Mining Operations

Mining Method

The mining method proposed by A&R is based on inclined standard room-and pillar working up-dip on a 20% slope with final rhomboid pillars of 1.6m by 5.7m which will result in less dilution and higher recovery of gold than the previous NGM method. The initial operations of the mine over the first 12-18 months will concentrate on the winning of ore readily available in the existing stopes, mining of remnant pillars and ore from development mining headings together with treatment of the ore from the harbour stockpile. It has been identified that a considerable amount of fines (-50µm) gold remains in all the worked out stopes as ore dust. This dust be will cleaned up and won by sweeping, dinting and wash down by hydraulic methods to maximise overall gold recovery. The A&R cut-off grade of 7g/t as against the NGM cut-off grade of 15g/t releases upwards of 20K tonnes of previously out of grade ore. Ore won from the remnant pillars is expected to have a grade of 13g/t. This initial ore allows early positive operational cash flow. Full stope mining will then commence as development accesses and proves the new mining areas.

Stope development will follow the following pattern:

- Development of the stope perimeter headings (Development Mining);
- Development drilling of the stope;
- Ore production from the stope (Production Mucking); and
- Backfilling of the worked out stope using waste rock from the mining and development operations.

This progressive mining process is illustrated in Figure 12.3.

All waste rock produced by the mining process and indeed from the process stream, including tailings, will be utilised in the system of backfilling the stopes. No waste of any sort will be removed from the mine.

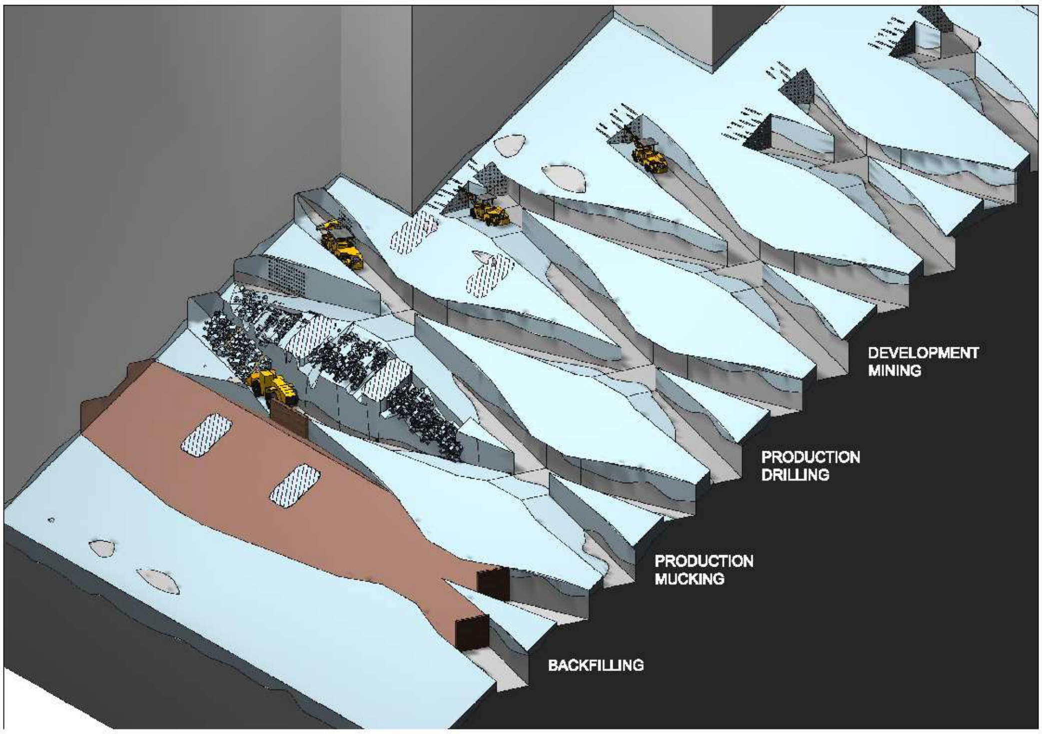


Figure 12.3 Diagrammatic Representation of Mining Method

Tailings/Backfill , Underground Water, Ventilation, and Geotechnical Design

Golder Paste Technology Europe Ltd have produced a comprehensive study for NGM, dated 15th July 2009, of the tailings/backfill system, underground water, ventilation and geotechnical considerations.

Golder Paste Technology Europe Ltd Report

Limitations

The study has been conducted at a preliminary stage in the NGM re-development. As such some of the information or engineering detail necessary to conclude the design is not yet available. The study has however focused on the development of feasible operating solutions which will be further defined as the mine nears operation and the necessary detail becomes available.

This report was prepared for the exclusive use of Angus and Ross PLC. The report, which specifically includes all tables, figures and appendices, is based on data and information received from Angus and Ross PLC and collected during any previous work conducted by Golder Paste Technology (Europe) Ltd (Golder PasteTec) and is based solely on the conditions of the property at the time of the work, supplemented by historical information and data obtained by Golder PasteTec as described in this report.

The services performed, as described in this report, were conducted in a manner consistent with that level of care and skill normally exercised by other members of the engineering and science professions currently practicing under similar conditions, subject to the time limits and financial and physical constraints applicable to the services.

Any use which a third party makes of this report, or any reliance on, or decisions to be made based on it, are the responsibilities of such third parties. Golder PasteTec accepts no responsibility for damages, if any, suffered by any third party as a result of decisions made or actions based on this report.

The content of this report is based on information collected during our investigation, our present understanding of the site conditions and our professional judgement in light of such information at the time of this report. This report provides a professional opinion and therefore no warranty is either expressed, implied or made as to the conclusions, advice and recommendations offered in this report. This report does not provide a legal opinion regarding compliance with applicable laws. With respect to regulatory compliance issues, it should be noted that regulatory statutes and the interpretation of regulatory statutes are subject to change.

The findings and conclusions of this report are valid only as of the date of this report. If new information is discovered in future work, Golder PasteTec should be requested to re-evaluate the conclusions of this report and to provide amendments as required.

Executive Summary

Angus and Ross PLC (A&R) are to undertake mining and mineral processing activity within the underground Nalunaq Gold Mine (NGL). The mineral processing, ore concentration, refinement, waste management and water management follows a different basis to that previously employed at NGM. The mineral processing operations will consist of two phases undertaken in the underground mine. The first phase is gravity based mineral processing circuit and the second (additional) phase is a cyanide based mineral processing circuit.

NGM intend to commence development based mining activity during the third quarter of 2009, installing the first phase of the processing circuit in the fourth quarter 2009. The second phase of the mineral processing circuit is planned for installation and subsequent commissioning in early 2010.

The mineral processing activity generates liquid and solid materials which have been identified and are described as process water with fines (tailings) from the mineral processing and dust / fumes associated with the mining and mineral processing.

Methodologies have been identified and preliminary design work undertaken to manage these waste materials produced. The aim being to minimise potential impact on the underground and surface environment that result from the mineral processing operations in the NGM.

All process water and fines from the mineral processing circuit (tailings) will be placed in previously mined areas in NGM. This is analogous to the conventional placement of tailings as mine backfill in underground mining operations. This application provides an effective means to fill old excavations within the mine whilst minimising materials that need to leave the mine. The tailings will be placed in old stopes with engineering controls designed and built to provide an opportunity to drain water from the material over the medium term thereby leaving the solids component in place in the mined out stopes.

Underground mine water will be managed in a controlled fashion to minimise discharge from the mine and maximise recirculation and recycling of the water in the mining and mineral processing operations. Engineered systems have been delineated to provide for appropriate means to control, clean and transport underground mine water in the operations and where necessary to discharge. These engineered controls are designed to handle the natural water flows recorded as well as those underground mine waters associated with the tailings placement, mineral processing and other mining activity.

The ventilation system at NGM has adequate capacity to manage the airborne dust and fumes associated with the mining and mineral processing activity. The ventilation system requires minor modifications to fan locations and ducting to manage ventilation flows adequately and ensure fresh air is blown in and exhaust air blown out of working areas (including the mineral processing area). A dust collection facility is recommended for the first phase of the mineral processing circuit. The second phase processing requires additional instrumentation, controls and procedures to be implemented to mitigate concentrations of the fumes associated with the electro-winning and chemicals associated with the leaching in the confined areas, ensuring these are exhausting out of the general mine environment.

Introduction

Background

Angus and Ross PLC (A&R) is planning to undertake mining and mineral processing activity within the underground Nalunaq Gold Mine (NGL). The mineral processing operations will consist of two phases. The first phase will be gravity based mineral processing circuit and the second phase a cyanide based mineral processing circuit.

NGM intend to commence development based mining activity during the third quarter of 2009, installing the first phase of the processing circuit in the fourth quarter 2009. The gravity mineral processing circuit will be running for approximately three months before commissioning the second phase of the mineral processing circuit in 2010.

The mineral processing phases of activity generate liquid and solid materials which have been identified during the design of the mining and mineral processing system. Specifically these waste materials are described as:

- Process water with fines from the mineral processing (tailings); and
- Dust and fumes associated with the mining and mineral processing.

Methodologies have been identified and preliminary design work undertaken to manage these waste materials produced. The aim being to minimise potential impact on the underground and surface environment that result from the mineral processing operations in the NGM.

Approach

The approach undertaken has identified effective methods to manage the process water and airborne waste materials generated by the mining and mineral processing operations. These methods suit both the gravity based and cyanide based phases of mineral processing activity.

The intent of this report is to provide descriptions of the adopted methodologies that have been incorporated in to the design planning by A&R for NGM.
A&R intend to advance these methods to manage waste to an appropriate level of design. This will be done in parallel to the advancement of the mine planning requirements toward the resumption of operations at the underground mine.

A geotechnical appraisal has also been undertaken for the planned mineral processing excavation within the NGM, recognizing the importance of this planned infrastructure to the operations. The geotechnical appraisal provides for stability analysis of the new excavation that incorporates the first and second phases of the mineral processing system at NGM. This appraisal of the preliminary design is included in this report.

Objectives

The objective of this report is to provide for a description of the solutions to manage the waste materials within the NGM operations.

This report provides three sections addressing the management of waste from the mineral processing and mining operations. The fourth section provides for the geotechnical appraisal of the mineral processing excavation. The sections are:

- Tailings;
- Underground water;
- Ventilation; and
- Geotechnical.

Each section provides a description of the methodology to be implemented at NGM to ensure efficient, safe, sustainable and effective management of its operations and the environment. The report also provides details of the preliminary design work for each aspect of the project.

Tailings

Tailings Management

Previously all ore was concentrated off site, removing any requirement to handle post concentration slurried waste material (tailings). The operation will concentrate and refine all the ore on site within the underground environment, and therefore it is necessary to manage the tailings within the underground mine. No external tailings storage facility is proposed. It is understood that development waste, i.e. material not passing through the concentration process will also be stored underground in previously excavated stope areas however this is not the focus of this study.

The overriding objective of the tailings management is the placement of material within the existing mined stopes, such that no tailings placement will be required externally from the mine. Furthermore, consideration has been given to minimising environmental, health and safety risk associated with the storage of the tailings.

Methodology Overview

The concentration process and hence tailings generation will occur on the 300 Level some 340m from the portal. The process will include the comminution of the gold bearing ore followed by gravity separation and then in 2010 additional cyanide based extraction process. To accommodate this latter addition to the process train, this report has considered two phases, namely;

- Phase 1 – Gravity based mineral processing; and
- Phase 2 – Cyanide based mineral processing.

Much of the engineering is similar between the two phases and is discussed as such, however where differences occur between the phases, these are noted. One key variation relates to production rate of the two concentration phases, such that Phase 1 has a nominal capacity of 5tph, whilst Phase 2 will operate at 10tph.

Tailings will be received from the gravity circuit (phase 1) or from a cyanide destruction process (phase 2) but in each scenario the solids concentration is likely to be similar. The relatively low rate of tailings production means it is necessary to dilute the received tailings with additional water sourced from the underground mine to enable effective pumping. This dilution will also reduce any residual cyanide concentration in the tailings following the cyanide destruction process, ensuring compliance with the International Cyanide Management Code (Cyanide Code).
The tailings will be pumped into previously mined areas where future access will not be required. Where necessary an engineered bulkhead will be employed to retain the placed tailings, ensuring their controlled separation from the operating mine. During the tailings placement water will either be displaced or released from the tailings slurry and this will be accommodated within the mine water management system.

Preliminary Tailings Management Design

Tailings Deposition Area Selection

An area has been identified within the mine for the placement of tailings as backfill and is indicated in Figure 12.4. The area is located at the lower end of the Target Block and extends from 280 Level through nominally to the 350 Level. The final elevation will be confirmed during later stage design work.

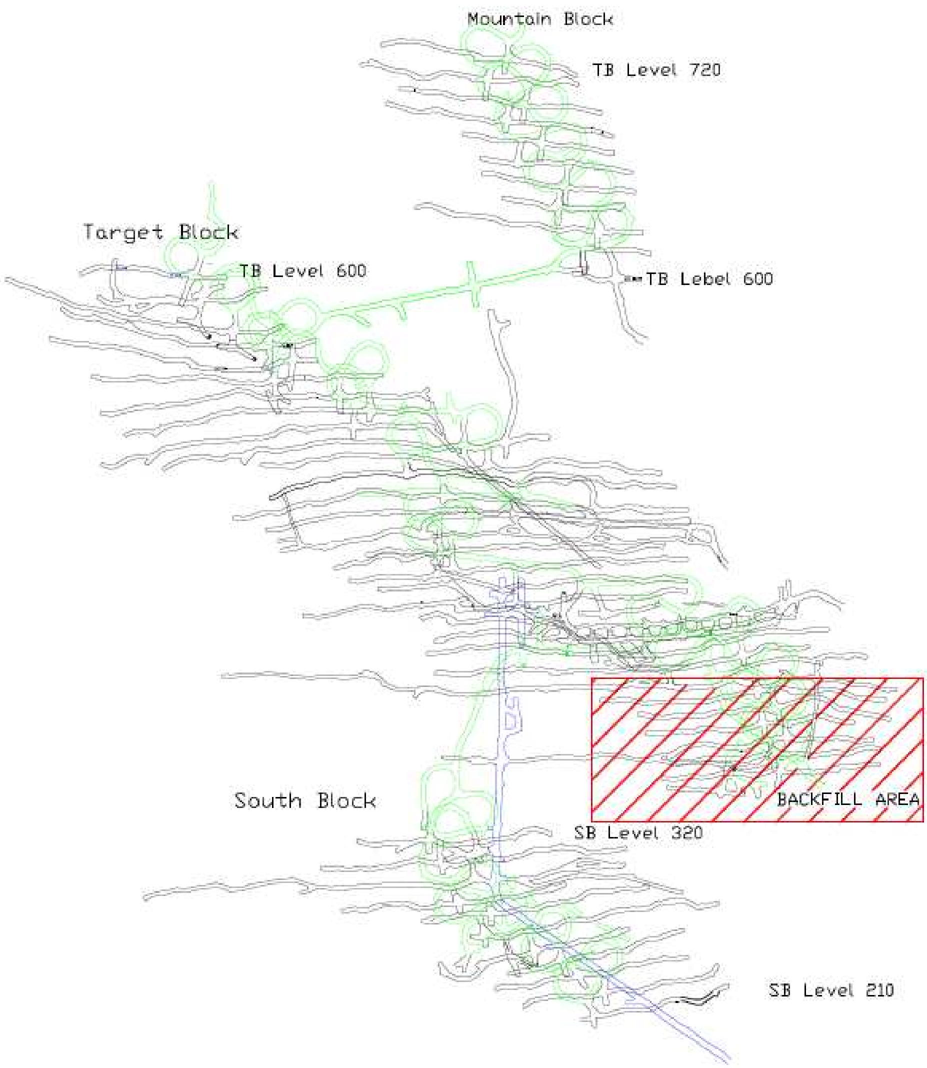


Figure 12.4: Long Section of underground mine showing tailings backfill area

The area identified is considered suitable for the following reasons:

- The area below the 300 Level can be used for deposition without the need for a bulkhead;
- The area is isolated, with access along a drive at the 300 Level and via the footwall ramp only, limiting the risk from inundation;
- Once above the 300 Level the tailings can be retained with a single bulkhead on the 300 Level access drive only;
- The ground below the filled area has no prospect of future mining owing to a fault structure, therefore there is no risk of sterilizing future ore;

- The area is close to the main concentration area and at a similar elevation, therefore minimizes pumping; and
- The area has been entirely mined and therefore no ore will be sterilized within the tailings.

It is estimated that the area identified, if filled to the 350 Level, would offer 44,500m^3 of storage capacity, equivalent to approximately one (1) year of operation at the higher production rate described in the following section, and shown in Table 12.1 below.

Table 12.1: Storage Capacities below 350 Level

Filling Area	**Stope Volume (m^3)**	**Sill Drive Volume (m^3)**	**Ramp Volume (m^3)**	**Total Volume (m^3)**
Below 300 Level	4,700	4,200	5,100	14,000
300 Level to 500 Level	13,600	8,000	8,900	30,500
Total	18,300	12,200	14,000	44,500

In completing the above storage assessment, the following assumptions have been made:

- All void space will be filled such that no future access to the area will be possible, including filling of the footwall decline ramp;
- The average true height of the mined out stopes is 1.7m;
- The sill drives have a cross sectional area of 10m^2; and
- The decline ramp has a cross sectional area of 25m^2.

An important assumption is the height of the stopes. A visual inspection of some stopes indicates areas of overbreak which could increase the average stope height, therefore increasing the available storage capacity. It is recommended that a full volumetric survey of the proposed filling area be completed to determine true stope volumes for mine planning and further detailed engineering design purposes.

Design Criteria

Further to information received from A&R, the tailings production for the respective operating phases will be as shown in Table 12.2.

Table 12.2: Mineral Processing Design Criteria

Criteria	Phase 1	Phase 2
Annual operating hours (hrs)	7,884	7,884
Solids throughput (tph)	5.0	10.1
Solids SG	2.9	2.9
Solids concentrations (wt%)	55	55
Slurry volume (m^3/hr)	5.8	11.8

Tailings Transportation

As highlighted previously, the anticipated production rates are low and therefore it is necessary to increase the flow rate through the addition of water to the tailings prior to pumping. Whilst the increase in volume to be transported will increase the operating cost, this is offset by the operational benefits of running a continuous system. The alternative would require campaign pumping of perhaps 20 minutes in every hour. To achieve this a storage vessel would be required to maintain the necessary buffer capacity and after each campaign the slurry line would need to drained and potentially flushed to avoid sanding in the line.

A&R has indicated it has an extensive inventory of 100 mm ND steel piping on site and therefore wishes to consider these as the media for slurry transportation. An alternative solution could be the use of a 75mm ND HDPE pipeline, this being the smallest practical line diameter. The two options are compared in Table 12.3 below.

Table 12.3: Tailings Pipeline Selection

Pipeline Option	Potential Advantages	Potential Disadvantages
100 mm ND Steel	▪ A&R has an existing inventory already on site ▪ Robust and common within the mine	▪ Potentially more labour intensive to install ▪ Poorer wear characteristics ▪ Greater water dilution required therefore greater operating cost
75 mm ND HDPE	▪ Can be easier to install if laid at the edge of a drive (with sandbags) ▪ Flexible enough to follow the profile of the decline without requiring couplings ▪ Better wear characteristics	▪ Will need to bought – no inventory available ▪ Less robust – can be damaged more easily by passing vehicles / rocks

Based on the above table, either option presents worthy merits. A third alternative maybe to use both steel and HDPE. The steel can be used in areas of the mine subjected to heavy traffic use, where its robustness and the option to hang it form the drive back would be of benefit. HDPE could then be used where the piping runs through lower traffic areas, notably within the footwall decline ramp adjacent to the identified tailings backfilling area. Traffic here would likely be limited to inspection work only and thus the HDPE piping could be laid on the floor of the drive with minimal protection. In addition, the flexibility of the HDPE would enable quick installation around the curve of the decline ramp.

Regardless of the line selection A&R make, the transportation of the slurry will comprise the same components. The slurry will be received either from the thickener, or from the cyanide destruction process at a suitably sized pump box. The required water addition to achieve the necessary flow volume will be introduced along with the tailings slurry to the pump box. Drawing from the pump box will be two centrifugal slurry pumps, one duty and one standby. It is expected that the necessary mixing of the slurry and the dilution water will be achieved in the pump box and as the mixture passes through the pump impeller, ensuring homogenous slurry leaves the pump discharge.

The modification to the flow volume will ensure the flow velocity within the selected pipe will exceed the critical settling velocity, therefore mitigating the

risk of sanding the line. Equally the flow velocity should not be so high as to increase the wear rate of the pipeline.
The preliminary pipeline routes are shown in Appendices 12.1 and 12.2 are identified as Route 1 and Route 2 respectively. Table 12.4 summarises key data for the routes.

Table 12.4: Key Pipeline Data

Key Data	Route 1	Route 2
Total Length (m)	602	814
Start Elevation (m)	312	312
Maximum Elevation (m)	312	362
Minimum Elevation (m)	293	293
Overall Elevation Variance (m)	19	19
Borehole requirement (m)	0	29

Route 1 presents an option for filling the stope areas below the 300 Level and would be consider viable only before the introduction of the bulkhead. The option takes the shortest possible route through existing infrastructure with a reasonably consistent gradient.

Route 2 is designed to accommodate filling from the 300 Level up to the 350 Level. The route selected requires the installation of a single borehole to connect from the 300 Level up to the main access ramp above. From here the piping would run within the main access ramp until it reached the Target Block decline. Following the target ramp decline down the pipeline would access each sub-level from where tailings would be discharged.

Based on the pipeline route selection, Golder has completed some preliminary hydraulic modelling based on the following assumptions:

- Design case represented by Route 2;
- A flow rate 42m^3/hr;
- A relative slurry density range between 1.2t/m^3 (21 wt% solids);
- Nominal 108mm ID pipeline (steel and HDPE assumed to be similar in characteristics);
- NPSH (Net Positive Suction Head) – 1m; and
- Pipeline routes as per Appendices 12.1 and 12.2.

Based on these assumptions, the following pumping characteristics and duties can be reported:

- Total discharge head (TDH) – 60m;
- Maximum pump flow rate – 42 m^3/hr;
- Anticipated pump selection – Warman 3 x 2 or similar; and
- Estimated operating power – 10 kW per pump.

Tailings Deposition

The tailings deposition will be a staged process, accounting for the period of time when deposition is below the 300 Level and then above it. This process can be described in five stages, namely:

- Stage 1 – Dewater the currently flooded area beneath the 300 Level and prepare the areas for receiving backfill;
- Stage 2 – Place backfill within the area below the 300 Level;
- Stage 3 – Before completion of Stage 2 the bulkhead along the 300 Level access drive should be constructed;
- Stage 4 – Commencing filling the area above the 300 Level behind the bulkhead; and
- Stage 5 – Complete filling to the designated elevation.

The area identified below the 300 Level is currently flooded and therefore this water should be removed in advance of tailings placement. The water will be pumped from the flooded area and directed to the mine water management system. Once dewatered, the area is to be inspected and prepared to receive backfill.

Tailings deposition is then commence. It is proposed that the tailings will be discharged from a single point from the sub-level immediately above the stope being filled. As a result of the need to dilute the tailings it is expected that there will be a brisk release of water from the deposited tailings, and this coupled with groundwater naturally occurring will result in the sub-aqueous disposal of the tailings. To optimise rapid settlement of the solids, the discharge pipeline should be extended as far into the stope (down dip) as possible. It will be necessary to review the discharge location to ensure even deposition of material across the strike length of each stope and maximise placement and backfilling of the old stopes voids. A number of discharge locations are to be considered for each sub-level.

It is anticipated that the area below the 300 Level will fill with supernatant water swiftly and as further tailings are deposited, the displaced water will be allow to travel down the 300 Level access drive where it will be intersected by the mine sedimentation system which is described further below. This operation will continue until such time as the area below the 300 Level is full and the water being displaced does not contain excessive suspended solids.

Prior to completion of filling below level 300 Level, it will be necessary to construct a bulkhead within the 300 Level access drive which will act to retain material once it is stored above the 300 Level. A description of the bulkhead is provided below. During construction it will be necessary to ensure dry working conditions within the 300 Level access drive and hence excess water within the lower levels will need to be mechanically removed with a pump rather than being allowed to simply be displaced by gravity. A submersible pump or similar may be employed to pump water via a pipe back to a suitable location where it can run back to the mine sedimentation trap whilst avoiding the bulkhead construction area. Once the bulkhead is construction is complete water can again run by gravity as the bulkhead will be design to allow the controlled passage of water through it.

Once the bulkhead is completed filling from the sub-level above the 300 Level can begin and in doing so the storage of tailings can advance above the 300 Level. The method for placement of tailings here will be similar to that described above; however the water management will vary.

Two principle options are available to manage the water. The first simply allows the water to fill up upon the placed tailings until such time as access to the deposition sub-level becomes compromised, at which time mechanical extraction of the water is required. A second, and potential more passive solution would be the installation of a decant (penstock) type system within the inclined stopes. The design will be such that is would discharge by gravity through the bulkhead to the 300 Level access drive.

A preliminary concept for such a solution would see a large diameter pipe pass through the bulkhead and then extend to the just below the first sub-level above the 300 Level. As tailings are discharged from this sub-level and the supernatant water level rises above the tailings, excess water would be collected within the decant, thereby preventing the water level from exceeding the level of the decant.

Once the tailings level reaches a height at which the supernatant quality is compromised, then the decant is to be extended to the next sub-level and the former sub-level abandoned. Tailings deposition would then commence from the new sub-level and thus the process would be repeated. The system would also ensure that the maximum height of supernatant water would be controlled by the height between the sub-levels.

Once filling of the entire area is complete, then the decant pipe should is cut off as close to the tailings level as possible, thereby preventing the development of excessive standing water in the placed tailings in perpetuity.
The design and specification for such a decant system are not within the scope of this study and therefore they should be developed in the next stage of the design.

Bulkhead Design

The bulkhead will be installed on the 300 Level drive to isolate the tailings storage from the 300 Level portal drive. The bulkhead will be installed along the 300 Level exploration drive, although the exact location of the bulkhead will be determined following examination of the ground conditions within the drive. The bulkhead is the primary engineered structure retaining the tailings based backfill with residual water in the older stoping area as previously discussed.

The design of bulkhead requires consideration of a number of elements to ensure its appropriateness as an engineered structure to control the forces acting on it and the environment in which it is constructed and in contact with. Thus, a range of factors are considered in its design.

Design Considerations

The potential failure mechanisms for bulkheads retaining the unconsolidated fill (tailings) are described below.

- Shear failure of the bulkhead and rock interaction as a result of pore pressures resulting in shear through the rock, through the concrete, or along the rock-concrete interface;
- High hydraulic gradient near the bulkhead resulting in leakage through the rock mass;
- Stress redistribution (or induced) stresses created as a result of the excavation around the drive perimeter; within the immediate rock itself;
- Hydro-fracturing mechanisms can pose a potential problem under some circumstances if the fill pressure exceeds the minor principal stress in the rock mass away from the immediate influence of the drift (i.e. that is not subjected to consolidation grouting), and this results in the opening or creation of fractures that transmit water and tailings to the downstream side of the bulkhead; and
- Chemical attack may reduce the integrity of concrete over time.

These aspects have been reviewed during the preliminary design determined for the NGM tailings deposition.

Design Parameters

The bulkhead is planned to be located beneath a 35° inclined stope and will be constructed to retain tailings up to a vertical height of 50m.

The following design parameters have been considered for design of the bulkhead:

- The drift has a rectangular geometry 3.5m wide by 3m high (Area = 10.5m^2 and Perimeter = 13m);
- Nominal vertical height of fill to bulkhead ≅ 50m;
- Tailings head (assumed saturated unit weight = 0.023MN/m^3) at bulkhead location = 1.15MPa

- Vertical Rock Cover ≅ 100m;
- The topography above the bulkhead is at a slope of approximately 20°;
- Maximum acceptable hydraulic gradient for longevity = 12 (~118kPa/m), corresponding to about 2(RMR – 5) = 2(64 - 5);
- Conservative rock-concrete interface shear strength of 0.15MPa for rock-concrete interface without grouting; and
- Overall Safety Factor of at least 3.0 required for permanent bulkhead longevity.

Rock Mass Description

The main orebody consists of quartz veining ranging up to 1m in width, located within predominant lithologies of mafic metavolcanics and dolerites. The orezone has an average plunge of 35° to the southeast, and is located beneath the east dipping face of a glaciated valley.

Based on the previous Golder report, the rock mass quality for the host mafic volcanics was assessed as having a mean RMR (Bieniawski, 1976) of 64 and values ranging from 56 (worst case) to 72 (best case), which classifies as fair to good rock mass quality. The intact rock shows an average uniaxial compressive strength (UCS) of 131MPa.

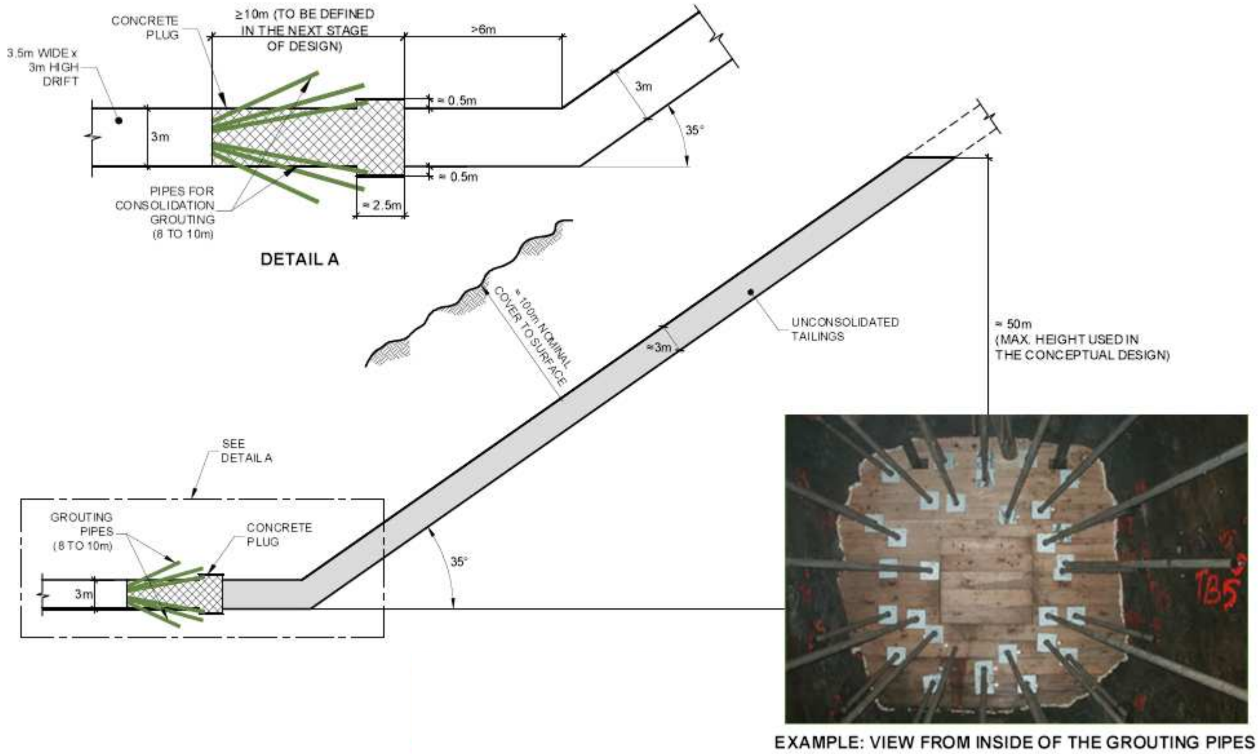
Mapping of the rock masses in the vicinity of the planned bulkhead site will be required so that the Nalunaq mine can assess the rock mass quality, rock mass fabric, and estimate the intact rock strength. The Norwegian Geotechnical Institute's (NGI) Q-System and Bieniawski's Rock Mass Rating system, RMR76 (Bieniawski, 1976) should be used to classify the rock mass quality. Field assessment of the intact rock hardness should be made according to guidelines described in ISRM, 1981.

Preliminary Bulkhead Design

A parallel (monolithic and "unhitched") type of bulkhead is considered to be most appropriate for this preliminary design at NGM.

The main bulkhead design parameter is the length of concrete that needs to be cast in the drift to ensure a stable structure. It must comply with all conditions discussed previously in this section; the selected length for the parallel bulkhead will be that which corresponds to the most critical of these five considerations.

Figure 12.5: Preliminary design concept for a bulkhead for tailings backfill



The design details and consideration will be advanced to a detailed design requirements as further work and data is assessed for the NGM as per previous discussions regarding the commencement of operations. In this further work, full design details and considerations will be made available for review presentation. For indicative purposes only at the stage in design, Figure 12.5 provides some details on the design concept. Based on the previous Golder report, there are no in situ stress measurements at the Nalunaq mine. The initial in situ stresses are assumed to be represented by the ratio of horizontal stress to vertical stress (k) of 2. The vertical stress is assumed to be equal to the unit weight (0.027 MN/m^3) of the rock, multiplied by the depth below surface. In addition, simplified two dimensional numerical analyses indicated that simulation of unloading due to glacial melting and erosion might have caused a rotation of the in situ stresses, with the projected horizontal component being equal to 0.9. Additional numerical analyses will be required during the next phase of the bulkhead design to better estimate the induced stresses around the bulkhead.

Re-handling of Tailings

It is understood that prior to the installation of the phase 2 mineral processing circuit, a portion of gold will be retained within the tailings and thus will need to be reintroduced to the concentration circuit once the phase 2 circuit is in operation. To facilitate this, the temporary storage of tailings will need to be achieved in such a way that it can be extracted and returned to the circuit.

The material will be stored either in a new excavation specifically designed for the purpose, or in an existing drive. Ideally the location will be a slight to moderately declining drive capable of containing approximately 15,000 m^3 of material assuming storage for three months of operation.

Once the material has been extracted and reprocessed, the excavation could act as an intermediate and emergency sedimentation trap, and would become part of the mine water management system or if planned appropriately form any other part of the future mine development planning and operational requirements.

Long-term Life of Mine Storage

It is estimated that the currently identified storage area would offer approximate two years of capacity at the proposed extraction rate. Additional capacity will be required in the future and additional areas can be identified with similar characteristic to that of the currently proposed option and these should be incorporated in the overall life of mine planning and design.

Underground Water

Mine water management at NGM focuses on managing the quantity and quality of water within the mine and exiting the mine. The inclusion of an underground concentration plant presents the opportunity to recycle ground water efficiently within the underground environment, minimising the quantity of water abstracted from other sources, and also the amount of water discharged from the mine.

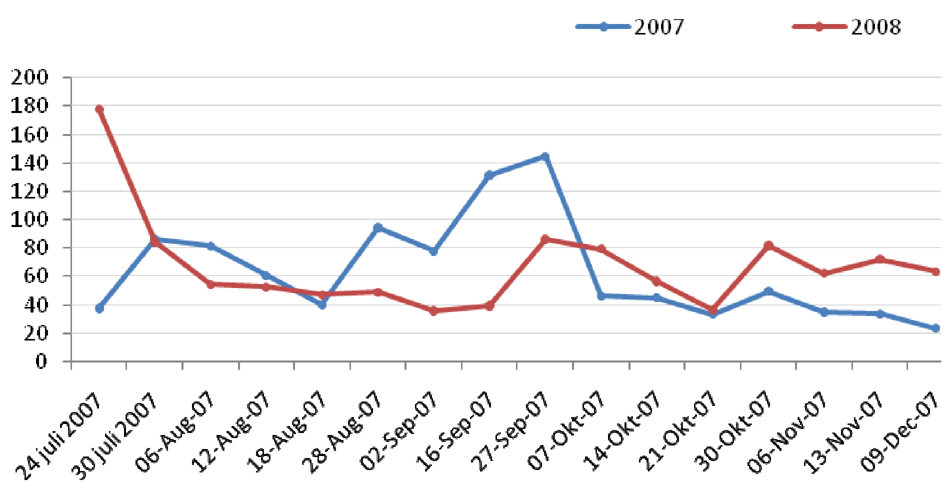
The water management focuses around a central water collection and sedimentation pond, where water from within the mine can be retained and re-circulated for use either in the mine, for the process or discharge externally. The correct design and sizing of the system will minimise suspended solids, thereby leading to an improvement in the current arrangement. This controlled management of the water within the mine, including specially design sedimentation traps will be used to minimise sediment loading within water discharged from the mine.

Conceptual Water Balance

A conceptual mine water balance has been developed to determine the average flows into, within and out from the mine. Using this water balance it is possible to develop preliminary estimations of where the water will flow within the mine and necessary controls and management required.

The water balance has been developed through limited site data, information from A&R and a series of assumptions. The site data was limited to two years of monitoring of mine water outflow. The water flow was observed within the valley beneath the 300 Level portal, and recorded using a V-notch weir. The results from that monitoring are shown on Figure 12.6 below.

Figure12.6: Mine Water Flow Data



Based on the above data, an average flow from the mine equalled 64m^3/hr, which includes all natural groundwater inflows and operational uses such drilling water. Based on this information, the mass balance data from A&R and an understanding of the future mining method and equipment operation, the following assumptions have been developed and used in the water balance modelling:

- Natural ground water inflow (average) – 50m^3/hr;
- Equipment water – 15m^3/hr;
- Other mine operational use (e.g. washing faces) – 10m^3/hr;
- Recycle water for processing – 36m^3/hr;
- All process water will report to tailings;
- Tailings will consolidate to 80wt% solids in the stopes;
- All water will be directed to the 300 Level; and
- Excess water will be released from the mine following sedimentation.

Based on the above assumptions, the preliminary water balance has been developed and is shown in Figure 12.7 below.

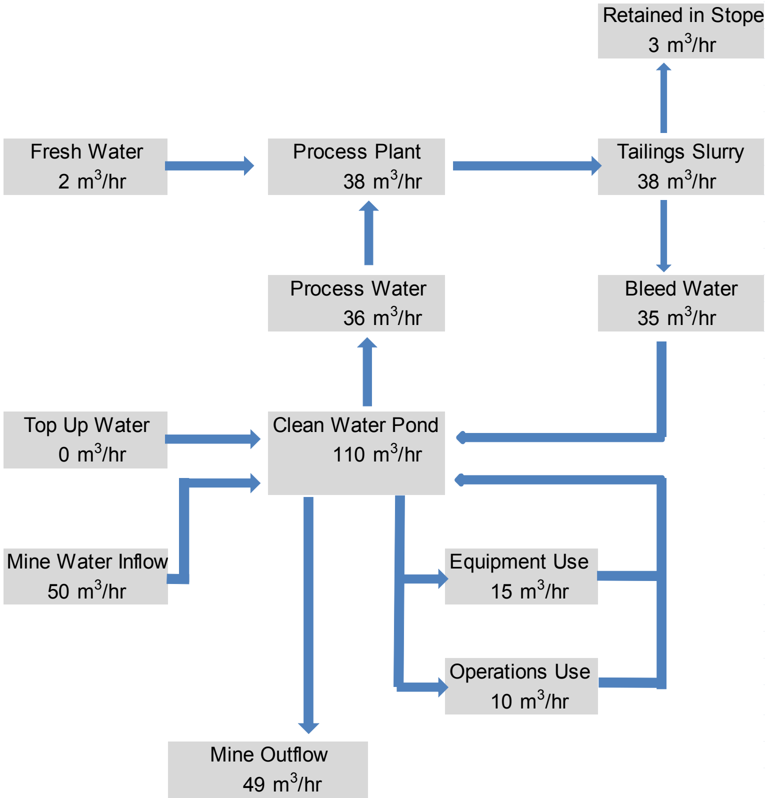


Figure 12.7: Preliminary Water Balance

From the model it is clear that within the mine there will be a recirculation of water, leading from the clean water pond to the concentration circuit, to the tailings and then back to the clean water pond. Within this circulation, it is assumed that only water retained within the tailings will be lost from the circuit. Additionally the model assumes that all water used within the mine for drilling etc will return to the clean water pond without loss. Consequently the mine water inflow approximately equates to the mine water outflow, less the water held within the tailings.

It should be noted that the model makes not allowance for the attenuation of water within mine between source and receptor. For example the model assumes that water used by a drill rig will immediately report to the clean water pond. This may have implications when considering short times frames of several hours where water may be extracted from the clean water pond but may not return for 24 hrs, leading to a potential for short-term draw down of the clean water pond. This issue would be mitigated by the inclusion of "top up" water which could be abstracted to cover any short-term issues that may occur infrequently in the operation.

Water Handling Systems

The water handling system is developed around a central mine water collection and sedimentation pond. The objective of this will be to retain adequate water within the mine to avoid the need for additional water to be pump in from the existing borehole located in the valley, but also to minimise suspended solids, the latter being essential for both use with equipment underground, and the responsible release of water to the environment when discharged from the mine as may be required.

Appendix 12.3 shows the proposed design for the water handling system. Water enters the system at the mouth of the declining ramp. Once within the large body of very slow moving water the sediments will fall from suspension and be retained at the base of the ramp. Water is then extracted from the sump at the far end of the decline, minimising disturbance to the settled solids. At times it is assumed the solids will need to be removed, and under these conditions the water level would be drawn down to allow the removal the settled material. This material would be re-handled within the processing plant, thereby ensuring its eventual storage within the tailings. This "clean up" process would likely be completed within the normal maintenance shut down period and thus would not impact on the operation of the mine. Alternative a second identical system may be installed allowing the first to be taken out of service without compromising the mining or concentration processes.

It is proposed that the water handling system be developed along the 300 Level portal drive, with all water from upper levels being directed to this level. Using the above water balance it is possible to estimate the minimum required retention time in the sump to ensure adequate removal of suspended solids. Golder has assumed an eight hour residence time as adequate for this purpose, and thus at the average inflow rate to the trap predicted by the water balance, an overall capacity of 884m^3 is required. The design presented on Appendix 12.3 provides for a capacity of 1000m^3.

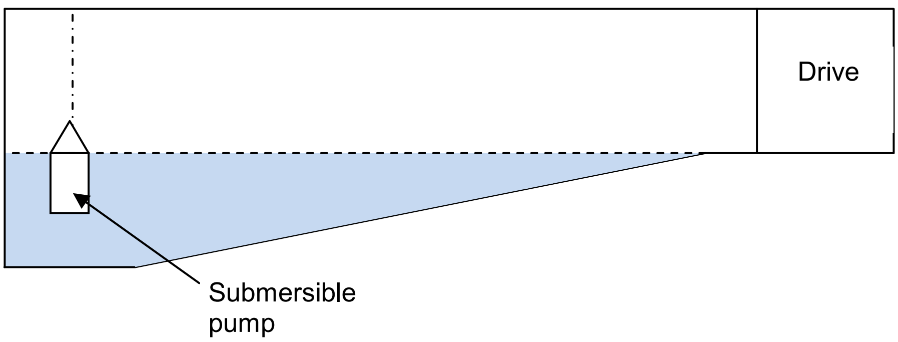
The data and anecdotal evidence suggests that high flows are expected during the spring thaw. In light of this the necessary capacity and / or number of systems installed should be reviewed during further detailed design and ongoing operation. Effort should also be placed on developing a water monitoring system. Such as system is described further below.

Mine Water Control and Discharge

As indicated by the preliminary water balance, it is anticipated that water will continually be discharged from the mine as is currently the case. The approached described herein is how NGM will manage the discharge of that water, which is something that is considered to be an improvement on the previous operation at NGM.

Water has been observed to exit the mine on at least two levels; however the majority discharges through the 300 Level portal. Through proper control of water flowing within the mine, water will be managed such that it is discharged through a single portal only, namely the 300 Level. The underground mine water management system is designed to collect water close to its source in small inter-level sediment traps (sumps), possibly on each sub-level. A simplified sketch of such a sump is shown in Figure 12.8 below for information.

Figure 12.8 – Inter-level Sedimentation Sump (Typical Cross Section)



The location of each sump can be determined after NGM complete a comprehensive survey to identify the sources of major water course entries to the mine. Conceptually a suspended submersible pump would transfer collected water to the central water handling system on the 300 Level, however the details for achieving this not provided for in this report. The option to utilise gravity may be an option, but where possible water should be run in pipes from each sedimentation sump, allowing the mine control over the water route and eventual discharge location.

Once water enters the central water handling system, suspended solids will be removed. The water balance confirms that there will be an excess of water and thus an overflow from the central water system will be necessary. This overflow is currently predicted to be approximately $50m^3$/hr, a reduction on the currently estimated discharge from the mine. Water will overflow from the sump end of the central water system ensuring it passes through the sedimentation process, and will pass immediately into a culvert or closed pipe to control its flow to the portal. Once at the portal, the water flow would enter an oversized pipe to control its flow into the valley floor, where it will be discharged in to the existing arrangements that NGM has in place.

Monitoring and Emergency Response Planning

As described previously, the current records for water entry into and out of the mine are scant. A new and robust system is recommended to be developed to monitor where and how water enters the mine, the flow across all seasons and finally the quality of the water as it exits the central water system. This monitoring should at least record the suspended solids content, pH and electrical conductivity. In addition, a regime should be developed to monitor the cyanide levels within supernatant water released from the tailings which should complement the normal monitoring of cyanide within the concentrator.

It is not known at the time of reporting what emergency response plans are in place at the mine, however it is important the that the water management is considered within in these as well as on its own. Issues to be considered may include:

- Unexpected inflows of water to the mine, inundating the central water handling system;
- Excessive sediment release from the central water system;
- High cyanide levels within the water when operating the CIL process; and

- Portions of the mine wide water system freezing.

Ventilation

Introduction

A review has been undertaken of the existing ventilation system at the NGM. This is based on the ventilation survey undertaken when the mine was in full scale operations in November 2008.

The following methodology is the basis of an effective underground mine ventilation system and the basis of this work. Flow through ventilation is the main ventilation circuit for the NGM. Fresh air enters the mine from surface via a shaft, ventilation raise or adit. The fresh air is distributed through the mine via internal ventilation raises and ramps, and flows are controlled by regulators and mounted ventilation fans. An auxiliary ventilation system takes air from the flow-through system and distributes it to the mine workings via ventilation fans, venturis and disposable fabric or steel ducting. The key requirements of a ventilation system is to maintain an appropriate volume and flow of fresh air to the working environment underground and thus to ensure dirty air is exhausted in a controlled fashion out of the underground mine.

This report is a technical review of the ventilation system at the underground mine. Specifically the review has included the following elements:

- To review of the status of the current ventilation system for maintaining adequate ventilation in the mine;
- To determine the required ventilation system for the proposed mining blocks; and
- To provide general considerations for future mining activity including the proposed mineral processing facility.

The ventilation review focuses on the three proposed mining blocks, referred to as:

- ‘A-B’ Southern Mining Block (SB) - down dip of general operations;
- ‘C’ Mountain Block (MB) - up dip of general mine operations; and

- 'D' Target Block (TB) - up dip of general mine operations.

These areas can be identified in Figure 12.4 and it is based on the data provided by A&R. A long section of the NGM with the salient points of

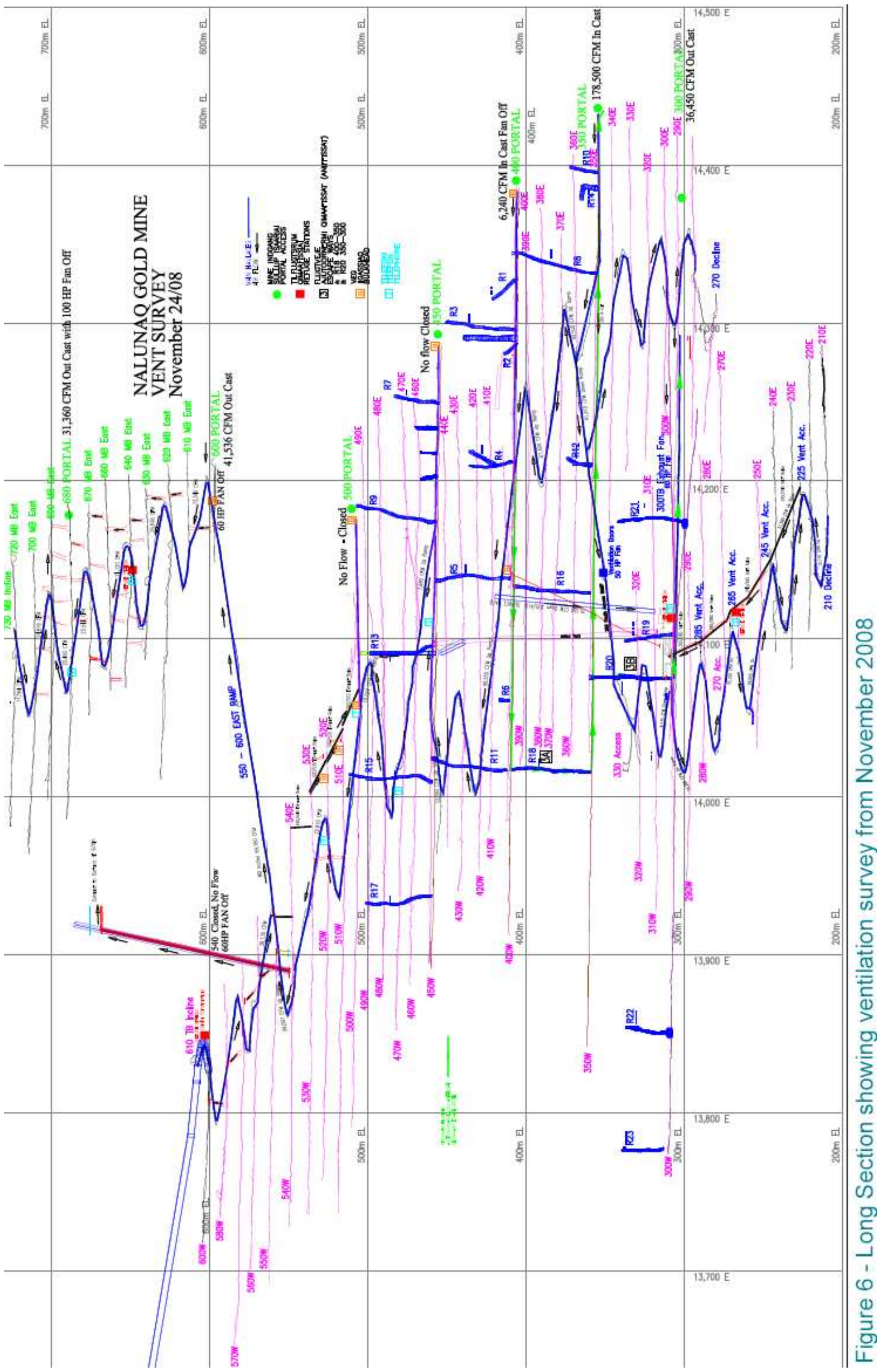


Figure 12.9 Long Section showing Ventilation Survey from November 2008

the ventilation survey from 2008 is shown in Figure 12.9. The proposed mining plan and mining equipment to be used underground in the operations start up in 2009/2010 has also been incorporated in to the review.

For the purposes of maintaining an appropriate and recognised standard of underground mine working environment the ventilation requirements have been based on those detailed in the Canadian Ontario Mining Regulations (1990) for underground mines.

Diesel Equipment

Diesel equipment is of prime significance to generating an understanding of the ventilation demand in the NGM operation like all other mines. Table 12.5 summarises the proposed mobile equipment NGM intends to use in the operation. This data is the basis for the diesel equipment analysis in the ventilation demand assessment.

<u>Table 12.5: Diesel equipment, power rating and air volume requirements</u>

Equipment		**Kilowatts**	**M^3 per second**
Model	**Type**		
JS 350	Loader	136	8.2
JS 220	Loader	75	4.5
JDT 413	Haulage Truck	102	6.1
JDT 415	Haulage Truck	136	8.2
MJM 21B	3 Boom Jumbo	69	4.1
MJM 20B	2 Boom Jumbo	46	2.8
M5700DTC	Kubota Tractor	43	2.6
Longhole Drill	Basket	Drill	2.8

Note: Diesel equipment information received from NGM.

To obtain ventilating air volume requirements for diesel equipment, the Canadian Ontario Mining Regulations (1990) were used (i.e. 0.06 m^3 per second per kilowatt).

Diesel Equipment Requirements per Mining Block

The mine plan at NGM is intending to work the three mining blocks previously discussed. A number of diesel powered mobile mining equipment is to be utilised for the mining operations. For any given mining block activity the equipment to be utilised and associated ventilation requirements are estimated in Table 12.6.

Table 12.6: Diesel equipment and airflow per mining block

Volume Equipment Model and Type	Number of Units	Total Air Required (M^3/sec)
JS350 Loader	2	16.3
JDT 415 Haulage Truck	2	16.3
MJM 21B Drill Jumbos (3 Boom)	3	12.4
Total Mining Block Air Volume Requirement	**-**	**45.1**

On the basis that the equipment listed above is the maximum utilised in any mining block and a further two trucks are utilised to provide ore haulage from the mining block to the mineral processing facility an additional air flow volume is required as below:

- JDT 415 trucks x two (ramp haulage) = 16.3m^3/sec

Thus a total ventilation requirement is estimated at 61.4m^3/sec for the underground mine.

The mine ventilation survey data reported by NGM in November 2008 details a total air volume in to the underground mine of 87.3m^3/sec. As such there is an acceptable and adequate level of air flow into the mine with the current system for the proposed mining plan.

The intent is to mine the three mining blocks consecutively as resource and mine planning indicates this to be an efficient approach. It is worthwhile to note that if future mine planning recognised a need to work the three mining areas at the same time with three times the amount of equipment in each block a total ventilation volume demand rises to 134.5m^3/sec plus 16.5m^3/sec ramp haulage giving a total demand of 151.0m^3/sec.

The current ventilation system should be maintained efficiently to minimise energy consumption and maximise its effectiveness in the underground operations. This can be maintained by:

- Periodic ventilation surveys;
- Planned ventilation control inspections and maintenance programmes;
- Erecting ventilation brattice or regulator controls on all disused drives and stoping areas to prevent short circuiting of any components of the system; and
- Erecting controls to ensure the intake (fresh) air circuit is separated from the exhaust ventilation circuit.

If further volume requirements are recognised through refinements in mine planning or a need to operate additional development headings additional options to those described above can be implemented in the underground operations to increase capacity of the system. These include:

- Portals from 450 to 500 Level be closed for airflow, unless required for access (which should be controlled to minimise circuit losses);
- Seal all inactive levels, if there is any losses of air from the ramp; and
- Run the fans on the 600 Level and/or the 680 Level portals (turned off in the November 2008 survey), to increase the exhaust air from the mine. Then taking volume and pressure readings on these fans to incorporate in to the overall ventilation system demand and supply balance.

The following points have been considered in this preliminary design review of the system:

- The 350 Level portal to ramp is clear for airflow;
- The 400 Level portal to ramp is clear for airflow;
- The main ramp is connected to 350 Level;
- Mining will be carried out in the A-B block, C block and D block only;
- The mine intake (fresh) air volume is presently supplied from 350 Level ramp portal, with a minor volume from the 400 Level ramp portal;
- The November 2008 survey data tabulated presents a discrepancy between intake and exhaust volumes in the survey; and
- The largest power rated requirements for a given piece of diesel equipment was used to determine air volume requirements.

Overall Ventilation Requirements for Mine Production Areas

Ventilation for 'A-B' Southern Stoping Block

To provide sufficient ventilation volumes for mining the Southern stoping block, a total of 61.4m^3/sec is required to be delivered by downcasting the ramp from 350 Level. There are two options discussed below to manage the ventilation requirements.

Option 1

To down cast 61.4m^3/sec on the ramp it will be necessary to exhaust the 225 Level through the 225/300 ventilation raise to the 300 Level and exhaust the ramp portal on 300 Level. This would require the installation of exhaust fans at the top of the 225/300 raise on 300 Level (fan sizes to be determined based on raise size). Level connections to the 225/300 ventilation raise between 300 and 225 Level must be sealed. Install two 1.2m diameter 113kw auxiliary ventilation fans on the ramp just above the 225 ventilation access and duct with 1.2m diameter ducting from each fan down the ramp and into the Southern stope block. This would require rigid wall duct with 1.2m diameter 113kw booster fans on each line, every 245m. These fans would supply a design air volume of 61.4m^3/sec to the southern stoping block and the ramp for truck operation. This option assumes the mineral processing area is ventilated from the Number 20 raise. The details of this option can be seen in Appendix 12.4.

Option 2

If the surface ramp is developed to 210 Level before mining commences, then the surface ramp from 210 Level can be used for exhaust air rather than the 225/300 raise. This would reduce the system resistance considerably and therefore lower fan horsepower requirements (i.e. install 1.2m fans at the entrance to the stoping block and therefore no fans and ducting required on the ramp). Exhaust air fans should be installed at the proposed surface ramp portal (two 1.2m diameter 113kw) fans to exhaust 61.4m^3/sec). All connections to the 225/300 ventilation raise should be sealed with brattice work or other appropriate controls. The details of this option can be seen in Appendix 12.5.

The following assumptions have been made regarding ventilation of this block:

- All other mining areas are idle;
- The largest power rated requirements for a given piece of diesel equipment was used to determine air volume requirements;
- The top of the 225 ventilation raise breaks into 300 Level; and
- The proposed ramp from surface will break into 210 Level.

Ventilation for the 'C' Block (Mountain Block) Mining Area

To provide sufficient air volumes for mining the 'C' block, a total of 61.4m^3/sec is required to exhaust the 720 Level. There are two options discussed below to manage the ventilation requirements.

Option 1

To provide sufficient air for mining the 'C' stoping block, install a 3m^2 raise from 720 Level to surface (27m perpendicular distance) and exhaust through this raise. It will be necessary to install an exhaust fan at the raise bottom on 720 Level and seal the 680 Level ramp portal. The 75kw fan on 680 Level at the ramp portal may be sufficient for this purpose and could be relocated to the bottom of the new raise. This fan should be started in the near future so as to permit the collection of volume and pressure readings and thereby enhance the ventilation data available as the bench mark for the NGM. Two 1.2m diameter 113kw fans should be installed at the ramp top on 720 Level (MB) and run with twin 1.2m diameter ducting into the stoping area. The details of this option can be seen in Appendix 12.6.

Option 2

Install two 1.2m diameter 113kw fans just below 680 Level and run 1.2m diameter rigid wall ventilation ducting from these fans into the 'C' stoping block on 720 Level. Booster fans ((113kw) will be required every 245m on each line. Airflow is to exhaust back down the ramp to 680 portal.

In this case a design air volume of 61.4m^3/sec is required to ensure sufficient air for trucks on the ramp above 680 Level. The 75kw fan presently on 680 Level may be suitable as it is currently reported as not in use.

An airtight brattice work ventilation control is also recommended to be installed on the 610 incline (TB) to ensure sufficient air volume upcasts the ramp from below.

As per the November 2008 ventilation survey, there is sufficient air volume on the 730 (MB) incline to complete development of the exhaust air raise. The details of this option can be seen in Appendix 12.7.

The following assumption has been made regarding ventilation of this block:

- All other areas in the mine are idle; and
- There are presently no raise breakthroughs from 720 Level (MB) to surface.

General Ventilation for the 'D' Block (Target Block) Mining Area

To provide sufficient air volumes for mining the 'D' block, a total of 61.4m^3/sec is required. There are two options discussed below to manage the ventilation requirements.

Option 1

To provide sufficient air volumes for mining the 'D' block a drive to connect the 610 (TB) incline to the 540 raise (in the refuge station area) is required to be developed to draw air up the 610 (TB) incline and exhaust to the 540 raise. The 75kw fan presently on the 540 raise may be adequate for this purpose. As commented previously, it is expedient to operate this fan as soon as possible and measure fan air volumes and pressures to provide confirmation for detailed mine planning and ventilation requirements in the future. Airtight ventilation brattice work should be installed on the 550-600 east ramp just above the 610 incline connection to ensure an upcast on the main ramp from below the 610 connection. Two 113kw fans should be installed on the 610 ramp just below the proposed connection drive with twin 1.2m diameter ducting into the stoping area. The details of this option can be seen in Appendix 12.8.

To develop the connection drive, a 30kw fan could be installed on the 610 incline just above the proposed drive and run a 0.9m diameter ducting into the drive. This would provide sufficient air.

Option 2

Install two 1.2m diameter 113kw fans just below 610 incline, on the main ramp, and run 1.2m diameter rigid wall ventilation ducting from these fans into the 'D' stoping block. Booster fans (113kw) will be required every 245m on each line. In this case a design air volume of 61.4m^3/sec is required to ensure sufficient air for trucks on the ramp incline. The airflow (61.4m^3/sec) will exhaust back down the ramp and to 680 portal. The details of this option can be seen in Appendix 12.9.

The following assumptions have been made regarding ventilation of this block:

- All other areas in the mine are idle; and
- There is no raise breakthrough at the top of the 610 ramp (incline) to surface. NGM should confirm there is no surface system connection at the top of the 610 (TB) incline;
- Surface photographs show a 670 ventilation raise breakthrough to surface. This raise may be at the top of the 610 (TB) ramp incline, in which case it may be used as an exhaust raise rather than developing a connection drive to the 540 raise. This is to be investigated further by NGM as the mine planning details develop; and
- As per the November 2008 air survey (17.9m^3/sec), there is sufficient air down casting the 610 (TB).

Typical Mining Block Ventilation

To ventilate a typical mining block 56m^3/sec is required. This includes a factor of 25% to account for leakage and losses. This equates to the installation of two 1.2m diameter 113kw fans in the upstream side of the mining block and running 1.2m diameter ducting from the fans. Each ventilation duct line will supply one-half the total air volume. Each 1.2m diameter ventilation duct can be split into three headings using ventilation duct laterals (i.e. routing of ducting dependent on sequence of mining).

Thereby giving the capability to ventilate six headings with fresh air. It makes intuitive sense that if fewer headings are worked at any point in time the local circuit may be made more efficient by not leaving fans on to unused headings or working areas. Conversely if more headings are worked consecutively then more fresh air intake is required in to the area.

Ventilation controlling brattice work should be installed in connection slots as required, to direct airflow through the active areas of diesel mobile equipment operations. This requires frequent inspections by appropriate personnel to ensure adequate ventilation of mining areas as is reasonable and expected in the mining operation. The details of this option can be seen in Appendix 12.10.

Mineral Processing Ventilation

The development of the additions mine openings for gravity phase of the mineral processing location will require 40 horsepower (30kw) fan at the bottom of the number 20 raise on 300 Level and ducting from the fan into the crushing/mill area. The fan and ducting will supply 15.9m^3/sec to the crusher area as per 300 to 400 Level plan. Airflow must exhaust to the 300 Level portal.
A dust collector (bag house) will be required in the crushing area with a pickup point over the crusher. Air exhausted from the bag house can recirculate in the crusher room.

The cyanide mineral processing phase in 2010 will require exhausting the air to the 300 Level portal to mitigate the potential of minor cyanide processing associated fumes escaping into the mine atmosphere if the pH of the leaching process drops. Air monitors are to be installed in the mineral processing vicinity to ensure detention of any such conditions is promptly made and mitigated. This area must have a continuous supply of ventilating air of approximately 2.4m^3/sec. Procedures for remedial action are to be developed and implemented if cyanide processing associated fumes are encountered.

If measures are to be taken to control the release of HCN gas into the mine environment, the waste pass can be used to draw (fresh) air from the 300 Level portal and through the mineral processing area. This will then become exhaust air after passing through the area. An exhaust fan will be required at the top of the waste pass to exhaust 8.3m^3/sec. This will then require appropriate ventilation controls to ensure the exhaust air from the mineral processing chamber is then channelled through a controlled exhaust system out of the underground mine.

In the case of the waste pass being utilised as the exhaust (and drawing in fresh air from the 300 portal) the 225/300 raise must be used as an intake raise with the two 1.2m 113kw fans on the ramp at the raise bottom to be installed in an airtight wall. These will draw fresh air down the 225/300 raise from the 300 Level portal. The air will need to be ducted with two rigid wall ventilation ducts, with booster fans every 245m on each line into the lower mining blocks. Air is to exhaust through the main ramp until such time as the planned lower mining block surface portal is installed which can then be utilised as an exhaust.

The fan sizes required will be determined for the 225/300 raise and the waste pass based on the geometry of these raises. It is likely the waste pass exhausting a small volume of air would require a fan in the size range of 0.9m diameter 5.3kw. Details of this option can be seen in Appendix 12.10.
The limited amount of electro winning that will take place with the mineral processing activity in NGM is undertaken in a controlled environment whereby the fumes are exhausted in to a controlled and sealed conduit. This conduit should be connected to ventilation duct and an exhaust fan to ensure the electro winning fumes are force ventilated out of the mine through the exhaust ventilation circuit.

Ore Pass Ventilation

Install a 0.9m 22.5kw fan on the main ramp just west of the 450 Level breakthrough and run a 0.9m diameter duct into the orepass access drive. This will supply 14.2m^3/sec to the orepass drive. If the orepass dump is on 450 Level, a fan and duct are not required.

Lower Mining Block Development Ventilation

Install a 1.2m 75kw fan 9m from the ramp entrance and run 1.2m diameter ventilation ducting from the fan down the ramp for development mining. This will supply 18.9m^3/sec to the face, which is sufficient air for a loader and truck. Details of this option can be seen in Appendix 12.11.

Recommendations

The following recommendations are provided on the basis of the analysis undertaken and the discussion presented in the previous sections of this report.

Southern 'A-B' Mining Block

Develop the proposed ramp from surface and install two 1.4m 56.3kw exhaust fans at the portal to exhaust air from the south mining block to surface.

Mountain 'C' Mining Block

Develop a $3m^2$ raise from 720 Level to surface and install two 1.4m 56.3kw fans at the bottom of the raise to exhaust air to surface.

The 75kw fan at the 680 portal may be adequate and could be relocated. This fan should be operated to permit the acquisition of volume and pressure measurements as soon as possible to support any decision on its suitability.

Target 'D' Mining Block

Develop a connection drive from the 610 ramp (beginning of the 'D' block mining area) to the 540 raise and install two 1.4m 56.3kw fans in the connection drive. This will up-cast the 610 incline and exhaust into the 540 raise to surface provided that there is no raise breakthrough at the top of the 610 ramp incline. Note: The 75kw fan presently at this raise may be adequate (volume and pressure measurements required to confirm suitability).

Crusher Ventilation

Intake air to be supplied from the fan at number 20 raise and duct in the crusher area. Air to exhaust to the 300 Level portal where $8.3m^3/sec$ is required. The fan should be started and volume and pressure readings taken to confirm suitability.

Geotechnical

Introduction

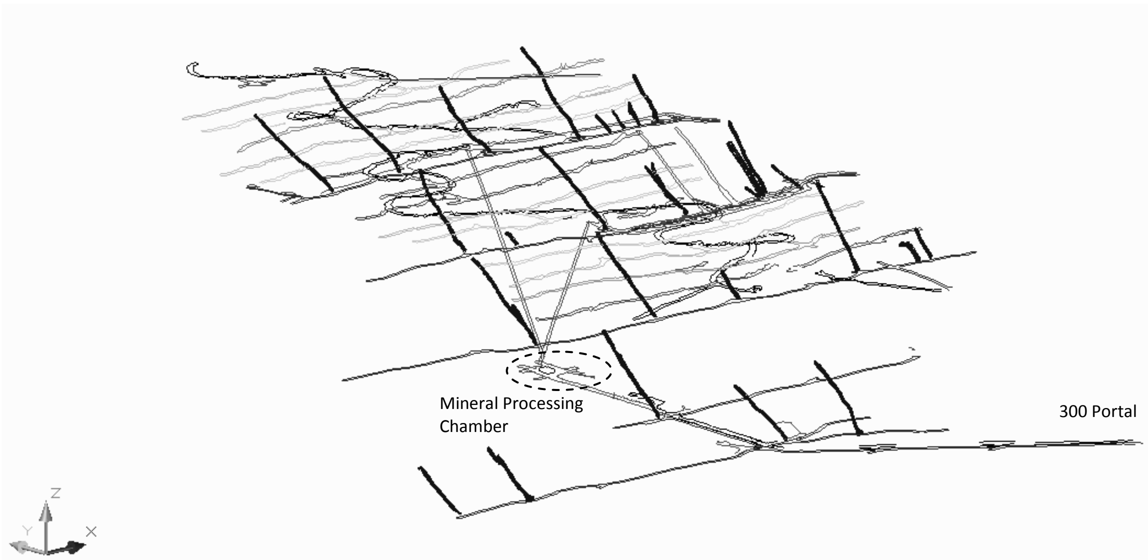
The mineral processing plant and operation requires additional excavation space in the NGM.

Based on the preliminary design of the mineral processing excavations an appropriate level of geotechnical analysis has been undertaken to assess the ground control requirements for the excavation. Preliminary confirmation is provided that excavation sizes are permissible and ground control requirements have been provided to manage risk to an acceptable level.

Mineral Processing Chamber Excavation

The mineral processing chamber consists of a range of existing and new drives to be developed in the NGM. The mineral processing area in the NGM can be seen in Figure 12.10. It is approximately 80m below the footwall of the orebody and is located on the 300 Level. Note – some mine openings and the surface contour has been excluded from the view to ensure clarity.

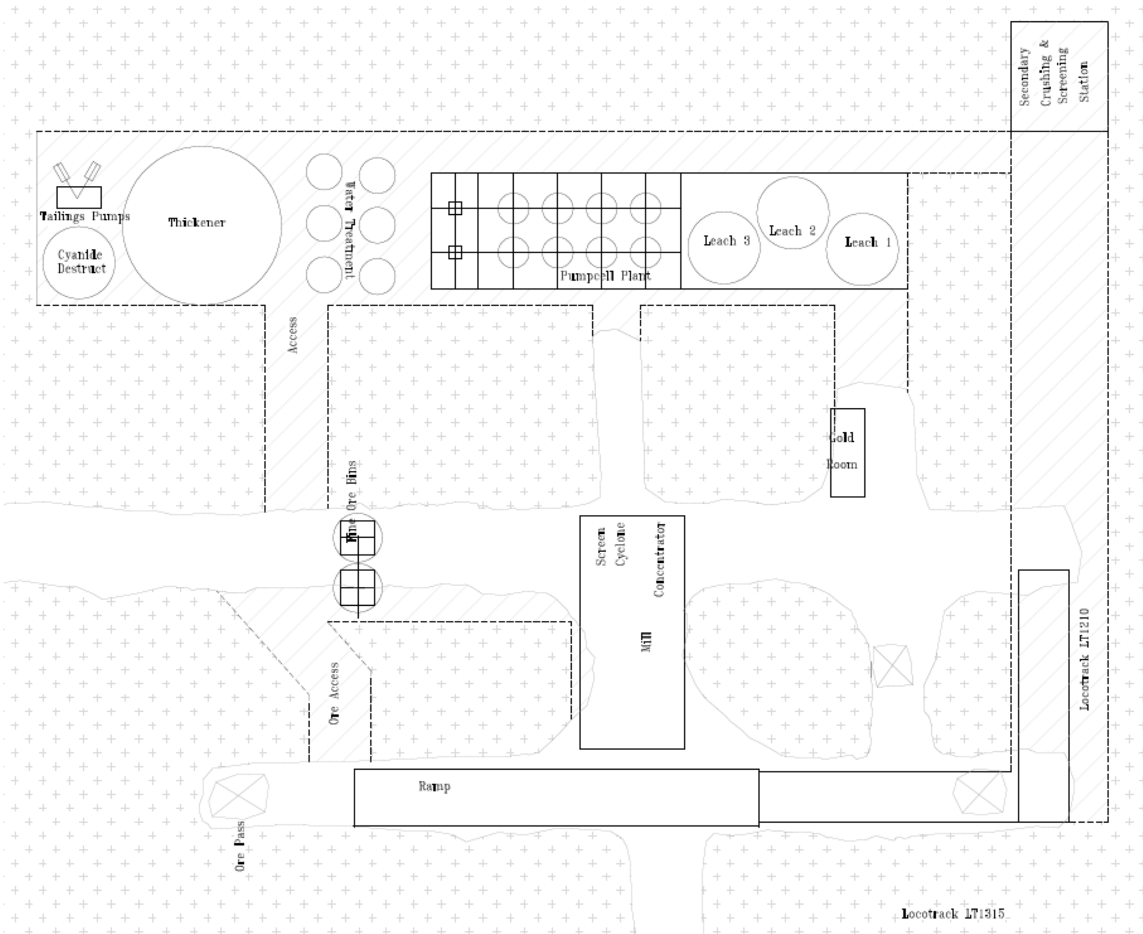
Figure 12.10 – Isometric view of horizontal and vertical development.



The existing drives form the major component of the gravity based mineral processing circuit. The new drives to be designed and excavated are predominantly for the chemical based mineral processing circuit. These form a large chamber in the new excavation measuring approximately 12m wide, 12m high and 65m long. Figure 8 provides a plan view of the existing drives, proposed drives and an indication of the general mineral processing plant layout in the mineral processing chamber. A larger scale drawing of the excavation with key dimensions and plant details can be seen in Appendix 12.12. Figure 12.11 shows the excavation geometry in plan view. Appendix 12.13 shows the excavation with all the plant and equipment removed.

The current crushing chamber is approximately 845m^2 in plan area. The additional plan area stripped, slashed and developed under the proposed design provides an additional 1,245m^2. This provides a total plan area of approximately 2,090m^2 for the mineral processing chamber. This represents an increase in plan area of some 247% on that currently in place.

Figure 12.11 - Plan view of mineral processing chamber excavation



Data Input and Review

The geometric data has been supplied by A&R for the existing mine geometry and the proposed additional excavation geometry to accommodate the mineral processing plant. In addition, A&R have provided the geological description from the Kvaerner Feasibility Study (2002) and the Golder Associates Geotechnical Audit (2004) for the NGM. These were commissioned by the previous owners of the NGM. The material in these reports is not repeated in this document and as such the reader should consider reviewing these documents for their content.

The evaluation of the geotechnical and geological data is based on the reports and data available through a desk study nature of assessment.

The data recorded from orebody and infrastructure geological and geotechnical mapping data in 2000 and 2004 is summarised in the 2004 Golder Associates Geotechnical report and of key note is the general commentary of the rock mass classification as being in the range 'fair' to 'good' with a mean Q value of 4. Low and high Q values range from 1 to 22.

The Q system of empirical rock mass classification is not discussed in length in this report, but for a sense of context the following salient points are made (Hoek, 2007):

On the basis of an evaluation of a large number of case histories of underground excavations the Norwegian Geotechnical Institute proposed a Tunnelling Quality Index (Q) for the determination of rock mass characteristics and tunnel support requirements. The system has been used extensively in mining and civil engineering tunnels (or drives). The numerical value of the index Q varies on a logarithmic scale from 0.001 to a maximum of 1,000 and is defined by:

$$Q = \frac{RQD \times Jr \times Jw}{Jn \times Ja \times SRF}$$

Where:
RQD is the Rock Quality Designation
Jn is the joint set number
Jr is the joint roughness number

Ja is the joint alteration number
Jw is the joint water reduction factor
SRF is the stress reduction factor

The rock tunnelling quality Q can now be considered to be a function of only three parameters which are crude measures of:

Block size (RQD/Jn)
Inter-block shear strength (Jr/ Ja)
Active stress (Jw/SRF)

The Q system goes on to evaluate and provide estimates on the ground control requirements for any given excavation based on over 1000 case histories.

Using the data reported in the 2004 Geotechnical Audit coupled to the preliminary excavation design for the mineral processing chamber, the ground control requirements are assessed. Table 12.7 provides a summary of the key points to the analysis of the excavation. Summary Q data from the 2004 reporting is included in this report in Appendix D.

Table 12.7 – Summary of key points from the Q system analysis for drives

Description	Value	Comments
Mean Q	4.37	Range from 1 to 22
ESR	1.6	For long term / permanent mine infrastructure
MU Span	5.8m	Maximum unsupported span estimate
Bolt Length	1.8m	Range from 1.6m to 2.8m
Bolt Spacing	1.5m	Ring and in-line spacing dimensions

Table Notes:

- Mean Q – is taken from the values reported in the 2004 report. This is summarized in the last page of the Appendix D;
- ESR – the value of 1.6 is used. This is representative for permanent mine openings, water tunnels for hydro power (excluding high pressure penstocks), pilot tunnels, drives and headings for large excavations;
- ESR – the value used in the 2004 work is between 3 and 5. This is representative of temporary mine openings and not relevant to the analysis of this mine infrastructure;

- ESR – NGM may wish to review the inclusion of an ESR of 1.3 when final design details are confirmed, as this infrastructure is intended to be relatively free from ground control risk. An ESR value of 1.3 is used for storage rooms, water treatment plants, minor road and railway tunnels, surge chambers and access tunnels. The use of an ESR of 1.3 equates to a smaller maximum unsupported span and bolts of 2.1m in length;
- MU Span – the maximum unsupported span permissible is estimated based on the equation below:

  $$\textit{Maximum span (unsupported)} = 2\ ESR\ Q^{0.4}$$

- Bolt Length – the bolt length is estimated for a range of excavations width and 1.8m length is quoted for a drive size of 5.4m width. The estimation is based on the equation below:

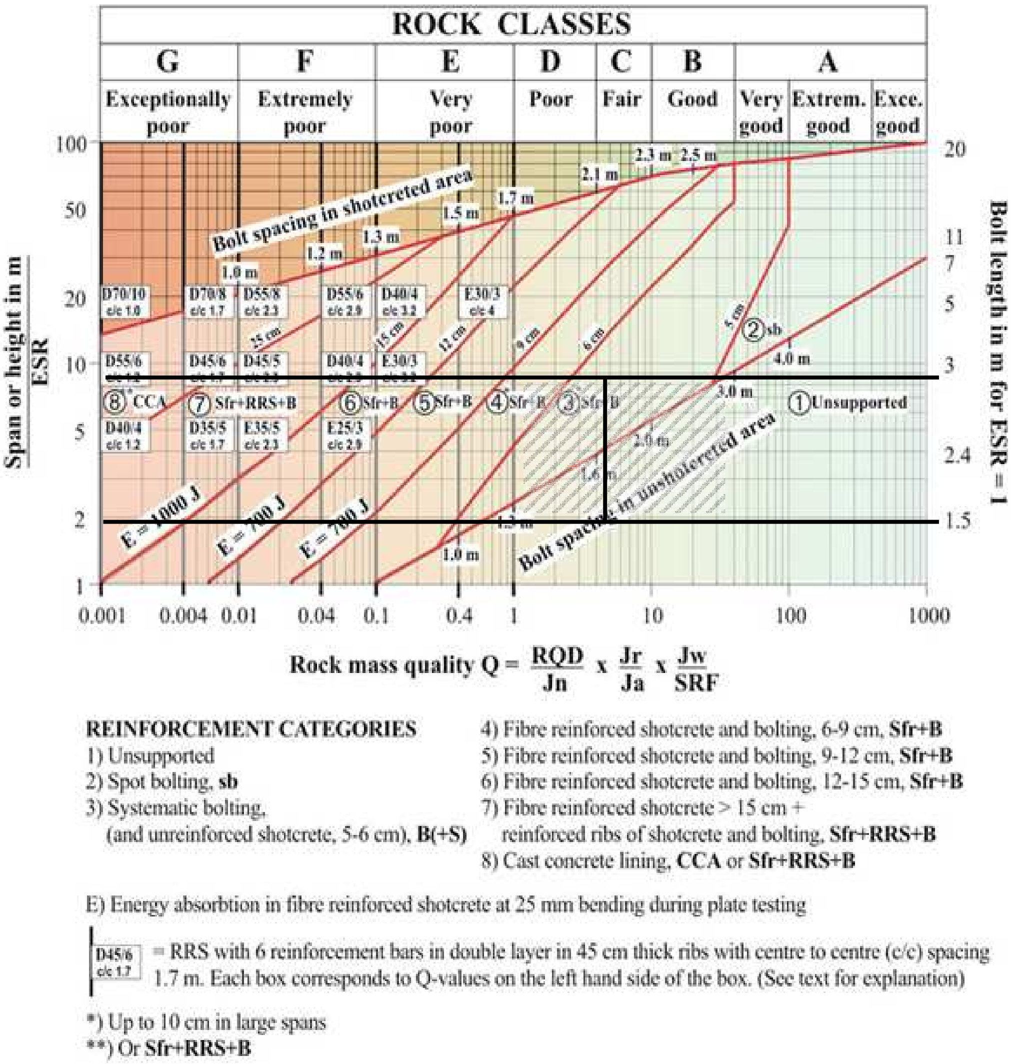
  $$L=[2 + \underline{(0.15B)}]/ESR$$

  Where B = excavation width

Bolt spacing – the bolt spacing estimate is scaled from the chart in Figure 9. This equates to in-line and ring spacing of the bolts.

It should be noted that the application of the Q system is largely based on the assumption that the orientation of the excavation has been designed such that it is at its most favourable orientation towards any faults and joints that may form wedges and blocks in the excavation. This is not confirmed as the case and as such a level of risk is presented by the larger size excavations in the mineral processing chamber. As such recommendations are made for ground control requirements to be installed and further site based assessment to be undertaken to confirm ground control requirements.

Figure 12.12 – Q system chart showing Q value of 4 and a range of drive spans



Preliminary Recommendations

Based on the review of the previous geotechnical assessment data, the update of the Q assessment and the proposed mineral processing chamber geometry the following recommendations are made:

- All backs of the excavations within the mineral processing chamber are bolted with 1.8m long fully encapsulated rock bolts, with surface plates and tensioned. Spacing at 1.5m intervals;
- Bolting is to extend across the back of the excavations and into the shoulder;

- The larger excavations (over 6m in width) as shown in Appendix D 12.14 and 12.15 are recommended to be bolted with 6m long fully encapsulated cable bolts, with surface plates and tensioned. The cable bolt should have a capacity of 25t. Spaced at 3m. If cable bolts are installed in sequence with rock bolts, the 1.8m patterned bolting may removed where it is in the exact same location as the cable bolt. This recommendation is given in the absence of site specific data that provides clear evidence of no hazard potential being created by large wedges or blocks that may cause looses in safe and efficient production at NGM;
- 6m cable bolts are recommended to be considered by NGM in any excavation design risk assessment to mitigate the potential for large wedges to fall / slide into the excavation. This should be at a similar spacing to the sidewalls. An excavation over 8m in height should have cable bolts installed from 5m height;
- Surface support is recommended from the Q system (as a shotcrete). In this application it is largely performing a function of managing hazard potential to the plant, equipment and personnel caused by loose rock and scats forming over time. As such 100mm steel mesh (5mm gauge) is appropriate in lieu of shotcrete. However, NGM may wish to consider the use of shotcrete as it will provide for complete scat control and provide a medium by which any deformation can be monitored (observing for any cracks). The mesh or shotcrete can then be whitewashed to improve the operating environment for the mineral processing and to aid monitoring and observation of any changes in ground stability during the life cycle of the excavation;
- NGM is recommended to risk assess whether it deems it necessary to bolt and mesh the sidewalls (down to 2m height or lower as deemed necessary) to manage risk to an acceptable level for plant and personnel as well as to manage any local defects that may warrant additional ground control measures to mitigate wedges and blocks of rocks that may become unstable;
- The new excavation maintains pillars at a 1:1 or greater width to height ratio. This permits a core of rock to exist which should enable no major issue to be experienced – unless local inspection reveals weakness in the pillars as a result of blast damage, discontinuities or

stress related changes. In such cases additional pillar strengthening measures may be required; and
- Any further refinements and changes to this design geometry must undergo a review to confirm the ground control requirements as appropriate.

The general bolt lengths and spacing arrangements concur with previous work undertaken and recommended to NGM.

The recommendations provided should be reviewed by site personnel based on local data available in the excavation area and considered in the context of A&R's risk management practices within its operations.

Further Mineral Processing Chamber Geotechnical Considerations

It is important to understand the limitations of rock mass classification systems. The use of such systems does not (and cannot) replace some of the more elaborate design procedures. However, the use of these design procedures requires access to relatively detailed information on in situ stresses, rock mass properties and planned excavation sequence, none of which may be available at an early stage in the project. As this information becomes available, the use of the rock mass classification schemes should be updated and used in conjunction with site specific analyses, such as the tasks listed below:

- Site specific discontinuity data;
- Block and wedge analysis with the discontinuity data, particularly the major fault structures in the NGM. These are the low (dip) angle faults ('Your' and 'Mosquito') and higher (dip) angle faults ('Clay' and 'Pegmatite');
- Numerical modelling should be considered for such an excavation of significance to the mine plan and operations. Such work should consider stress analysis for the proposed excavation profile over the life cycle of the NGM's full life of operation and consider the proximity to other drives, including the surface;
- NGM should consider the mining excavation and installation of ground control sequence so as to manage risk to an acceptable level during

the excavation process – including catering for any unanticipated geotechnical issues as they may arise. Such a sequence example to consider could be to develop half the heading width, bolt and cable that section. Then strip out the side to develop to the excavations full design width. This may be followed by stripping out the floor to create the full height. This example is to promote consideration of the excavation process and is not suggested as the best method or approach. NGM should determine that as a function of the equipment and personnel resources in place at the operation. In the excavation sequence – 6m cable bolts should be installed on a cut by cut basis to mitigate risk. It is not recommended they are installed after a large portion or the whole of the excavation has been mined; and

- Subject to observations during the mining, the brows into the larger excavations may require additional brow bolting to mitigate risks associated with blocks and wedges that may become potentially unstable.

Operational Geotechnical Considerations

NGM are recommended to consider the adaption of recommendations made in this report to the refinement and implementation of the following systems in the NGM operation, the identification of hazards and management of risk:

- Risk assessment of design process and operational practices for the excavation;
- Ground control standard specification be developed to ensure quality control and consistency in implementation;
- Ground control requirements for drive intersections in the mine to be risk assessed based on site specific geotechnical and geological data and to cater for the consideration of production interruption and personnel exposure risks;
- A routine check scaling and inspection program is developed and implemented at NGM for all major travel ways and areas of significant mine infrastructure; and
- Consideration is given to the development and implementation of a Ground Control Management Plan (GCMP) for the NGM operations. A GCMP will define the process, accountabilities, tools and all

fundamental aspects for the NGM to operate within a framework to manager risk to an acceptable level from the hazards associated with ground control.

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