Chemical composition of water extracts from tailings samples.

Table CB-4: Chemical composition of tailings decant samples.

Table CB-1: Acid forming characteristics and ARD classification of tailings samples.

Sample Location	**Sample Code**	**Sample Description**	**pH$_{1:2}$**	**EC$_{1:2}$**	**ACID-BASE ANALYSIS**							**NAG TEST**			**ARD Class.**
					Total %S	**SO4 %S**	**Sulph %S**	**MPA$_{(1)}$**	**ANC**	**NAPP$_{(1)}$**	**ANC/ MPA$_{(1)}$**	**NAGpH**	**NAG$_{4.5}$**	**NAG$_{7.0}$**	
Discharge Point for NTSF	NTSF-1	Primary Tailings	8.2	1.99	0.71	0.12	0.67	21	64	-85	3.1	8.2	0	0	NAF 🟦
North Beach of NTSF	NTSF-2	Primary Tailings	8.2	4.71	0.37	0.32	0.26	8	76	-84	9.5	8.8	0	0	NAF 🟦
West Beach of NTSF	NTSF-3	Primary Tailings	8.2	2.02	1.13	0.20	1.06	32	78	-110	2.4	8.5	0	0	NAF 🟦
Western Cell of STSD	STSF-1	Oxide Tailings	8.5	6.41	0.24	0.77	0.01	0.3	16	-16	52	8.9	0	0	NAF 🟦
Western Cell of STSF	STSF-2	Salt Precipitate	8.1	34.00	0.94	0.39	0.81	25	78	-102	3.1	8.5	0	0	NAF 🟦

<u>KEY</u>

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

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

MPA = Maximum Potential Acidity (kgH$_2$SO$_4$/t)

ANC = Acid Neutralising Capacity (kgH$_2$SO$_4$/t)

NAPP = Net Acid Producing Potential (kgH$_2$SO$_4$/t)

<u>ARD Classification Code</u>

🟦 NAF = Non-Acid Forming

🟥 PAF = Potentially Acid Forming

<u>NOTES</u>

NTSF - Northern Tailings Storage Facility

STSF - Southern Tailings Storage Facility

(1) Calculated using concentration of sulphide S

Table CB-2: Multi-element composition and geochemical abundance indices (GAI) of tailings samples.

Element	**Unit**	**Detection Limit**	**Sample Description/Code**					***Mean Crustal Abundance**	**Sample Description/Code**				
			Primary Tailings			Oxide Tailings	Salt Precipitate		Primary Tailings			Oxide Tailings	Salt Precipitate
			NTSF-1	NTSF-2	NTSF-3	STSF-1	STSF-2		NTSF-1	NTSF-2	NTSF-3	STSF-1	STSF-2
Al	%	0.005%	0.704%	1.170%	1.130%	0.705%	0.437%	8.2%	-	-	-	-	-
As	mg/kg	5	47	82	77	63	69	1.5	**4**	**5**	**5**	**5**	**5**
Ba	mg/kg	10	110	130	110	200	220	500	-	-	-	-	-
Be	mg/kg	1	<	<	<	<	<	2.6	-	-	-	-	-
Ca	%	0.005%	2.400%	3.120%	3.010%	0.824%	0.270%	4.0%	-	-	-	-	-
Cd	mg/kg	1	2	4	3	2	2	0.11	**4**	**5**	**4**	**4**	**4**
Co	mg/kg	2	9	14	15	19	18	20	-	-	-	-	-
Cr	mg/kg	2	10	12	16	9	6	100	-	-	-	-	-
Cu	mg/kg	5	77	130	144	175	134	50	-	1	1	1	1
Fe	%	0.005%	3.540%	4.320%	4.690%	4.010%	3.560%	4.1%	-	-	-	-	-
Hg	mg/kg	0.1	<	<	<	<	<	5.0%	-	-	-	-	-
K	%	0.005%	0.136%	0.174%	0.142%	0.154%	0.176%	2.1%	-	-	-	-	-
Mg	%	0.005%	0.871%	1.120%	1.200%	0.412%	0.258%	2.3%	-	-	-	-	-
Mn	mg/kg	5	1620	1810	1760	1570	1810	950	-	-	-	-	-
Mo	mg/kg	2	<	<	<	<	<	1.5	-	-	-	-	-
Na	%	0.005%	0.078%	0.251%	0.071%	0.624%	1.780%	2.3%	-	-	-	-	-
Ni	mg/kg	2	3	5	6	8	4	80	-	-	-	-	-
P	mg/kg	50	1120	1100	1350	780	670	1000	-	-	-	-	-
Pb	mg/kg	5	39	84	72	29	64	14	1	2	2	-	2
Sb	mg/kg	0.1	0.2	0.5	0.2	0.4	0.2	0.2	-	1	-	-	-
Se	mg/kg	1	<	<	<	<	<	0.05	<3	<3	<3	<3	<3
Sn	mg/kg	5	<	<	<	<	<	2.2	<1	<1	<1	<1	<1
Sr	mg/kg	2	126	183	180	78	64	370	-	-	-	-	-
Ti	mg/kg	10	10	20	20	20	10	5600	-	-	-	-	-
Tl	mg/kg	5	<	<	<	<	<	0.6	<2	<2	<2	<2	<2
V	mg/kg	5	19	38	42	53	20	160	-	-	-	-	-
Zn	mg/kg	5	336	507	498	333	407	75	2	2	2	2	2

< element at or below analytical detection limit.

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

Table CB-3: Chemical composition of water extracts from tailings samples.

Parameter		Detection Limit	Sample Description/Code				
			Primary Tailings			Oxide Tailings	Salt Precipitate
			NTSF-1	NTSF-2	NTSF-3	STSF-1	STSF-2
pH	pH Unit	0.1	8.2	8.2	8.2	8.5	8.1
EC	dS/m	0.01	1.99	4.71	2.02	6.41	34.00
Alkalinity (CaCO$_3$)	mg/L	1	15	17	12	5	1
CN	mg/L	2	32	18	18	6	14
CN$_{WAD}$	mg/L	2	2	4	2	4	<
Al	mg/L	0.01	0.04	0.04	0.16	0.08	0.04
As	mg/L	0.001	0.002	0.003	0.002	0.002	<
Ba	mg/L	0.001	0.012	0.001	0.006	0.001	0.026
Be	mg/L	0.001	<	<	<	<	0.006
Ca	mg/L	10	660	1380	720	3100	1100
Cd	mg/L	0.0001	<	0.0009	0.0003	0.0005	0.0173
Cl	mg/L	10	1240	4340	1220	19040	49800
Co	mg/L	0.001	0.005	0.016	0.006	0.064	0.562
Cr	mg/L	0.001	<	<	<	0.001	0.004
Cu	mg/L	0.001	0.003	0.005	0.004	0.003	0.012
Fe	mg/L	2	<	<	<	<	<
Hg	mg/L						
K	mg/L	10	240	460	260	580	1180
Mg	mg/L	10	120	280	120	440	1620
Mn	mg/L	0.001	0.273	0.532	0.275	0.038	2.04
Mo	mg/L	0.001	0.06	0.064	0.034	0.029	0.222
Na	mg/L	10	920	3080	1120	11620	28800
Ni	mg/L	0.001	<	0.001	<	<	0.003
P	mg/L	20	<	<	<	<	<
Pb	mg/L	0.001	<	<	<	<	<
Sb	mg/L	0.001	0.002	0.002	0.002	0.001	0.001
Se	mg/L	0.01	<	<	<	<	<
Sn	mg/L	0.01	<	<	<	<	<
SO4	mg/L	10	2640	5880	3240	10840	6880
Sr	mg/L	1	14	36	32	26	16
Ti	mg/L	2	<	<	<	<	<
Tl	mg/L	0.001	<	<	<	<	<
V	mg/L	0.01	<	<	<	<	<
Zn	mg/L	0.005	<	0.007	0.008	0.006	0.007

< element at or below analytical detection limit.

Table CB-4: Chemical composition of tailings decant samples.

Parameter		**Detection Limit**	**Sample Description/Code**	
			Primary Tailings Decant	Oxide Tailings Decant
			NTSF-4	STSF-3
pH	pH Unit	0.1	8.0	7.2
EC	dS/m	0.01	11.90	44.70
Alkalinity ($CaCO_3$)	mg/L	1	155	34
CN	mg/L	0.004	0.0561	0.504
CN_{WAD}	mg/L	0.004	0.041	0.0336
Al	mg/L	0.1	<	<
As	mg/L	0.01	<	<
Ba	mg/L	0.1	<	<
Be	mg/L	0.01	<	<
Ca	mg/L	1	469	1110
Cd	mg/L	0.005	<	<
Cl	mg/L	1	2130	13900
Co	mg/L	0.01	0.07	0.4
Cr	mg/L	0.01	<	<
Cu	mg/L	0.01	0.96	<
Fe	mg/L	0.05	<	0.05
K	mg/L	1	313	336
Mg	mg/L	1	210	516
Mn	mg/L	0.01	0.49	0.12
Mo	mg/L	0.01	0.2	0.07
Na	mg/L	1	1860	9000
Ni	mg/L	0.01	<	<
P	mg/L	1	<	<
Pb	mg/L	0.01	<	<
Sb	mg/L	0.01	0.01	<
Se	mg/L	0.01	0.01	<
Sn	mg/L	0.01	<	<
SO4	mg/L	1	3700	5780
Sr	mg/L	0.1	11.2	22
Ti	mg/L	0.01	<	<
Tl	mg/L	0.01	<	<
V	mg/L	0.01	<	<
Zn	mg/L	0.01	<	<

< element at or below analytical detection limit.