MAIN PIT BACKFILLING CONCEPT APPROACHES, RUM JUNGLE

Submitted to:

Northern Territory Government
Department of Mines & Energy

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EXECUTIVE SUMMARY

This report describes conceptual backfilling approaches for the Main Pit at the former Rum Jungle Mine Site (Rum Jungle). It was prepared in support of the Rum Jungle Rehabilitation Planning Project by Andy Thomas with input from Christoph Wels, Jack Caldwell, and Andy Robertson of Robertson GeoConsultants (RGC).

Study Objectives

The preferred rehabilitation strategy for Rum Jungle includes backfilling Main Pit with Potentially Acid Forming (PAF) material from Intermediate WRD, Main WRD, and material sourced from Dysons (backfilled) Pit. PAF materials would be placed below the minimum expected groundwater level to remain saturated. This would prevent future oxidation and limit the generation of Acid and Metalliferous Drainage (AMD). The unsaturated zone of the pit would be filled with Non Acid Forming (NAF) waste material or clean fill. NAF material would be used to construct the final raised landform. The East Branch of the Finniss River would be routed around the landform.

Main Pit is currently flooded with relatively clean water that is partially flushed each year by the East Branch of the Finniss River. The pit also contains tailings that were discharged into the pit sub-aqueously during mining operations, a limited quantity of waste rock/soil placed during earlier rehabilitation works and residual amounts of untreated pit water.

No specific information is available on the in-pit backfill material characteristics and therefore data are used from Dysons Pit and an analogous site. There is evidence of pit wall instability at the time of mining in historical photographs and also interpreted from recent bathymetric surveys. This is one of a number of geo-hazards which give rise to a suite of potential construction risks associated with backfilling.

The DME requested that RGC identify approaches with consideration of the following project objectives/priorities, as established in consultation with the DME and their other consultants:

- protect the health and safety of personnel
- technically feasible with reasonable probability of success
- meet construction timeframes
- optimize dewatering prior to backfilling
- optimize the quantity of PAF material backfilled into the pit
- deliver the project in a cost-effective manner
- efficient and allows for the addition of lime during backfilling
- create a stable, long-term landform
**Preliminary Approaches**

The following preliminary approaches were developed:

- **Approach 1: Dewatering and Dry Placement.** This involves dewatering the pit entirely and then placing material by conventional earthworks.

- **Approach 2: Crest Dumping.** This involves dewatering the pit entirely and then dumping material from the pit crest, moving equipment out and over the advancing front.

- **Approach 3: Overwater Dumping with Dewatering and Placement.** This involves dumping material from the pit lake using barges or floating conveyors without first dewatering the pit.

- **Approach 4: Pit Edge Stacker Dumping.** This involves dewatering the pit entirely and using long-boom conveyor stacker/s to dump material from the edge.

- **Approach 5: Partial Dewatering and Floating Conveyor Dumping.** This involves partially dewatering the pit and dumping material initially with a floating conveyor (to establish a trafficable layer), then dewatering and using conventional earthworks.

The options were assessed via a simple multiple attribute rating (SMART) method by comparing scores for each of the selection criteria (which reflect the project objectives/priorities). Scoring was done based on qualitative estimation of the selection criteria. The options were then ranked according to their total score and the highest scoring approach was select as the preferred option (Approach 5).

**Refined Approaches**

Approach 5 (denoted 5A for clarity) was used as the basis for a refined approach (5B) with increased initial clean water volume removed from the pit and quantity of PAF material backfill by; accelerating tailings settlement during backfilling, and having more of this material placed using conventional earthworks (resulting in higher density). The tonnage of waste rock, volumes of impacted and unimpacted water, and time to backfill were quantified for these options by way of backfill schedules and material balances. The two approaches were then scored and compared using the SMART method.

**Preferred Approach**

Approach 5A overall scored highest and was selected by RGC as the recommended advanced conceptual backfilling option. Note that if different weightings were applied to the selection criteria, the result may be different. Also, if one or more criteria had absolute precedence (e.g. stable long-term landform) then other backfill methods that were not considered in this study (e.g. dredging of tailings) could be better suited.
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LIST OF ACRONYMS AND ABBREVIATIONS

AHD - Australian Height Datum

AMD - Acid and Metalliferous Drainage

DME - Department of Mines and Energy

EFDC - East Finniss Diversion Channel

NAF - Non Acid Forming

NT - Northern Territory

PAF - Potentially Acid Forming

RGC - Robertson GeoConsultants Inc.

WRD - Waste Rock Dump

1 INTRODUCTION

1.1 GENERAL

This report describes candidate approaches to backfilling the Main Pit at the former Rum Jungle Mine Site (Rum Jungle). Robertson GeoConsultants (RGC) identified a preferred approach from a number of alternatives in consultation with the Northern Territory (NT) Department of Mines and Energy (DME) and their other project consultants. Identification of a preferred approach was based predominantly on technical considerations. The project objectives/priorities and a description of the alternatives are discussed. For the preferred approach, RGC recommends further investigation and analysis to advance the design. A recommended scope of work for further study is provided.

1.2 TERMS OF REFERENCE

1.2.1 Site Details

Rum Jungle is located about 105 km by road south of Darwin, NT, near the town of Batchelor (Figure 1-1). The mine operated between 1952 and 1971.

The site is characterized by a sub-tropical wet-dry climate and typically receives about 1,500 mm of annual rainfall. Most rainfall occurs during a distinct wet season that typically lasts from November to April, with little rainfall between May and October. Mean monthly maximum temperatures at the Batchelor Airport range from 31°C in June to 37°C in October (during the ‘build up’ to the wet season).

The East Branch of the Finniss River flows through the mine site. It was diverted to allow open pit mining of the Main and Intermediate ore bodies. Peak flows are currently routed through Main and Intermediate Open Pits. Non-peak flows are diverted through the East Finniss Diversion Channel (EFDC). Surface water enters the site from the east via the upper East Branch of the Finniss River and from the south east via Fitch Creek.
1.2.2 Mining History

According to Davy (1975), the mine produced 3,530 tonnes of uranium oxide, 20,000 tonnes of copper concentrate, and lesser amounts of cobalt and lead concentrate. The Main ore body was mined by underground methods from 1950 to 1953 and then by surface mining methods until 1958. Dysons ore body was mined in 1957 to 1958 and the Intermediate ore body was mined in the mid-1960s, both via open pits.

Waste rock removed from the open pits is stored in three WRDs: Main, Intermediate, and Dysons. Untreated, finely-ground tailings of unknown characteristics were discharged into the Old Tailings Dam or Dysons Open Pit from 1953 to 1965.

The Old Tailings Dam received un-neutralised slurried tailings from 1953 to 1961. It was located in a flat 30 hectare area north of Main Pit. Tailings accumulated behind a series of small dams. Supernatant liquor and drainage from the area reported to the East Branch of the Finniss River. According to Houghton (2009), the Old Tailings Dam had been subjected to many years of seasonal flooding which removed large volumes of the fine fraction and slimes.

The tailings were deposited into Dysons Pit from 1961 to 1965 via slurry pumping discharged from the south western end, resulting in a beach forming at this end. Saturated, finer materials (slimes) accumulated north east in the deeper sections of the cut. About 600,000 tonnes of tailings were deposited into this pit.

During operations, a processing plant was located near the Main Pit and a heap leach pad between the Main and Intermediate Pits. The Copper Extraction Pad was used to process copper ore from Intermediate Pit.

According to the Department of Transport and Works, Volume 2 (1981), un-neutralised tailings were sub-aqueously discharged at the northern perimeter of the Main Pit from 1965 to 1971. The document states that 700,000 tonnes of tailings were discharged.

1.2.3 Initial Rehabilitation

The mine was not rehabilitated at the time mining operations ceased in 1971. During 1977 and 1978 initial attempts were made to clean up the treatment plant area (Verhoeven, 1988). Severe impacts of AMD occurred until 1984 when large-scale rehabilitation began. Rehabilitation works were undertaken in 1984 and 1985. According to Verhoeven (1988), rehabilitation primarily involved:

- Re-shaping and covering the Main WRD and the majority of the Intermediate WRD and Dysons WRD.
- Re-locating a portion of the Main North WRD and covering the remainder in-situ.
- Treating highly-contaminated water that had filled the Main and Intermediate Pits.
• Backfilling Dysons Pit with additional tailings, low-grade ore and contaminated sub-soils collected from the former copper extraction area and the Old Tailings Dam.
• Covering the backfilled Dysons Pit with drainage and plant-growth mediums.

Monitoring in the period shortly after rehabilitation demonstrated that these measures achieved the objectives that were established under the Rehabilitation Agreement (see Verhoeven, 1988). However, current monitoring and conditions of waste facilities at the site indicate that further rehabilitation is needed to meet contemporary environmental performance standards and also to address the concerns of the traditional Aboriginal owners of the land.

1.2.4 Current Rehabilitation Planning

In 2009, the Commonwealth Government of Australia and the NT Government entered into a four-year National Partnership Agreement (NPA) for the management of the site. As part of the NPA, the Mining Performance Division of the NT Government’s Department of Resources (now the DME) was tasked with characterizing current conditions of the site and developing a revised rehabilitation strategy. That strategy is intended to:

• Achieve locally-derived water quality objectives (LDWQOs) for the East Branch of the Finniss River.
• Ensure the long-term physical and geochemical stability of waste rock and tailings stored at the site.
• Achieve the post-closure land-use aspirations of the Traditional Owners.

To achieve these objectives, the DME selected a rehabilitation strategy during Phase 1 of the Rum Jungle Rehabilitation Project. The preferred strategy was selected from a series of five alternatives after consultation with various government departments and Traditional Owner representatives (see RGC, 2013). The preferred strategy includes:

• Backfilling Main Pit with waste rock.
• Building a new Waste Storage Facility (WSF) north east of the Old Tailings Dam area.

Main Pit would be backfilled to above ground surface with the East Branch of the Finniss River to flow around the covered pit. PAF waste rock would be placed in the groundwater saturated zone so it remains submerged year-round. Because this waste rock will have no contact with atmospheric oxygen, it will not generate AMD after rehabilitation. NAF waste rock and clean fill would be placed above the saturated zone as the final landform and cover system.
1.3 **STUDY OBJECTIVES AND SCOPE OF WORK**

The primary objective of this study is to develop and evaluate alternative approaches to backfill the Main Pit to select a preferred approach. The study comprised the following activities:

- Research the mining, post-mining and rehabilitation history of Main Pit.
- Interpret the current pit geology, configuration and materials.
- Define the backfill project objectives/priorities, established by the DME and project team.
- Identify the main pit geo-hazards and risks associated with backfilling.
- Brainstorm potential backfilling approaches.
- Assess and compare approaches against selection criteria.
- Refine and compare options to develop an advanced conceptual approach.
- Recommend future investigations and analyses to advance the design.

This report is a deliverable in accordance with the RGC document titled ‘Request for Variation 2 – Phase 5 Groundwater Investigations at the Rum Jungle Mine Site’, dated 16 November 2015.
2 MAIN PIT INFORMATION

To develop a feasible solution, it is important to identify the potential geo-hazards in Main Pit. This can only be achieved with an understanding of the geological and anthropogenic history of the pit.

2.1 DOCUMENTS REVIEWED

The following key documents were reviewed for specific information on the pit:


In addition, the following resources were used:

- Bathymetry survey – supplied by the DME, March 2015.
- LiDAR survey – supplied by the DME, April 2015.
- Aerial imagery and photos of the site (1957 - 1958) viewed on the DME website (DME, 2012).

2.2 GEOLOGY

2.2.1 Regional

The regional geology is described in detail in a number of documents, such as Berkman (1968). In brief, the site is situated in a triangular area of the Rum Jungle mineral field that is bounded by the Giants Reef Fault to the south and a series of east-trending ridges to the north (Figure 2-1). This triangular area is known as ‘The Embayment’ and it lies on the shallow-dipping limb of a north east-trending, south west plunging asymmetric syncline that has been cut by northerly-dipping faults.

The main lithological units in The Embayment are the Rum Jungle Complex and meta-sedimentary and subordinate meta-volcanic rocks of the Mount Partridge Group. The Rum Jungle Complex consists mainly of granites and occurs primarily along the south eastern side of the Giants Reef Fault, whereas the Mount Partridge Group occurs north of the fault and consists of the Crater Formation, the Geolsec Formation, the Coomalie Dolostone, and the Whites Formation.
2.2.2 Main Pit

According to Williams (1963), no detailed geological structural analysis was carried out in Main Pit at the time of operations. The structure around the periphery is very complex, having been subjected to at least four generations of movement with brecciation in the later stages. The following sections provide a brief description of the geology of Main Pit relevant to this study.

The ore body was located on the northern limb of a tightly folded syncline, on the contact between the Coomalie Dolostone and the Golden Dyke Formation. It was hosted in the carbonaceous pyritic slate member of the Golden Dyke Formation which is a basal mudstone sequence comprising Mudstone, Schist, and Slate. It is a quartz-sericite material with a strong foliation, due to at least two generations of micro-folding.

As shown on the composite pit geology map in NT DTW (1981), generally speaking, the northern half of the pit mostly consists of Mudstone and the southern half is Slate except for a significant zone of Dolostone in the south-south east. An intensely sheared zone trending approximately east-west, bisects the pit which is called the ‘main shear zone’. A north-south fault and east-west fault associated with tectonic shattering truncated the ore body at depth.

A simplified interpretation of the main geological units and structural features exposed in the pit walls are shown on the marked-up bathymetry survey plan in Figure 2-2.

2.3 Pit Geometry

Summary details of the historic and current pit geometry and composition are provided in this section. More detailed discussion can be found in the RGC document titled ‘Rum Jungle Mine Site – Open Pit Backfilling Evaluation’, dated May 2015 (Appendix A).

Figure 2-3 is the idealized pit shell at the end of mining taken from Berkman (1968) showing the current in-pit materials which are discussed in this section. The section is aligned approximately north-north-west to south-south-east as shown on the pit plan view in Figure 2-2.
2.3.1 End of Mining

Main Pit was mined to about 110 m depth (Fitzgerald and Hartley, 1965) below ground surface, roughly circular in plan with a diameter of about 350 m. According to Berkman (1968) the average pit slope angle was about 40°. A haul road (see Figure 2-4) spiraled down clockwise starting at the southern perimeter of the pit and switched-back anticlockwise at approximately three quarters of the way down, presumably to avoid the shear zone exposed on the eastern wall.

On the basis of historical photos, catch-bench spacings were between 10 m and 30 m. Bench and batter geometry was poorly defined over much of the pit due to wasting causing filling-in of the benches. Filling occurred even during mining operations as evidenced in the historic photo in Figure 2-4.
2.3.2 Current

Based on the historical photos (NT DME, 2012) and by comparing the site in 1977 and 2010, we may conclude that the pit surface footprint has not noticeably changed with the exception of the northern perimeter. Over about a 50 m length of the perimeter the rim appears to have moved approximately 10 m towards the center; this is inferred to be due to backfilling during rehabilitation in the 1980s.

On the basis of the bathymetry survey, we conclude that the upper half-spiral of the former haul road is relatively intact. Beyond this the original haul road is covered by backfill and scree. The former benches are also mostly indistinguishable due to filling-in resulting in relatively uniformly sloped pit walls with overall slope angles ranging from between 25° to 30° in the mudstone and 28° to 38° in the slate. Flattening of the pit walls since mining suggests that the slope has deteriorated.
The floor of the pit in the south is mapped to be at about 16 m AHD which is thought to be the distal extent (slimes zone) of the tailings. There is a slight slope in the central and southern areas which becomes progressively steeper to the north and over the former pit walls, up to about 15°. The slight slope increase over the central and southern areas is consistent with fine-grained tailings sedimentation, whereby progressively finer components drop out of suspension with increasing distance from the discharge point. The geometry of the cone in the north is suggestive of coarser-grained material rather than tailings.

There is anecdotal evidence that soil from the Old Tailings Dam and waste rock was end-dumped into the pit during the rehabilitation works which likely accounts for all or some of the depositional cones. There is evidence of an end-dumped fill zone on the eastern side of the pit which pre-dates the tailings deposition (covered by tailings at the toe). A scree cone on top of the tailings to the east is inferred to be due to failure of materials in the main shear zone after tailings deposition.

Using the bathymetry survey and post-mining pit shell, RGC constructed a pseudo three-dimensional model (Figure 2-5). The bathymetry data is from below pit lake level so the pit freeboard above is not shown. Volumes were calculated using the site LiDAR data to capture the zone impounded by this freeboard. Approximately 800,000 m$^3$ of backfill was calculated to be in the pit bottom, with about 2,800,000 m$^3$ of space above. These numbers are consistent with those presented in Verhoevan (1988).
According to Verhoevan, treatment precipitates (sludge) generated from the in-pit liming operations during rehabilitation overly the tailings; however the composition and configuration is unknown. Decomposed organics may also have settled to the lake floor since completion of rehabilitation works.

2.4 MATERIAL PROPERTIES

There has been no intrusive in-pit investigation post mining; therefore information on the current pit materials is largely inferred from historical documents and similar sites. Provided below is information on the pit walls and in-pit backfill materials which predominantly consists of tailings but may also include soils and waste rock.

2.4.1 Pit Walls

Berkman (1968) states that the material in the pit is generally soft slates or intensely fractured chloritic rock. Such argillaceous rocks are particularly susceptible to swelling and softening when saturated. Observations of the walls around the perimeter of the pit by Andy Thomas of RGC in July 2015 support this; in wetted zones of the walls it was evident that the rock had softened, in some cases to a clayey soil-strength material. The exposed wall material strength ranged from very low (Coomalie Dolostone) to medium in the other units. Figure 2-6 is a photo of the highly weathered, low strength argillaceous rock in the walls above the pit lake level, taken during the RGC site visit.

![Figure 2-6 - Photo showing softened rock in the wetted zone of the pit walls](image)

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There is no information available regarding the shear strength of the pit wall rocks or the stability during mining. The deterioration in pit slope materials during mining and since flooding is not well understood.

2.4.2 In-pit Backfill Materials

The backfill materials already in Main Pit are poorly understood as regards their properties and geotechnical characteristics. Historical documents note that the backfill is predominantly tailings, but there is some uncertainty particularly about conditions around the cone in the north. On the weight of evidence, RGC’s interpretation is that there is coarser-grained material (possibly soil) overlying the tailings in this area. The only way to resolve this is would be to do intrusive investigations in the pit.

Recognising that most of the backfill is tailings, discussion of tailings behavior and properties is presented below.

Segregation

The sub-aqueous tailings are likely to have appreciably less segregation than sub-aerially deposited tailings. This is consistent with observations of the tailings surface in the central and southern areas.

Grain Size

Allen & Verhoeven (1986) indicate that the tailings type was generally finely ground, acid leached waste from the processing of uranium and copper ore. The mine deposits were hosted in fine grained metamorphic rock which would be expected to result in fine grained tailings and hence potentially very low permeability materials.

Consolidation

Given the length of time since deposition, the tailings in the main pit are expected to be normally consolidated due to their own (buoyant) unit weight; excess porewater pressures have probably dissipated. The material is expected to have relatively high void ratios and low shear strength near the tailings surface where the effective stress in the submerged tailings is very low. They have the potential to be highly compressible, generate excess pore pressures under backfill loading, and consequently have low shear strength under conditions of rapid loading.

Shear Strength

The shear strength of the tailings will determine if it will support material being placed onto it. Tailings with the characteristics presented above would be weak and not trafficable. The geotechnical characteristics must be known to allow assessment of what loadings can be applied and the allowable rate of backfilling, if slumping and failure is to be avoided.
2.4.2.1 Analogous Tailings Deposits

In the absence of actual data, tailings properties and loading response from Dyson’s Pit and an analogous site (IAA Helmsdorf) where RGC has experience are discussed.

Dysons Pit

Houghton (2009) studied the tailings deposited in Dysons Pit. He explains that the lower tailings layer was sub-aerially deposited at the time of mining, up to the capacity of the facility. Subsequently 8.5 m of tailings/contaminated soil from the Old Tailings Dam were placed on top during rehabilitation. The tailings of the latter materials were likely the remnant coarser components because the slimes are thought to have been mostly washed away.

Houghton noted that the largest settlement occurred at the eastern end of the pit; the slimes zone of the facility during operations. Approximately 0.9 m settlement was measured in the first 18 months post-rehabilitation, followed by an additional 2.0 m in the subsequent 22 years. This total settlement represents 6 % of the tailings thickness at this location.

IAA Helmsdorf

The IAA Helmsdorf site is a legacy of the former Soviet-German mining company, Wismut, located in former East Germany. The very fine-grained uranium ore tailings were placed sub-aqueously in a large valley tailings facility, up to 45 m thickness. Despite about 35 years of self-weight consolidation, investigations in the 1990s revealed high void ratios and low shear strengths which varied significantly despite the uniformity of the material (Jakubick, 1990).

Modeling done for the reclamation cover design predicted large variations in total settlement, depending on the tailings properties. For a 2 m thick surcharge load, settlement modeling predicted relative settlement to range from around 4 % of total thickness for a relatively coarse-grained (sandy) tailings profile up to 22 % for a uniformly fine grained (clayey) tailings profile. The predicted time of consolidation in this scenario ranged from about one year for a sandy tailings profile to more than 30 years for the fine slimes profile (Wels et.al, 2000).

Further detailed technical discussion of the soil mechanics of tailings settlement with reference to the IAA Helmsdorf site is provided in Appendix B and Section 5.1.2.

2.5 GROUNDWATER

A brief description of groundwater is provided in this section for context in the latter calculations in this report.

Historical data on the groundwater regime during the period of mining is virtually non-existent, and there is very little in the mining records (NT DTW, 1981). Pumping records provide total pumped
volumes from the pit during mining but details are sparse on the location or rate of inflows. However some pertinent details in NT DTW (1981) are summarized below:

- A principal aquifer was observed in the Coomalie Dolostone in the south eastern corner of the open cut at about 27 m below ground surface.
- In April 1956 the seepage rate was reported at 34 L/s.
- In May 1958 seepage from the Coomalie Dolostone unit exposed in the north perimeter of the pit was collected on a bench at 70 m depth.
- In June 1958, seepage was observed from the western end of the main shear zone 76 m below the surface.

The tailings has probably effectively sealed the deeper pit walls (below the tailings) and possibly under the fill cones, from the surrounding bedrock aquifers.

Groundwater modeling completed recently by RGC for rehabilitation planning predicted that the lowest seasonal groundwater level after full backfilling of Main Pit would be at approximately 58.5 m AHD (in the north western area of the pit). This elevation is therefore a conservative (lower) elevation to which PAF waste rock could be backfilled and remain permanently saturated. Sensitivity analyses indicated that the lowest seasonal groundwater level could be higher in all or at least the eastern portion of the pit, potentially allowing optimization of the PAF backfill limit.

In summary, groundwater inflows to the pit can be expected to be concentrated in certain zones corresponding to more transmissive geological units, and the rates will fluctuate seasonally. The groundwater response will vary depending on the backfill approach taken. For instance, lowering the pit lake level increases pit wall exposure and hence the rate of inflow. Therefore the design of the initial dewatering and pumping during backfilling must take into account the pit groundwater regime.

2.6 Pit Lake Water Quality

RGC (April 2012) provides data on the quality of the water with depth in the pit lake, from sampling conducted in 2010/2011. In general, the water quality in the upper portion of the pit lake shows very little mining impact while high concentrations of the AMD products SO₄ and metals are observed in the untreated water near the bottom. The transition from dilute, unimpacted water to more saline water at depth is relatively sharp and is referred to as a ‘chemocline’. Electrical Conductivity (EC) profiles of the Main Pit provided in RGC (April 2012) indicate that the chemocline is gradually moving downward from about 38 m AHD in 1990 to about 22 m AHD in 2008. This ‘erosion’ of the chemocline is believed to be due to seasonal flushing of the pit lake with river water.

The source of the contamination is thought to be highly-contaminated residual pit lake water, likely seepage from the former Copper Extraction Pad (to the west of the pit) where heap leaching was
previously done. This type of contaminated water probably flooded into the pit when mining dewatering ceased.

Residual pore water in the sub-aqueously placed tailings is also believed to be highly impacted (due to processing in an acid leach circuit) and may be a secondary contaminant source which is contributing to the poor quality pit water near the bottom.

Dewatering of the pit lake and dumping material into the pit lake could disturb the chemocline and potentially disperse the contaminants of the deeper impacted layer throughout the entire water column. This would impact the quality of the pit water and dictate how it needs to be managed, possibly requiring expensive treatment for discharge. Similarly, backfill dumping of PAF waste rock could potentially contaminate the pit lake by releasing stored ARD products. The extent of contamination and hence possible requirement for treatment is a critical consideration in the backfill design.

2.7 GEO-HAZARDS

There are a number of geo-hazards present, or potentially present, in the pit. Each backfilling approach (see Section 3) involves different risks, and risk levels. Some of the main unknowns and plausible hazards are:

- Variable thickness of treatment sludge and/or organic deposits which are probably of very low strength.
- Unknown and variable tailings composition (both laterally and vertically), likely to be of low strength, high void ratio, and low permeability over all or some of the footprint.
- Steep backfill/scree cones founded on tailings in a meta-stable condition potentially exacerbated by dewatering.
- Compressible materials which could take many years to consolidate.
- Tailings susceptible to liquefaction and sudden loss of strength.
- Unstable, low strength and highly fractured (in shear zones) pit wall materials since softened from pit flooding, susceptible to sliding and slumping.
- Clayey pit wall skin impeding dewatering of the walls leading to wall pressurization and instability.
- High decay rate and possible solution channeling/undercutting in the Coomalie Dolostone exposed in the pit.
- Zones of contaminated pit lake water and highly impacted tailings porewater.

These geo-hazards are discussed further for the backfill approaches in Section 3 to 5.
3 CANDIDATE BACKFILLING APPROACHES

This section describes the preliminary potential backfilling approaches and their appraisal.

3.1 OBJECTIVES

To aid in selecting a preferred approach, the following backfilling objectives/priorities were established in consultation with the DME and their consultants:

- protect the health and safety of personnel
- technically feasible with reasonable probability of success
- meet construction timeframes
- optimize dewatering prior to backfilling
- optimize the quantity of PAF material backfilled into the pit
- deliver the project in a cost-effective manner
- efficient and allows for the addition of lime during backfilling
- create a stable, long-term landform

These were considered when developing the candidate approaches and used as the criteria to compare them.

3.2 APPROACHES

Conceptual backfilling approaches were initially developed during a brainstorming session by the RGC project team and later refined in a meeting in Darwin attended by staff from DME, RGC and O’Kane Consultants (OKC). Although there are a multitude of possible iterations, the general steps in each method are described for screening level assessment.

The construction approaches are limited to the backfill of PAF material only. It is common to all approaches that NAF material/clean fill would be placed above the elevation of 58.5 m AHD by conventional earthworks.

Note that the backfill material is collectively termed ‘waste rock’, but in part may also include contaminated soils and spent heap leach material from Dyson’s Pit.

3.2.1 Approach 1: Dewatering and Dry Placement

3.2.1.1 Description

This approach involves dewatering the pit entirely and placing waste rock by conventional earthworks. To access the pit floor with earthmoving machinery, the low strength tailings surface will need to be strengthened. There are industry-established methods of doing this, such as that
undertaken by RGC (Wels et. al., 2000) at Wismut in the early 1990s on tailings with shear strengths as low as 3 kPa. The general steps are:

1. Dewater the entire pit using submersible pumps on floating pontoons and continue pumping to maintain a dry surface.
2. Pit push-back to establish a haul road to the bottom.
3. Working from the access point, if necessary remove surficial sludge with a long-boom excavator.
4. Working in incremental forward advances, place geosynthetic layers and initial drainage layer as needed to improve the surface strength and continue pumping to maintain a dry surface.
5. Haul waste rock and place initially in thin lifts with light earthworks equipment.
6. Place and compact in thicker lifts using dozers and rollers to spread and compact (conventional earthworks).

Figure 3-1 is a schematic of this approach.

![Figure 3-1 - Approach 1 schematic](image)

### 3.2.1.2 Discussion
The need for geosynthetics as well as the placement methodology will depend on the bearing strength of the tailings surface, which is predicted to be low in the slimes zone in the south and increase towards the beach zone to the north. For surface shear strengths less than 50 kPa, geosynthetic reinforcement becomes increasingly necessary. For shear strengths below 20 kPa heavy geotextile (or geogrid or both) use is typically required with thin layer cover advancement using special light spreading equipment. Below 5 kPa, special placement methods such as the use of long boom excavators may be required in addition to the use of geotextiles to achieve the initial layer
placement. For sandy tailings, with relatively high shear strengths, cover placement may be achieved without reinforcement.

Assessment of the bearing strength of the tailings and other pit floor materials would be required and could be completed prior to cover placement (e.g. from a pontoon) and as construction proceeds.

To prevent puncturing of the geotextiles, initial lifts of waste rock would need to use select material or screened/processed waste rock from which the oversize particles have been removed. Layer thickness and compaction effort would need to be designed to prevent damage. The material particle size gradation would need to be such that the generated seepage water can freely drain into this layer.

Tailings are typically very low permeability materials and as such drainage can take a long time, particularly in the fine grained materials of the slimes zones. Loading will generate excess porewater pressures which will dissipate over time, principally as a function of the drainage properties. Consolidation of the tailings will occur as excess porewater pressures dissipate. By reducing the length of the drainage path using wick drains the rate of consolidation can be increased significantly, conceivably so that the majority occurs during construction. Installation of wick drains could be included in this approach. Wick drains would be installed after placing the geotextiles; a small crane rig would be used. An additional advantage of doing this is that the strength gains would allow thicker backfill lifts. Further discussion of wick drains is provided in Section 4.2.2.1.

3.2.2 Approach 2: Crest Dumping

3.2.2.1 Description
This approach involves dumping waste rock from the pit crest and then advancing machinery out and ‘crossing over’ the pit crest/slopes onto the advancing backfill. The general steps are:

1. Dewater the pit lake to the desired level which may not necessarily be the pit bottom, depending primarily on water quality considerations.
2. Back end dumping waste rock into the pit from trucks or pushed using dozers, around the entire pit perimeter.
3. If and when a platform is established at the dump crest, progressively push material towards the centre of the pit.
4. When the toes of the dumps converge in the middle and the dumps are overlapping, use conventional earthworks to place and compact the remaining space.

Figure 3-2 is a schematic of this approach.
3.2.2.2 Discussion

Surcharge loading would occur on the pit wall slopes if waste rock were end dumped from the pit crest. The natural angle of repose of clean permeable waste rock is typically about 38° or 1.5H:1V. If waste with an angle of repose of 38° is advanced over a pit wall slope of 30° it creates a wedge of material on the upper part of the slope. This wedge effectively steepens the slope (to 38°) increasing the shear stress on the interface between the base of the waste rock and the pit wall. There is risk of this wedge sliding on the weathered pit wall slope.

Where the pit wall slopes are steeper than 38° the dump will be continuous to the pit bottom. Because the material will be at angle of repose as well as founded on low strength tailings and weathered pit walls, in addition to the risk of failure along the wall, there is also risk of basal failure in the tailings below the base. Regardless, operating on the waste rock, at angle of repose, will be a risk to machinery and personnel.

Surcharge loading of the pit crest from waste rock piles and earthworks machinery would act to destabilise the crest and could cause failure of the walls. This is particularly a problem if the walls have softened or weathered.

From review of the historical photos and the bathymetry survey, it is evident that localized pit wall sliding has previously occurred. The possibility of sudden crest failure would present an unacceptable risk to operators and machinery. The risk to personnel could potentially be eliminated by using remote-controlled machinery but there remains the risk of significant financial loss associated with loss of machinery and associated delays.
3.2.3  **Approach 3: Overwater Dumping with Dewatering and Placement**

3.2.3.1  Description

This approach involves initially dumping waste rock from the water surface (from barges or floating conveyors) to establish a trafficable layer, without initially lowering the pit lake level. Steps include:

1. Establish floating equipment (barge or conveyor).
2. Establish waste rock transfer facility from the pit crest to the floating equipment.
3. Dump waste rock into the water by continuous conveyor operation or by loads using bottom-dumping barges, to cover the pit bottom.
5. When sufficient load bearing thickness is established (predicted to be at least 5 m), secondary dewatering of the pit to the level of the waste rock and continue pumping to maintain a dry surface.
6. Pit pushback to establish a haul road to the backfill surface.
7. Haul waste rock and use conventional earthworks to place and compact.

Figure 3-3 is a schematic of this approach.

![Figure 3-3 - Approach 3 schematic](image)

3.2.3.2  Discussion

Overwater dumping without dewatering greatly reduces the risks to personnel and machinery resulting from pit wall instability. There remains the potential hazard of development of a displacement wave from wall or tailings failure which could damage floating equipment. Regardless this approach is a relatively safe way to initially cover the tailings and build a trafficable working surface, thus also addressing the risks associated with working on a low bearing strength layer.
To permit drainage and trafficking of the tailings cover layer, the dumped material would need to be select material or screened/processed waste rock.

The method of barge dumping would be slow, limited by barge size and time to load and move the waste rock. The use of a floating conveyor would be more efficient, plus the dumping rate could be moderated to gently ‘rain’ the material onto the tailings. More discussion on this method is provided in Section 3.3.4.

### 3.2.4 Approach 4: Pit Edge Stacker Dumping

#### 3.2.4.1 Description

This approach could be done with either a flooded or dewatered pit, however in reality some pit freeboard may be necessary to protect against overtopping of the rim from waves set up by dumping. The general steps include:

1. Dewater the pit lake to the desired level which may not necessarily be the pit bottom, depending primarily on water quality considerations.
2. Construct stacker pads around the pit perimeter, reinforcing the pit walls if and where necessary.
3. Establish long boom conveyor/s to the site and set up at a safe distance from the pit.
4. Discharge waste rock in a series of overlapping radial arcs at locations around the perimeter such that the dump footprints overlap.
5. Infill the zone between the dump crests and the pit edge to establish a platform to access the pit.
6. Haul waste rock and use conventional earthworks to place and compact.

Figure 3-4 is a schematic of this approach.

![Figure 3-4 - Approach 4 schematic](image)
3.2.4.2 Discussion
This approach involves using long-boom conveyor stacker/s. Long boom stackers have been built with boom lengths ranging up to about 125 m. The largest mobile stackers have a boom length up to about 60 m. Larger than this the equipment is custom built and relatively immobile. Establishing such large and exotic equipment to the site would be challenging, costly and take time. Obtaining and using smaller equipment would be more feasible but the waste rock dumps would not meet in the centre of the pit, necessitating ‘crossing’ the pit crest and onto the loosely-placed dumps. This would expose the equipment and personnel to hazards associated with potential failure of the waste rock.

3.2.5 Approach 5: Partial Dewatering and Floating Conveyor Dumping

3.2.5.1 Description
This approach was developed to decrease the volume of pit water in contact with waste rock and requiring treatment; essentially an optimisation of Approach 3. It involves partially dewatering the pit lake and depositing waste rock initially with a floating conveyor.

1. Dewater the pit lake to the desired level, which would need to account for the logistics of establishing a decline conveyor from the pit crest level to the floating equipment (considering pit access and decline length limitations).
2. Minor pit pushback to establish a haul road to the desired pit lake level.
3. Dump waste rock into the water by continuous conveyor operation and backfill as high as practicable.
4. Pump and treat displaced pit water from and during backfilling to maintain a static lake level.
5. Secondary dewatering (and treatment) of the remaining pit water to establish a dry surface and continue pumping as required to maintain a dry and stable working platform.
6. Haul waste rock and use conventional earthworks to place and compact.

Figure 3-5 is a schematic of this approach.
3.2.5.2 Discussion
This approach is a compromise to better balance achieving the different project objectives. It allows the removal of a significant amount of pit water before contact with contaminated waste rock. At the same time, a sizeable portion of the backfill will be placed with conventional techniques to maximize the tonnage placed while minimizing the risks associated with pit access and in-pit earthworks.

3.3 DISCUSSION OF COMMON ELEMENTS OF THE APPROACHES

The following is information on important aspects common to more than one of the approaches.

3.3.1 Dewatering

Pressurization

During mining, the pit was excavated at a slow rate and the surrounding groundwater table probably fell consistently with the increasing depth of the pit. If the pit is now dewatered, a ‘rapid drawdown’ scenario will develop. If the pit is dewatered quickly the phreatic surface in the walls will be transient and the pore pressures will be greater than was the case when the pit was excavated. These pore pressures may destabilize parts of the pit walls. The water in the pit currently acts as a slope ‘stabilizing’ force that would be removed on rapid dewatering.

To minimize drawdown instability, dewatering the pit lake would need to be at a slower rate than the rate of fall of the surrounding groundwater surface. Geological units and structural features have unique drainage characteristics therefore the response will not be uniform around the pit. Slope stability analyses will be required for a range of different pit wall characteristics (hydraulic and geotechnical) to determine a safe dewatering rate.
The pit lake drawdown rate could potentially be increased by pumping out groundwater from perimeter dewatering wells; however, the feasibility of this approach would need to be assessed. A distinct advantage in doing this would be to reduce the inflow of clean groundwater before it comes into contact with contaminated waste rock and pit water.

The phreatic surface would need to be monitored and controlled once the pit is dewatered to maintain pit wall stability. This issue is an important consideration for the approaches where personnel and equipment need to enter the pit.

**Contamination**

Water that contacts the waste rock in the overwater dumping approaches is predicted to become contaminated. Approach 3 for example, would cause the entire pit lake to become contact water. Similarly, water near the bottom of the pit water column and in the tailings pore space is anticipated to require treatment to meet the criteria for discharge. Water treatment can be costly and potentially put constraints on the dewatering and hence progress of backfilling. As such it is desirable to minimize the volume of contact water.

The tailings porewater is probably highly contaminated, requiring intensive water treatment. This water will seep out of the tailings as they consolidate in response to backfill loading. The approaches incorporating pit water pumping to remove the pit lake and maintain a dry working surface (Approach 1, 3 and 5) will entrain this water. Whereas for Approaches 3 and 5 it will be mixed and potentially diluted with the comparatively clean pit lake water, this cannot occur in Approach 1. The required rates of dewatering will depend on a number of factors including; groundwater inflow rate, seasonal precipitation and evaporation, dewatered pit lake level, rate of backfill and length of construction.

Secondary dewatering of the residual pit lake would need to be in part via sumps in strategic locations over the pit floor to be able to drawdown the phreatic surface sufficiently for a dry working surface.

**3.3.2 Push-back/Haul Road**

Accessing the pit for Approaches 1, 3 and 5 requires the development of a haul road. This could be by rehabilitating the former road, or constructing an entirely new road. Both options will require some extent of push-back between the access ramp and the pit rim to create stable slopes.

Depending on the required access level, pit wall conditions and other factors, it may be feasible to re-establish the relatively intact portion of the former haul road which is at about 5 % grade. Alternatively, and where the former road is unable to be re-established, a new road could be constructed at a steeper grade closer to contemporary designs; up to about 10 % grade.

The push-back overall angle may need to be flatter than previous; this is because materials have deteriorated since mining. Due to the low strength and highly fractured nature of the pit wall...
materials, stabilization measures such as meshing/shotcreting and rock bolting are unlikely to be suitable or economic.

Construction is anticipated to involve blasting techniques. Spoil generated could be end dumped over the advancing haul road to avoid haulage. A different construction approach would be required if the haul road goes through the zones of scree/backfill because this material will be unstable both above and below the advancing level.

For durability, the road would be constructed with a wearing course of engineered material which could potentially be local screened/processed material.

### 3.3.3 Waste Rock Dumping/Placement

All approaches, with the exception of Approach 1, involve a dumping method (i.e. conveyor, barge dumping or end tipping) for a portion of the total backfilling. Dumping material can be fast and efficient but the resulting waste rock piles will be in a loose state i.e. with high void ratios. Due to loading and with time, the particles will reorient and compact to some degree, reducing the void ratios and resulting in settlement. The rate and amount of settlement will be a function of the waste rock material properties and loading; however, it will be largely uncontrollable. In contrast, backfill by placement in lifts and using compaction equipment increases the density and hence the tonnage of material backfilled. It also reduces post-construction settlement.

For Approach 3, and also to a lesser extent Approach 5, overwater dumping should not be concentrated in certain locations and done slowly enough so that it does not ‘punch’ through the tailings surface. The backfill approach could be analogous to ‘raining’ the material, at least initially, until a firm contiguous layer can be established.

### 3.3.4 Conveyor

Conveying material is highly efficient over large distances, particularly if it eliminates hauling by truck i.e. if it was used all the way from the material source. It also allows for efficient lime addition because the material is spread along the belt, allowing thorough coating.

The concept is to use a number of floating conveyor segments that could be articulated to enable coverage of the entire pit (see Figure 3-6). Segment movement could be via on-board motors or with cables attached to the shore. It has been used successfully for overwater material movement elsewhere but its applicability in this scenario would need to be properly assessed.
A constraint of using conveyors in this instance is that the materials must suit certain particle size specifications subject to the conveyor feed mechanism and belt size. As such, the waste rock may require some level of processing/screening. These requirements would need to be assessed in more detail if this method were selected.

### 3.4 SUMMARY

A summary of the advantages and disadvantages of the approaches is in Table 3-1.
<table>
<thead>
<tr>
<th>Approach</th>
<th>Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
</table>
| Dewatering and Dry Placement                 | • Technically feasible with rigorous and specialized design and construction practice  
• Maximizes volume of non-contact water        | • Major level of ground investigations                                       |
|                                              | • Promotes tailings consolidation during backfilling                       | • Pit push-back and wick drain installation takes a long time                  |
|                                              | • High backfill density                                                    | • Major pit push-back generates large spoil volume                           |
|                                              |                                                                            | • Potentially highly contaminated tailings pore water requiring intensive treatment |
|                                              |                                                                            | • High exposure to all pit geo-hazards                                       |
|                                              |                                                                            | • AMD products from exposed pit walls                                       |
|                                              |                                                                            | • Ongoing pumping and treatment of contact water during construction         |
|                                              |                                                                            | • Very high cost                                                            |
| Crest Dumping                                | • Minor level of ground investigations                                     | • AMD products from exposed pit walls                                       |
|                                              | • Maximizes volume of non-contact water                                    | • Dangerous for personnel and machinery working on pit crest and unstable waste rock slopes |
|                                              | • Fast and uncomplicated waste rock dumping                               | • Low backfill density                                                      |
|                                              | • Contact water does not need to be pumped and treated                    | • Large settlement after backfilling                                        |
|                                              | • Low cost                                                                 |                                                                            |
| Overwater Dumping with Dewatering and Placement | • Technically feasible if rigorous design and construction followed      | • Major level of ground investigations                                       |
|                                              | • Initial dumping uncomplicated and safe for personnel with low exposure to pit geo-hazards for machinery | • Select material or some waste rock screening/processing for cover layer dumping |
|                                              | • Low exposure to pit bottom geo-hazards                                   | • Gentle backfill dumping and pit push-back takes a long time                 |
|                                              | • Partial tailings consolidation from cover layer dumping and dewatering  | • Large volume of contact water                                              |
|                                              | • High backfill density                                                    | • Major pit push-back generates large spoil volume                           |
|                                              |                                                                            | • AMD products from partially exposed walls                                  |
|                                              |                                                                            | • Ongoing pumping and treatment of contact water during backfilling          |
|                                              |                                                                            | • Exposure to pit wall hazards during backfill placement                      |
|                                              |                                                                            | • High cost                                                                  |
| Pit Edge Stacker Dumping                     | • Minor level of ground investigations                                     | • High to extremely high cost if long-boom conveyor used                      |
|                                              | • Very low personnel and machinery exposure to pit geo-hazards             | • Dangerous for personnel and machinery if mobile conveyor/s used             |
|                                              | • Uncomplicated lime addition to conveyed waste rock                      | • AMD products from exposed walls                                            |
|                                              | • Maximizes volume of non-contact water                                    | • All waste rock to be screened/processed to conveyor requirements           |
|                                              | • Contact water does not need to be pumped and treated                    | • Low backfill density                                                      |
| Partial Dewatering and Floating conveyor Dumping | • Technically feasible with conventional design and construction practice | • Moderate level of ground investigations                                     |
|                                              | • Significant volume of non-contact water                                 | • Some pit push-back spoil                                                   |
|                                              | • Low exposure to pit geo-hazards                                         | • AMD products from partially exposed walls                                  |
|                                              | • Minor pit push-back                                                     | • Waste rock screening/processing for conveyor dumping                       |
|                                              | • Fast material movement and dumping                                      | • Some pit lake contact water                                                |
|                                              | • Partial high density backfill                                           | • Ongoing pumping and treatment of contact water during backfilling          |
|                                              | • Low cost                                                                |                                                                            |
4 SCREENING LEVEL ASSESSMENT

The approaches were assessed with a high-level comparison method. It was done using the Simple Multi-Attribute Rating Technique (SMART) which uses the following stages, as summarized from Olson (1996):

- Stage 1: Identify the decision-makers
- Stage 2: Identity the context and purpose of the decision
- Stage 3: Identify the alternatives
- Stage 4: Establish the criteria
- Stage 5: Assign values for each criterion
- Stage 6: Determine the weight of each of the criteria
- Stage 7: Calculate a weighted average of the values assigned to each alternative
- Stage 8: Make a provisional decision
- Stage 9: Perform sensitivity analyses

The decision-maker in this assessment is RGC (Stage 1). The context and purpose of the study (Stage 2) is explained in Section 1.3.

4.1 ALTERNATIVES

The backfilling alternatives (Stage 3) described in Section 3 are:

1. De-watering and Dry Placement
2. Crest Dumping
3. Overwater Dumping With Dewatering and Placement
4. Pit Edge Stacker Dumping
5. Partial Dewatering and Floating Conveyor Dumping

4.2 CRITERIA AND WEIGHTING

The project objectives were used as the assessment criteria (Stage 5) with the addition of one extra to capture the level of ‘Site Investigation’ that would be necessary for each approach.

Criterion were ranked based on their relative importance (Stage 6) and a weighting value was applied (out of 100) (Stage 6). Each weighting was then calculated as a proportion of the summation of the individual weightings (weighted average) (Stage 7). This allows normalization of the relative importance into weights summing to 1. The weightings and weighted averages for each criterion are in Table 4-1.
### Table 4-1 - Criteria details

<table>
<thead>
<tr>
<th>Criteria</th>
<th>Details</th>
<th>Weighting</th>
<th>Weighted Average</th>
</tr>
</thead>
<tbody>
<tr>
<td>Health and Safety</td>
<td>Of construction personnel</td>
<td>100</td>
<td>0.19</td>
</tr>
<tr>
<td>Technical Feasibility</td>
<td>Likelihood of success</td>
<td>100</td>
<td>0.19</td>
</tr>
<tr>
<td>Time</td>
<td>Total time for the project</td>
<td>75</td>
<td>0.14</td>
</tr>
<tr>
<td>Cost</td>
<td>Total cost of the project</td>
<td>75</td>
<td>0.14</td>
</tr>
<tr>
<td>Water Quality</td>
<td>Extent of impact to pit water</td>
<td>50</td>
<td>0.09</td>
</tr>
<tr>
<td>Waste Rock</td>
<td>Tonnage of waste rock backfilled</td>
<td>50</td>
<td>0.09</td>
</tr>
<tr>
<td>Settlement</td>
<td>Predicted total during construction</td>
<td>50</td>
<td>0.09</td>
</tr>
<tr>
<td>Liming</td>
<td>Ease and efficiency of addition</td>
<td>25</td>
<td>0.05</td>
</tr>
<tr>
<td>Investigations</td>
<td>Extent and complexity necessary</td>
<td>10</td>
<td>0.02</td>
</tr>
</tbody>
</table>

### 4.3 Scoring

Scores for each criterion for each approach are given in the range of 0 to 1 (Stage 8). Because criteria values/quantities are not well defined, or cannot be quantified (i.e. ‘Health and Safety’ and ‘Technical Feasibility’), scores were assigned to the qualitative categories shown in Table 4-2.

### Table 4-2 - Category scores

<table>
<thead>
<tr>
<th>Category</th>
<th>Score</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very bad</td>
<td>0</td>
</tr>
<tr>
<td>Bad</td>
<td>0.25</td>
</tr>
<tr>
<td>Moderate</td>
<td>0.5</td>
</tr>
<tr>
<td>Good</td>
<td>0.75</td>
</tr>
<tr>
<td>Very Good</td>
<td>1</td>
</tr>
</tbody>
</table>

The scores given to each alternative are presented in Table 4-3.
### Table 1-3 - Screening level assessment - scoring

<table>
<thead>
<tr>
<th>Criteria</th>
<th>Approach</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1</td>
</tr>
<tr>
<td>Health and Safety</td>
<td>0.25</td>
</tr>
<tr>
<td>Technical Feasibility</td>
<td>0.50</td>
</tr>
<tr>
<td>Time</td>
<td>0.00</td>
</tr>
<tr>
<td>Cost</td>
<td>0.00</td>
</tr>
<tr>
<td>Water Quality</td>
<td>1.00</td>
</tr>
<tr>
<td>Waste Rock</td>
<td>1.00</td>
</tr>
<tr>
<td>Settlement</td>
<td>1.00</td>
</tr>
<tr>
<td>Liming</td>
<td>0.25</td>
</tr>
<tr>
<td>Investigations</td>
<td>0.00</td>
</tr>
</tbody>
</table>

### 4.4 RESULTS

Results of the assessment are shown graphically in Figure 4-1 and the weighted component scores for each criterion are detailed in Table 4-4. The relative ranking of the approaches are also provided in Table 3.
### Table 4-4 - Screening level assessment - component values

<table>
<thead>
<tr>
<th>Approach</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Health and Safety</td>
<td>0.05</td>
<td>0.00</td>
<td>0.09</td>
<td>0.19</td>
<td>0.09</td>
</tr>
<tr>
<td>Technical Feasibility</td>
<td>0.09</td>
<td>0.00</td>
<td>0.14</td>
<td>0.09</td>
<td>0.14</td>
</tr>
<tr>
<td>Time</td>
<td>0.00</td>
<td>0.14</td>
<td>0.07</td>
<td>0.04</td>
<td>0.07</td>
</tr>
<tr>
<td>Cost</td>
<td>0.00</td>
<td>0.14</td>
<td>0.04</td>
<td>0.00</td>
<td>0.07</td>
</tr>
<tr>
<td>Water Quality</td>
<td>0.09</td>
<td>0.09</td>
<td>0.00</td>
<td>0.09</td>
<td>0.05</td>
</tr>
<tr>
<td>Waste Rock</td>
<td>0.09</td>
<td>0.05</td>
<td>0.05</td>
<td>0.00</td>
<td>0.07</td>
</tr>
<tr>
<td>Settlement</td>
<td>0.09</td>
<td>0.02</td>
<td>0.02</td>
<td>0.00</td>
<td>0.05</td>
</tr>
<tr>
<td>Liming</td>
<td>0.01</td>
<td>0.02</td>
<td>0.02</td>
<td>0.05</td>
<td>0.02</td>
</tr>
<tr>
<td>Investigations</td>
<td>0.00</td>
<td>0.01</td>
<td>0.01</td>
<td>0.01</td>
<td>0.01</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>0.43</strong></td>
<td><strong>0.48</strong></td>
<td><strong>0.44</strong></td>
<td><strong>0.47</strong></td>
<td><strong>0.57</strong></td>
</tr>
<tr>
<td><strong>Rank</strong></td>
<td>5</td>
<td>2</td>
<td>4</td>
<td>3</td>
<td>1</td>
</tr>
</tbody>
</table>

**Figure 4-1 - SMART analysis, Approach 1 to 5**
Partial dewatering and floating conveyor dumping (Approach 5) scored the highest.

4.5 Sensitivity Analysis

Criteria weighting is subjective, and with the exception of ‘Health and Safety’ and ‘Technical Feasibility’, could be different depending on the decision-maker. To investigate the potential influence of this, a sensitivity analysis of the total approach scores was conducted by varying the weightings of a number of key criteria to a range of nominal values; 0, 25, 50, 75 and 100. Graphs of the resultant total score are shown in Figure 4-2. Note that Approach 2 was excluded from the study because it was deemed to be technically infeasible with a score of 0 for ‘Technical Feasibility’ in the screening level assessment.
Figure 4-2 - Screening level assessment - weighting sensitivity analysis for key criteria
Observations from the sensitivity study are:

- Approach 1 is highly sensitive to all criteria.
- Approach 3 is highly sensitive to Water Quality but relatively insensitive to the other criteria.
- Approach 4 is highly sensitive to all criteria with the exception of Time.
- Approach 5 is relatively insensitive to all criteria.

Approach 5 scored the highest for all criteria regardless of the weighting value used. This occurs even when the weighting is zero because it scores high enough in the other criteria to maintain the highest overall score. This indicates that Approach 5 is robust as the most promising backfilling option.

4.6 DISCUSSION AND LIMITATIONS

Although the technique produces a numerical value, the ranking system is subjective and scoring was conducted in a relative sense i.e. a result twice the score does not necessarily imply double the benefit. The importance of the analysis and final score is for relative ranking of the alternatives, whereby the approach with the highest score is the most promising to meet the project criteria.

The assessment was conducted at a screening level only in a non-quantitative sense, using experience and engineering judgment. The approaches were not developed sufficiently to be able to conduct a reliable quantitative analysis. The purpose of the assessment was to identify the most promising candidate to subsequently refine.

Notwithstanding the above, based on the combination of very low scores for the critical criteria of health and safety and technical feasibility for Approach 2, RGC considers that at this level of assessment, this option is fatally flawed. The principal hazard of this approach is associated with operating equipment in the pit on meta-stable waste rock dumps. It would need to be proven that this hazard could be successfully mitigated to meet these important project criteria to justify developing this approach further.
5 SCREENED BACKFILLING OPTIONS

Based on the results of the screening level assessment, the methodology with the highest score (Approach 5) was selected for further assessment and refinement. Two sub-options were developed using this methodology and then compared against the selection criteria. This was done by defining the approaches in more detail and then evaluating the DME high-priority (and suitable) project criteria in a quantitative sense. The result is a semi-quantitative overall assessment, whereby the scoring of the criteria of Waste Rock, Water Quality and Time was done with quantified comparisons.

The overall objective was to evaluate candidate approaches in more detail than was possible, or warranted, at the screening level and ultimately select a preferred concept backfill approach.

5.1 OPTIONS

The refined sub-options were given the post-script ‘A’ and ‘B’ to differentiate them from the original.

5.1.1 Approach 5A

This approach is the same as described for Approach 5. Initial dewatering would be to 43 m AHD; the level of the extent of the relatively intact portion of the former haul road.

Backfilling via floating conveyor would be done to a nominal level of 40 m AHD, which would leave a 3 m layer of water over the backfilled waste rock for the equipment to operate. This level could be optimized subject to the draft requirements of the floating equipment.

The remainder of the Main Pit (from 40 m AHD to surface) would be backfilled and compacted using standard earthmoving equipment.

5.1.2 Approach 5B

The project team expressed an interest in accelerating tailings consolidation to: (i) maximize waste rock backfill tonnage, and (ii) achieve the majority of settlement during backfilling to reduce long-term settlement of the final landform and cover. To this end, this option includes installation of wick drains in the tailings to accelerate consolidation.

Initial pit dewatering would also be to 43 m AHD; however, this approach uses a newly constructed haul road at 10 % grade to reduce the push-back excavation and improve waste rock hauling efficiency.

Initial backfilling would be via floating conveyor from a trafficable surface to a nominal level of 22 m AHD (i.e. to 5 m above the lowest tailings surface level). To prevent generating contact water during this initial tailings access layer, NAF material could be used for backfill. Regardless, for ease of wick drain pre-drilling and to allow free drainage from the wick drains into this layer, the material would need to be predominantly coarse-grained (i.e. sand and gravel) with no oversize.
Processing/screening waste rock could be an option if no other suitable material could be economically sourced.

Following initial backfilling and secondary dewatering to the top of the tailings surface, push-back and haul road construction would be continued at 10% grade to the backfill surface (22 m AHD). Predrilling through the backfill layer and wick drain installation would then be done from this surface. Additional wick drain details are provided in Section 5.1.2.1.

After installing a drainage blanket, backfilling with waste rock using conventional earthworks would commence. The backfill lift rate (loading rate) would need to be designed to optimize tailings consolidation. To this end, consolidation of the tailings during backfilling would have to be monitored.

5.1.2.1 Tailings Settlement

When saturated tailings are loaded, the load is initially carried by the water in the pores (Terzaghi, 1943). This results in a rapid increase in excess porewater pressure. Then there is a period of consolidation as these pressures dissipate.

The rate of consolidation of fine grained materials is a function of hydraulic conductivity (permeability) which is determined by the grain size of the material; the finer the material, the longer it will take for excess porewater pressure to dissipate. Very low hydraulic conductivity materials can take decades to consolidate.

Wels et. al. (2000) showed that increasing the surcharge load three-fold only increased the total magnitude of settlement by about 20% and had a negligible increase on the rate of consolidation. The only practicable and efficient way to accelerate the consolidation of tailings is to reduce the length of the drainage paths by using wick drains. They work by providing a shorter distance for the water to travel to exit the tailings.

Wels et. al. (2000) considered using fully-penetrating wick drains at 4 m spacing in very fine, clay-rich uranium tailings. The modeling predicted that 85% of the total settlement occurred within one year; this represented a decrease in the total time of consolidation by at least an order of magnitude. The actual wick drain design (spacing and depth) would have to be based on the specific material properties of the Rum Jungle tailings.

See Appendix B for details of consolidation principles.
5.1.2.2 Implications for Rum Jungle

With consideration to the above, general implications on the settlement and possible tailings response used in the analysis for Approach 5B are summarized below:

- The initial nominal 5 m backfill layer could generate substantial settlement.
- The relationship between load and settlement is not linearly proportional; increasing the cover load (by placing a thicker layer and/or dewatering) would need to be weighed against the negatives of increased loose backfill and contaminated water to be treated.
- Differential settlement is expected over the footprint, with the largest settlement being near the centre corresponding to the greatest depth of tailings.
- For fine-grained tailings, wick drains could significantly reduce the time for settlement.
- Final settlement is anticipated to be in the range of 5 % to 20 % of the tailings thickness (based on consolidation modeling completed by RGC for the Wismut uranium tailings and elsewhere); 10 % was thought to be appropriate and was adopted for this study.

It is important to note that the actual response will depend on the tailings properties (as demonstrated in the IAA Helmsdorf study) which could vary significantly both laterally and vertically. To this end it is important to characterize the properties of the tailings in Main Pit for; wick drain design and construction planning, estimation of the shear strength that governs the rates of construction, and prediction of total settlement.

5.2 Quantitative Criteria Analysis

To quantify the key components (Waste Rock, Water Quality and Time) for the two alternatives, preliminary schedules and material balances were developed. Whist key information has been provided for the reader to have a general understanding of the inputs to the comparison graphs, there are many additional variables/unknowns not presented and many variable interdependencies (some of which are not known at this time) that would affect the quantities. As such, the comparisons should be viewed in a general sense for the purpose of guiding selection only.

5.2.1 Inputs

Site-specific water and material input values were used to calculate the quantities. The key inputs are presented in the relevant sections below, and the supporting calculations are presented in the calculation package in Appendix C. Where site-specific data were not available, reasonable assumptions were made.
5.2.2 Waste Rock

The total PAF waste rock backfill tonnage depends on the following main variables:

- Waste rock properties, primarily the density for different backfill stages.
- Pushback geometry, which determines the spoil volume and total void space.
- Tailings and waste rock settlement before the backfill reaches 58.5 m AHD.
- Method of backfill, which dictates the material density.

Taking into account these and other variables, the approach schedules showing backfill tonnage over time, are shown in Figure 5-1.

Figure 5-1 - Waste rock backfill tonnage (dark grey zones are wet seasons)

An additional approximately 24,000 tonnes of PAF waste rock can be backfilled using Approach 5B instead of 5A, which represents an increase in total backfill capacity of about 6%. This is predominantly due to the larger proportion of backfill by conventional earthworks (higher density due to compaction), and to a lesser extent the additional pit volume available from settlement of the tailings.
5.2.3 Water Quality

Dewatering volumes and rates of the impacted and non-impacted pit water were based on the following main variables:

- Average groundwater inflows for the wet and dry seasons (from RGC modeling).
- Incident precipitation and evaporation outflow for the wet and dry seasons, accounting for the variable pit lake surface area in the case of evaporation.
- Volume of water in the pit reservoirs above the level of dewatering, which depends on the approach taken.
- Rates of dewatering and backfilling (water displacement), which impacts on the time dependent precipitation and evaporation variables.
- Waste rock void space, which determines the volume of water stored in the backfill and is different depending on the approach.
- Tailings settlement seepage water which depends on the amount of settlement.

Volumes of unimpacted and impacted water pumped from the pit were calculated for each option (see Figure 5-2). Unimpacted water is that pumped prior to the push-back and backfill (initial dewatering). Anything after this is considered to be impacted and assumed to require treatment. Although the spoil dumped into the pit lake from the push-back may not be contaminated, the fragile chemocline currently existing at depth in the pit lake will be disturbed likely causing highly contaminated pit water at depth to mix with the unimpacted water above.
Because both Approach 5A and 5B are initially dewatered to the same level, the total volume of unimpacted water is the same (about 1.65GL).

However, Approach 5B is estimated to generate around 570 ML (33%) more impacted water than Approach 5A, principally because the backfilling takes longer and hence more groundwater enters the pit. Other, less significant, reasons are: (i) compaction of a greater proportion of waste rock in Approach 5B reduces void space and hence increases water to be treated in 5B; and (ii) Approach 5B releases more contaminated tailings pore water during construction (due to tailings settlement) requiring treatment.

The rates of dewatering are presented in Figure 5-3. From a technical perspective, pumping rates could be adjusted widely depending on the number of pumps. The following key constraints/assumptions need to be kept in mind:

- Discharge to the creek is permitted only in the wet season, hence initial dewatering was assumed only over this time.
- An allowance of two months was assumed following initial dewatering for pit wall depressurization, but it could actually take considerably longer.
- Backfilling will not be done in the wet season, but it was assumed that dewatering would continue at the net inflow rate.
• The pool for floating conveyor dumping was assumed to be maintained at a constant level, requiring that the dewatering rate was set to match the net inflow and displaced volume rate.

• The displaced volume during overwater dumping is dependent on the conveyor backfill rate. For this assessment it was set at the conveyor material movement rate; however, this could be modified for example to suit the treatment plant constraints (it is understood that water treatment plants are designed based on the peak inflow rates and lower rates correspond to a more cost efficient solution).

• During conventional earthworks the surface must remain dry to support vehicle movement and as such the dewatering rate was set to match the net inflow rate.

• Dewatering was taken to be complete once the PAF material backfill had reached the final level of 58.5 m AHD.

![Dewatering Rates](image)

**Figure 5-3 - Dewatering rates**

Similarly to the dewatering volume, the initial dewatering rate of unimpacted water is the same for Approach 5A and 5B (about 160 L/s). There is also no significant difference in the peak dewatering rate for impacted water (about 65 L/s). Pumping for Approach 5B continues for a year longer than 5A. As noted above, the dewatering rates are dependent on many variables which need to be considered in unison to optimize the solution.
5.2.4 Time

As for any bulk earthworks project, the schedule is critically dependent on the rate at which material is moved. Ultimately this will be decided by project economics and by the contractor responsible. Some of the main variables in calculating the schedules are:

- Method of backfill; as conveyor and conventional earthworks have different rates.
- Source location; which determines the haul distance.
- Achievable conveying rates, based on material properties and conveyor performance.
- Hauling logistics, such as size and number of trucks, transfer times and vehicle speed.

For conventional earthworks, values were selected based on RGC’s general earthworks experience. For the conveyor and wick drains, specialist contractors were consulted to establish reliable rates.

Note that the schedule in Figure 5-4 is for the entire backfill project, including the pit volume above 58.5 m AHD and the final landform. Clean fill for the landform is assumed to be clayey material which is planned to be sourced from the Woodcutters Mine which is located about 12 km from Main Pit.

![Backfill Schedule](image-url)

**Figure 5-4 - Backfill schedule**
Approach 5B is expected to take 12 months (21 %) longer than 5A. This is due to the additional work associated with pit push-back and installation of wick drains; a full construction season was assigned for these tasks.

5.3 **Refined Approaches Assessment**

An options comparison of Approach 5A and Approach 5B was done as the basis for selection. As per the screening level options assessment, the SMART technique (described in Section 4) was used with the same criteria and weighting (Table 4-1).

5.3.1 **Scoring**

The same category scores (Table 4-2) were used to score the approaches against the project criteria. The scores given to each alternative are presented in Table 5-1.

<table>
<thead>
<tr>
<th>Criteria</th>
<th>Approach</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>5A</td>
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<tr>
<td>Health and Safety</td>
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</tr>
<tr>
<td>Technical Feasibility</td>
<td>0.75</td>
</tr>
<tr>
<td>Time</td>
<td>0.75</td>
</tr>
<tr>
<td>Cost</td>
<td>0.75</td>
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<tr>
<td>Waste Rock</td>
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</tr>
<tr>
<td>Settlement</td>
<td>0.50</td>
</tr>
<tr>
<td>Liming</td>
<td>0.75</td>
</tr>
<tr>
<td>Investigations</td>
<td>0.75</td>
</tr>
</tbody>
</table>

5.3.2 **Results**

Results of the assessment are shown graphically in Figure 5-5 and Figure 5-6 and the weighted component scores for each criterion as well as relative ranking are detailed in Table 5-2.
Figure 5-5 - SMART Analysis, Approach 5A and 5B component scores

Figure 5-6 - SMART Analysis, Approach 5A and 5B component and total scores
<table>
<thead>
<tr>
<th>Approach</th>
<th>5A</th>
<th>5B</th>
</tr>
</thead>
<tbody>
<tr>
<td>Health and Safety</td>
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</tr>
<tr>
<td>Technical Feasibility</td>
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<td>0.09</td>
</tr>
<tr>
<td>Time</td>
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<td>0.04</td>
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<td>Water Quality</td>
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<td>Waste Rock</td>
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<td>0.09</td>
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<tr>
<td>Settlement</td>
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<td>0.09</td>
</tr>
<tr>
<td>Liming</td>
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</tr>
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<td>Investigations</td>
<td>0.01</td>
<td>0.00</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>0.66</strong></td>
<td><strong>0.54</strong></td>
</tr>
<tr>
<td><strong>Rank</strong></td>
<td>1</td>
<td>2</td>
</tr>
</tbody>
</table>

Approach 5A scored higher than 5B in total and for all criteria with the exception of ‘Water Quality’, ‘Waste Rock’ and ‘Settlement’. This is a logical result given that these were the criteria for which Approach 5B was developed.

### 5.3.3 Sensitivity Analysis

Similarly to the screening level assessment, sensitivity analysis of the total approach scores was conducted by varying the weightings of key criteria to values of 0, 25, 50, 75 and 100. Graphs of the resultant total score are shown in Figure 5-7.
Figure 5-7 – Refined level assessment - weighting sensitivity analysis for key criteria
Observations from the sensitivity study are:

- Approach 5A is highly sensitive to ‘Water Quality’ and less sensitive to the other criteria.
- Approach 5B is more sensitive than Approach 5A.
- Approach 5B is highly sensitive to all criteria with the exception of ‘Water Quality’.

Approach 5A scored the highest for all criteria regardless of the weighting value. This occurs even when the weighting is zero because it scores high enough in the other criteria to maintain the highest overall score. This indicates that Approach 5A is robust as the most promising backfilling approach.

5.3.4 Discussion and Limitations

The comments provided in Section 4.6 are also relevant to this assessment; the assessment enables relative comparisons of the criteria whereby the actual score numbers are semi-quantitative at best due to subjective weighting of the criteria.

On the balance of all the criteria, Approach 5A is preferable. The specific benefits of Approach 5B, namely faster settlement, less contact water and a relatively small increase in available PAF material storage, need to be critically weighed up against the ‘downsides’ such as higher risk profiles for health and safety, lower chance of project success and significantly higher cost.

Specifically relating to settlement, it is important to mention that in the long-term the total magnitude of tailings settlement will eventually be the same, however using Approach 5A a larger proportion will have been completed after the backfilling has finished.
6 CONCLUSIONS: PREFERRED OPTION

6.1 CONCLUSIONS

The objective of this study was to develop and evaluate alternative methods for backfilling the Main Pit. During an initial brainstorming session, five candidates (Approach 1 to 5) were developed.

The five candidates were designed to a conceptual level and then compared using qualified assessment of the key project criteria. A combination of initial placement of waste rock from a floating conveyor followed by ‘dry’ placement using conventional earthmoving equipment (Approach 5) scored the highest and was selected for further assessment.

Subsequently, two sub-options were developed to a more detailed conceptual level:

- The original design using only partial dewatering and no wick drain installations (Approach 5A), and
- Full dewatering and use of wick drains to increase the PAF waste rock backfill and accelerate the tailings consolidation (Approach 5B).

For these two sub-options preliminary construction schedules and material balances were developed and selected scoring criteria (waste rock, water quality and time) were estimated. A semi-quantitative comparison of these approaches was done. On the basis of this comparison, RGC recommends adopting Approach 5A as the preferred advanced conceptual design.

6.2 POTENTIAL RISKS OF PREFERRED OPTION

The risks discussed below are not necessarily unique to the preferred approach. They share common risks to some extent, depending on the interaction with the geo-hazards listed in Section 2.7. The discussion pertains to the highest risks of the proposed approach. Others, for example conveyor mechanics, would need to be assessed by others with appropriate relevant knowledge.

6.2.1 Pit Wall Failure

Utilizing an overwater dumping method at a higher level in the pit rather than working closer to the tailings surface significantly reduces the risks to personnel and equipment. However it should be noted that any approach that incorporates dewatering and exposes pit walls further has the potential to introduce instability hazards.

The pit wall materials are susceptible to swelling and softening when saturated. Site observations near the pit crest indicate that this has produced a ‘skin’ of very low strength rock with high clay content. If this is indicative of the rest of the pit walls, this layer could be a partial groundwater barrier impeding the dissipation of excess porewater pressures (following rapid drawdown).
6.2.2 Cavities

The Coomalie Dolostone unit is susceptible to dissolution. This natural phenomenon may have been accelerated due to the low-pH pit water, leading to undercutting/channeling. If this unit, predominantly surficial in the south of the pit, is surcharge loaded at the crest it could lead to sudden cover-collapse and sink-hole formation. This phenomenon may also be important from a groundwater and dewatering perspective, potentially a conduit for concentrated (and potentially high) pit water inflow.

6.2.3 Pit Wall Pressurization

Drawing down the pit lake faster than the surrounding groundwater table would generate excess porewater pressures and destabilizing forces on the walls. Failure of the pit walls could set up displacement waves on the lake during overwater dumping, or mass material movement into the pit during the dry backfill, endangering personnel and equipment. The consequence depends on the nature and extent of the failure, but it is conceivable that if it were to occur during dry backfilling (when people will be in the pit) it could cause a fatality. The risk level associated with this geo-hazard could be reduced with appropriate investigation, design and construction.

6.2.4 Tailings Failure

Loading the tailings too much or too quickly during overwater dumping could cause rapid failure through the tailings. This would cause displacement waves in the pit lake and endanger personnel and machinery. The critical time for this is during the overwater backfilling when works will be largely remotely done by machinery; as such the consequence of this would be primarily limited to financial loss (due to loss or damage to equipment). The consequence would clearly depend on the scale of the occurrence.

A risk assessment workshop involving all the project stakeholders would be a good way to identify all of the project risks.

6.2.5 Settlement

Settlement of the final landform is likely to occur over time after backfilling is complete. The extent and spatial variability of the settlement could be reliably predicted prior to backfilling using the data gathered from the site investigations (Section 7).

Regardless, the project objective of a stable long-term landform could still be achieved, with relatively minor post-construction earthworks on occasion to fill in areas that have settled.
7 RECOMMENDATIONS

7.1 SITE INVESTIGATIONS

RGC recommends investigation of the pit walls and in-pit backfill materials before advancing the design. The purpose would be to characterize the geotechnical, hydrogeological and geochemical properties. The possible scope of investigations is provided below as ‘Stage 1’ and ‘Stage 2’ programs.

7.1.1 Stage 1

Stage 1 investigations would achieve a general understanding of the geotechnical, hydrogeological and geochemical conditions of the pit walls and in-situ backfilled materials. Results would support detailed decision-making on the backfilling approach. It would provide the information to design the dewatering program and pit push-back to reinstate the haul road.

It should be recognized that investigating pit wall slumping and sloughing can only be covered in a general sense using a borehole investigation program. Such failures are highly dependent on very local conditions on benches and slopes and are not amenable to accurate determination from spatially-limited and isolated information from boreholes. Rather, a more holistic approach would be needed, such as a trial pit dewatering with slope observation/monitoring (see Section 7.1.3).

7.1.1.1 Site Investigations

A possible scope for Stage 1 site investigations for the pit walls, in-situ backfill and waste rock materials should any of the options presented in this report be adopted could include:

**Pit Walls**

- Site walkover to map the exposed pit walls and geometry.
- Oriented diamond core drilling to below the tailings surface level at about five locations around the pit.
- Geotechnical/geological core logging and photographing.
- Sample collection for subsequent laboratory testing including; unconfined compressive strength (UCS), direct shear test, point load strength, slake durability and moisture content.
- Installation and development of monitoring wells/vibrating wire piezometers in completed boreholes.
- Hydrogeological slug and pumping tests.
- Groundwater sampling for subsequent laboratory geochemical testing.
- Overwater CPT probing of the pit walls, including sampling if possible.
Borehole locations would be selected based on the site walkover information and the concept design to best meet the objectives and geological/spatial coverage. Critical to this approach is the pit push-back design; this area may require special attention. It may be warranted to incline boreholes to optimize interception of geological structures, the inclination of which can be determined from the initial site walkover.

CPTs on the walls would be done as part of the overwater in-situ backfill investigations (see below) with the objective of assessing the degree of softening of the surfaces. Depth of investigation will be limited by the possible down-force; dependent on the barge size and rig capability. Testing should be conducted around the pit in all of the geological units, with additional focus on the former pit haul road/push-back extent. A total of about 20 CPT locations should provide coverage.

Hydrogeological testing would also utilize existing groundwater wells in the area to help in characterizing the broader groundwater response.

**In-situ Backfill**

Overwater in-situ backfill investigations:

- Piezocone with porewater pressure dissipation CPTù to practical refusal at about 10 locations over the pit bottom.
- Porewater dissipation testing at regular depth intervals (say 5 m).
- Mud rotary boreholes adjacent to each location for shear vane testing and ‘undisturbed’ thin-walled tube/piston sampling at 1 m depth intervals.
- Field pocket penetrometer and Torvane testing on the recovered samples.
- Geotechnical laboratory testing for classification tests such as; particle size distribution, Atterberg limits, hydrometer, consolidometer, moisture content, dry density and specific gravity.
- Geochemical laboratory testing of porewater quality.

Further specialist geotechnical testing (e.g. rheology, finite-strain consolidation tests) could also be scheduled depending on the design data requirements.

Data would be obtained by overwater probing using a rig mounted on a barge. The barge would need to be secured with guy wires to maintain position and so that down force could be applied. Note that a floating barge that is only stayed by cables is a poor platform from which to do geotechnical investigations. Vertical thrust on the CPT rods is limited, thus limiting the depth to which probing can be done, and unsteady support conditions influence the results of the in-situ porewater pressure dissipation tests. The existing pit lake water depth is too great to use a spud supported barge until dewatering is done to a shallow height above the tailings (see Stage 2). Consequently,
porewater dissipation test results should be critically reviewed early in the program to judge if platform movement is affecting the reliability of the data.

**Backfill Materials**

It is understood that backfill materials and their backfilling order will be selected based on their geochemical properties and AMD considerations. Their geotechnical properties also need to be included in this selection process pertaining to backfill material screening and earthworks design. Previous studies of the site waste materials would be a valuable resource for these design components.

SRK (2012) is a geochemical characterisation program of the materials in the Main, Intermediate and Dyson’s WRDs and Dyson’s (backfilled) Pit (SRK, 2012). In 2014, RGC conducted additional geochemical characterization of the WRD materials also including Particle Size Distribution (PSD) tests and compaction trials. Missing from these programs, relevant to the backfilling design, is detailed geotechnical information such as field and laboratory strength and compaction data.

If additional characterization of the WRD materials is needed for the design, the following field tasks could be undertaken:

- Large test pits/costean excavations.
- Geotechnical logging, sampling and photographing.
- Laboratory testing including; particle size distribution, point load strength, slake durability, moisture content, maximum dry density and acid/base accounting.

**Analysis/Design**

With the data above, the following analysis could be done to predict material behavior:

- Develop a pit model incorporating geological, geotechnical and hydrogeological properties.
- Hydrogeological pit dewatering analysis.
- Slope stability analysis of structural and mass stability of the pit walls.
- Liquefaction assessment for platform stability and backfill/loading rates.
- Settlement modeling, incorporating the in-situ backfill and waste rock materials.

Note that RGC has already developed a site-wide hydrogeological model for groundwater flow and contaminant transport. Additional data would be used to update the calibration of the existing site-wide model and develop a refined pit-scale groundwater flow model.

Using results of the analysis, the following general design elements could proceed:

- Dewatering and pumping plan.
- Limnology modeling of chemocline disturbance/contaminant dispersion.
- Water treatment facility design.
- Push-back, wall stabilisation and road design.
- Waste rock material processing/screening requirements.
- Conveyor establishment, configuration and operational details.
- Backfill construction staging plan, including backfill/loading rates.
- Backfill earthworks specification.
- Final landform design.

7.1.2 Stage 2

General information and considerations for the Stage 2 investigations will be based on information from the Stage 1 program. The purpose of the program would be to further, and more reliably, characterize the materials to enable improved predictions of consolidation and stability.

As previously mentioned, investigations over deep water are constrained by available equipment and the capabilities of such equipment. More effective investigations could be performed if the pit was substantially dewatered such that the depth of water was relatively shallow.

It is envisaged that the Stage 2 investigation program would include instrumented trial loading of the surface and additional investigations performed either from waste rock roads or shallow floating barge platforms (with spud legs for stability).

About 10 CPTu test locations in a grid array (north-south and east-west) to generate sections through the pit would be reasonable. The tests would be progressed into the tailings to practical refusal which will be dictated by the tailings strength and available down force. Porewater dissipation tests would be conducted at regular intervals to measure dissipation time ($t_{50}$) for settlement calculations. Mud rotary drilling adjacent to these locations would enable shear vane testing (at 1 m intervals) at a number of the locations to enable direct comparison with CPT values. Samples would be collected at intervals for geotechnical and geochemical laboratory testing.

7.1.3 Dewatering Trial

A dewatering trial (full or partial) before backfill would allow thorough assessment of the pit wall conditions and hydrogeological regime in proximity of the Main Pit. Slope monitoring of the pit walls during dewatering, as well as the ability to inspect pit slopes visually would add considerably to the understanding of the pit behavior, particularly the stability of slaked sedimentary rocks subject to the pit wall water pressure conditions that will develop during dewatering.

For maximum benefit from the study, groundwater monitoring bores and other instrumentation (e.g. level-loggers) would be installed prior to dewatering.
7.1.4 Earthworks Trials

The different methods and different materials will result in different backfill material densities. Dry dumped will have the lowest density, placed and compacted in lifts will have the highest density, and dumped over water is expected to be somewhere in-between. Field trials with the specific backfill materials and using these different methods could be conducted to support calculation of the total backfill tonnage. Earthworks optimisation could be investigated for example using different lift thickness and equipment sizes. A method-based earthworks approach would be suited to this project i.e. developing a prescribed methodology (lift thickness and number of passes). Earthworks trials would assist in developing the optimal methodology to meet the backfill objectives.

7.1.5 Optimisation of Backfill Approach

Due to the large number of variables and the inter-dependency of these variables, there is considerable scope to optimize the preferred approach for any, or all, of the project criteria. The optimal solution, on the balance of all criteria and stakeholder needs, may indeed be somewhere in-between e.g. overwater dumping at a level between 22 m AHD and 43 m AHD.

The recommended optimization approach is to collect site information (i.e. Stage 1 program) to eliminate the identified unknowns and better define the variables. With this information the approach could be better detailed such that the criteria could be quantified, possibly with input from earthworks contractors. Then by refining that approach, principally by changing the level of overwater dumping, the different refinements could be compared in a quantified sense using a multi-attribute rating technique.

Key to the optimisation is obtaining accurate site specific data of the pit, tailings and waste rock properties.

7.2 Construction Approach

For a problem of this complexity, even with thorough and detailed investigations there will be residual unknowns that will become apparent during construction. In this scenario the ‘Observational Method’ would be well suited. The observational method involves designing for the most likely conditions, having realistic contingencies worked out in advance, and modification of the program during construction.

The steps in the Observational Method from Peck (1969) (in italics) and examples of the possible role in the backfilling works are described below:

- Exploration sufficient to establish at least the general nature, pattern and properties of the deposits, but not necessarily in detail - undertake Stage 1 investigations and Stage 2 if necessary.
- **Assessment of the most probable conditions and the most unfavorable conceivable deviations from these conditions** - make a model/s of the pit geology, geochemistry, hydrology and hydrogeology.

- **Establishment of the design based on a working hypothesis of behavior under the most favorable conditions**

- **Selection of quantities to be observed as construction proceeds and calculation of their anticipated values on the basis of the working hypothesis** - monitoring items such as; porewater pressures, pit wall condition and stability, water quality and settlement.

- **Calculation of values of the same quantities under the most unfavorable conditions compatible with the available data**

- **Selection in advance of a course of action or modification of design for every foreseeable significant deviation of the observational findings from those predicted on the basis of the working hypothesis** - for example if the walls of the pit push-back prove to be unstable, a course of action could be to adopt a flatter angle, or use reinforcement.

- **Measurement of quantities to be observed and evaluation of actual conditions**

- **Modification of design to suit actual conditions**
8 CLOSURE

Robertson GeoConsultants Inc. is pleased to submit this report titled ‘Main Pit Backfilling Concept Approaches Study’ to the Northern Territory Department of Mines and Energy. This report was prepared for use by the DME and prior consent by the DME is to be given before the contents of this report are considered by any third party.

We trust that the information provided in this report meets your requirements at this time. Should you have any questions or if we can be of further assistance, please do not hesitate to contact the undersigned.

Respectfully Submitted,

[Signature]

J A Caldwell

Andy Thomas
Geological Engineer

Jack Caldwell
Civil Engineer
9 REFERENCES


Main Pit Bathymetry Survey
with Marked-up Interpretations

Rum Jungle Mine Site
Scale 1:2,500

Legend
Inferred Fill Materials
- Soil
- Tailings
- Waste Rock

Lithology
- Quartzite Breccia (Geolsec Formation)
- Mudstone Sequence (Golden Dyke Formation)
- Black Slate Sequence (Golden Dyke Formation)
- Coomalie Dolostone

Symbology
- Pit Wall Angle
- Gully - Indicative of Localized Slumping
- Geological Unit Boundary
- Feature Boundary
- Material Movement
- Scree / Backfill Cone
- Inferred Batter Slopes

Notes:
- Bathymetry data from C.D.U. survey - supplied by the D.M.E. March 2015
- Geology from Berkman (1968)
- 1m contour interval
- Profile shown below pit lake level only
- Pit lake level approximately 59m AHD at the time of survey
Surface Geology
Rum Jungle Mine Site
Scale: 1:15,000

Groundwater Level Contours

Legend:
- Existing Groundwater Bore
- Screen Depth < 5m
- Screen Depth 5 - 15m
- Screen Depth > 15m
- 2014 Monitoring Bore
- Static groundwater level on 24-Nov-14 (m AHD)

Drainage
Road
Fault
Confirmed
Indefinite
Inferred Groundwater Flow Direction

GG

Section Line

Figure: 2-1
Project No: 18300
6/4

Original File: RJ_Litho_AllAreas_21Sep2015.mxd

Client: Lithology

Coomalie Dolostone
Crater Formation
Geolsec Formation
Quartz Vein
Rum Jungle Complex
White's Formation

Report: RGC 1830064
Last Update: Sep 21, 2015

Drawn: L.R.
RUM JUNGLE MINE SITE
OPEN PIT BACKFILLING EVALUATION

Submitted to:
Northern Territory Government
Department of Resources - Minerals and Energy

Prepared by:
Robertson GeoConsultants Inc.
Consulting Engineers and Scientists for the Mining Industry
www.robertsongeoconsultants.com

May 2015
Executive Summary

At the request of the Northern Territory (NT) Department of Mines and Energy (DME), Robertson GeoConsultants (RGC) evaluated alternative approaches to backfilling the Main Open Pit at the Rum Jungle Mine Site with waste rock.

This document describes and assesses risks for the following alternative pit backfilling approaches:

- **Approach 1** – De-water the pit; construct an access road to the level of the tailings backfill in the pit; spread the waste rock in successive layers over the tailings.
- **Approach 2** – Dump waste rock from the perimeter of the pit into the pit lake; advance the crest of the dump out into the pit as filling proceeds.
- **Approach 3** – Procure a barge; construct a loading platform; place the rock into the barge, and dump the rock from the barge evenly around the pit.

A risk assessment establishes that both Approaches 1 and 2 are fraught with risks to workers and the environment. Specifically with regard to Approach 1, de-watering the pit will require water treatment prior to discharge; an access road into the pit will not necessarily be cheap or safe; the surface of the saturated tailings may not support equipment or the increasing thickness of waste rock; and continued control of groundwater inflow during backfilling will be difficult.

With regard to Approach 2, the primary risk is that the tailings may not provide a suitable foundation for the advancing face of dumped waste rock. Slope failures of the advancing waste rock would imperil equipment and workers in delivery trucks.

Additional characterization of the Main Open Pit and waste rock would be required to refine the risk estimates provided here and to formulate mitigation measures. Because such work will be costly, RGC does not recommend this.

By comparison, Approach 3, involves conventional barge operation - something not done before at the mine, but common in ports and harbours in Australia. Loading, transporting, and dumping rock into a hopper to load the barge are conventional operations; the risks of which are readily mitigated and managed.

Accordingly RGC recommends detailed evaluation of Approach 3 involving loading barges with waste rock and dumping it from the barge evenly around the pit. Further evaluation would involve compiling a cost estimate and preparation of an operations plan and schedule.
REPORT NO. 183005/2

OPEN PIT BACKFILLING EVALUATION

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OPEN PIT BACKFILLING EVALUATION

1 INTRODUCTION

1.1 GENERAL

This report describes an evaluation of alternative approaches to filling the Main Open Pit at the former Rum Jungle Mine Site. It was completed by Robertson GeoConsultants Inc. (RGC) for the Northern Territory (NT) Department of Mines and Energy (DME).

The DME intends to backfill the Main Open Pit with waste rock from the adjacent Main and Intermediate Waste Rock Dumps (WRDs) as part of site rehabilitation. RGC was asked to provide advice on a strategy for backfilling the pit. Accordingly, in this report, we describe alternative backfilling approaches, their relative potential risks, and key considerations in proceeding with planning pit backfilling.

This report was prepared by Andrew Thomas and reviewed by Jack Caldwell, both of RGC.

1.2 SCOPE AND OBJECTIVES

The scope of the work is as follows:

- Review documentation on the pit history, geology and configuration.
- RGC personnel discuss conceptual backfill approaches.
- Compile a concept-level risk assessment.
- Recommend a preferred approach.
- Discuss closure design issues.

The focus of the study is on possible backfill construction methods. Consideration and comparison is primarily from a construction risk perspective. The scope does not include a detailed discussion of the associated water management or environmental concerns. The objective of this letter is to provide the DME with a conceptual understanding and comparison of possible concept backfill approaches and RGC’s recommendation of the preferred method.
2 PROJECT BACKGROUND

2.1 THE MINE

The former Rum Jungle Mine Site is 105 km (by road) south of Darwin, NT. It became Australia’s first uranium mine after the deposit was discovered in 1949. Underground mining commenced in 1952 followed in 1954 by development of Main Open Pit (formerly known as White’s Open Pit). The ore treatment plant was built in 1954. The plant was directly north of the pit. Mining of the main pit continued until 1958 and elsewhere on the site until 1969 (Verhoeven, 1988).

Mining and processing operations had ceased by 1971. The site was initially rehabilitated in 1984/1985 and the DME is currently developed final rehabilitation strategy that is consistent with current standards for mining.

2.2 PIT GEOLOGY AND CONFIGURATION

2.2.1 Documents Reviewed

Key documents reviewed in the study are:


In addition, the following resources were consulted:

- LiDAR survey – supplied by DME, email Rider/Ferguson, 22 April 2015.

2.2.2 Geology

The regional and site geology is described in a number of documents, such as Berkman (1968). In brief and as relevant to this study, the geology of the pit is as follows.

The Main ore body was on the contact between the Coomalie Dolomite and the Golden Dyke Formation. The ore was hosted in the carbonaceous pyritic slate member of the Golden Dyke Formation which is a basal mudstone sequence comprising mudstone, schist, and black slate. An intensely sheared zone trending approximately east-west, bisects the pit; this is called the ‘main shear zone’. A north-south fault and east-west fault associated with tectonic shattering truncated the ore body at depth.
Interpretation of zones of the main geological units and structural features are shown on the marked-up bathymetry survey plan of Attachment 1.

### 2.3 **Final Pit Configuration**

Based on review of historical photos and documentation, key details of the final pit configuration (prior to any backfill) and associated interpretation is as follows:

- The surface footprint was roughly semi-circular in the southern half and slightly lenticular in the northern half.
- The maximum surface footprint was about 340 m wide (east-west) and 350 m long (north-south).
- A haul road spiraled down clockwise, with the entrance aligned with the southern extent of the pit (azimuth 160°).
- At approximately three-quarters of the total depth in the south-west of the pit the haul road switched-back and continued to the base in an anti-clockwise down direction. This was done presumably to avoid passing through the weak main shear zone on the eastern wall.
- Maximum depth is reported to have been 105 m below crest ground surface level.
- Inter-ramp bench stacks generally comprised four batters over the first spiral and reduced by one batter each spiral thereafter.
- Bench and batters were poorly defined over many sections of the pit, presumably due to sliding filling in the benches. Slope deterioration was occurring even during the time of mine operation. These zones appear to be due to mass wasting rather than along discrete geological structure. Areas particularly affected are those coincident with the zone of Coomalie Dolostone. Such zones are more prevalent in isolated zones of the Mudstone than in the Black Slate Sequence.

### 2.4 **Current Pit Details**

Based on the bathymetry and LiDAR survey, interpretation of the current pit configuration is provided herein with corresponding reference locations, where applicable, marked on Figure 2-1.
Figure 2-1 - Rum Jungle main pit marked-up bathymetry survey
2.5 Pit Configuration

Information on the current pit configuration is as follows:

- The surface footprint dimensions and shape have not noticeably changed since cessation of mining.
- Parts of the main haul road are distinguishable except where backfill or scree slopes have filled in the road and adjacent.
- The floor is approximately 50 m below the crest ground surface level.
- Generally the former benches are indistinguishable which is likely due to material sliding and forming a relatively uniform wall profile.
- The inter-ramp angle generally ranges between 35° and 43°.
- The overall slope angle (crest to floor) generally ranges between 25° and 30° in the Mudstone Sequence, and between 28° and 38° in the Black Slate and Coomalie Dolostone Sequences.
- Pit lake surface at the time of the bathymetry survey was approximately 9 m below the pit crest in the north and about 3 m in the south.

2.6 Tailings Backfill

There is a distinct zone of uniform and shallower slope angle (15°) over an approximately 80 m length of the crest in the north of the pit (azimuth 360° to 020°) [Fill Zone 1 - Reference 1]. The slope of this zone transitions smoothly to the pit floor; this indicates that the material is fill. Such deposition shape is consistent with the ‘flowing’ of fine-grained materials such as tailings.

An extract from Department of Transport and Works, Volume 2 (1981) provides supporting evidence that this material is tailings, deposited during mine operation:

“Tailings material was disposed of at the water surface of the open cut from 1965 to 1971…”

This document states that 700,000 tonnes of tailings material were dumped in the pit. Calculations indicate that this amount could account for the current level of tailings in the pit.

2.7 Waste Rock Backfill

There is a less distinct zone of relatively uniform slope (overall 38°) in the east of the pit (from azimuth 080° to 100°). This is indicative of backfill [Fill Zone 2 - Reference 2]. This zone is approximately 90 m wide at the head and 50 m wide at the toe adjoining the tailings surface. The surface is somewhat undulating; this suggests that it is coarse, granular material.

A platform approximately 10 m below the pit crest is consistent with the material having been dumped in stockpiles and pushed over the edge with a dozer. The platform height was possibly near that of
the pit lake surface at that time. The contact with the pit floor is defined by the tailings; this indicates that rock dumping precedes or is coincident with tailings deposition.

2.8 SHEAR ZONE SLIDE

There is a scree cone (azimuth 080°) on top of the tailings surface. It is coincident with the approximate location and width of the main shear zone that is reported to traverse generally east-west. The slide scar is about 20 m wide with the head of the slide in the order of 5 m to 10 m below the pit crest level.

The scree cone extends over about 50 m diameter on the pit floor. It is possible that this comprises fractured black slate of the Golden Dyke Formation associated with the main shear zone.

The presence of the cone on top of the tailings surface indicates that the slide occurred after tailings deposition.

2.9 SUMMARY

The following points summarise the interpretation and key findings from the desktop study:

- It may be possible to use the remnant pit haul road for access to azimuth 360° approximately 20 m below crest level.
- The walls have deteriorated and mass sliding has occurred, particularly throughout the Mudstone and Coomalie Dolostone Sequences. Material softening from the continued saturated conditions is highly likely.
- The Coomalie Dolostone unit is vulnerable to solution channeling when exposed to acidic conditions, such as that of the pit lake water.
- Material in the northern fill zone and extending into the former bottom of the pit is tailings that were placed during mine operation up until 1971.
- Tailings in the bottom are saturated and are probably normally consolidated.
- The eastern fill zone contains coarse-grained material, thought to be waste rock, which extends below the depth of the tailings surface.
- The main shear zone is weak and unstable which is evidenced by sliding failure on the eastern side of the pit.
- Backfill and scree materials are in a loose, meta-stable state.
3 ALTERNATIVE BACKFILL APPROACHES

3.1 GENERAL

Senior RGC personnel brainstormed alternative pit backfill approaches. The alternative approaches are discussed below. A figure illustrates each.

The background pit cross-section drawing used was modified from that in Figure 2 of Berkman (1968) in which the section line is aligned north north-west to south south-west, looking east north-east. This configuration is idealized and representative of the pit circa 1968.

3.2 REQUIREMENTS

Information relevant to evaluation of alternative pit backfilling methods is in Table 3-1.

<table>
<thead>
<tr>
<th>Detail</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Backfilling should be done over no more than three construction seasons.</td>
<td></td>
</tr>
<tr>
<td>Potentially acid forming (PAF) rock used in pit backfilling should be below the elevation of the seasonally lowest local groundwater table.</td>
<td></td>
</tr>
<tr>
<td>Non-acid forming (NAF) rock should be used in backfilling above the PAF rock in the zone of fluctuating groundwater levels.</td>
<td></td>
</tr>
<tr>
<td>The backfilled pit will be covered with a suitable cover.</td>
<td></td>
</tr>
<tr>
<td>The local river may be routed to flow over the backfilled pit and its cover.</td>
<td></td>
</tr>
<tr>
<td>It may be necessary to amend or provide for amendment of longer-term waste quality form the waste in the pit beneath the permanent water table. Lime addition during rock placement and/or lime addition to pit waters during backfilling may be feasible.</td>
<td></td>
</tr>
</tbody>
</table>
3.3  **APPROACH 1 – DE-WATERING AND PLACEMENT**

This approach would involve:

1. De-watering the pit.
2. Cutting an access ramp down to the pit floor for equipment access or in part using the existing haul road.
3. Working from the bottom of the access ramp outwards, progressively constructing a trafficable platform on top of the tailings surface; it will probably be necessary to place geogrid or strong geotextile to support the advance lifts. It may also be necessary to install wick drains to speed up tailings strength increase.
4. Placing and compacting (where possible) waste rock in horizontal lifts using conventional earthworks plant.

![Figure 3-1 - Approach 1, De-watering and Placement](image-url)
3.4 **APPROACH 2 - CREST DUMPING**

This approach would involve:

1. Dumping waste rock from the crest around the pit perimeter.
2. Push waste rock by dozer over the advancing crest, and move progressively towards the pit centre.

![Figure 3-2 - Approach 2, Crest Dumping](image)

3.5 **APPROACH 3 - OVERWATER DUMPING WITH OPTIONAL DEWATERING AND PLACEMENT**

This approach would involve:

1. Constructing barge access to the pit lake surface and a waste rock loading area.
2. Dumping waste material from the barge over the full tailings surface area.
3. Once the rock approaches the water level, it may be possible to convert to earthworks plant that traverses the upper surface of the rock. This would be done to place the upper layers of NAF rock.

![Figure 3-3 – Approach 3, Overwater Dumping with Optional Dewatering and Placement](image)
4 ANALYSIS AND COMPARISON OF APPROACHES

4.1 RISK ASSESSMENT

The three alternatives are compared using a risk assessment approach. Appendix A is the Risk Register. Appendix B describes the terminology, hazard and consequence classification.

4.2 DISCUSSION

On the basis of the hazards and consequences for the three approaches as noted in Attachment 2, it is clear that Approach 3 involves the least risk, and the risks it does involve are most easily mitigated. The major risks associated with Approach 3 are the issues of locating, transporting, and safely using a barge. In practice, barges are available and widely used in Australia and with careful equipment selection and provision of suitable transport to site these risks can be addressed.

With this approach there is risk associated with possible rapid settlement of the waste rock (due to foundation or dumped rock failure) setting up waves on the pit lake surface. It is thought that this can be managed by dumping in a systematic way so that the pit floor level is evenly raised. A residual similar risk is present from possible mass sliding of material in the pit walls which is practically impossible to eliminate.

Both Approaches 1 and 2 are fraught with uncertainty, hazards and consequences that could halt implementation of the approach or result in severe danger to human health and equipment. Specifically with regard to Approach 1, the need to de-water the pit is problematic; access may not be possible or safe; the tailings surface may be inaccessible or require significant reinforcing; and the overall risk to workers and equipment is considered untenable.

Similarly with regard to Approach 2 the probability is very high that the tailings will not be able to function as a secure foundation and support the advancing face of the waste rock dump. Foundation failure would make implementation of the approach impossible or involve high danger to workers and equipment.
4.3 **Detailed-Level Comparison of Methods**

If the DME requires a detailed-level comparison of the approaches, depending upon the level of detail, some or all of the following data will be required:

- Site investigations, comprising:
  - Probing of tailings to obtain material and strength information using a barge-mounted drilling rig.
  - Mapping of exposed pit walls for slope stability analysis using a boat to access the walls.
  - Coupled geophysics and borehole investigation program to investigate the presence of solution channels in the Coomalie Dolostone.
  - Drilling program around the pit crest for access ramp and/or loading facility design and/or stockpile foundation conditions.

- Preliminary project design to develop the construction methodology and schedule.
- Environmental performance evaluation.
- Consultation with contractors and suppliers for project cost estimation.

The difficulty and expense of these investigations is likely to be high.
5 RECOMMENDATIONS

5.1 PREFERRED APPROACH

Based on the information at hand, Approach 3 appears to involve the lowest construction risk. Accordingly RGC is of the opinion that Approach 3 should be seriously considered for implementation.

If the DME are of the same opinion, the type of barge and detailed methodology of loading and dumping etc. should be investigated. The practical aspects of procuring and providing a barge should be researched. An estimate of the time it may take to fill the pit with a barge should be established. In addition the need, if any, to de-water the pit should be evaluated.

A study should be undertaken to define a suitable loading area platform location. Depending on the location and the barge loading methodology, ground investigation comprising site mapping and shallow intrusive methods (e.g. test pits) may be required.

If reliable estimates of the waste rock settlement and facility behavior are required for cover or landform design purposes, a study could be undertaken to investigate the tailings; however the need for this should be carefully weighed up against the likely high cost.

5.2 CONSIDERATIONS FOR CLOSURE DESIGN

Possible design issues relating to backfilling the pit so that they can be addressed in the closure design are noted in Table 3-1.
Table 6-1 – Pit Backfill Design Considerations

<table>
<thead>
<tr>
<th>Design Issue</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>Settlement</td>
<td>Due to consolidation of both the tailings material in the pit and the waste rock, the finished surface will settle. Cursory estimates of the magnitude are in the order of five to ten metres over time. Differential settlement could be expected to be about half this. This behavior would need to be incorporated into any cover design. Settlement will occur with all approaches however the magnitude will be reduced where the waste rock is placed in horizontal lifts.</td>
</tr>
<tr>
<td>Flooding</td>
<td>The pit is within the flood plain and as such will be inundated to some level in flood events. This water should either be prevented from entering the backfill materials or only be in contact with clean fill.</td>
</tr>
<tr>
<td>Contamination</td>
<td>Groundwater level fluctuation needs to be accounted for in the backfill design. Clean fill needs to be located in the zone of fluctuation so that oxidation and resultant groundwater contamination does not occur.</td>
</tr>
<tr>
<td>Groundwater</td>
<td>The waste rock is expected to be highly permeable and significantly more permeable than the surrounding bedrock. Therefore it can be expected that the pit would act as a significant groundwater sink regardless of the backfill approach taken.</td>
</tr>
<tr>
<td>Pit Lake Water</td>
<td>Any water removed from the lake (either pumped or displaced) will likely be contaminated (i.e. elevated TDS and TSS) and needs to be captured and treated. If the pit water level rises due to displacement, seepage into the surrounding groundwater is likely to occur. The acceptability of this situation needs to be evaluated to determine the possible requirement for dewatering.</td>
</tr>
</tbody>
</table>

5.3 Additional Evaluation

RGC recognizes that there are be issues related to Approach 3 that require further assessment, such as getting a barge to site, cost constraints, and the performance of a cover after settlement of the backfill over time. Also required would be assessments of:

- Water management strategies during backfilling, including the use of the Intermediate Open Pit for water storage.
- The degree of tailings disturbance by backfilling (and the condition of pit water with respect to TDS and TSS during the process).

Accordingly, it may be worthwhile to undertake a detailed cost-benefit study of the approaches. Such a study would have to be based on collection of additional site data as noted in Appendix 1, and a careful consideration of the cost of mitigating the risks, relative to the DME’s risk tolerance.
6 CLOSURE

Robertson GeoConsultants Inc. is pleased to submit this report. It was prepared by RGC for the use of the Northern Territory Department of Mines and Energy and prior consent by the Department should be given before the contents of this report are considered by any third party.

We trust that the information provided in this report meets your requirements at this time. Should you have any questions or if we can be of further assistance, please do not hesitate to contact the undersigned.

Respectfully submitted,

ROBERTSON GEOCONSULTANTS INC.

Prepared by:

Andrew Thomas
Senior Consultant, Geological Engineering CPEng

Reviewed by:

Jack Caldwell, P.Eng.    Dr. Paul Ferguson
Civil Engineer         Senior Geochemist
7 REFERENCES


APPENDIX A

Risk Register
<table>
<thead>
<tr>
<th>Approach</th>
<th>Subsystem</th>
<th>Element</th>
<th>Expected Performance</th>
<th>Hazard (Possible Failure Mode)</th>
<th>Probability Quantification</th>
<th>Consequences</th>
<th>Consequence Quantification</th>
<th>Risk Quantification</th>
<th>Mitigation and/or Response Measures</th>
</tr>
</thead>
<tbody>
<tr>
<td>Approach 1</td>
<td>Pit Lake Water</td>
<td>Contamination</td>
<td>Water effectively treated</td>
<td>Release of contaminated water to the environment</td>
<td>3</td>
<td>9</td>
<td>Well designed water treatment facility and/or water storage</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Contamination</td>
<td>Water effectively treated</td>
<td>Release of contaminated water to the environment</td>
<td>3</td>
<td>9</td>
<td>Well designed water treatment facility and/or water storage</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Access Road (Existing Haul Road or New Road)</td>
<td>Utility</td>
<td>Trafficable by conventional earthworks plant</td>
<td>Delayed access, impeded access, plant bogged</td>
<td>3</td>
<td>12</td>
<td>Regrade and construct trafficable surface, regular road maintenance</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Stability</td>
<td>Slopes supporting road are unstable</td>
<td>Release of contaminated sediments to the environment</td>
<td>3</td>
<td>9</td>
<td>Well designed access road, avoid shear zone and fill zones, slope stability assessment and support measures where required</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Pit Wall</td>
<td>Overall stability</td>
<td>Stable</td>
<td>Destroy access road, engulf plant and personnel, accidents</td>
<td>5</td>
<td>10</td>
<td>Stability assessment, flatten slopes, dewater perimeters to reduce porewater pressures, support slopes with shotcrete, rock bolting</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Localised sluffing</td>
<td>Minor sluffing</td>
<td>Block or damage access road, strike plant and personnel</td>
<td>4</td>
<td>12</td>
<td>Stability assessment, site-specific design, protective barriers, delineate exclusion areas such as shear zones and fill zones</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Earthquake response</td>
<td>No earthquakes during site works</td>
<td>Earthquake causes slope mass sliding, excessive sluffing</td>
<td>1</td>
<td>5</td>
<td>Early warning system, flatten slopes, dewater perimeter to reduce porewater pressures, support slopes with shotcrete, rock bolting, delineate exclusion areas such as shear zones</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Tailings</td>
<td>Surface strength</td>
<td>Accessible, some deformation</td>
<td>Inability to access and advance waste rock, plant bogging or sinking</td>
<td>4</td>
<td>16</td>
<td>Place geogrids, strong geotechnics, and/or install wick drains, establish earthworks methodology</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Settlement</td>
<td>Slow, predictable, controllable, uniform consolidation</td>
<td>Plant and personnel sinking, bogging or damage, loss of integrity of placed waste rock</td>
<td>4</td>
<td>12</td>
<td>Place geogrids, strong geotechnics, and/or install wick drains, establish rapid response strategy</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Seismic response</td>
<td>No earthquake during pit filling</td>
<td>Liquefaction in earthquake and/or loss of foundation strength</td>
<td>5</td>
<td>10</td>
<td>Early warning system, place geogrids, strong geotechnics, and/or install wick drains</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Groundwater</td>
<td>Inflow</td>
<td>Consistent rate of inflow, can be directed with drains and pumped away</td>
<td>Infow exceeds reasonable or practical drainage and pumping capacity</td>
<td>2</td>
<td>10</td>
<td>Install perimeter dewatering wells, increase pumping capacity, maintain pit floor drainage</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Approach</td>
<td>Subsystem</td>
<td>Element</td>
<td>Expected Performance Hazard (Possible Failure Mode)</td>
<td>Probability Quantification</td>
<td>Consequence Quantification</td>
<td>Consequence Risk Quantification</td>
<td>Mitigation and/or Response Measures</td>
<td></td>
<td></td>
</tr>
<tr>
<td>----------</td>
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<td>-------------------------------------------------</td>
<td>---------------------------</td>
<td>-----------------------------</td>
<td>-----------------------------</td>
<td>----------------------------------</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Approach 2</td>
<td>Pit Lake Water</td>
<td>Contamination</td>
<td>Water effectively treated</td>
<td>3</td>
<td>Release of contaminated water to the environment</td>
<td>5</td>
<td>Well designed water treatment facility and/or water storage</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Contamination</td>
<td>Entrained sediments settled out and backfilled into pit</td>
<td>3</td>
<td>Release of contaminated sediments to the environment</td>
<td>5</td>
<td>Well designed water treatment facility including settling ponds</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Pit Lake Water Level</td>
<td>Contamination</td>
<td>Pumping rate matches displacement to maintain water level</td>
<td>2</td>
<td>Contaminated pit water seeping into surrounding groundwater</td>
<td>6</td>
<td>Establish earthworks and pumping methodology, monitoring and additional pumping if required</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Pit Perimeter</td>
<td>Overall stability</td>
<td>Stable</td>
<td>2</td>
<td>Loss of perimeter crest, loss of plant and personnel</td>
<td>10</td>
<td>Stability assessment, monitor perimeter for cracking and movement, delineate exclusion areas such as shear zone and fill zones</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Localised sluffing</td>
<td>Excessive and extensive sluffing</td>
<td>3</td>
<td>Unsafe working conditions for earthworks, plant, damage to plant</td>
<td>12</td>
<td>Conduct trial to assess whether method will work, dewater and continue dumping</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Tailing</td>
<td>Foundation stability</td>
<td>Sufficient strength to support advancing waste-rock</td>
<td>4</td>
<td>Rapid slope failure of advancing face or mass slide of waste rock with loss of working surface, loss of plant and personnel, unsafe working conditions</td>
<td>20</td>
<td>Monitor waste rock movement, establish earthworks methodology and backfill rate to allow porewater pressure dissipation, increase pumping rate to increase freeboard</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Porewater pressure</td>
<td>Excess porewater pressure dissipates at rate commensurate with advance of front face of waste-rock</td>
<td>4</td>
<td>Delayed advance of wastewater front while waiting for porewater pressure to dissipate</td>
<td>12</td>
<td>Monitor waste rock settlement, adjust earthworks methodology and backfill rate to manage settlement</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Waste Rock</td>
<td>Stockpiles</td>
<td>Sufficient stockpile foundation strength</td>
<td>2</td>
<td>Rapid pit crest failure, damage to plant and personnel, unsafe working conditions</td>
<td>6</td>
<td>Stability assessment, delineate exclusion areas such as shear zone and fill zones, reduce stockpile volume, monitor pit perimeter for cracking and movement</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Working platform</td>
<td>Unstable and advances at the rate of rock dumping</td>
<td>3</td>
<td>Method will not work</td>
<td>15</td>
<td>Conduct trial to assess whether method will work, dewater and continue dumping</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Approach 3</td>
<td>Barge</td>
<td>Availability</td>
<td>Available, transportation and cost effective</td>
<td>4</td>
<td>Higher project cost, delayed project start</td>
<td>12</td>
<td>Seek competitive bids, consult several barge contractors</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Operation</td>
<td>Difficult and efficient to operate</td>
<td>3</td>
<td>Delayed project execution, long project life, higher project cost</td>
<td>10</td>
<td>Review different barge types and sizes, select equipment to achieve highest efficiency, establish works methodology</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Siting</td>
<td>Stable, easy access</td>
<td>1</td>
<td>Loss of plant and personnel</td>
<td>5</td>
<td>Implement standard care and maintenance</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Waste Rock</td>
<td>Stockpiles</td>
<td>Sufficient stockpile foundation strength</td>
<td>2</td>
<td>Rapid pit crest failure, damage to plant and personnel, unsafe working conditions</td>
<td>6</td>
<td>Stability assessment, delineate exclusion areas such as shear zone and fill zones, reduce stockpile volume, monitor pit perimeter for cracking and movement</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Backfill</td>
<td>Uniform placement, stable backfill layers</td>
<td>2</td>
<td>Unstable barge, personnel injury</td>
<td>6</td>
<td>Uniform dumping over the entire pit floor surface, allow time for waste rock to consolidates, implement standard care and operating practices</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Loading Platform</td>
<td>Overall stability</td>
<td>Stable</td>
<td>2</td>
<td>Loss of perimeter crest, loss of plant and personnel</td>
<td>10</td>
<td>Stability assessment, careful selection of location, monitor for cracking and movement, ground improvement / support as required</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Localised sluffing</td>
<td>Excessive and extensive sluffing</td>
<td>3</td>
<td>Unsafe working conditions for earthworks, plant, damage to plant</td>
<td>12</td>
<td>Stability assessment, careful selection of location, monitor for cracking and movement, ground improvement / support as required</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
APPENDIX B
Risk Method and Terminology
RISK REGISTERS

General
This Appendix describes the risk registers for the alternative pit backfilling approaches. The following is a discussion of elements of the registers.

Systems and Sub-systems
Selection of the systems and sub-systems is based on consideration of the key (significant) activities involved in backfilling a pit and the system that may respond to such work.

Elements
The term element is used to include what, in essence, may be considered the functionality of a systems and/or sub-system. Thus the ‘element’ column in the risk register focuses on those aspects of performance generally considered when undertaking work to backfill a pit.

Expected Performance
Expected performance is that the system and/or macro component and/or its subsystems perform in accordance with design criteria and intentions, or with what may be termed standard expectations for the performance of systems and components. Possible failure modes generally refer to physical failure of a system or component, excess production of constituents, or the absence of people and equipment to effect standard, routine surveillance and maintenance.

Failure Modes
The possible failure mode is the way in which malfunction or failure of the element associated with a specific system and/or subsystem may occur. For example a spillway designed to pass the probable maximum flood, may fail if a flood greater than the probable maximum flood occurs. While this and other possible failure modes may, by definition, be impossible, we nevertheless quantify such failure modes on the basis of reality - even structures designed to cope with worst case events have failed.

Probability Quantification
The probability of the occurrence of a given failure mode is quantified on a scale of 1 to 5, as follows:

1 = Most unlikely
2 = Unlikely
3 = Possible
4 = Likely
5 = Will occur

Numbers in Tables 1, 2, 3 and 4 are assigned by RGC staff on the basis of their experience and opinions.

Consequences and Consequence Quantification
The consequence of failure of a component is quantified on a scale from 1 to 5, as follows:

1 = Not significant - work-around possible

2 = Interference with safe work - could be mitigated at expense

3 = Possible intolerable consequences to human health and safety or environment

4 = Significant accidents and possible worker accidents

5 = Intolerable accidents and/or worker deaths

Numbers are assigned by RGC staff on the basis of their experience and opinions.

**Risk Quantification**

Risk quantification is the product of the probability quantification and the consequence quantification. Colors shown in the table generally grade from blue (low risk quantification) to orange and red (high risk quantification). The following is the general color scheme and risk quantification that results:

<table>
<thead>
<tr>
<th>Likelihood</th>
<th>Not Likely</th>
<th>Low</th>
<th>Moderate</th>
<th>High</th>
<th>Expected</th>
</tr>
</thead>
<tbody>
<tr>
<td>Extreme</td>
<td>5</td>
<td>10</td>
<td>15</td>
<td>20</td>
<td>25</td>
</tr>
<tr>
<td>High</td>
<td>4</td>
<td>8</td>
<td>12</td>
<td>16</td>
<td>20</td>
</tr>
<tr>
<td>Moderate</td>
<td>3</td>
<td>6</td>
<td>9</td>
<td>12</td>
<td>15</td>
</tr>
<tr>
<td>Low</td>
<td>2</td>
<td>4</td>
<td>6</td>
<td>8</td>
<td>10</td>
</tr>
<tr>
<td>Negligible</td>
<td>1</td>
<td>2</td>
<td>3</td>
<td>4</td>
<td>5</td>
</tr>
</tbody>
</table>

**Response Measure and Mitigative Measures**

Response measures and mitigative measures generally include conservative design, ongoing surveillance and maintenance, and provision of backup or redundant facilities.
APPENDIX B – PRESENTATION OF TAILINGS CONSOLIDATION
CONSOLIDATION DISCUSSION - General

- **Loading Saturated Tailings (Terzaghi, 1943)**
  - Load is initially carried by water in the pores
  - Resulting in rapid increase in pore water pressure
  - Excess pore water pressure gradually dissipates over time
  - Expulsion of water results in a volume change = primary consolidation
  - Load is gradually transferred to the tailings matrix
  - Time for consolidation critically dependent on drainage path length and permeability - parameters of the tailings material and dam configuration

- **Wick Drains**
  - Allow enhanced dissipation of excess porewater pressure by reducing drainage path length (Robertson et al., 2011; Wels. et al., 2000)
  - Provided drainage can occur, settlement magnitude the same with or without wick drains
  - Settlement time can be accelerated by up to an order of magnitude (Wels et al., 2000)
  - Spacing typically 1m to 1.5m for shallow drains (~ 5m depth) and 3m to 5m for deeper drains
CONSOLIDATION DISCUSSION – Rum Jungle

- Dysons (Houghton, 2009)
  - Lower layer deposited during mine operations
  - Upper layer from Old Tailings Dam, deposited during rehabilitation – coarse-grained tailings, slimes washed away by flooding
  - Observed settlement in response to backfill in 1984 (~ 8.5m contaminated soils)
    - Largest settlement observed in slimes zone (eastern end)
    - Approximately 0.9m settlement during first 18 months post-rehabilitation followed by an additional 2.0m settlement between 1986 to 2008 (22 years)
    - Total settlement of ~2.9m or 6% of maximum tailings deposit thickness

- Tailings in Main Pit
  - Tailings discharged sub-aqueously
  - Steeper beach observed on the north side (up to 43m AHD) possibly comprising coarser tailings; slimes likely accumulated in central and southern portion
  - Maximum thickness of tailings about 50m
  - No information on consolidation properties and/or degree of self-weight consolidation

Note: Settlement response is critically dependent on the tailings properties and in-situ pore pressure conditions which are unknowns at this time

RGC CASE STUDY - IAA Helmsdorf (Wismut)

- Coring of uranium tailings in slimes zone
  - 20m to 25m thick sub-aqueously placed tailings
  - 35+ years self weight consolidation (1960-1995)
  - Significant variation in void ratio (density) with depth
  - Implications for consolidation and settlement (see next slides)
Consolidation properties for uranium tailings in slimes zone:

- **HW11 & HW12:**
  - Clay rich, very compressible
  - Very low hydraulic conductivity

- **HW14**
  - Predominantly clay, very compressible
  - Moderate effective hydraulic conductivity due to sand lenses

- **HW13:**
  - Sandy, low compressibility
  - Higher hydraulic conductivity

Predicted consolidation & settlement in slimes zone (for 2m cover load and no wick drains):

- Large variation in total settlement (0.8m to 4.3m)
- Large variation in relative settlement
  - 4% for sandy profile (HW13)
  - 22% for clayey profile (HW11, HW12)

Large range in predicted time of settlement

- ~ 1 year for sandy profile
- ~ 10 years for clayey/sandy profile
- 30+ years for uniform clayey profile
Predicted influence of cover thickness on consolidation and settlement in slimes zone:

- Some increase in predicted settlement with greater surcharge
  - Three times cover load increases settlement by 0.6m (or 23%)
- No significant increase in rate of consolidation/settlement

Influence of surcharge load:

- 2m cover = 36 kPa
  - $e = 1.1$ to $1.3$
- 6m cover = 108 kPa
  - $e = 1.0$ to $1.25$

Rum Jungle backfill:

- 50m backfill (buoyant)
  - $e = 0.75$
- 50m backfill (dewatered)
  - $e = 0.65$
- Dewatering backfill provides only limited additional settlement
**RGC CASE STUDY - IAA Helmsdorf (Wismut)**

- Use of wick drains to accelerate consolidation/settlement (in fine tailings):
  - Assume fully penetrating wick drains at 4m spacing
  - 85% settlement in ~1 year
  - Wick drains predicted to increase rate of settlement by an order of magnitude (from 10 years without deep drains to ~1 year)

**RUM JUNGLE IMPLICATIONS**

- Slimes zone of IAA Helmsdorf likely a good analogue for sub-aqueously placed tailings in Main Pit
- Future consolidation/settlement is critically dependent on tailings material properties (as demonstrated in IAA Helmsdorf)
- Tailings properties likely to vary significantly both laterally and vertically in Main Pit
- Initial cover placement under water (say 5m thick) would result in significant settlement; temporary “dewatering” of this interim cover would result in relatively small further increase in settlement
- Final settlement estimated to be in the range of 5% to 20% of tailings thickness in Main Pit
- Due to the varied thickness and material properties, differential settlement is expected (highest settlement likely in center)
- Wick drains could significantly accelerate time of settlement
- Wick drain design would require detailed knowledge of the tailings
**RUM JUNGLE IMPLICATIONS**

![Graph showing Additional Waste Rock vs. Settlement]

- Using plausible range of settlement (5% to 20%) from Helmsdorf, additional volume available for waste rock backfill estimated to range from 2% to 8.5% of total waste rock backfill volume.
- With consideration to analog sites and data from Dysons, average final tailings settlement = 10% of deposit depth used for preliminary scheduling and quantities (see subsequent slides).

---

**REFERENCES**


Many RGC papers on tailings studies can be found at:

[https://www.rgc.ca/?page=publications](https://www.rgc.ca/?page=publications)
APPENDIX C – MAIN PIT BACKFILL ADVANCED CONCEPT APPROACH –
CALCULATION PACKAGE
REPORT NO. 183006/3

CALCULATION PACKAGE

MAIN PIT BACKFILLING ADVANCED CONCEPT APPROACH

Submitted to:

Northern Territory Government
Department of Mines and Energy

Prepared by:

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Consulting Engineers and Scientists for the Mining Industry
www.robertsongeoconsultants.com

December 2015
REPORT NO. 183006/3

CALCULATION PACKAGE

MAIN PIT BACKFILLING ADVANCED CONCEPT APPROACH

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REPORT NO. 183006/3

CALCULATION PACKAGE

MAIN PIT BACKFILLING ADVANCED CONCEPT DESIGN

1 INTRODUCTION

Backfilling the Main Pit is a component of the overall Rum Jungle rehabilitation project. The plan is to backfill the existing open pit with waste material from various deposits on the site. The main reasons for doing this are to prevent acid generation from these Potentially Acid Forming (PAF) materials, and to reinstate the land to a dry surface.

An options study (RGC, December 2015) was conducted by Robertson GeoConsultants (RGC) in which candidate approaches (Approaches 1 to 5) were initially screened and the most promising (Approach 5) was selected using a qualitative selection process. Two approaches were then refined (Approach 5A and 5B) and comparison of these was done via a semi-quantitative technique.

This document presents the calculations for the criteria that were quantified in the comparison of Approach 5A and Approach 5B.

The overall intent of the study was to produce a rational basis for comparison and to inform scheduling and costing.

It was compiled by Andy Thomas and reviewed by Jack Caldwell, both of RGC.

2 BACKFILL APPROACHES

Details of the methodology for Approach 5A and Approach 5B, as well as the pit geometry, are in RGC (December 2015). Both approaches include PAF waste rock backfill with overwater conveying and conventional earthworks methods. The main distinctions are that a greater proportion is backfilled via conventional earthworks in Approach 5B and that wick drains are incorporated to accelerate settlement of the tailings.
3 CRITERIA ANALYZED

Key criteria for which quantified estimates could be made at this time, with the current site knowledge, are described in Table 1.

Table 1 – Criteria details

<table>
<thead>
<tr>
<th>Criteria</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>Time</td>
<td>Length of the component tasks to estimate the total construction period</td>
</tr>
<tr>
<td>Water Quality</td>
<td>Contamination impact on pit water, separated into ‘Impacted’ and ‘Unimpacted’ and calculated for both total volume and dewatering rates with time</td>
</tr>
<tr>
<td>Waste Rock</td>
<td>Tonnage of PAF waste rock backfilled with time</td>
</tr>
</tbody>
</table>

4 METHOD

The method used to calculate the quantity for each of the criteria components analyzed is summarized in this section.

4.1 TIME

A nominal time allowance was made for the dewatering tasks. In reality this depends on the pit walls phreatic surface drawdown.

Total pit push-back time was calculated based on typical earthworks rates from RGC experience on similar projects.

The time for the floating conveyor dumping backfill component depends on the operating speed and limitations of the conveyor. A conveyor rate, incorporating allowance for maintenance and downtime, was selected in consultation with a conveyor contractor.

The time for the conventional earthworks backfill component depends on the rate that the material can be carted by truck and shovel from the source locations to the pit and the machinery limitations in the pit. The earthworks machinery fleet was selected based on experience on similar projects and considering such variables as; material density, truck load volume, load time, average truck speed, distance and grade of the route, dump time etc.

4.2 WATER QUALITY

The volume and rate of dewatering were calculated for both impacted and unimpacted water. Impacted water is defined as water that has come into contact with backfill material, pit push-back spoil or tailings porewater seepage; effectively this includes all water within or entering the pit after the completion of initial dewatering of the existing (unimpacted) pit water.
4.2.1 Volumes

The volume of unimpacted water was obtained from the pit volume-height curve (Figure 1). The volume of impacted water includes the pit water, calculated from this curve, as well as the net inflow (groundwater plus precipitation, minus evaporation) volume and seepage from the tailings (in the case of Approach 5B).

Dewatering was assumed to cease once the PAF waste rock backfill reached 58.5m AHD level.

4.2.2 Rates

The sizing of wastewater treatment facilities is typically based on the peak inflow rate. This could be attenuated with a storage facility, so the rates presented in this report do not necessarily need to be the treatment rates.

In the case of unimpacted water, the pumping rates were calculated using the total volume and the time allowed for pumping to get an average rate.

For impacted water, rates depend on the net inflow, backfill pore space and the seepage flow from the tailings due to settlement. The rates were calculated to maintain a constant pit water level throughout the year. In reality the level could potentially be allowed to rise with the backfill level providing the working surface remained dry for earthworks.

4.3 WASTE ROCK

The important measure in maximizing the PAF waste rock backfill is the tonnage placed rather than the volume. Consequently it is the density that is of primary importance. Notwithstanding, Approach 5B generates a larger volume available for waste rock during backfilling due to settlement of the tailings.

Typical values were assigned for the different backfill approaches; dumped over water, and placed and compacted (conventional earthworks). Using the different proposed levels of backfill for each method and the pit volume-height curve (Figure 1), the different available volumes were obtained. Multiplying by the material density yielded the total backfill tonnage.

5 DATA

Available site data, and their sources, used in the calculations are presented herein.

5.1 PRECIPITATION

An average annual precipitation value of 1,652mm was used; averaged from 30 years of data for the region by the Australian Government Bureau of Meteorology (AGBM).
5.2 EVAPORATION

Average monthly pan evaporation for the region was sourced from the AGBM (Table 2). The values are reported to be based on at least 10 years of data.

<table>
<thead>
<tr>
<th>Season</th>
<th>Month</th>
<th>Pan Evaporation (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wet</td>
<td>January</td>
<td>162.5</td>
</tr>
<tr>
<td></td>
<td>February</td>
<td>137.5</td>
</tr>
<tr>
<td></td>
<td>March</td>
<td>162.5</td>
</tr>
<tr>
<td></td>
<td>April</td>
<td>137.5</td>
</tr>
<tr>
<td>Dry</td>
<td>May</td>
<td>187.5</td>
</tr>
<tr>
<td></td>
<td>June</td>
<td>187.5</td>
</tr>
<tr>
<td></td>
<td>July</td>
<td>187.5</td>
</tr>
<tr>
<td></td>
<td>August</td>
<td>225.0</td>
</tr>
<tr>
<td></td>
<td>September</td>
<td>225.0</td>
</tr>
<tr>
<td></td>
<td>October</td>
<td>225.0</td>
</tr>
<tr>
<td></td>
<td>November</td>
<td>275.0</td>
</tr>
<tr>
<td></td>
<td>December</td>
<td>187.5</td>
</tr>
</tbody>
</table>

6 PROPERTIES

Measured, inferred or modeled properties used in the calculations are presented in this section.

6.1 Pit

Details of the pit were sourced from the document RGC (December 2015).

6.1.1 Dimensions

The pit is roughly circular with a crest diameter of about 380m and total plan area of about 113,000m$^3$.

The pit lake area after initial dewatering to 43m AHD for both approaches is about 50,000m$^2$. 
6.1.2 Volumes

A pit volume-height graph was generated from a pit three-dimensional model which was generated by meshing pit bathymetry, historic pit shell survey and recent LiDAR data. The graph (Figure 1) is of the pit space above the tailings surface. It was used to estimate volumes for the different materials.

Material volumes used in the calculations for each approach are in Table 3. Note that the water volumes are for the in-pit water only; groundwater and precipitation/evaporation fluxes are accounted for in the calculations in Section 8.

![Main Pit: Volume - Height](image)

Figure 1 – Main pit volume-height curve
Table 3 – Pit material volumes

<table>
<thead>
<tr>
<th>Stage</th>
<th>Material</th>
<th>Levels (m AHD)</th>
<th>Volume (m$^3$)</th>
<th>Approach 5A</th>
<th>Approach 5B</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Dewatering</td>
<td>Unimpacted water</td>
<td>60 to 43</td>
<td>1,400,000</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Conveyor Dumping</td>
<td>Loose waste rock</td>
<td>17 to 40</td>
<td>900,000</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>17 to 22</td>
<td>-</td>
<td>150,000</td>
<td></td>
</tr>
<tr>
<td>Secondary Dewatering</td>
<td>Impacted water</td>
<td>43 to 40</td>
<td>150,000</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>43 to 22</td>
<td>-</td>
<td>900,000</td>
<td></td>
</tr>
<tr>
<td>Conventional Earthworks</td>
<td>Compacted PAF Waste Rock</td>
<td>40 to 58.5</td>
<td>1,300,000</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>22 to 58.5</td>
<td>-</td>
<td>2,140,000</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Clean Fill</td>
<td>58.5 to final landform</td>
<td>1,000,000</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

6.1.3 Haul Road

Interpreted from the main pit bathymetry survey, the former haul road is relatively intact to about 43m AHD, at which point it is truncated by the deposition cone in the north and after which it appears to be in poor condition or filled in.

The grade of the former road is approximately 5%.

6.1.4 Wall Materials

The push-back will involve excavating material from the pit walls. The majority of the walls comprise metamorphosed argillaceous rocks; shale, slate and schist. Typical properties of these used in the calculations are provided in Table 4.

Table 4 – Wall material properties

<table>
<thead>
<tr>
<th>Property</th>
<th>Shale/Slate/Schist</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density (t/m$^3$)</td>
<td>2.5</td>
</tr>
<tr>
<td>Bulking Factor (%)</td>
<td>40</td>
</tr>
<tr>
<td>Void Space (%)</td>
<td>15</td>
</tr>
</tbody>
</table>
6.2 GROUNDWATER

Groundwater inflows were obtained from RGC groundwater modeling done for lake levels of 43m AHD and 22m AHD, reflective of the two approaches. The resultant inflow rates for the two seasons are shown in Table 5.

<table>
<thead>
<tr>
<th>Dewatered Pit Lake Level (m AHD)</th>
<th>Average Groundwater Inflow (L/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Dry Season</td>
</tr>
<tr>
<td>43</td>
<td>6</td>
</tr>
<tr>
<td>22</td>
<td>8</td>
</tr>
</tbody>
</table>

6.3 BACKFILL

Waste backfill is planned to come from (in order) the Intermediate Waste Rock Dump (WRD), Dysons Pit and the Main WRD. Above 58.5m AHD clean backfill is proposed to consist of clayey laterite material. Typical properties for these units are provided in Table 6.

<table>
<thead>
<tr>
<th>Property</th>
<th>Waste Rock</th>
<th>Laterite Clay</th>
</tr>
</thead>
<tbody>
<tr>
<td>Loose Density (t/m$^3$)</td>
<td>1.60</td>
<td>1.50</td>
</tr>
<tr>
<td>Void Space (%)</td>
<td>15</td>
<td>N/A</td>
</tr>
<tr>
<td>Compacted Density (t/m$^3$)</td>
<td>1.76</td>
<td>1.65</td>
</tr>
<tr>
<td>Water-placed Relative Density</td>
<td>0.4</td>
<td>N/A</td>
</tr>
<tr>
<td>Water-placed Density (t/m$^3$)</td>
<td>1.66</td>
<td>N/A</td>
</tr>
</tbody>
</table>

Waste rock compacted density was obtained using data from site compaction trials undertaken by RGC in 2014. The data indicates that the waste rock from the Main WRD has a compaction factor ranging from about 5% to 20% depending upon the thickness of lift (800mm and 200mm respectively). From the Intermediate WRD, the compaction factors range from about 8% to 25%. For the calculations, a compaction factor of 10% was adopted, deemed to be realistically achievable with lift heights of about 500mm.

The water-placed relative density of the waste rock was estimated based on the premise from Gerwick (2007) that ‘Relative densities of cohesionless materials (sands and gravels) placed through a substantial depth of water may vary from 40% to 60%’. For the calculations, a value of 40% was used.

6.4 FINAL LANDFORM

The final landform configuration was taken from the Water Technology channel design option model diagrams provided to RGC in September 2015. The concept design (Figure 2) is a conical-shaped structure with a flat crest at the maximum height of 74m AHD.
7 ASSUMPTIONS

7.1 CONSTRUCTION SCHEDULE

The wet season was taken to be over the months January to April (inclusive), during which it was assumed that earthworks will not be possible. The remaining eight months was taken to be the dry season construction period.

It was assumed that the construction schedule would be seven days per week, 10 hours per day except for conveyor operation which was assumed to be 20 hours per day (making reasonable allowance for maintenance/shut-down).

A month was taken to be a yearly-averaged value of 30.25 days.

7.2 PIT LAKE LEVEL

The pit lake surface level varies up to several metres throughout the year. For the design it was assumed that the initial lake level would be 60m AHD.
7.3 **GROUNDWATER**

Consideration was not made for the influence on the groundwater inflow rate due to the presence of backfill against the pit walls. Intuitively the inflow rates would reduce as the pit is filled. Therefore the inflow rates adopted for the life of the project are probably higher than if backfill was incorporated in the modeling.

7.4 **DEWATERING**

The time for initial pit dewatering for both approaches was assumed to be over four months of the wet season. If the time for full depressurization of the pit walls is longer, the dewatering rate may need to be decreased which would impinge on the dry season, or more time would need to be allowed before entering the pit for the push-back.

The quality of the pit water from the initial dewatering was assumed to be acceptable for direct discharge to Finniss River during the wet season.

Starting from commencement of pit push-back the pit water was assumed to be impacted either by the materials being dumped, or due to disturbance of the chemocline at the bottom mixing with the overlying water. As such this water is expected to require treatment prior to discharge.

Secondary dewatering was assumed to commence immediately after conveyor backfilling, which for both approaches is in the dry season. Because this water is expected to need treatment and there will likely be water storage associated with the treatment system, it was not restricted to the wet season as per the initial dewatering.

The dewatering rate selected for the secondary dewatering was designated to be similar to the peak dewatering rate (about 70L/s) from displacement during conveyor backfilling. The resulting timeframe is two months for Approach 5A and six months for Approach 5B. This rate may need to be decreased, or more time allowed following if the time for pit wall depressurization is longer.

7.5 **PUSH-BACK**

The former haul road would be reinstated at the existing 5% grade for Approach 5A and a new road at 10% for Approach 5B, over a total length of about 400m for both.

A pit wall angle of 30° was assumed for the push-back. This relatively shallow angle was selected for safety and to avoid the requirement for complex ground support measures.

It was assumed that 20% of the former haul road width could be utilized and the new haul road would be 20m wide.

Construction of the push-back was assumed to be using drill and blast method to break up the rock and then shoveled and carted using three Caterpillar 777 haul trucks (see Table 7 for specifications). It was assumed that the spoil would be back-end dumped from the haul trucks into the pit from the road edge.
7.6 BACKFILL

7.6.1 Conveyor

Conveyor backfill rates were derived from discussion with a conveyor contractor who thought that 500 tonnes per hour would be reasonably achievable.

7.6.2 Conventional Earthworks

Key details assumed for the conventional earthworks component are provided in Table 7.

Table 7 – Conventional earthworks key details

<table>
<thead>
<tr>
<th>Item</th>
<th>Waste Rock</th>
<th>Clean Fill</th>
</tr>
</thead>
<tbody>
<tr>
<td>Haul Truck Type</td>
<td>Caterpillar 777</td>
<td></td>
</tr>
<tr>
<td>Capacity (t)</td>
<td>100</td>
<td></td>
</tr>
<tr>
<td>Number of Active Trucks</td>
<td>5</td>
<td>8</td>
</tr>
<tr>
<td>Load Efficiency (%)</td>
<td>90</td>
<td></td>
</tr>
<tr>
<td>Average Haul Speed (km/hr)</td>
<td>20</td>
<td>30</td>
</tr>
<tr>
<td>Haul Distance (km)</td>
<td>2</td>
<td>12</td>
</tr>
<tr>
<td>Combined Load and Dump Time (mins)</td>
<td>12</td>
<td>12</td>
</tr>
<tr>
<td>Average Cycle Time (mins)</td>
<td>24</td>
<td>60</td>
</tr>
</tbody>
</table>

7.7 SETTLEMENT

Discussion of the settlement behavior of the tailings, based on data from Dyson’s Pit and an analogue site (IAA Helmsdorf) is in RGC (December 2015). A settlement value of 10% of the tailings depth was assumed for the calculations in the main pit.

Based on consultation with a wick drain contractor, the assumed time required for wick drain installation would be about three months.

It was assumed that the 10% total settlement would be achieved in the first 12 months which is before the PAF waste rock reaches final level, hence the total of this additional volume could be taken up by PAF waste rock.
8 CALCULATIONS

8.1 SURFACE WATER

8.1.1 Precipitation
Total average annual precipitation was multiplied by the total pit surface area for a total volume of about 165ML which equates to an average wet season monthly inflow of 18L/s (assuming constant, uniform inflow).

8.1.2 Evaporation
The seasonal average rates were multiplied by the exposed pit lake surface area for a total volume of about 113ML which was averaged to get a wet season monthly outflow of 2.8L/s and an average dry season outflow of 4.0L/s (assuming constant uniform outflow).

8.2 PUSH-BACK
The volume of material was calculated by taking the average cross sectional area of the push-back at the mid-distance of the haul road and multiplying it by the total length. It was calculated to generate 120,000m$^3$ and 250,000m$^3$ of bank volume spoil in Approach 5A and Approach 5B respectively. Using the assigned density value, this equates to a total tonnage of about 270,000 tonnes and 560,000 tonnes. Using the assigned bulking factor, this converts to a total volume of 168,000m$^3$ and 350,000m$^3$.

Push-back for Approach 5A was calculated to take five months and eight months for Approach 5B; additional three months is for the haul road extension constructed after secondary dewatering. Based on these timeframes, average spoil backfill was calculated by averaging the total bulked volume over the period of construction.

Although it was assumed that construction would only be during day shifts, for simplicity, the dumping rate was taken to be uniform and constant, averaged over the full construction period. It equates to an equivalent volume rate of 13L/s and 17L/s for Approach 5A and Approach 5B respectively. Using the assigned void ratio, the displaced pit volume from the spoil backfill was calculated to be an equivalent of 11L/s and 14L/s.

8.3 BACKFILLING
With the assumptions and properties assigned, the length of time for material movement and backfill tonnage can be estimated for each methodology.

8.3.1 Conveyor
Using the assumed conveyor rate and shift length, the time to backfill to the prescribed levels for Approach 5A would be five construction months and Approach 5B would be one month.

Using the loose material density value, about 1.5 million tonnes of PAF waste rock would be backfilled in Approach 5A and 250,000 tonnes in Approach 5B.
8.3.2 Conventional Earthworks

Using the truck material movement values and shift length, it was calculated that about 9,000 tonnes of PAF waste rock could be backfilled per shift. At this rate, it would take nine construction months to backfill using conventional earthworks for Approach 5A and 19 months for Approach 5B.

Using the assumed placed density, about 2.3 million tonnes of PAF waste rock would be placed in 5A compared to about 3.8 million tonnes in 5B. Some of the additional tonnage in 5B is attributed to the extra pit volume from tailings settlement (see Section 8.4).

Clean fill backfill above 58.5m AHD could be done at a rate of about 5,300 tonnes per shift, equating to eleven construction months for both approaches. This equates to about 1.6 million tonnes of material backfilled.

8.4 Settlement

Total settlement is equal to the volume of water expelled from the tailings.

Installation of wick drains is incorporated in Approach 5B to accelerate tailings settlement. The total settlement was calculated to generate an additional 63,000m$^3$ of pit volume which can then be used for PAF waste rock backfill.

The settlement volume essentially represents the volume of water that has been displaced from the tailings; hence an additional 63,000m$^3$ of impacted water is generated in Approach 5B. Calculating this at a constant and uniform average rate over the settlement time equals 2L/s.

8.5 Dewatering

The component inflows and outflows to calculate the dewatering rates are explained for Approach 5A and Approach 5B in Table 8 and Table 9 respectively. These rates were used to calculate the total impacted water volumes of 1.7GL and 2.3GL respectively.

Note that the construction season details only are provided. In the wet season, net inflow comprizing groundwater inflow and precipitation minus evaporation would be pumped out at the inflow rate.

<table>
<thead>
<tr>
<th>Stage</th>
<th>Components</th>
<th>Timeframe (months)</th>
<th>Equivalent Rate (L/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Dewatering</td>
<td>Pit lake + groundwater inflow (wet) + precipitation (wet) – evaporation (wet)</td>
<td>4</td>
<td>157</td>
</tr>
<tr>
<td>Pit Push-back</td>
<td>Groundwater inflow (dry) + push-back displacement – evaporation (dry)</td>
<td>5</td>
<td>13</td>
</tr>
<tr>
<td>Conveyor Dumping</td>
<td>Groundwater inflow (dry) + conveyor displacement – evaporation (dry)</td>
<td>5</td>
<td>63</td>
</tr>
<tr>
<td>Secondary Dewatering</td>
<td>Pit lake + groundwater inflow (dry) – evaporation (dry)</td>
<td>2</td>
<td>31</td>
</tr>
<tr>
<td>Conventional Earthworks</td>
<td>Groundwater inflow</td>
<td>9</td>
<td>6</td>
</tr>
</tbody>
</table>
Table 9 – Approach 5B dewatering components

<table>
<thead>
<tr>
<th>Stage</th>
<th>Components</th>
<th>Timeframe (months)</th>
<th>Equivalent Rate (L/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Dewatering</td>
<td>Pit lake + groundwater inflow (wet) + precipitation (wet) – evaporation (wet)</td>
<td>4</td>
<td>157</td>
</tr>
<tr>
<td>Pit Push-back</td>
<td>Groundwater inflow (dry) + push-back displacement – evaporation (dry)</td>
<td>5</td>
<td>13</td>
</tr>
<tr>
<td>Conveyor Dumping</td>
<td>Groundwater inflow (dry) + conveyor displacement – evaporation (dry)</td>
<td>1</td>
<td>65</td>
</tr>
<tr>
<td>Secondary Dewatering</td>
<td>Pit lake + groundwater inflow (dry/wet) – evaporation (dry/wet)</td>
<td>6</td>
<td>61/67</td>
</tr>
<tr>
<td>Pit Push-back</td>
<td>Groundwater inflow (dry) + push-back displacement – evaporation (dry)</td>
<td>3</td>
<td>17</td>
</tr>
<tr>
<td>Wick Drain Installation</td>
<td>Groundwater inflow (dry) + Seepage water</td>
<td>5</td>
<td>10</td>
</tr>
<tr>
<td>Conventional Earthworks</td>
<td>Groundwater inflow</td>
<td>14</td>
<td>8</td>
</tr>
</tbody>
</table>

9 SCHEDULES

The time and material schedules for each of the backfilling approaches are shown in Appendix A.

10 LIMITATIONS

Because limited site data was available, for the most part, the values were selected using engineering judgment and experience. The actual final quantities will depend critically on many interdependent variables. The accuracy of the calculations could be improved by incorporating values representative of the actual site conditions and materials.

11 REFERENCES


APPENDIX A – TIME AND MATERIAL SCHEDULES
## Approach SA

### Main Pit Backfilling Advanced Concept Approach - Calculation Package

#### Robertson GeoConsultants Inc.      RGC Report 183006/3

<table>
<thead>
<tr>
<th>Year</th>
<th>Month</th>
<th>Rate (m³/ month)</th>
<th>Rate (m³/d)</th>
<th>Rate (m³/month)</th>
<th>Cumulative (m³)</th>
<th>Percent Total (%)</th>
<th>Rate (L/q)</th>
<th>Rate (L/month)</th>
<th>Cumulative (L)</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 Nov-17</td>
<td></td>
<td>5,623</td>
<td>182.188</td>
<td>102.166</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>Start Initial Dewatering</td>
</tr>
<tr>
<td>2 Nov-18</td>
<td></td>
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<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
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<td>End Initial Dewatering</td>
</tr>
<tr>
<td>3 Oct-19</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>Start Clean Fill Backfilling</td>
</tr>
<tr>
<td>4 Oct-20</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>End Clean Fill Backfilling</td>
</tr>
</tbody>
</table>

### Conversion Table

- **Rate (m³/ month):**
  - 1 month = 30 days
  - 1 month = 365 days

### Backfilled Volume

<table>
<thead>
<tr>
<th>Year</th>
<th>Month</th>
<th>Rate (m³/ month)</th>
<th>Rate (m³/d)</th>
<th>Rate (m³/month)</th>
<th>Cumulative (m³)</th>
<th>Percent Total (%)</th>
<th>Rate (L/q)</th>
<th>Rate (L/month)</th>
<th>Cumulative (L)</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 Nov-17</td>
<td></td>
<td>5,623</td>
<td>182.188</td>
<td>102.166</td>
<td>0</td>
<td>0</td>
<td>5,623</td>
<td>182.188</td>
<td>102.166</td>
<td>Start Initial Dewatering</td>
</tr>
<tr>
<td>2 Nov-18</td>
<td></td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>End Initial Dewatering</td>
</tr>
<tr>
<td>3 Oct-19</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>Start Clean Fill Backfilling</td>
</tr>
<tr>
<td>4 Oct-20</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>End Clean Fill Backfilling</td>
</tr>
</tbody>
</table>

### Details

- **Start Initial Dewatering:** 5,623 m³/month
- **End Initial Dewatering:** 0 m³/month
- **Start Clean Fill Backfilling:** 0 m³/month
- **End Clean Fill Backfilling:** 0 m³/month
<table>
<thead>
<tr>
<th>Year</th>
<th>Month</th>
<th>Rate (m³/day)</th>
<th>Rate (m³/month)</th>
<th>Cumulative (m³)</th>
<th>Percent Total (%)</th>
<th>Rate (t/day)</th>
<th>Rate (t/month)</th>
<th>Cumulative (t)</th>
<th>Rate (L/s)</th>
<th>Volume (L)</th>
<th>Cumulative Volume (L)</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>3 May</td>
<td>2023</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>End Establishment etc.</td>
</tr>
<tr>
<td>1 Jan</td>
<td>2017</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>Start Initial Dewatering</td>
</tr>
<tr>
<td>1 Jul</td>
<td>2017</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>Start Initial Dewatering</td>
</tr>
<tr>
<td>1 Aug</td>
<td>2017</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
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<tr>
<td>1 Sep</td>
<td>2017</td>
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<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>End Secondary Pit Push-back</td>
</tr>
<tr>
<td>1 Nov</td>
<td>2017</td>
<td>0</td>
<td>0</td>
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**Approach SB**

**Conventional Earthworks**

**Conveyor Bumping**

**Backfilled Volume**

**Backfilled Tonnage**

**Dewatering**

Robertson GeoConsultants Inc.  
RGC Report 183006/3