ALEXCO KENO HILL MINING CORP.

MINE DEVELOPMENT AND OPERATIONS PLAN

Revision 0

November 2021

ALEXCO KENO HILL MINING CORP.

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1. INTRODUCTION

1.1 PROJECT BACKGROUND

Alexco Keno Hill Mining Corp. (AKHM) continues to develop the mineral resources of the Keno Hill Silver District (KHSD). The company has been actively developing the District since 2006. A new mill complex was constructed in 2010 and operated for three years, processing material from the Bellekeno Mine. Mining operations were suspended in 2013, and Alexco maintained the District on a care and maintenance status and focused on additional exploration which led to increases in the estimated Mineral Resources for the Bermingham and Flame & Moth deposits. Phased underground development of these deposits commenced in 2018 and includes a new portal and decline at the Bermingham deposit, and a new portal and ramp at the Flame & Moth deposit. In Q4 of 2020 production from Bellekeno resumed and the District mill returned to operation and Alexco began shipping concentrate in Q1 2021.

1.2 SCOPE OF THIS DOCUMENT

This document serves as the comprehensive AKHM Mine Development and Operations Plan for the permitted operations, fulfilling the requirement for amendment of AKHM Quartz Mining License QML 0009 (2019). The document replaces previous versions of the Bermingham Development and Operations Plan (2019), and the Flame & Moth Mine Development and Operations Plan (2017).

The Bellekeno Mine Development and Operations Plan (2007) remains valid. Some sections, such as site wide management practices, are updated herein. Where there are differences, this document is the current document. The Bellekeno reserve has been mined out as of November 2021 and is entering into temporary closure.

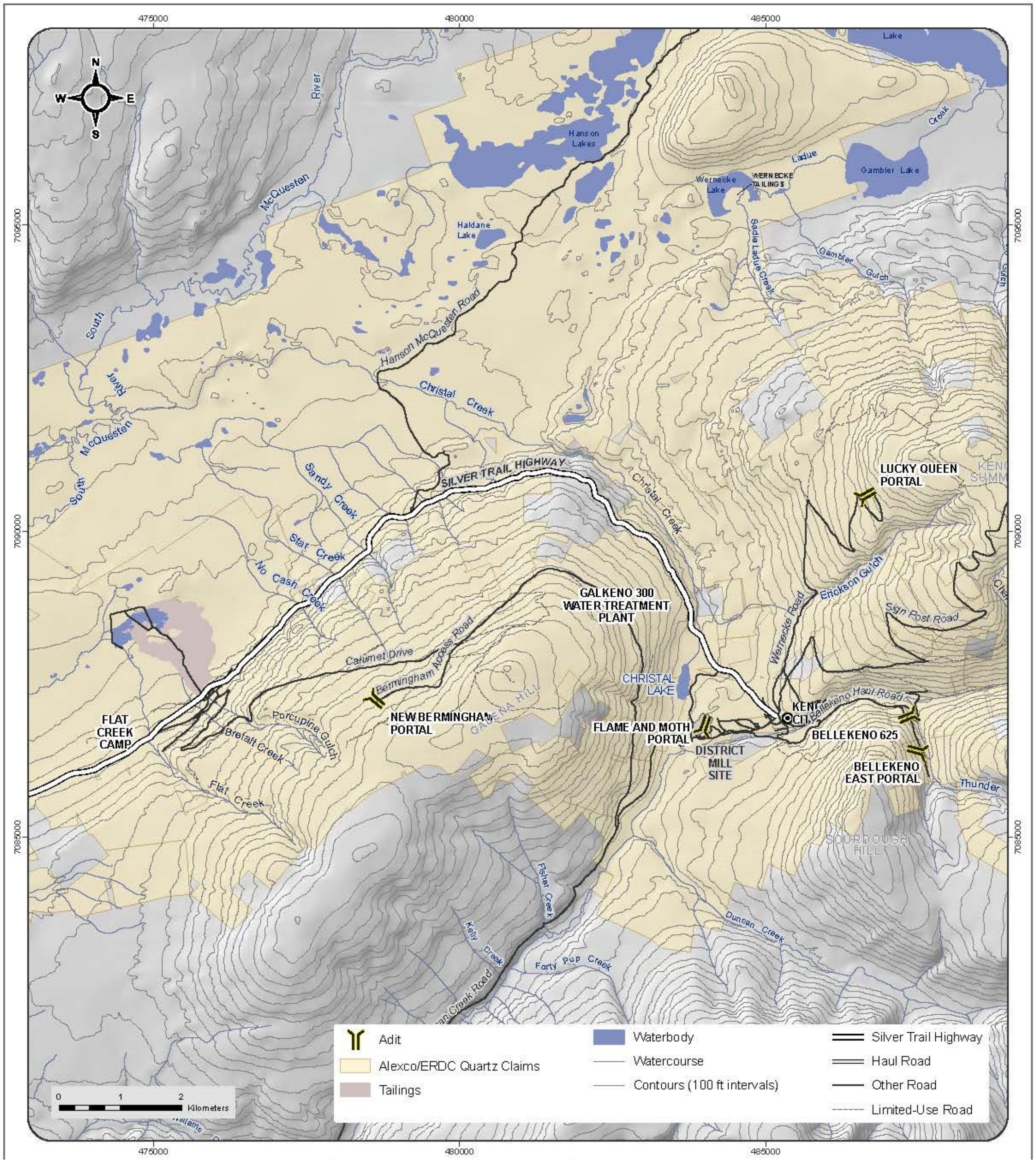
The Lucky Queen and Onek deposits are not currently in the scope of this document as neither are in the LOM mine plan nor authorized with a water use licence.

The Owner of the Mine Development and Operations Plan is the Operations Manager.

1.3 LOCATION AND ACCESS

The KHSD is located in the central Yukon approximately 350 km north of Whitehorse, Yukon Territory, Canada. The area is covered by NTS map sheets 105M/14. The Bellekeno, Bermingham and Flame & Moth deposits are within the KHSD, as is shown in Figure 1-1. The Bellekeno adit is located at UTM 7087062 N, 487363 E Zone 8, NAD83. The Bermingham adit is located at UTM 7087233 N 478607 E Zone 8, NAD 83. The Flame & Moth adit is located at UTM 7086787 N, 484002E Zone 8, NAD 83.

Access to the property is via the Alaska, Klondike, and Silver Trail highways from Whitehorse to Mayo (407 km), and an all-weather gravel road northeast from Mayo to Elsa (45 km); a total distance of 452 km. The closest sizable town for services is Mayo, which is located on the Stewart River, approximately 40 km to the southwest of the Project location. Mayo is accessible from Whitehorse via a 460 km all-weather road and is also serviced by the Mayo airport, which is located just to the north of Mayo. An all-weather gravel road known as the Silver Trail Highway leads from Mayo to the Project, the historic company town of Elsa, and the village of Keno City.



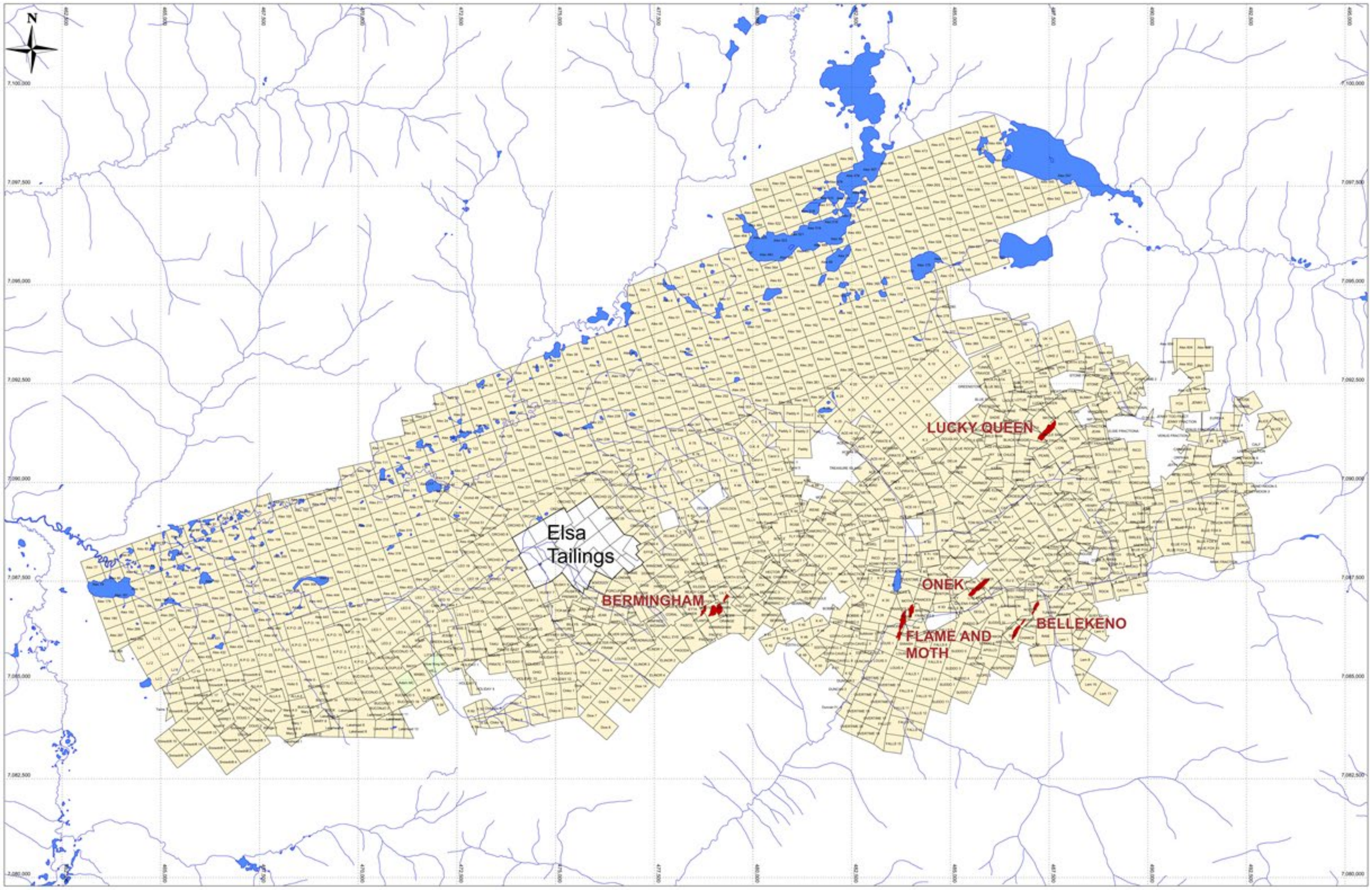
Source: Alexco 2020

1.4 MINERAL TENURE

Mineral exploration at the KHSD was initially permitted under the terms and conditions set out by the Government of Yukon (YG) in the Class 3 Quartz Mining Land Use Permit LQ-00186, issued on July 5, 2006 and valid until July 4, 2011. Alexco subsequently obtained a Class 4 Quartz Mining Land Use Permit – LQ-00240 on June 17, 2008. The two permits were amalgamated on December 8, 2008 under LQ-00240, which has subsequently been renewed as Class 4 Mining Land Use Approval LQ00476 on June 17, 2018.

All quartz mining leases and Crown Grants have been legally surveyed; the quartz mining claims have not been legally surveyed. The KHSD quartz mining claims and quartz mining leases are held by one of two wholly-owned subsidiaries of Alexco: Elsa Reclamation & Development Company Ltd. (ERDC) or Alexco Keno Hill Mining Company Ltd. (AKHM), except for holding a 50% share with third party individuals in three leases (Rico, Kiddo and Argentum).

The AKHM quartz mineral holdings as of December 22, 2020 cover an area of 238.12 km^2, and comprises 703 quartz mining leases, 867 quartz mining claims, and two Crown Grants as shown in Figure 1-2. This does not include the mineral claims that are the subject of the separate technical report titled Mineral Resource Estimation Elsa Tailings Project Yukon, Canada, by SRK dated June 16, 2010.



Source: Alexco 2020

Figure 1-2: Alexco Quartz Mining Claim and Lease Holdings in the Keno Hill Silver District Excluding the Tailings Property

1.5 MAJOR PERMITS AND REGULATORY CONTEXT

The Bellekeno, Bermingham, and Flame & Moth mines have all permits and authorizations in place for mine production. Although the QML authorization is also in place for Onek and Lucky Queen, there are currently no plans to bring the Onek mine into production. An amendment to Water Licence QZ18-044 would be required to bring Lucky Queen and Onek into production. The existing approvals are for the mill throughput of 400 tpd (based upon a 12-month average). The existing approvals and assessments for exploration, mining activities, and for ERDC activities are summarized in Table 1-1.

Table 1-1: Relevant Assessment and Regulatory Approvals

Purpose	YESAA Approval	Quartz Mining Act Approval	Water Use Licence
Alexco Keno Hill Mining Permits			
Bellekeno Advanced Exploration	Project #2008-0039 Decision Document	Class 4 Mining Land Use Approval (LQ00476, expires 2028)	Type B Water Use Licence QZ07-078/Amendment 1 QZ10-060, licence cancelled in 2015. Replaced by amended type A Water Licence in 2015[2]
Bermingham Advanced Exploration	Project#2017-0086 Decision Document	Class 4 Mining Land Use Approval (LQ00476, expires 2028)	Schedule 3 Notice of Water Use/Deposit of a Waste without a Licence
Bellekeno Mine Production	Project #2009-0030 Decision Document	Quartz Mining Licence (QML-0009, Amendment 2, expires 2037)[1]	Type A Water Use Licence QZ18-044, expires 2037[2]
Onek and Lucky Queen Mine Production	Project#2011-0315 Decision Document	Quartz Mining Licence (QML-0009, Amendment 2, expires 2037)[1]	Use of water and the deposit of waste into water is not authorized
Flame & Moth Mine Production	Project #2013-0161 Decision Document	Quartz Mining Licence (QML-0009, Amendment 2, expires 2037)[1]	Type A Water Use Licence QZ18-044, expires 2037[2]
Bermingham Mine Production	Project#2017-0176 Decision Document	Quartz Mining Licence (QML-0009, Amendment 2, expires 2037)[1]	Type A Water Use Licence QZ18-044 issued, Expires 2037[2]
Elsa Reclamation and Development Company Permits			
Care and Maintenance	Project #2006-0293 and 2012-0141	N/A	Type B Water Use Licence QZ17-076 expires 2022[2]
Reclamation Plan	Project #2011-0187 Decision Document (land treatment facility) Project #2012-0077 Decision Document (building demolition) Project #2018-0169 Decision Document (Reclamation Plan implementation)	N/A	Submitted application QZ21-012 to Water Board following issuance of YESAB Decision Document for Project #2018-0169

Notes
1. https://emr-ftp.gov.yk.ca/emrweb/COMM/major-mines/keno-hill/mml-keno-qml-0009-nov2019.pdf
2. http://www.yukonwaterboard.ca/waterline/

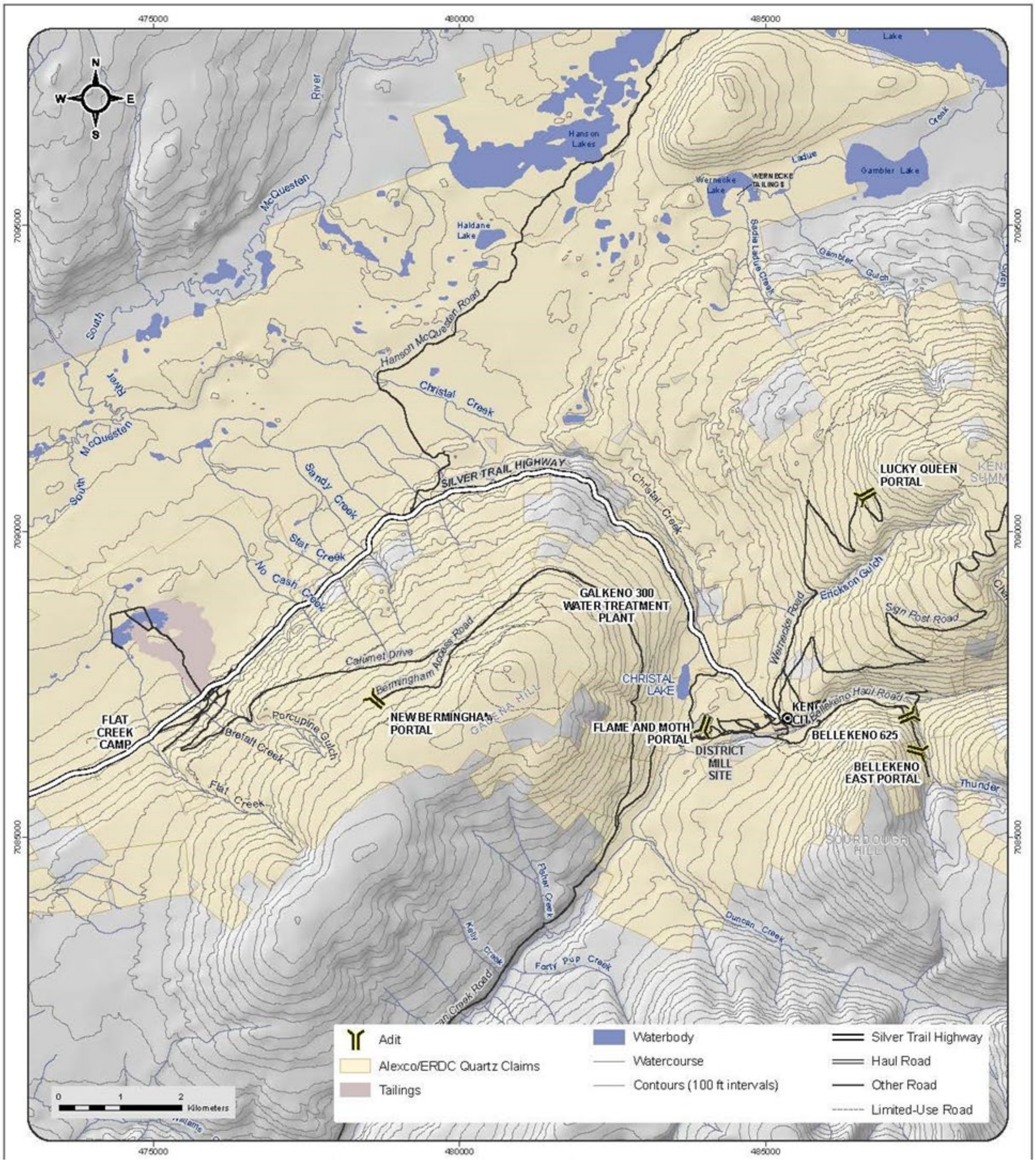


Figure 1-3: Keno Hill Silver District Mining Operations Area Overview

2. SITE DESCRIPTION

2.1 HISTORY

The KHSD is situated in the traditional territory of the First Nation of Na-Cho Nyäk Dun. The site is located within 5 km of Keno City and 50 km from the community of Mayo. There is a long history of mineral development over the past one hundred years. Silver and lead ore deposits were discovered on Keno Hill in the early 1900s, and the area has since seen fluctuating levels of ongoing quartz and placer mining and exploration. Today the area supports not only mineral development, but also tourism, recreational and traditional land uses.

The KHSD is a historic mining district, with the first production recorded in 1913. Since that time, an estimated 200 million (M) oz of Ag has been produced from over 30 small mines across the KHSD. Due to the high grade, steeply dipping veins which host the mineralization, the historic mines were typically small underground operations "chasing the vein", followed by open pit operations beginning in the 1970's to recover selected crown pillars.

In the late 1980's, the then-owner United Keno Hill Mines Limited (UKHM) declared bankruptcy and the site was eventually declared abandoned in 2001, reverting to the Government of Canada. Alexco was the successful bidder in a commercial sale and purchase process and in 2006 became the 100% owner of the assets. Through this transaction, Alexco has the right to mine the deposits and the obligation to develop, permit and implement a reclamation and closure plan for the legacy liabilities across the District. Alexco is fully indemnified for the historic liabilities.

2.1.1 FLAME & MOTH HISTORY

Claim staking and prospecting began at Flame & Moth in 1920. By 1923, numerous surface workings and a 13 m inclined shaft had been sunk with a 4.6 m crosscut developed from it on the Moth claim. It is believed that a second shaft to a depth of 30.5 m was also sunk in this vicinity. An adit was developed to a length of 12.2 m on the Frances 7 claim. Production for this period is not known.

Subsequent to this early work, little or nothing appears to have happened on the property until the acquisition by UKHM just prior to 1950. A 27.4 m inclined shaft was sunk to a vertical depth of 21.3 m along the footwall of what was likely the Moth vein. A crosscut, through the zone 13.7 m below surface and 42.7 m of drifting 22.9 m below surface, identified quartz-carbonate vein hosted mineralization averaging 343 g/t Ag, 1.6% Pb, and 5% Zn developed in quartzite and greenstone along a zone approximately 30.5 m long and up to 9.1 m wide. Thirteen horizontal core drill holes totalling 193 m were drilled from the drift, but the core recovery was poor.

During 1954 and 1955, mineralization of pyrite and minor arsenopyrite was reported up to 240 m along strike to the north. This was explored by bulldozer trenching, soil sampling, and ground geophysics, but was unsuccessful because of the depth of gravel overburden, reported to a 12 m depth.

UKHM returned to Flame & Moth in 1961 with a program of soil sampling and ground geophysics (self-potential, magnetics, Ronka EM), and drilled five surface core drill holes located near the shaft to test the mineralization at depth. The soil samples and geophysics yielded little information, and no veining was intercepted in the drilling.

In 1965, 28 vertical overburden drill holes were drilled, along with another attempt at soil sampling and geophysics. A proposal to excavate an open pit was first made at this date, based on a calculated resource of 3,360 t grading 573 g/t silver (Ag), 1.4% lead (Pb), and 5.6% zinc (Zn). The pit would have reached to 18.3 m below the surface.

In 1974, four lines of angled overburden drill holes totaling 989 m were drilled for extensions along a 180 m strike length with limited success due to deep overburden and broken ground conditions, although a weakly mineralized structure was located at 76 m in the footwall of the main vein.

More overburden drilling was completed along strike in 1984 and four core drill holes were sited to test the downward projection of the known mineralization. The deeper drilling (60 to 90 m below surface) returned only very low values from a wide but diffuse pyritic vein zone. A small amount of vein material (368 t grading 699 g/t Ag, 1.39% Pb, and 0.72% Zn) was sent to the mill, which may have come from vein material exposed during stripping of overburden in preparation for the open pit development. In May 1987, the open pit Mineral Resources were re-evaluated at 12,600 t grading 699 g/t Ag and 4.0% Pb to a depth of 24.4 m. The key assumptions used to estimate this historical estimate are not known. This historical estimate was prepared before the adoption of NI 43-101 and therefore should not be relied upon.

Surface core drilling by Alexco in the Flame & Moth resource area totalled 14 drill holes (3,986.2 m) in 2010, 32 drill holes (7,149.2 m) in 2011, and 48 drill holes (10,106.5 m) in 2012, eight drill holes (1,835 m) in 2013, and 49 drill holes (12,166.4 m) in 2014.

2.1.2 BERMINGHAM HISTORY

The first claims in the Bermingham area were staked in 1921, within a decade of commercial production starting in the Keno Hill Silver District. Shallow underground workings were initiated in 1923 with the discovery of vein float and limited production of high-grade silver and lead from the Bermingham vein ensued. When TYC optioned the Mastiff claim group in 1928, a 30 m shaft and 223 m of drifting had been completed on three separate levels. The underground workings showed a structure with a maximum width of 17 m on the 100 level that contained multiple bands of mineralization with interstitial waste that was cut off at its southwest extent by the Mastiff fault.

The TYC optioned the ground in 1928 and completed additional underground workings and identified a fault offset vein portion but dropped the lease in 1930 due to low silver prices and a lack of ore grade material. Trenching and prospect shafts identified the offset vein approximately 91 m to the west-northwest, where TYC sank the No. 1 shaft and completed 22 m of drifting. An oxidized siderite-pyrite vein with some galena was located below the position of the future main Bermingham pit but no mineralized material was reported from 127 m of drifting completed on the 200 level. TYC relinquished the lease in 1930 due to low silver prices and the absence of economic grade material. A variety of individual workers extracted another 676 t grading 7,875 g/t Ag and 70% Pb between 1930 and 1940. This work was poorly documented but is known to include considerable trenching, shafting, and drifting during 1930, 1932 to 1937, and 1939 to 1940.

UKHM subsequently purchased the property as part of the district consolidation, and during 1948 to 1951 drove an adit and drift approximately 9 m below the bottom of the TYC workings. In 1952, many of the old Treadwell workings were surveyed and sampled, but the adit level was subsequently abandoned in 1954 after very little ore grade material was realized. During this time, UKHM reportedly milled 5,165 tons of ore at 47.3 oz/ton (opt) Ag, 8% Pb, and 1.3% Zn, of which all but 60 tons was recovered from the old dumps.

Between 1965 and 1982, 874 overburden drill-holes totalling 19,931 m, and 27 core holes totalling 2,407 m, were drilled in the Bermingham area; a small portion of which occurred in the present resource area. Poor ground conditions prevented many of these holes from adequately penetrating the vein zone; however, they outlined an open pit resource and stripping began in 1977.

The exploration conducted by Alexco is the first comprehensive exploration effort in the district since 1997. The first holes drilled by Alexco in the Bermingham area were in 2009 (two core holes totalling 523 m), targeting the Bermingham vein at depth in the hanging wall of the Mastiff Fault below an area with a historic shallow open pit resource. Results of this drilling were sufficiently encouraging to continue exploration in 2010 and 2011. Alexco conducted further diamond drilling programs at Bermingham in 2012, 2014, 2015 and 2016 in the Bermingham deposit and surrounding area.

Between 2009 and 2018, a total of 56,324 m surface core diamond drilling has been completed by Alexco at Bermingham, with a total of 169 drill holes, including two drill holes (523 m) in 2009, nine drill holes (3,046 m) in 2010, 25 drill holes (6,888 m) in 2011, 17 drill holes (5,576 m) in 2012, eight drill holes (2,668 m) in 2014, eight drill holes (2,606 m) in 2015, 50 drill holes (17,371 m) in 2016, 38 drill holes (13,277 m) in 2017 and 12 drill holes (4,369 m) in 2018. In addition, 24 underground drill holes (4,214 m) were completed from the exploration decline in 2018. All holes were diamond cored in HQ/HTW apart for a few reduced to NQ/NTW because of ground conditions.

2.2 GEOLOGY

2.2.1 DISTRICT GEOLOGY

The KHSD is located in the northwestern part of the Selwyn Basin in an area characterized by the Robert Service and Tombstone Thrust Sheets that are overlapping and trend northwest. The area is underlain by Upper Proterozoic to Mississippian rocks that were deposited in a shelf environment during the formation of the northern Cordilleran continental margin. The KHSD geology is dominated by the Mississippian Keno Hill Quartzite comprising the Basal Quartzite Member and conformably overlying Sourdough Hill Member. The unit is overthrust in the south by the Upper Proterozoic Hyland Group Yusezyu Formation and is conformably underlain in the north by the Devonian Earn Group (McOnie and Read, 2009).

The Yusezyu Formation of the Precambrian Hyland Group comprises greenish quartz-rich chlorite-muscovite schist with locally clear and blue quartz-grain gritty schist and is separated from the Keno Hill sequence by the regionally extensive Robert Service Thrust Fault that occurs immediately south of the area.

The Earn Group, formerly mapped as the “lower schist formation” (Boyle, 1965), is typically composed of recessive weathering grey graphitic schist and green chlorite-sericite schist with an upper siliceous graphitic schist found locally.

Within the Keno Hill Quartzite Formation, the Basal Quartzite Member that is the dominant host to the silver mineralization, comprises commonly calcareous, thick to thin-bedded quartzite and graphitic schist and may be up to approximately 1,100 m thick where structurally thickened. The overlying Sourdough Hill Member, formerly mapped as the “upper schist formation” (Boyle, 1965), is up to approximately 900 m in thickness and comprises predominantly graphitic and sericitic schist, chloritic quartz augen schist some of which may be of volcanogenic origin, and minor thin bedded limestone.

The Earn Group and Keno Hill Quartzite are locally intruded by stratigraphically conformable, although lensoidal, Middle Triassic greenstone sills, for which any feeder dykes are unrecognizable. The sequence was metamorphosed to greenschist facies assemblages during Cretaceous regional deformation at about 100 My, and subsequently intruded by aplite sills or dikes considered to be related to the Tombstone intrusive suite.

Three phases of folding are identified with the two earliest phases consisting of isoclinal folding with subhorizontal, east or west trending fold axes, the axial plane forming the dominant regional foliation. The later fold phase displays subvertical axial planes and moderate southeast-trending and plunging fold axes. The first phases of folding formed structurally dismembered isoclinal folds of which the Basal Quartzite Member outlines synforms at Monument Hill where the Lucky Queen mine is located and at Caribou Hill, while between Galena Hill and Sourdough Hill the Bellekeno mine, the Flame & Moth and Bermingham deposits are located on the upper limb of a large-scale anticline that closes to the north.

Up to four main periods of faulting are recognized with the oldest fault set consisting of south dipping foliation parallel structures that developed contemporaneously with the first phases of folding, sometimes shown as "low angle bedding faults". The Robert Service Thrust Fault truncates the top of the Keno Hill Quartzite Formation and sets the Precambrian schist of the Yusezyu Formation above the Mississippian Sourdough Hill Member. The silver mineralization in the KHSD is hosted by a series of northeast oriented, southeasterly dipping veins formed in pre- and synmineral faults referred to as vein-faults (Boyle, 1965) that display left lateral normal oblique displacement. There are two related sets locally recognized as either a more easterly trending "longitudinal" vein set that, depending on the competency of the host rock, can form up to a 30 m wide zone of anastomosing subparallel veins, or a more northerly trending "transverse" vein set that can reach up to 5 m in thickness.

The mineralized vein-faults are commonly offset by northwest striking, steeply southwest dipping, post-mineral cross faults, that display right lateral normal oblique displacement.

Mineralization is of the polymetallic silver-lead-zinc vein type that typically exhibits a succession of hydrothermally precipitated minerals from the vein wall towards the vein centre. However, in the KHSD, multiple pulses of hydrothermal fluids or fluid boiling, probably related to repeated reactivation and breccia formation along the host fault structures, have formed a series of vein stages with differing mineral assemblages and textures. Supergene alteration may have further changed the nature of the mineralogy in the veins. Much of the supergene zone may have been removed due to glacial erosion.

In general, common gangue minerals include (manganiferous) siderite and, to a lesser extent, quartz and calcite. Silver predominantly occurs in argentiferous galena and argentiferous tetrahedrite (freibergite). In some assemblages, silver is also found as native silver, in polybasite, stephanite, and pyrargyrite. Lead occurs in galena and zinc in sphalerite, which at the KHSD can be either an iron-rich or iron-poor variety. Other sulphides include pyrite, pyrrhotite, arsenopyrite, and chalcopyrite.

The Keno Hill mining camp has long been recognized as a polymetallic silver-lead-zinc vein district with characteristics possibly similar to other well-known mining districts in the world. The largest accumulation of minerals of economic interest occurs in areas of increased hydrothermal fluid flow in structurally prepared competent rocks such as the Basal Quartzite Member and Triassic Greenstone. Incompetent rocks like phyllites tend to produce fewer and smaller (if any) open spaces, limiting fluid flow and resulting mineral precipitation.

The geology of the KHSD area is shown in Figure 2-1.

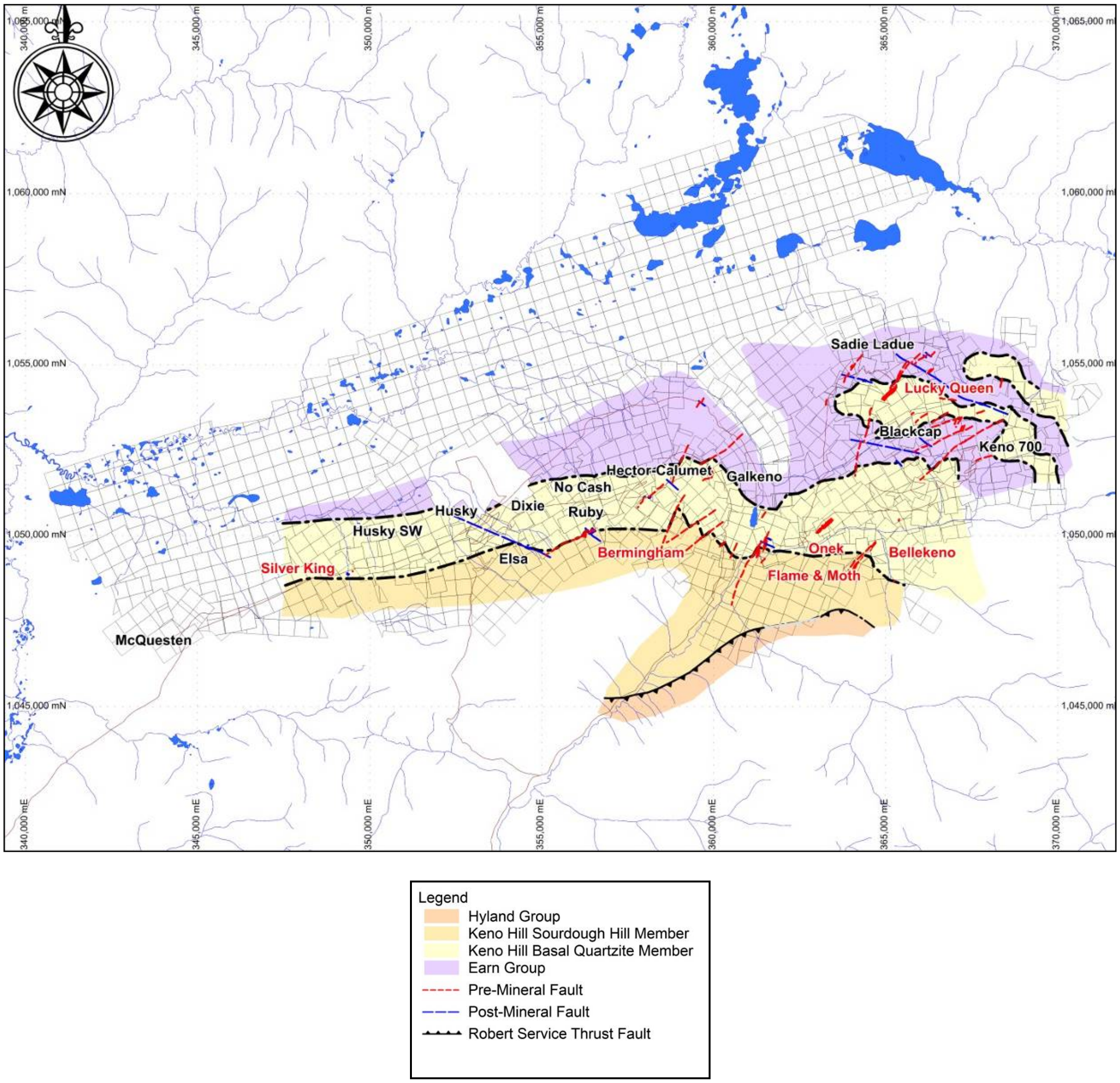


Figure 2-1: Geology of the Keno Hill Silver District

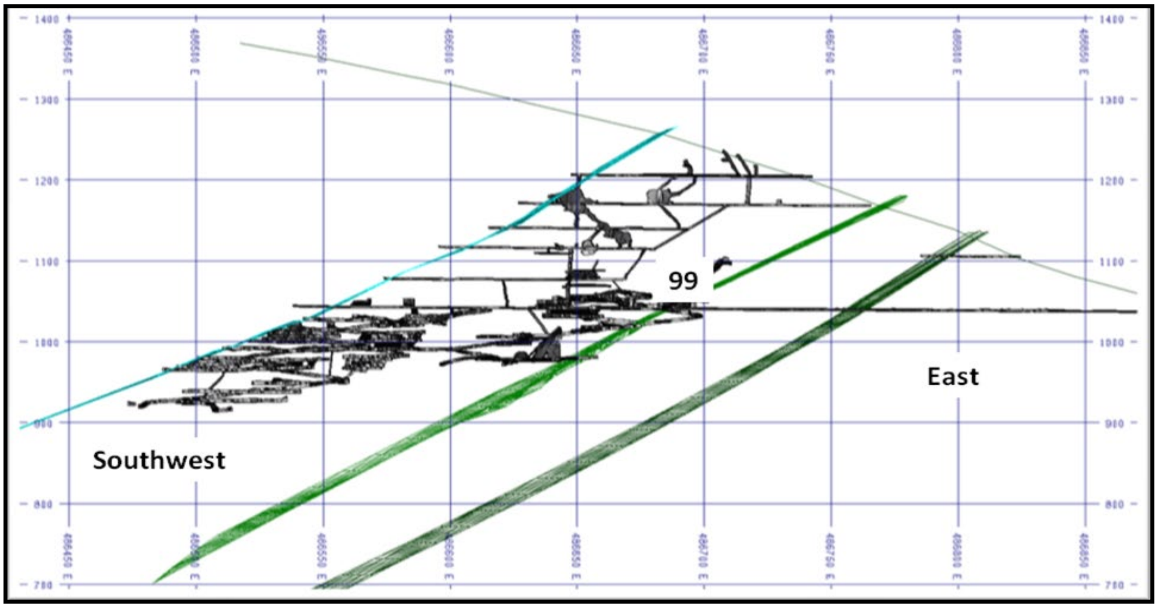
2.2.2 BELLEKENO MINE GEOLOGY AND MINERALIZATION

The Bellekeno vein system consists of ten known veins with variable characteristics. Vein material has been extracted from the Ram, Eureka, Tundra, 48, 49, and 50 veins that generally strike 030° to 040°, with dip directions varying 60° southeast to 80° northwest. Recent mechanized mining has focused on the stronger 48 Vein structure, while conventional historical narrower mining focused on the smaller, higher grade vein structures.

There are three main zones within the 48 Vein structure: the Southwest, 99, and East zones, each with distinctive silver to lead ratios, zinc content, and accessory mineral assemblages (as shown in Figure 2-2).

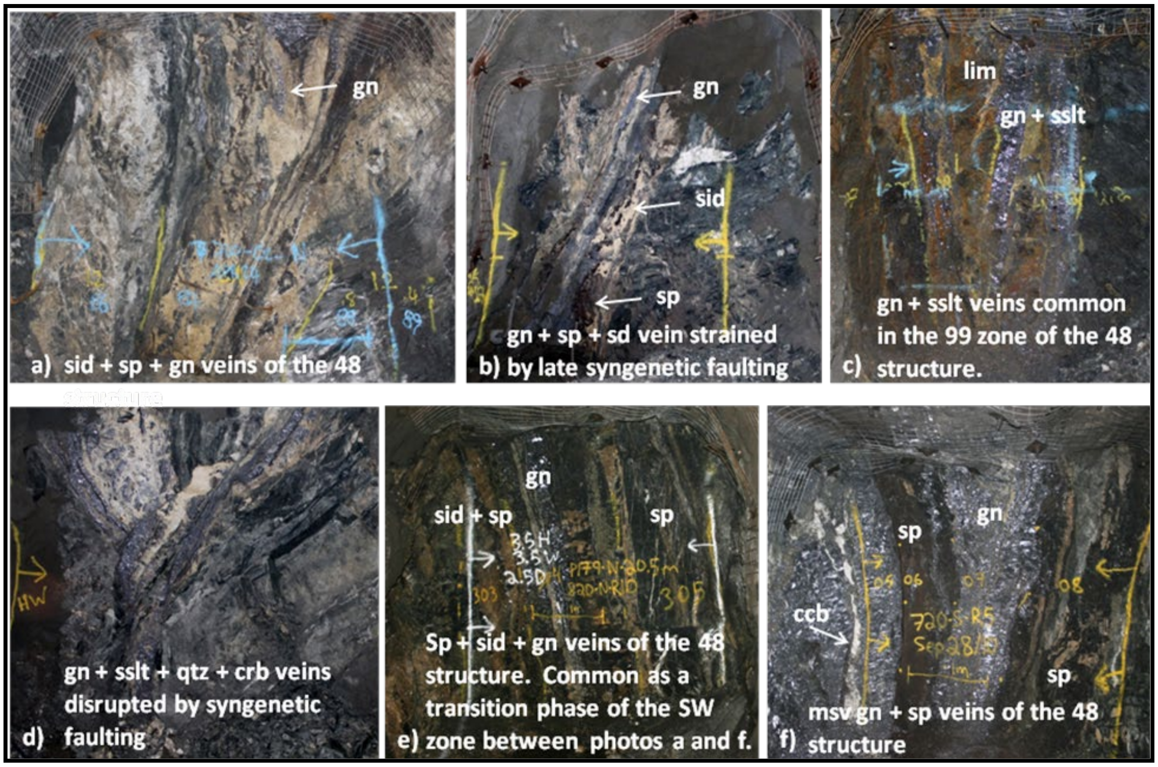
The thickness of the vein ranges between a few cm to upwards of 5.5 m. Post-mineral faulting typically shows intense iron carbonate alteration and local brecciation while the distribution of syn-mineral faulting is observed to have a strong impact on silver grades and mineral textures as can be seen in Figure 2-3. Left oblique-normal movement along the 48 Vein structure is estimated from stratigraphic offset to be approximately 35 m.

The mineralized zones appear as discontinuous steeply plunging shoots, hosted within manganese-rich siderite vein structures, and may have pervasive secondary limonitic alteration where exposed to groundwater. Minerals of economic interest include very fine-grained silver-bearing sulphosalts associated with galena and sphalerite. Common accessory minerals include pyrite, arsenopyrite, and chalcopyrite while anglesite, cerrussite, smithsonite, malachite and azurite have been occasionally observed.



Source: SRK 2013

Figure 2-2: Schematic Long Section of the 48 Vein Bellekeno Mine Showing Workings



Source: SRK 2013

Note. (gn) galena; (sid) manganese rich siderite; (sp) iron rich; (Fe 65) sphalerite; (lim) limonitic oxidation of carbonate facies; (sslts) non-specific sulphosalts; (qtz) siliceous floods and concretions associated with late breccias; (crb) white carbonate

Figure 2-3: 48 Vein Structures and Mineralogy, Bellekeno Mine

2.2.3 BERMINGHAM GEOLOGY AND MINERALIZATION

In the Bermingham area, five mineralized veins have been identified (Aho, Bermingham, Bear, Bermingham Footwall, and West Dipper veins) within a structurally complex network of fault and vein structures related to the through-going northeast striking, southeast dipping, Bermingham vein-fault system. Less extensive north-northeast striking vein geometries are also observed within the mineralized system. The combined displacement in the Bermingham area associated with the Bear and West Dipper veins has displaced the hanging wall of the vein system approximately 165 m along a vector 095°/ -60° to the southeast. While dip separation of stratigraphy across the Aho vein ranges from 50 m to 80 m. The mineralized veins are affected by numerous post-mineral faults. The early Aho vein comprises predominantly quartz and occurs over several metres width within a wide halo of structurally damaged rocks. Minor sulphides are present with arsenopyrite and pyrite being the most abundant, with accessory galena and sphalerite.

The Bermingham vein has a strike between 029° and 042° and dips between 40° and 64° to the southeast. The structure accommodates approximately 65 m of the total Bermingham displacement. In the Etta Zone (in the hanging wall of the post-mineral Mastiff fault), the Bermingham vein at its most southwestern extent, is observed to converge with the Aho vein structure, while to the northeast, it converges with the Bermingham Footwall vein.

The Bermingham Footwall vein has a strike of between 040° and 060°, and dips between 67° and 73° to the southeast. The structure accommodates approximately 70 m of the total Bermingham displacement. In the Etta Zone, the Bermingham Footwall vein terminates against the Bermingham vein up-dip, and this intersection plunges moderately steeply to the northeast into the Arctic Zone (in the footwall of the post-mineral Mastiff fault). At depth, the Bermingham Footwall vein terminates against the Aho vein along a steep plunging north-easterly trajectory.

The Bermingham vein and Bermingham Footwall vein typically exist within a wide 5 m to 10 m wide structurally damaged zone containing numerous stringers, veinlets, breccias, and gouge. In most cases, a discrete vein 0.5 m to 2.5 m wide exists within this zone, consisting predominantly of carbonate (dolomite, ankerite, and siderite), quartz and calcite gangue, and sulphides: sphalerite, galena, pyrite, and arsenopyrite, with accessory, chalcopyrite, argentian tetrahedrite (freibergite), jamesonite, ruby silver, and native silver.

The Bear vein strikes between 010°and 050° and dips between 65°and 80° to the southeast. The structure accommodates approximately 30 m of the total Bermingham displacement. It occupies a position in the footwall of the system beneath a major flexure in the Bermingham vein, to which it joins up dip. At depth and to the southwest, the Bear vein junctions with the Bermingham Footwall vein. Early phase mineralization is absent, and the Bear structure is considered a late response to the slip-impeding flexure in the Bermingham vein noted above. Wide high grade mineralization is positioned on more northerly striking and steeper dipping areas.

First recognized in 2016, the West-Dipping vein strikes 020° and dips 50° to the west, it is situated between the Bear and Bermingham veins. It displays only minor displacement and is considered to represent an adjustment in the Bear vein hanging wall to a pronounced curvature in the sliding path. Similarly, oriented veins were observed historically in the Keno Hill district at Elsa, Husky, Runer, Black Cap, and are also interpreted at Hector-Calumet and Lucky Queen (Boyle, 1965; Cathro, 2006; UKHM, 1997 unpublished). The Bear and West dipping veins are structurally and mineralogically similar to the Bermingham veins but quartz and calcite (considered early mineral phases) are less abundant or absent whilst sulphosalts are more abundant. This difference is considered a product of a shorter duration of activity on both the Bear and West Dipper veins allowing for deposition of only the later stages of the mineralization. Wide, high grade veining is spatially associated with vein-fault domains exhibiting steeper dip and/or more northerly strike. The post-mineral faults that are recognized within the resource area include the Mastiff, Hanging wall, Cross and Super faults. The attitudes of post-mineral faults appear bimodal, with one set striking between 280° and 293°, and the other at 314° to 317°, although they may represent end members of a single fault set. These northwest trending structures cut and displace all mineralized veins, and while they are typically non-mineralized, it is sometimes observed that mineralization may have been drawn into the later fault.

The Mastiff fault strikes at 137°, dips 51° to the southwest, and displaces the hanging wall obliquely 131 m down to the northwest along a vector 302° / -23°. The location of the Mastiff fault is well constrained by drilling and exposure in the main pit. When discussing the Bermingham, Bermingham Footwall and Aho veins, the vein zones located in the footwall of the Mastiff fault are referred to as the "Arctic" Zone (to the west) and "Etta" Zone in the hanging wall (to the east).

The Hanging wall fault strikes between 000° and 025° and dips between 53° and 65°to the east, and is represented in drill-core by very wide zones (10 m -30 m) of unconsolidated fault breccia and gouge, mineralization is sporadic and weak and occurs as trails of fragmented clasts that are interpreted to represent pre-fault material. The Hanging wall fault extends to surface where it was intersected by historic trenching northeast of the current resource area.

The Cross fault strikes between 120° and 130° and dips between 45° and 68° to the south. The fault displaces all veins 76 m down to the south along a vector 274° / -29°. The Cross fault includes two sub-parallel splays, and their generation is considered a response to a strong flexure in the main fault shape.

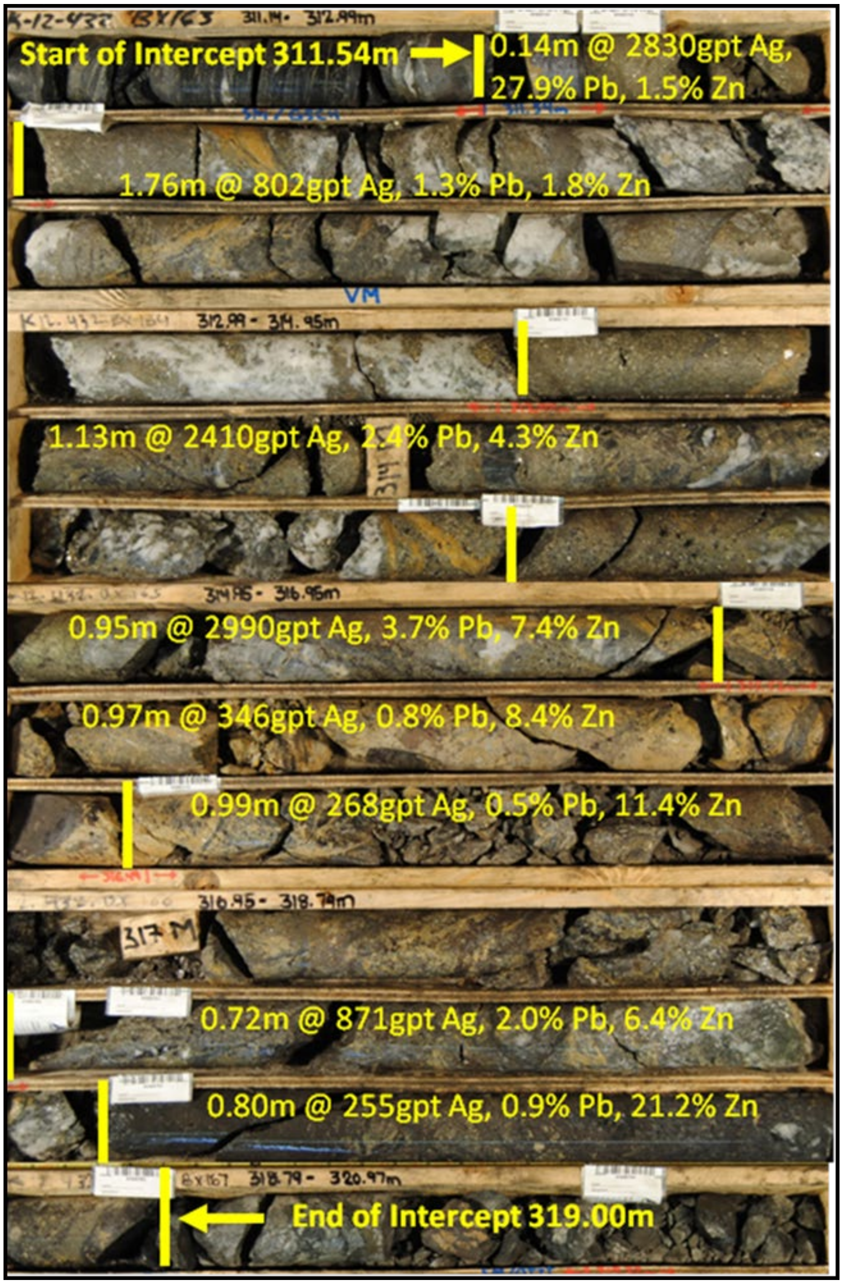
The Super fault strikes 133° and dips 25° to the southwest with the hanging wall displaced approximately 42 m downward to the south along a vector 272° / -15°. The structure dislocates the historic workings and open pit from the current resource area that is wholly situated in the footwall. The fault structure is well represented by drill-core and is exposed in the north end of the historic main pit where it has also been referred to as the Mirror fault.

2.2.4 FLAME & MOTH GEOLOGY AND MINERALIZATION

The Flame vein is unique because of its uniformly singular form, width, grade and length. It occurs over a one kilometre (km) strike length orientated at strike of 025° and dipping approximately 65° southeast and has been traced by drilling over to least 300 m depth extent. Two main styles of mineralized veining commonly with multiple banding, internal brecciation and often rehealed textures are observed (Figure 2-4). The early phase comprises dominantly quartz gangue with abundant but irregular amounts of pyrite, pyrrhotite, sphalerite and arsenopyrite, while a later phase comprises predominantly siderite containing abundant sphalerite, pyrite, galena, with minor chalcopyrite and trace amounts of tetrahedrite, pyrargyrite, jamesonite, boulangerite and cassiterite as identified in thin section samples.

The vein is divided into two parts by an approximate 90 m right lateral offset on the post-mineral Mill Fault that are referred to as the Lightning Zone in the southeast and the Christal Zone in the northwest.

The associated Moth vein, the subject of historic prospecting, is considered to represent a footwall splay of the Flame vein, although the relationship is not fully understood.



Source: SRK 2013

Figure 2-4: Vein-Fault Intercept in Drill Hole K-12-0432, Flame & Moth

2.2.1 EXPLORATION AND DRILLING

The exploration conducted by Alexco since 2005 is the first comprehensive exploration effort in the KHSD since 1997. The work has included a program of geologic data compilation, aerial geophysical surveying, and surface core drilling. Alexco converted the historic maps and documents from nearly 70 years of mining in the District to digital form. The digital data has been used to construct district scale maps and three-dimensional (3D) mine models.

Since acquiring the Keno Hill property (up to July 12, 2021), Alexco has completed a total of 822 surface diamond drill holes for a total of 215,625 m. In addition, a total of 433 underground holes for 30,199 m has also been completed, mainly at Bellekeno, but also includes 24 holes for 4,213 m drilled in 2018 from the Bermingham exploration decline.

Exploration drilling by Alexco has primarily been conducted to test targets immediately adjacent to historic resource areas and, to a lesser extent, to evaluate targets based on interpretation of exploration data. The objective has been to locate structurally controlled vein mineralization.

Standard logging and sampling conventions are used to capture information from the drill core. Since 2010 all core logging data has been directly digitally entered to the geology database with data including comments captured in separate tables including lithology, structure, mineralization type, intensity of oxidation, phases and abundance of veining, alteration, stratigraphy, and geotechnical.

2.3 ENVIRONMENT

2.3.1 SETTING

The property is located within the Yukon Plateau (North) Ecoregion and is characterized by rolling upland areas and wide open valleys. Vegetation communities include Northern boreal forests along the lower slopes and valley bottoms, and open scree slopes above treeline. Many of the valley bottoms include open peatlands, fens, and meadows. A variety of wildlife, birds, and fish species are present in the area. The landscape around the Project is characterized by rolling hills and mountains with a relief of up to 1,975 meters above seal level (masl). The Bermingham deposit is located on Galena Hill, Bellekeno deposit on Sourdough Hill and Flame & Moth is located at the valley between Galena Hill, Sourdough Hill, and Keno Hill. Slopes are gentle except the north slopes of Keno Hill and Sourdough Hill.

The central Yukon Territory is characterized by a sub-arctic continental climate with cold winters and warm summers. Average temperatures in the winter are between minus fifteen and minus twenty degrees Celsius, but can reach minus sixty degrees Celsius. The summers are moderately warm with average temperatures in July approximately fifteen degrees Celsius. Mining operations are carried out year-round.

Because of its northern latitude, winter days are short; north-facing slopes experience ten weeks without direct sunlight around the winter solstice. Conversely, summer days are very long, especially in early summer around the summer solstice. Annual precipitation averages twenty-eight centimetres ("cm"); half of this amount falls as snow, which starts to accumulate in October and remains into May or June.

The KHSD falls in the subarctic clime of the Koppen climate classification. The closest current long-term climate record is at the Mayo Airport, which had an average daily temperature of -2.4°C and average annual precipitation of 313.5 mm, with 203.8 mm falling as rain for the 1981 to 2010 period (the public record is updated every 10 years). The wet season occurs in summer/fall with drier winters. Meteorological data have been collected in the KHSD for three locations: since 2007 at the Calumet weather station as part of the development of the reclamation studies for historic liabilities; since 2011 at the Keno Hill District mill meteorological station as part of Bellekeno mining operations; and since 2012 at the Valley Tailings Facility meteorological station. The monthly and annual temperatures are, on average, colder at the three KHSD stations than at Mayo Airport, which is expected given the higher site elevation.

The environmental setting of the site is summarized in Table 2-1. The KHSD has a long mining history and is a brownfields site. The current environmental conditions reflect the brownfield conditions and recent improvements as a result of Care and Maintenance water treatment upgrades and interim reclamation undertaken by ERDC for the historic liabilities.

Table 2-1: Keno Hill Silver District Environmental Setting Summary

Drainage Region	Stewart River drainage region
Local Catchments	No Cash Creek, Flat Creek, Christal Creek, Lightning Creek
Ecoregion	Yukon Plateau (North)
Study Area Elevation	900-1,350 masl
Vegetation Communities	Northern boreal forests occupy lower slopes and valley bottom; spruce, pine and alder; grasses and sedges, mosses occupy forest floor; heavy moss and lichen growth resident as ground cover; understory of shrub willow; open and forest fringe areas of willow and scrub birch, and various flowering plant species.
Wildlife Species	Moose, grizzly and black bear, caribou, beaver, wolf, lynx, marten, wolverine, western tanager, magnolia warbler, white-throated sparrow, bald eagle, furbearers and small animals. Committee on the Status of Endangered Wildlife in Canada (COSEWIC) listed species include: Common Nighthawk (Threatened); Rusty Blackbird and Olive-Sided Flycatcher (Special Concern).
Fish Species	Bering and Beaufort Sea salmonids and freshwater species including: Arctic grayling, Arctic char, lake trout, trout perch, lake whitefish, broad whitefish, burbot, inconnu, Arctic Cisco, Northern pike, slimy sculpin.

2.3.2 HYDROLOGY

The KHSD contains two main watersheds: Lightning Creek watershed and the Christal Creek watershed, which is a sub-watershed of the South McQuesten River.

Christal Creek flows northwest from Christal Lake for approximately 22 km before it flows into the South McQuesten River. Water chemistry and aquatic resources in the creek have been influenced by previous mine and milling operations including tailings deposition and adit discharge. Christal Creek receives input from treated water from Galkeno 900 adit, Galkeno 300 adit, and seepages (surface and groundwater) from workings on the west face of Keno Hill. Christal Lake has been a receptor for effluent from various mines including Galkeno 900 and the Mackeno Mill area and Mackeno tailings, contributing to metal loading in Christal Creek.

Lightning Creek is situated within a narrow valley with a steep gradient flowing from the north side of Sourdough Hill into Duncan Creek, which drains into the Mayo River. Hope and Thunder gulches flow into Lightning Creek within the bounds of the KHSD. Lightning Creek has also been the site of extensive placer mining upstream of Keno City both historically and at present time. Treated mine adit discharge from Bellekeno and Onek report to the Lightning Creek drainage.

The Bermingham portal and associated infrastructure is in the No Cash Creek Catchment. The No Cash Creek is situated on the northwest slope of Galena Hill and flows down the hillside towards the wetlands northeast of Flat Creek. There is no direct connection between No Cash Creek and either Flat Creek or the South McQuesten River, as No Cash Creek ends in a bog. From the headwaters on Galena Hill to dispersion in the bog, the distance is roughly 2.3 km.

2.3.3 WATER QUALITY

The KHSD has an extensive database of environmental monitoring and environmental impacts assessments, going back twenty years in some areas. This is in large part due to the historic operations and the reclamation planning requirements. The additional environmental requirements for both operations and closure have been clearly defined through the permitting processes.

Geochemical and water quality studies consistently show that the site is not a source of acid rock drainage. However, oxidation of sulphides and metal leaching under circumneutral conditions does occur, with local zones of acidity in areas of higher sulphide material, particularly proximal to the mineralized veins. The tailings are neither net acid generating nor a source of metal leaching. Tailings are deposited in a lined dry stack tailings facility which is progressively reclaimed during operations. There are comprehensive waste management, water management and monitoring programs defined by permits and in effect on site to ensure compliance with water licence requirements.

The site is permitted to discharge water from both the mill pond and from the different mines. The Water Licence has varying standards for discharge water chemistry depending on the receiving environment. There is a substantial network of environmental monitoring stations, combined with many years of data and water modelling that shows that operations are not constrained by the net positive water balance, although water must be carefully managed across the site. Details are documented in the issued Water Licence QZ18-044.

The AKHM management plans required under that licence specify the site monitoring, surveillance, reporting and physical inspection requirements to support that plan. An adaptive management plan has also been designed to guide responses to unforeseen or contingency events with respect to water quality in the receiving environment.

2.3.4 METEOROLOGICAL DATA

Meteorological data have been collected in the KHSD since 2007 at the Calumet weather station as part of the development of the ESM Reclamation supporting studies), since 2011 at the Keno District Mill meteorological station (installed as part of Bellekeno mining operations) and since 2012 at the Valley Tailings meteorological station. A detailed description of all site monitoring is available in the Monitoring and Surveillance Plan. Briefly, all three stations collect air temperature, relative humidity, rainfall, solar radiation, wind speed and wind direction. In addition, the Keno District Mill station has a snowfall conversion adaptor and calculates evapotranspiration, while the Valley Tailings station collects barometric pressure and soil water content. The Calumet station collects soil temperature. A Yukon Government monitored snow course station located at 1,310 masl elevation also exists in the area and has been monitored for over 30 years. Snow survey locations have been established at Keno Hill Silver District Mill Site, Bellekeno and New Bermingham sites. Information collected is submitted as part of the annual report.

3. ENVIRONMENTAL MANAGEMENT

3.1 AKHM MANAGEMENT PLANS

The Water Licences and Quartz Mining Licence require a series of management plans for operations. Some plans are cited within this document. The complete list of required plans is shown in Table 3-1.

Table 3-1: Plans Under the Existing Mine Licenses

QML-009 Plans	WL QZ18-044 Plans
Adaptive Management Plan	Adaptive Management Plan
Dry Stack Tailings Facility Construction and Operation Plan	Attenuation Study Plans
Dust Abatement and Monitoring Plan	Bioreactor Design and Operation Plan
Emergency Response and Health and Safety Plan	Environmental Monitoring, Surveillance and Reporting Plan
Environmental Monitoring, Surveillance and Reporting Plan	Groundwater Monitoring Plan
Explosives Management Plan	Hydrogeology Monitoring Plan
Hazardous Materials Management Plan	Operations and Maintenance Plan
Heritage Resources Protection Plan	Physical Inspections and Reporting Plan
Mill Development and Operation Plan	Reclamation and Closure Plan
Mine Development and Operation Plan	Sludge Management Plan
Noise Management Plan	Spill Contingency Plan
Reclamation and Closure Plan	Tailings Characterization Plan
Road Development and Operations Plan	Water Management Plan
Sediment and Erosion Control Plan	Water Treatment System Operations Manuals
Spill Contingency Plan	
Tailings Management Plan	
Traffic Management Plan	
Waste Management Plan	
Waste Rock Management Plan	
Wildlife Protection Plan	

3.2 MINE WATER MANAGEMENT

The Bellekeno mine is located above the valley floor; therefore, total groundwater inflow is expected to be limited and is within 3-4 lps. Dewatering will still be required to remove service water from the mine. Bermingham and Flame & Moth mines are both expected to have higher groundwater flows. For the Flame & Moth mine, the maximum groundwater flow is expected to be 33 L/s. For Bermingham mine, the maximum inflow of 11 L/s has been used for engineering purposes.

For all the mines, the dewatering strategy is to use electric submersible pumps to collect water from sumps near the active mining areas and pump it in stages to the dirty water sump located on the ramp. The dirty water will decant to a clean water sump where a clean water pump will pump the water to the surface for recycling or treatment and eventual discharge. Dirty water sumps will include the use of a Sturda wier filter cloth curtain to separate solids from clean water that is then pumped to surface for additional water treatment.

3.2.1 BERMINGHAM

The new Bermingham underground development and production will require up to 140.1 m^3/day which includes a contingency of 25%. Daily water usage during ongoing underground mine development and operation is estimated at 112.5 m^3/day when mining at an estimated maximum rate of 400 tonnes per day (tpd). This water is used for percussion drilling, dust suppression, equipment cooling, and minor use for sanitation. The surface water management structures for the new Bermingham mine include storage ponds, diversion ditches, and a water treatment plant. During advanced exploration and ramp development, water was pumped from the bottom of the underground decline to a series of underground sumps before being conveyed to the surface sump, which then discharged to ground.

The underground workings at Bermingham will not be connected to the historical underground workings. A comprehensive hydrogeological investigation was completed at the Bermingham deposit to assess the hydrological conditions associated with development of the deposit. The hydrogeological investigation estimates that up to 960 m^3/d or 11.1 L/s of groundwater inflow could be expected to be encountered during the operation of the Bermingham mine. Water Licence QZ18-044 authorizes the Bermingham mine to discharge up 1,200 m^3/d, which incorporates 25% contingency from the groundwater inflow estimate.

The Bermingham mine requires continual dewatering, with discharge flows dependent on mine depth. The Bermingham mine has been licenced to discharge a maximum of 1,200 m^3/d or 13.9 L/s. A conventional water treatment plant using lime addition for metals removal and break point chlorination for ammonia treatment is located adjacent to the Bermingham portal.

Recycle of water from underground workings is the primary source of water for the Bermingham mine, similar to the Bellekeno and Flame & Moth mine operations. If required volumes for underground workings are not available water will be sourced from the Bermingham water treatment pond and as a contingency water will be sourced from a groundwater well near the portal.

Bermingham mine water chemistry will be similar to that observed at the historical Bermingham 200 adit. Discharge from the underground is conveyed through the water treatment plant into a lined settling pond, which decants to the No Cash Creek catchment. Water will be treated to meet effluent quality standards before being discharged. The pond has a total capacity of 206 m^3 including freeboard volume of 46 m^3 that accommodates the 24 hour maximum rain event of 48.7 mm.

If excess water accumulates in either the P-AML WRSF or the temporary ore/P-AML pad, the water will be collected by a Vac truck and transported to the water treatment plant as done at the Bellekeno mine. The amount of water collected for transport per week is expected to total less than one vac truck volume. This estimate is due to the size of the WRSF and pad footprints and the amount of precipitation, evaporation and sublimation that occurs.

A diversion ditch is currently excavated up gradient of the new Bermingham portal, which diverts surface water runoff from reaching the portal pad, infrastructure, and collection pond. This will be extended around the N-AML waste rock disposal area to divert clean water around the portal pad and waste rock disposal area. The water within the diversion ditch will infiltrate to ground approximately 0.65 km uphill from the headwaters of No Cash Creek. The diversion ditch has been designed to convey the 24 hour maximum rain event of 48.7 mm.

3.2.2 FLAME & MOTH

The Flame & Moth deposit extends below the valley floor and for that reason there is potential for increased inflows of water. The groundwater investigations and modelling predicted the maximum potential mine water inflow of 35 L/s or 3,024 m^3/d at a mine depth of 270 m below ground surface.

A dirty water and clean water sump have been constructed as part of the current decline development, located approximately 100 m down the decline from the surface portal. Metso dirty water centrifugal pumps are planned in two parallel banks of three pumps each in series. Underground sumps are planned as follows: 794, 724, and 651 at the lowest area of Lightning and one at the lowest area of Christal. These sumps will be equipped with 45 kW dirty water submersible pumps to pump to the main sump. The water is pumped from underground into a water treatment plant located within the mill, for pH adjustment, and removal of both dissolved metals and suspended solids. The treated water discharges to a lined pond for sampling prior to discharge to either Lightning or Christal Creek.

3.3 WASTE ROCK MANAGEMENT

Studies conducted throughout the KHSD provide a detailed foundation for understanding the weathering behavior or 'geoenvironmental' characterization of rock in the KHSD. These studies were used to design a classification system for waste rock management and inform the waste rock management plan.

Rock is classified as to the potential for acid generation and/or metal leaching (P-AML or N-AML). Geochemical screening criteria are defined for each deposit, and proportions of P-AML and N-AML material by rock type estimated to plan materials handling and meet licence conditions for the proposed development activities. The AKHM Waste Rock Management Plan methodology is used to classify waste rock as P-AML or N-AML during active mining, and each round is managed accordingly.

The majority of the waste rock excavated is expected to be N-AML. Waste rock field classified as N-AML will be stored in designated locations at each of the mine sites. P-AML waste rock is categorized as waste rock and mineralized waste rock of no economic interest with increased likelihood for acidic or metal leaching. Rock field-classified as P-AML (mainly pyrite rich graphitic schist) will be stored in designated P-AML waste rock storage facilities or permanently stored underground as cemented back fill within excavated stopes. Any water that enters a P-AML waste rock storage facility will be collected and treated. Additionally, monitoring upgradient and downgradient of each P-AML facility is done as part of ongoing site monitoring.

Based on mine planning, the projected waste rock amounts for each of the mine sites is presented in Table 3-2. The actual quantities and the predicted life of mine (LOM) totals will change as mining advances. The current mine plan and schedule will be considered to have the correct values.

Table 3-2: Waste Rock Storage

	Bellekeno	Bermingham	Flame & Moth
Backfill	21,900	147,100	422,600
Max N-AML on surface (tonnes)[a]	0	190,000	125,000
Max P-AML on surface (tonnes)[a]	2,073	16,000	12,000
Total Waste Rock to be produced (tonnes)	0	409,518	392,630
Total Waste Rock to Backfill (tonnes)	0	381,652	210,859
Remaining Waste Rock on Surface (tonnes)	0	27,867	118,033

The specifics of waste rock management characterization, handling and final deposition are documents in the AKHM Waste Rock Management Plan Revision 6.4. The mine operations measures are detailed in the following chapter of this document.

4. MINERAL RESOURCES AND RESERVES

Alexco issued the results of a Pre-Feasibility Study (PFS) for the Keno Hill Silver project in April 2019, which was then replaced with the current report, entitled NI 43-101 Technical Report on Updated Mineral Resource and Reserve Estimate of the Keno Hill Silver District (Alexco, 2021).

The mineral reserves within the PFS comprises mining four deposits (also referred to as "mines"): Bermingham, Flame & Moth, Bellekeno, and Lucky Queen. The current Alexco Life of Mine operating plan only includes Bellekeno, Bermingham and Flame & Moth mines. The majority of the mill feed (over 94%) will come from Bermingham and Flame & Moth deposits. Two mines will be operating at any given time, with the exception of the initial ramp up period of ore from Bellekeno only. The Bellekeno reserves have been mined out at the time of this document and the mine is entering a temporary closure period. Although the Lucky Queen mine is included in the mineral reserves, it is not currently in the Alexco LOM plan until it is authorized with a water use licence.

The estimated Mineral Resource for the KHSD includes the Bellekeno, Lucky Queen, Flame & Moth, Onek, and Bermingham deposits. The total estimated Mineral Resources inclusive of estimated Probable Mineral Reserves is shown in Table 4-1.

Table 4-1: Keno Hill Mineral Resources at January 01, 2021

Category	Tonnes (t)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)	Contained Ag (oz)
Indicated	3,826,800	596	0.34	2.1	5.4	73,352,000
Inferred	1,719,600	442	0.2	1.4	3.9	24,413,000

Notes:

1. All Mineral Resources are classified following the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) of NI 43-101.
2. Indicated Mineral Resources are inclusive of Probable Mineral Reserves estimates.
3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All numbers have been rounded to reflect the relative accuracy of the estimates.
4. The Mineral Resource estimates comprising Lucky Queen and Flame & Moth, Onek and Bermingham are supported by disclosure in the news release dated March 28, 2019 entitled "Alexco Announces Positive Pre-Feasibility Study for Expanded Silver Production at Keno Hill Silver District" and the Technical Report filed on SEDAR dated February 13, 2020 with an effective date of March 28, 2019.
5. The Mineral Resource estimate for the Bermingham deposit is based on Mineral Resource estimates having an effective date of March 28, 2019.
6. The Mineral Resource estimate for the Lucky Queen, Flame & Moth and Onek deposits have an effective date of January 3, 2017.
7. The Mineral Resource estimate for the Bellekeno deposit is based on an internal Mineral Resource estimate completed by Alexco Resource Corp. and externally audited by SRK Consulting Inc., having an effective date of January 01, 2021. This Mineral Resource estimate has been depleted to reflect all mine production from Bellekeno to the end of December 2020.

The estimated Probable Mineral Reserves calculated by the Qualified Person (from Mining Plus Canada) for this Project are 1.44 Mt grading 804 g/t Ag, 2.64% Pb, 3.84% Zn and 0.31 g/t Au for an overall Ag equivalent (AgEq) grade of 1,035 g/t as of April 1, 2021.

The Mineral Reserves (Table 4-2) show the total Mineral Reserves for the KHSD; all Mineral Reserves are Probable Mineral Reserves. External dilution and mineable recovery has been applied to the Mineral Reserves. Please note that rounding of tonnes, average grades, and contained metal may result in apparent discrepancies with totals rounded.

Table 4-2: Mineral Reserves, Alexco Resource Corp. – Keno Hill Silver District Project

Deposit[3]	Category	Tonnes	Ag (g/t)	Pb (%)	Zn (%)	Au (g/t)	Contained Metal			
							Ag (000 oz)	Au (000 oz)	Pb (M lbs)	Zn (M lbs)
Bellekeno	Proven	-	-	-	-	-				
	Probable	12,809	936	13.00	7.30	0	385	0	4	2
Bellekeno Surface Stockpile	Proven									
	Probable	3,397	1150	21.70	4.50	0	126	0	2	0
Lucky Queen	Proven	-	-	-	-	-	-		-	-
	Probable	70,648	1,269	2.71	1.56	0.13	2,883	0	4	2
Flame & Moth	Proven	-	-	-	-	-	-		-	-
	Probable	721,322	672	2.69	6.21	0.49	15,590	11	43	99
Bermingham	Proven	-	-	-	-	-	-		-	-
	Probable	630,173	899	2.26	1.30	0.13	18,209	3	31	18
Total	Proven	-	-	-	-	-	-		-	-
	Probable	1,438,349	804	2.64	3.84	0.31	37,193	14	84	122

Notes:

1. Mineral Reserves are reported herein based on an NSR cutoff value using estimated metallurgical recoveries, assumed metal prices and smelter terms, which include payable factors, treatment charges, penalties, and refining charges.
2. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces.
3. The Bellekeno, Lucky Queen, Flame & Moth and Bermingham deposits are incorporated into the current mine plan supported by disclosure in the news release dated May 26, 2021 entitled “Alexco Announces 22% Increase to Silver Reserves; Updated Technical Report Demonstrates Robust Economics at Keno Hill”.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

The surface drill holes used to compile the Flame and Moth resource are shown in Figure 4-1.

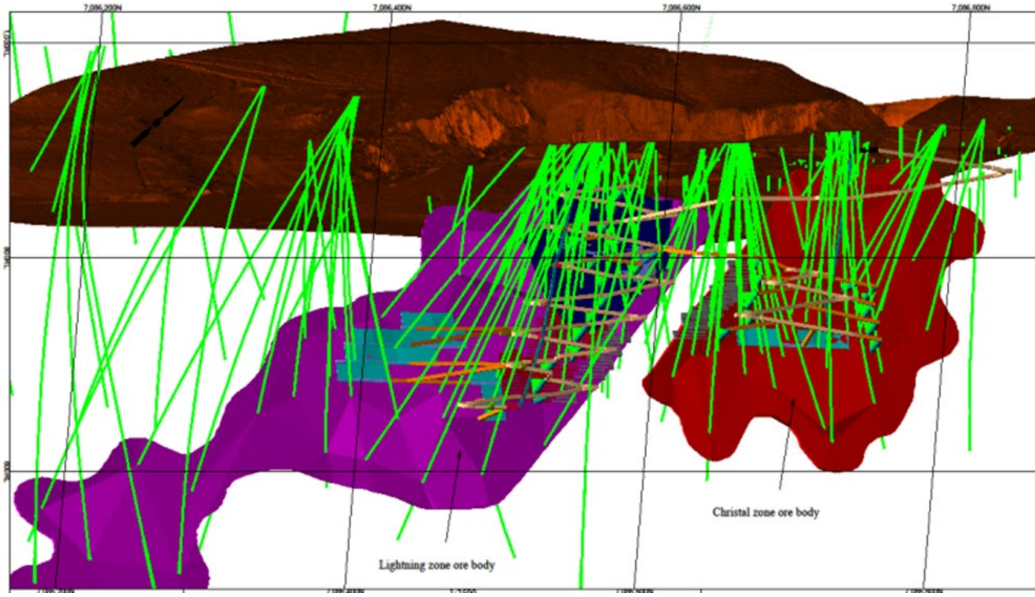


Figure 4-1: Flame & Moth Deposit Summary of Drill Holes

The underground and surface drill holes used to compile the Bermingham resource are shown in Figure 4-2.

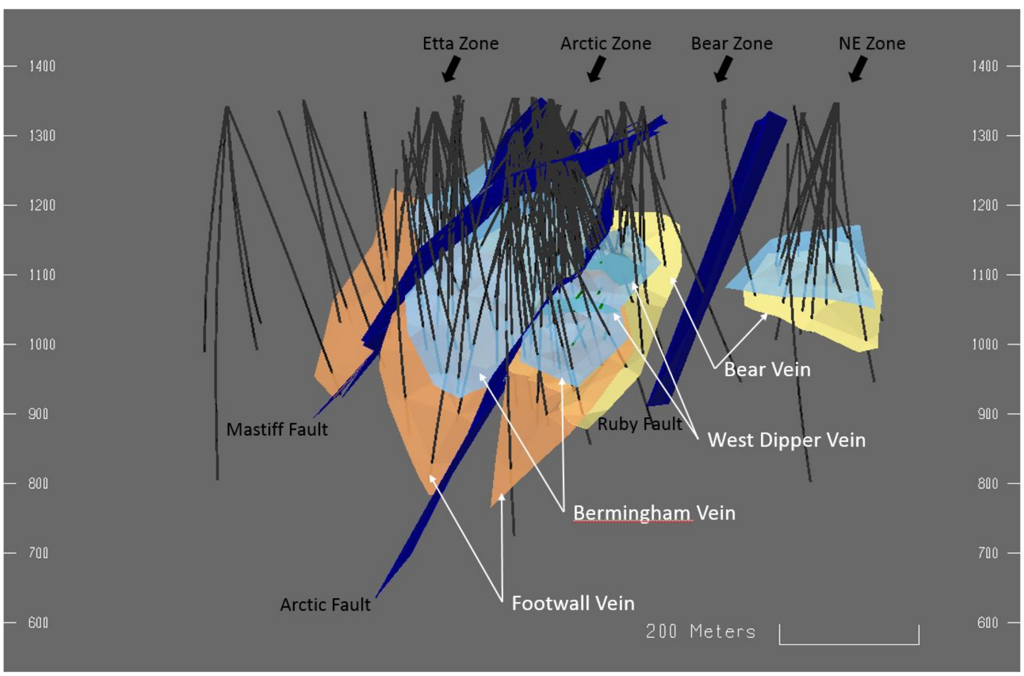


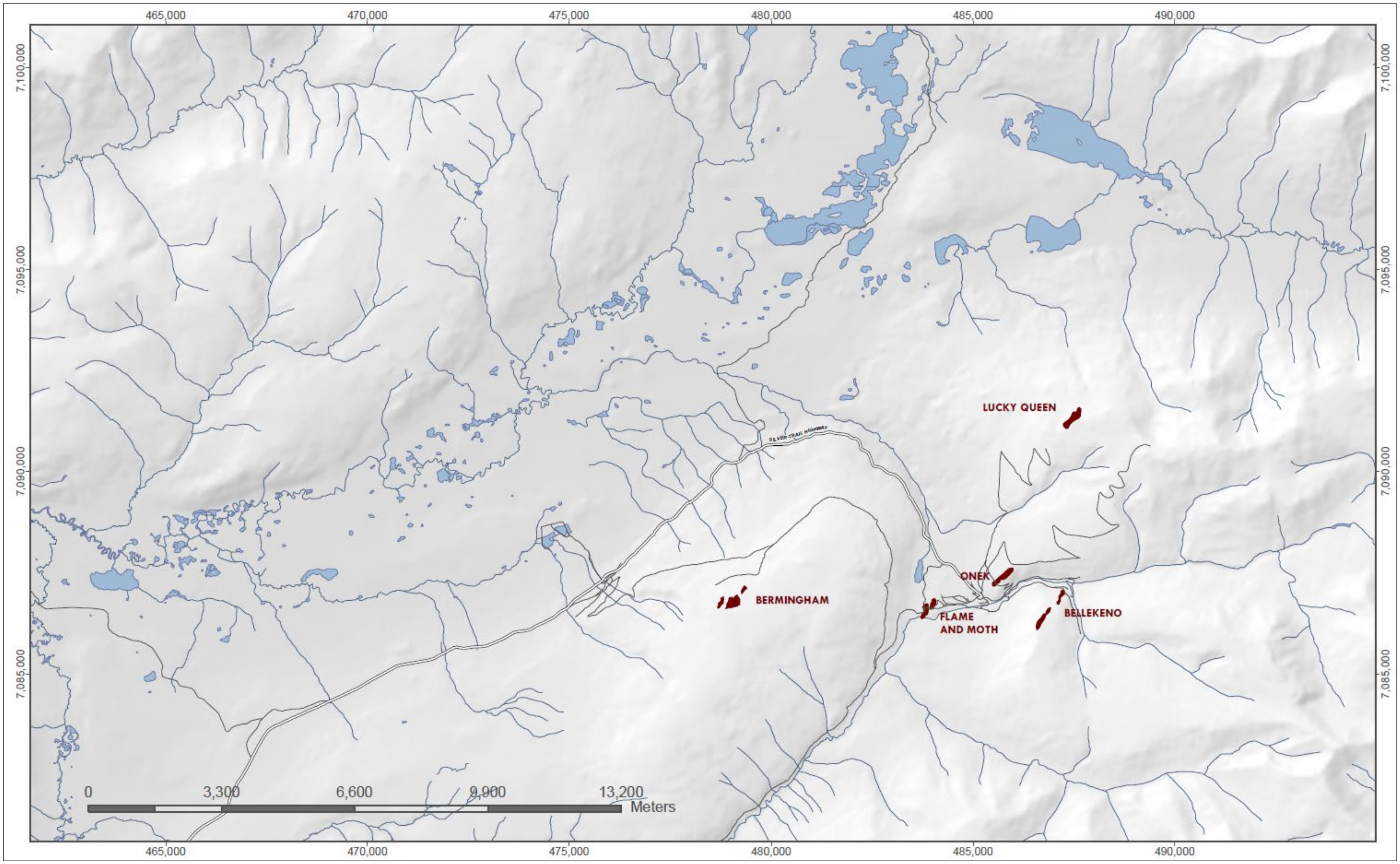
Figure 4-2: Bermingham Deposit Summary of Drill Holes

5. MINE DESIGN AND MINING METHODS

5.1 INTRODUCTION

The Keno Hill Silver District (KHSD) project currently contemplates mining from four separate deposits: Bellekeno, Bermingham, Flame & Moth, and Lucky Queen. The locations are shown in Figure 5-1; the mill, administration, and shop complexes are located near the Flame & Moth Deposit.

All are characterized by high-grades, narrow vein widths, and challenging ground conditions. Historical mining methods used in the KHSD have included cut and fill, small scale longhole stoping, shrinkage stoping, and square set stoping. All deposits will be mined by mechanized underground mining methods of cut and fill (MCF) and longhole open stoping (LHOS) with both methods utilizing cemented rock fill and unconsolidated rock fill as required.



Source: Alexco 2020

Figure 5-1: Deposit Location

5.2 GEOTECHNICAL ANALYSIS

The KHSD Project is composed of four silver / lead / zinc narrow-vein hosted deposits located throughout the district; Bellekeno Deposit (BK); Lucky Queen Deposit (LQ); Flame & Moth Deposit (FM); and Bermingham Deposit (BM). It is noted that the Flame & Moth and Bermingham deposits are comprised of several zones. All deposits will be extracted using overhand mechanized cut and fill (MCF) and longhole open stoping (LHOS) underground mining methods with cemented rock fill (CRF) and unconsolidated rock fill (URF) backfill types as required. Extraction will be bottom-up within panels; however, panels will be developed top-down and temporary sills will be utilized to allow extraction to proceed as ore is accessed via the decline.

Production mining has previously been undertaken at the Bellekeno deposit by Alexco. However, the Bermingham and Flame & Moth deposits are 'greenfield', with decline development being completed at both deposits to enable underground resource definition and geotechnical drilling to be undertaken. Numerous geotechnical studies have been carried out on the KHSD most recently by Jacobs Engineering (Jacobs, 2019).

5.2.1 GEOTECHNICAL DOMAINS

To understand the ground conditions at the KHSD Project, geotechnical domains were created for the Bermingham and Flame & Moth deposits. Preliminary geotechnical parameters were assessed using major lithology types as identified by Alexco geology personnel. The geotechnical domains are outlined below on which ground support designs have been based:

- Domain 1: Quartzite (waste development);
- Domain 2: Schist (waste development);
- Domain 3: Faults (waste and production development); and
- Domain 4: Mineralization (production development).

The rock mass descriptions for Keno Hill are outlined in Table 5-1.

Table 5-1: General Summary of Rock Mass for Keno Hill

Rock Mass Quality	Domain	GSI			Q Value
		Structure	Surface	Value	
Fair to Good	Domain 1 - Quartzite	Block Very Blocky	Rough Smooth	45 - 60	2.0 - 6.0
Poor to Fair	Domain 2 - Schist	Very Blocky Seamy	Smooth Weathered	30 - 45	0.3 - 2.0
Poor to Fair	Domain 3 - Mineralization	Very Blocky Seamy	Smooth Weathered	30 - 45	0.3 - 2.0
Extremely Poor to Poor	Domain 4 - Faults	Disintegrated Foliated	Slickensided	20 - 30	0.05 - 0.3

5.2.2 PILLAR SIZING

Based on previous production experience at the KHSD Project, the following controls are used to form stable rib and sill pillars:

- Rib Pillars: lateral extent of production development will be controlled by; deposit geometries; the placement of central access development; and the positioning of rib-pillars in uneconomic material throughout the deposits. Conversations with Alexco personnel indicate this to be achievable based on previous production experience at the KHSD Project; and
- Sill pillars: the use of a top-down extraction approach will result in sill pillars being formed. Sill pillar stability will be controlled by; the construction of sill matts (based on previous experience), timely emplacement of cementitious backfill, and careful mining practices.

5.2.3 GROUND WATER

The Bellekeno, Lucky Queen, and Bermingham mining zones are located above the valley floor. Consequently, this tends to limit the occurrence and effect of adverse hydrogeological conditions. However, the Flame & Moth deposit is located in a valley floor such that there is a possibility of higher water inflow to the planned workings. Preliminary investigations suggest that ground water may be structurally compartmentalized in the deposit-scale Mill Fault.

5.3 MINING METHODS

While production mining at KHSD project will predominantly use MCF mining method, several zones of the Bermingham and Flame & Moth deposits will be extracted using LHOS. In these areas, all available geotechnical drill hole data proximal to the planned stope hanging wall was used for stability analysis. Within the MCF mining method production areas, there will also be short uphole stopes used to extract the temporary sill during retreat. The primary LHOS zones include the following:

- Bermingham Deposit planned LHOS zones:
    - NE Zone; and
    - Arctic Zone (Lower).
- Flame & Moth Deposit planned LHOS zones:
    - Christal Zone (Lower); and
    - Lightning Zone (West).

Figure 5-2 and Figure 5-3 show the Flame & Moth and Bermingham mines with the associated zones and mining methods.

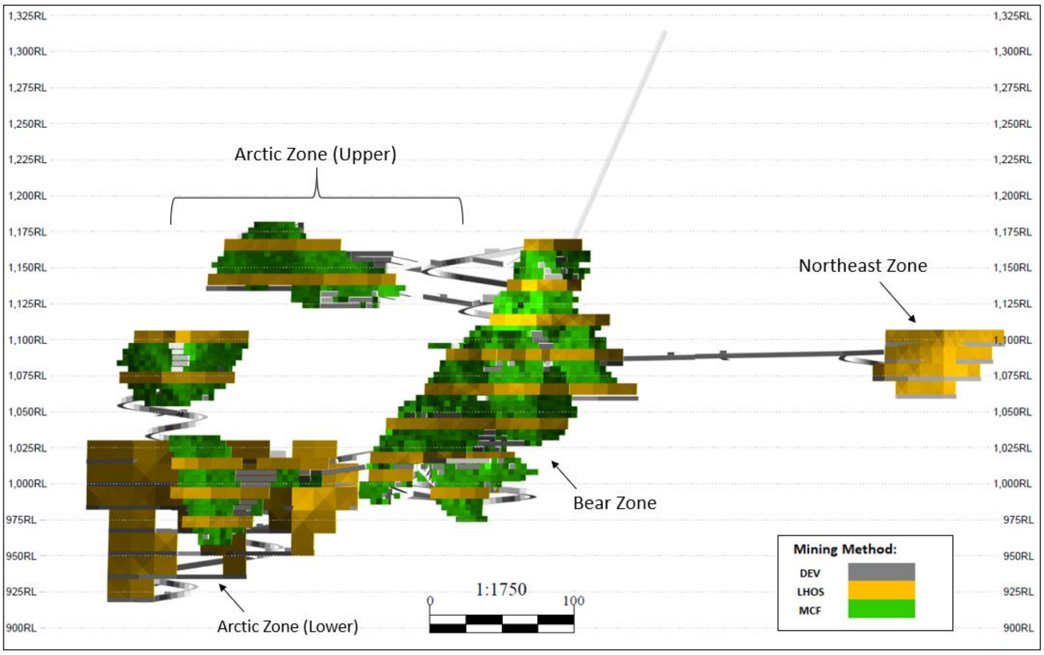


Figure 5-2: Bermingham Zone and Mining Method Domains (Looking NW)

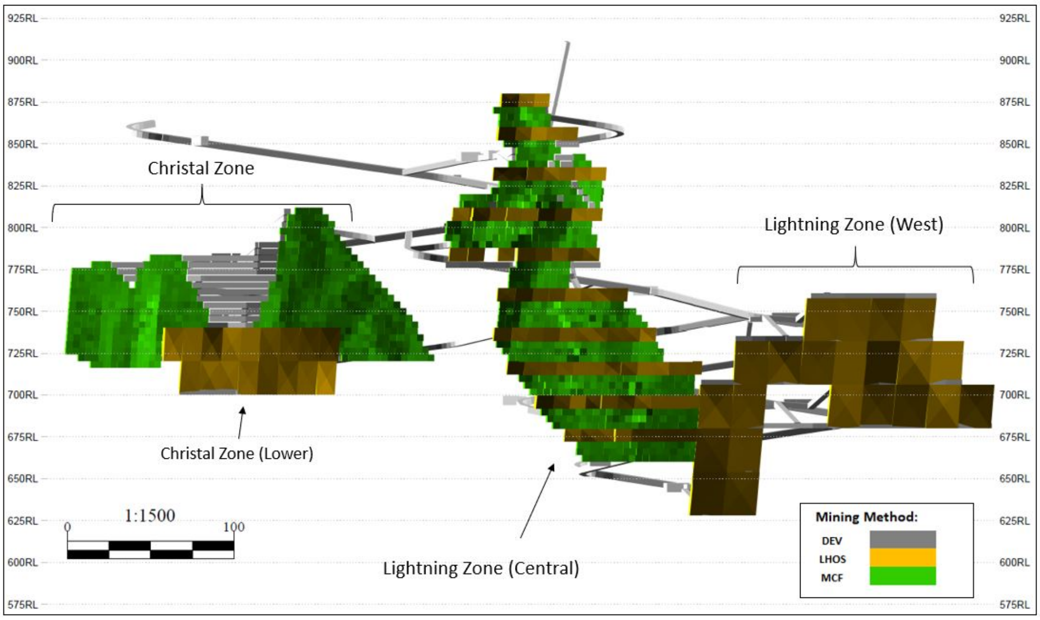


Figure 5-3: Flame & Moth Mine Zone and Mining Method Domains (Looking SW)

The Mining Methods selected and applied are:

- Overhand Mechanized Cut and Fill (MCF):
    - Drift and Fill (D&F) method in the wider areas of the orebody with cemented rock fill (CRF).

- Longhole open stoping (LHOS):
  - LHOS will be standard longitudinal retreat with CRF;
  - Downhole LHOS method will be used where the ground conditions and ore body dip allow; and
  - Uphole LHOS method with no backfill where permitted.

Mining sequences for downhole LHOS stopes are bottom-up with CRF backfill. As for the MCF method – mining sequences are bottom-up, using temporary sills with CRF backfill containing higher cement binder content used only for sill levels. The use of temporary sills will allow the extraction of individual MCF panels to proceed as they are accessed and allow multiple active production fronts. The remainder of the MCF lifts will utilize unconsolidated rock fill for bottom-up mining sequences. Uphole LHOS stopes will be mined with no backfill using a longitudinal retreat sequence. Cemented rockfill mix design and curing times to achieve required strengths were determined by an independent consultant and are detailed in Alexco's standard operating procedures.

These mining methods were chosen due to the narrow steeply dipping nature of the orebodies and to maximize safety and productivity. The various deposits require the use of mining methods that can adequately support the vein and that are flexible and selective while minimizing the direct mining costs.

The main factors driving the mining method selection process are:

- Proven mining methods used at the Bellekeno mine;
- Ground conditions in the vein and along the vein contacts range from good to very poor;
- Ground conditions can vary substantially over short distances (five metres);
- Vein continuity is good; however, the vein geometries vary greatly between deposits;
- Metal content and distribution varies significantly between deposits and varies over the stope mining scale;
- The footwall is often characterized by competent quartzite but can be weak in some areas;
- The hangingwall varies from competent quartzite to weak layers of quartz breccia with clay filled shear bands, graphitic schists, or sericite schists;
- Geological contacts at the hangingwall and footwall can often be visually identified but can be faulted or fractured contacts with gouge and breccias;
- Mineralization contacts are less clearly defined and are based on a combination of structure, vein mineralogy, and metal grades;
- Vein systems can be locally water bearing and required time to drain when they are first crosscut by development; and
- Vein depths are shallow with a low-stress regime, high-stress issues are not a factor in mine planning, but lack of clamping forces contributes to the poor ground conditions.

The mine design strategy was to design as many areas as practical using small scale longhole mining methods while planning mechanized overhand cut and fill for areas where ground conditions were poor, or where the combination of vein dip and true width was not compatible with longhole stoping methods.

5.3.1 MECHANIZED CUT AND FILL (MCF)

MCF will be the dominant mining method for the Flame & Moth, Bermingham, and the principal mining method for the Lucky Queen deposit. In MCF method, an attack ramp is developed from the main ramp at a gradient of -15%. Upon reaching the orebody, an intersection is developed and a lift is developed in both directions along strike, following the geological contact of the orebody. At the end of the lens, the void is backfilled using either unconsolidated rock fill or cemented rockfill (CRF) with a Load Haul Dump (LHD) machine. The LHD utilizes a rammer-jammer plate (a dozer plate modified to be attached to a scoop to push waste tight to the back) to ensure that the backfill is placed tight to the back of the drift.

Once the level has been completely backfilled, the next lift above the previously mined lift is accessed by slashing down the back of the attack ramp and working off the muck pile/horizon. Figure 5-4 illustrates the sequence of activities with MCF mining.

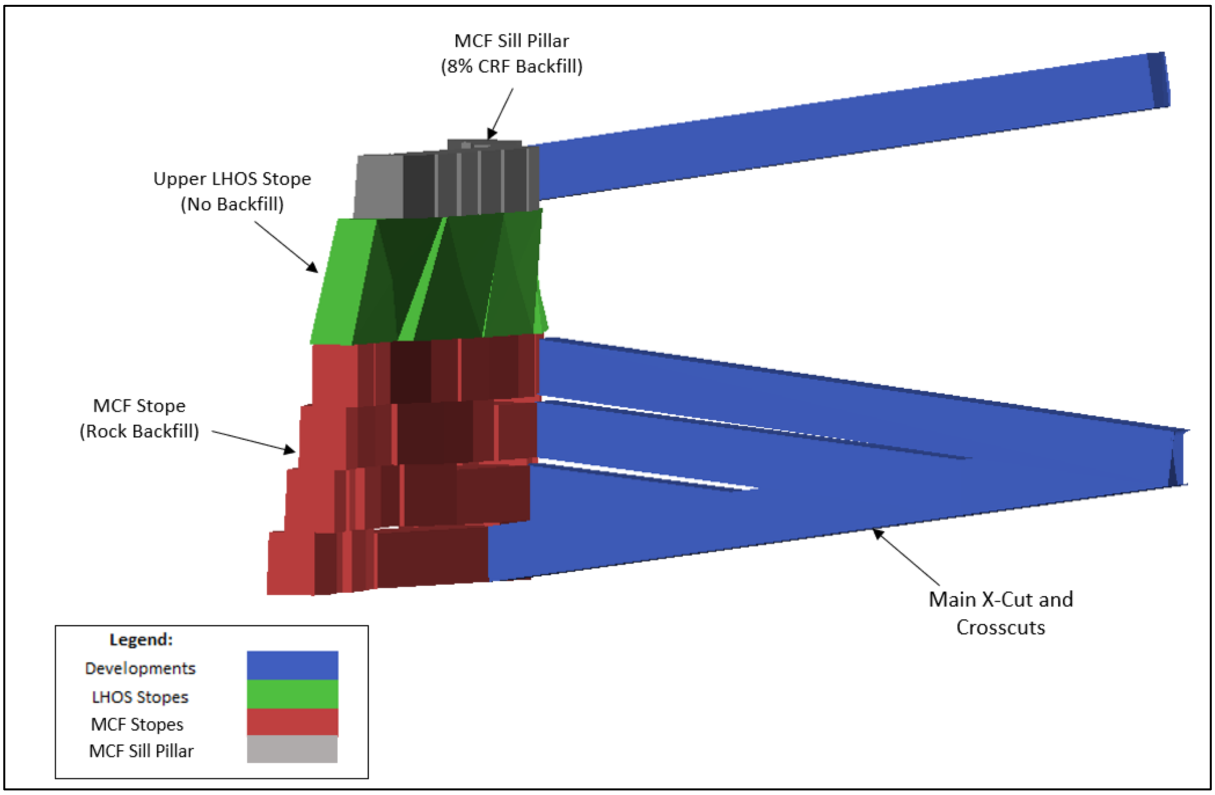


Figure 5-4: MCF lifts with Uphole Stopes Section

MCF drift are on average 4.0 m high with varying widths, based on the deposit geology. For areas wider than development equipment is capable of mining or supporting, a second parallel drift will be mined beside the backfilled drift to fully extract the orebody width prior to accessing the lift above. In this situation, the first drift will be completely backfilled with cemented rock fill to ensure a stable wall to allow adjacent mining activity.

For the Bermingham and Flame & Moth deposits, the lifts are sequenced bottom up within each panel; however, to maximize productivity the panels are mined from the top down as they are accessed by the ramp spiraling down. As such, a pillar will remain between the top lift of one panel and the bottom lift of the panel above. These pillars will be extracted using an uphole drill and blast method discussed in Section 5.3.2.

A variable width shanty-back drift profile has been used to create the MCF method minable shapes for the KHSD mines. The primary reason for implementing a shanty MCF profile is to potentially reduce dilution; however, the development and blasting practices will require strong quality control in order to ensure that additional waste is not mined. The hanging wall (HW) and footwall (FW) are restricted to the following stope dip angle parameters, 90 degrees for FW and minimum 60 degrees minimum/90 maximum degrees for HW. Figure 5-5 depicts a section view of an example shanty back MCF profile versus standard vertically aligned hanging walls.

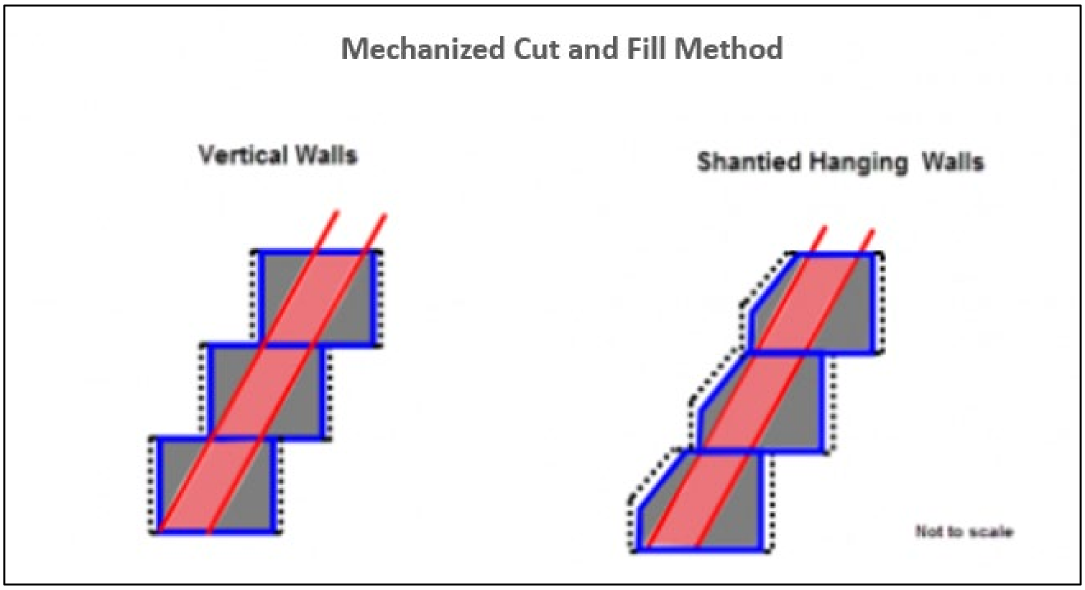


Figure 5-5: Shanty MCF Development Shapes Example

5.3.2 LONGHOLE OPEN STOPING (LHOS)

Longhole Open Stoping (LHOS) will be the preferred mining method when the ground conditions and the lens geometry allow. In LHOS, two drifts are developed along the strike of the orebody at a vertical spacing selected based on geotechnical constraints for that zone. After development is completed, blasting rings are drilled in parallel from the top level to the bottom level. Hole diameter and blast design follow industry best practice and are detailed in Alexco's standard operating procedures. Several rows will also typically be pre-loaded to minimize the loading crew's exposure to the open stope brow.

An initial slot is developed by drilling and blasting a drop raise made up of multiple holes in close spacing. Hole diameter and blast design follow industry best practice and are detailed in Alexco's standard operating procedures. Once this initial slot has been blasted (retaining a minimum pillar below the top drift) the entire stope is blasted and mucked using a LHD. All remote mucking will be carried out using a LHD equipped with a remote package.

Stope strike lengths are based on the geotechnical analysis that has been performed and is detailed in the sites' Ground Management Control Plan (Appendix B and Appendix C). Typical stope lengths vary from approximately 8 m to 20 m.

Once the stope is empty, the stope is backfilled with Cemented Rock Fill (CRF). Cemented rockfill mix design and curing times to achieve required strengths were determined by an independent consultant and are detailed in Alexco's standard operating procedures. Figure 5-6 illustrates the LHOS Method. LHOS will be used in the Flame & Moth deposit at Christal and Lightning zones, as well as NE and Arctic zones of the Bermingham deposit.

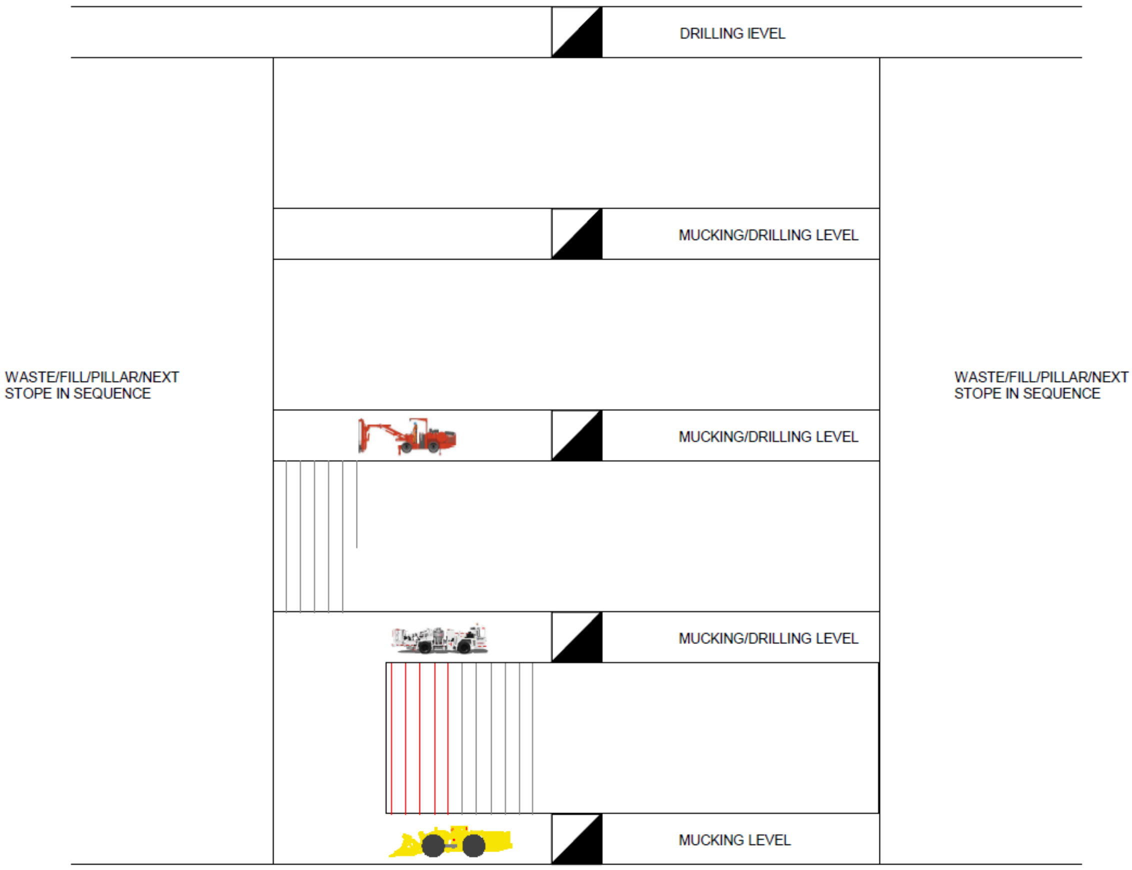


Figure 5-6: (Downhole) Longhole Stoping

A modified version of LHOS will be used at the top of a MCF level to extract the sill pillar between MCF panel and the panel above, or in areas where there is no access for a top drift. Cemented rockfill mix design and curing times to achieve required strengths for the CRF pillar above the uphole stope were determined by an independent consultant and are detailed in Alexco's standard operating procedures. In uphole stoping, a series of parallel rings are drilled from the bottom drift into the back, to the limit of the lens. An inverse raise is drilled and blasted on the extremity of the stope. The longhole rings are then blasted into the void created by the raise and mucked using a remote-operated LHD. No backfill is necessary in this method. Figure 5-7 illustrates the uphole stoping method.

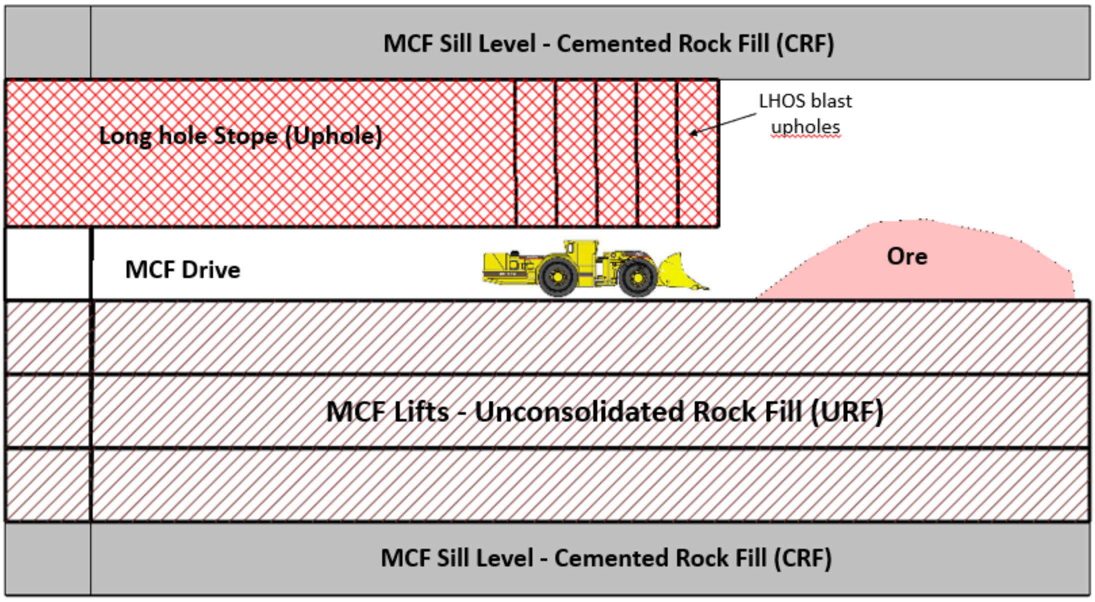


Figure 5-7: Uphole Longhole Stoping

The only mining method used at Bellekeno Mine will continue to be uphole LHOS to extract a mostly developed panel at the beginning of mine production. Uphole LHOS will also be used to extract sill pillars in the Bermingham and Flame & Moth mines. LHOS method will not be used at Lucky Queen Mine.

5.4 MINE DESIGN

5.4.1 FLAME & MOTH MINE PLAN

Flame & Moth mine is located adjacent to the existing mill and administration facilities. Access to the mine will be through an already developed portal and ramp under development. The portal consists of a steel multiplate culvert half arch anchored to concrete blocks and pinned with rock bolts to bedrock. The initial slope of the portal entrance will be at +1% grade to eliminate any surface water from outside the portal flowing down the decline ramp. Secondary drifts including but not limited to a maintenance shop, remucks where required, safety bays, sumps and ore accesses will be completed as the ramp advances.

Development in the Flame & Moth mine will be sized based on development type, equipment dimensions, and geotechnical conditions. Drift dimensions are usually within the 3.5 m to 4.5 m range. The primary mining method is MCF method, except Lightning West and Lower Christal zones which is LHOS method.

Emergency egress and ventilation for the Flame & Moth mine will be provided through a vent raise driven to surface. Figure 5-8 shows a long section of the existing and planned workings. Figure 5-9 shows a typical ramp development section profile with auxiliary services installed.

Ore and waste will be handled by 22 tonne capacity haulage trucks underground and backfill will primarily be handled by smaller 16 tonne capacity haulage trucks. Trucks will be loaded at remuck bays on the ramp systems and will be hauled directly to the surface ore pad.

Life of mine development waste rock broken underground will be hauled to surface. Part of this waste rock will be used for construction and backfill, both as rockfill and CRF. Surface handling and backfill of Flame & Moth waste rock is based on geochemical characterization and an approved waste rock characterization and management as required by the Water Licence.

Ore from the Flame & Moth mine will be sourced from the Lightning and the Christal zones, which contains approximately 68% and 32% of total ore tonnes, respectively. Table 5-2 shows the annual production schedule for the Flame & Moth mine. A distribution of total ore tonnes by mining method for the Flame & Moth mine is 49% MCF and 51% LHOS.

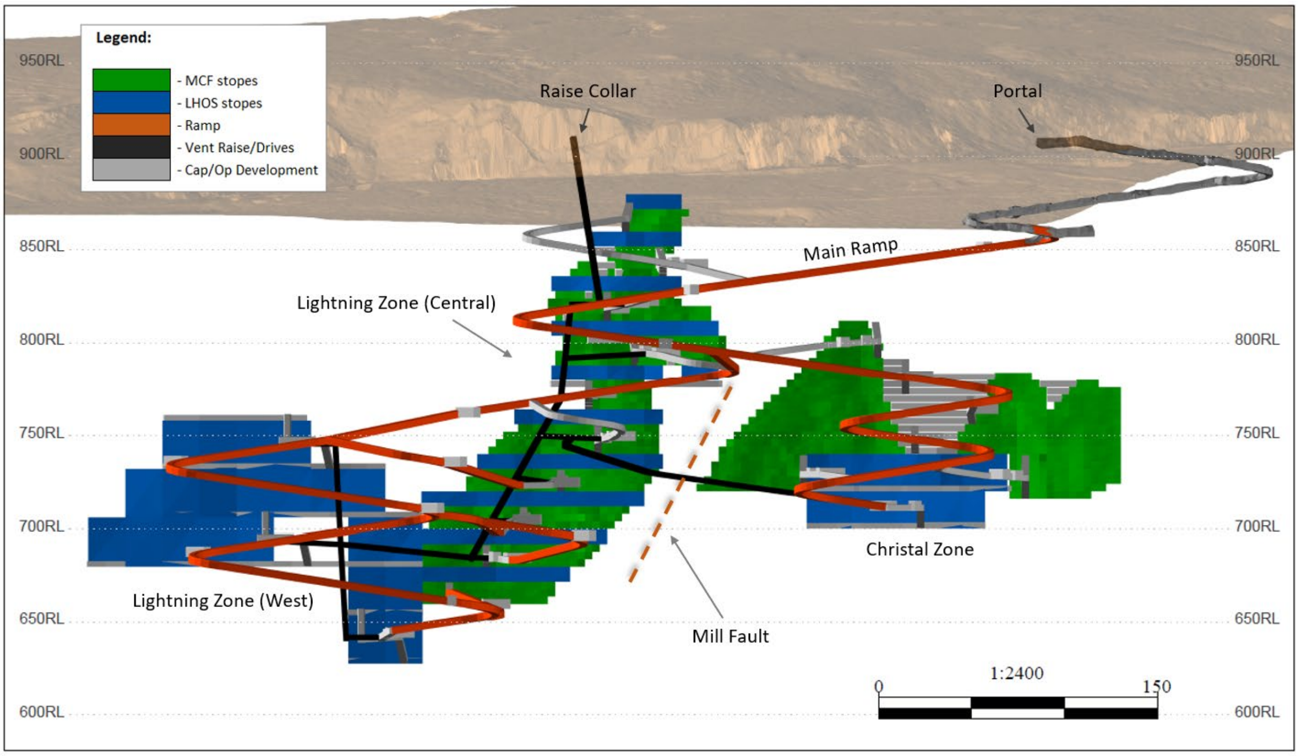


Figure 5-8: Flame & Moth Isometric View (Looking NW)

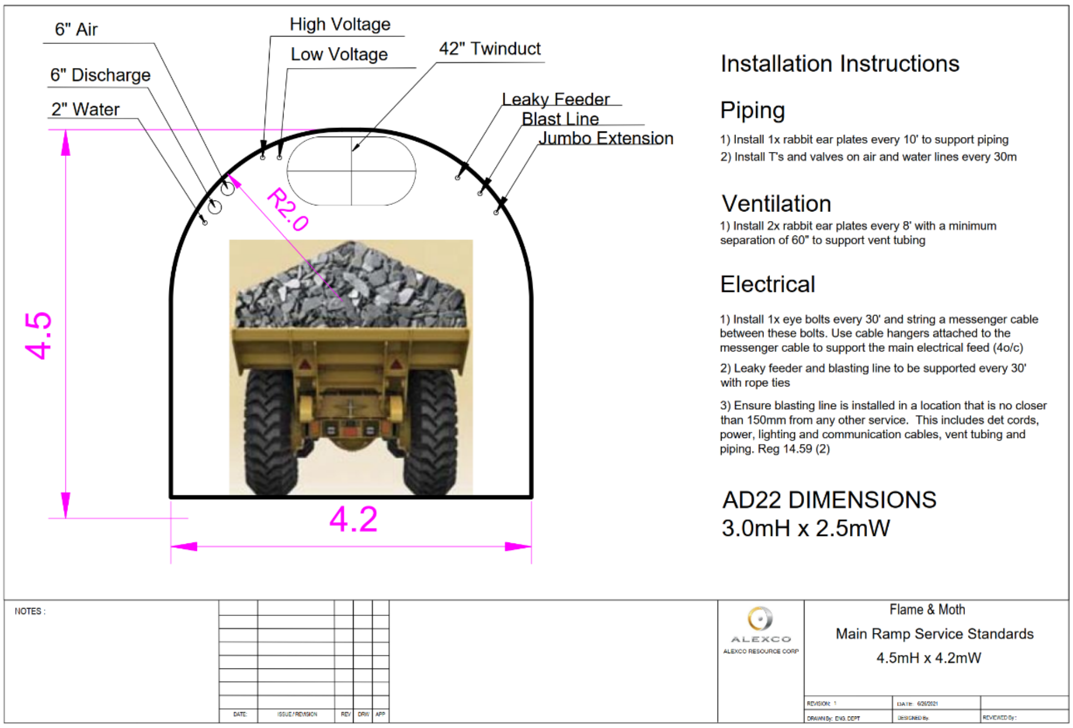


Figure 5-9: Main Access Ramp Development Section Profile

Table 5-2: Flame & Moth Production Quantities

Flame & Moth	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Mill Feed Tonnes	721,322	25,684	64,406	76,761	82,458	82,038	82,308	114,704	158,646	34,315
Waste Tonnes	385,709	47,779	81,247	93,359	102,387	45,943	8,957	6,037	-	-
Total Tonnes	1,107,031	73,463	145,653	170,120	184,846	127,981	91,266	120,741	158,646	34,315
MCF Tonnes	356,198	25,684	42,101	42,438	81,299	69,646	45,050	34,158	15,821	-
LHOS Tonnes	365,124	-	22,306	34,323	1,159	12,391	37,258	80,546	142,826	34,315
MCF Backfill Tonnes	239,514	20,352	29,915	28,812	52,472	51,722	24,103	16,873	15,265	-
LHOS Backfill Tonnes	153,116	-	-	-	-	-	-	42,996	79,989	30,131
Total Backfill Tonnes	392,630	20,352	29,915	28,812	52,472	51,722	24,103	59,868	95,254	30,131
Ag (g/t)	672	648	698	751	961	802	674	550	537	489
Au (g/t)	0.49	0.34	0.43	0.44	0.66	0.64	0.59	0.46	0.38	0.38
Pb (%)	2.69	2.60	2.80	3.13	5.84	3.85	2.09	1.79	1.53	1.09
Zn (%)	6.21	6.72	7.98	5.93	5.05	7.27	5.95	5.80	5.96	6.64
Ag (Oz)	15,590,371	535,432	1,444,769	1,852,558	2,548,967	2,116,083	1,784,214	2,028,692	2,740,173	539,482
Au (Oz)	11,284	279	894	1,075	1,760	1,687	1,555	1,680	1,929	425
Pb (lbs)	42,796,111	1,470,316	3,970,042	5,290,545	10,607,930	6,955,586	3,799,287	4,519,278	5,359,236	823,891
Zn (lb)	98,812,459	3,805,943	11,327,902	10,032,737	9,184,196	13,140,030	10,800,289	14,665,205	20,830,765	5,025,392
Development (m)	8,939	1,127.76	1,740.53	2,228.38	2,333.22	1,099.23	245.62	163.88		-

Notes:
1. Development (m) are lateral and vertical metres.
2. Mill feed tonnes are all Probable Mineral Reserves.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
4. Tonnages are diluted and recovered.
5. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounce.

5.4.2 BERMINGHAM MINE PLAN

The Bermingham mine is located approximately 4 km from the mill underneath the historical Bermingham Pit. Access to the mine will be through an already developed portal and ramp under development. The portal consists of a steel multiplate culvert half arch anchored to concrete blocks, and pinned with rock bolts to bedrock. Secondary drifts including but not limited to a maintenance shop, remucks where required, safety bays, sumps and ore accesses will be completed as the ramp advances.

Development in the Bermingham mine will be sized based on development type, equipment dimensions, and geotechnical conditions. Drift dimensions are usually within the 3.5 m to 4.5 m range. The primary mining method is MCF method, except NE and Arctic zones which is LHOS method.

Emergency egress and ventilation for the Bermingham Mine will be provided through a vent raise driven to surface. Figure 5-10 and Figure 5-11 show an isometric view of the existing and planned workings of Bermingham mine. Typical ramp development section profile and service installation are similar to that outlined for the Flame & Moth.

Ore and waste will be handled by 22 tonne capacity haulage trucks underground and backfill will primarily be handled by smaller 16 tonne capacity haulage trucks. Trucks will be loaded at remuck bays on the ramp systems and will be hauled directly to the surface ore pad. From the surface ore pad all ore will be loaded on 30 tonne articulated trucks to be transported to the mill.

Life of mine development waste rock broken underground will be hauled to surface. Part of this waste rock will be used for construction and backfill, both as rockfill and CRF. Surface handling and backfill of Bermingham waste rock is based on geochemical characterization and an approved waste rock characterization and management as required by the Water Licence. Detailed waste production schedule and waste handling is outlined in Alexco's Waste Management Plan (as referenced in Section 3.1).

Ore from the Bermingham Mine will be sourced from the Footwall, Bear and North East zones as shown in Figure 5-11. The production schedule is based on the assumption that 85% of the geotechnical pillars will be mined as the pillars are located within high grade areas of the deposit and will pay for the additional costs associated with their extraction in challenging ground conditions. Table 5-3 shows the production quantities and table for the Bermingham Mine. The distribution of total ore tonnes by mining method for the Bermingham mine is 48% (MCF) and 52% (LH).

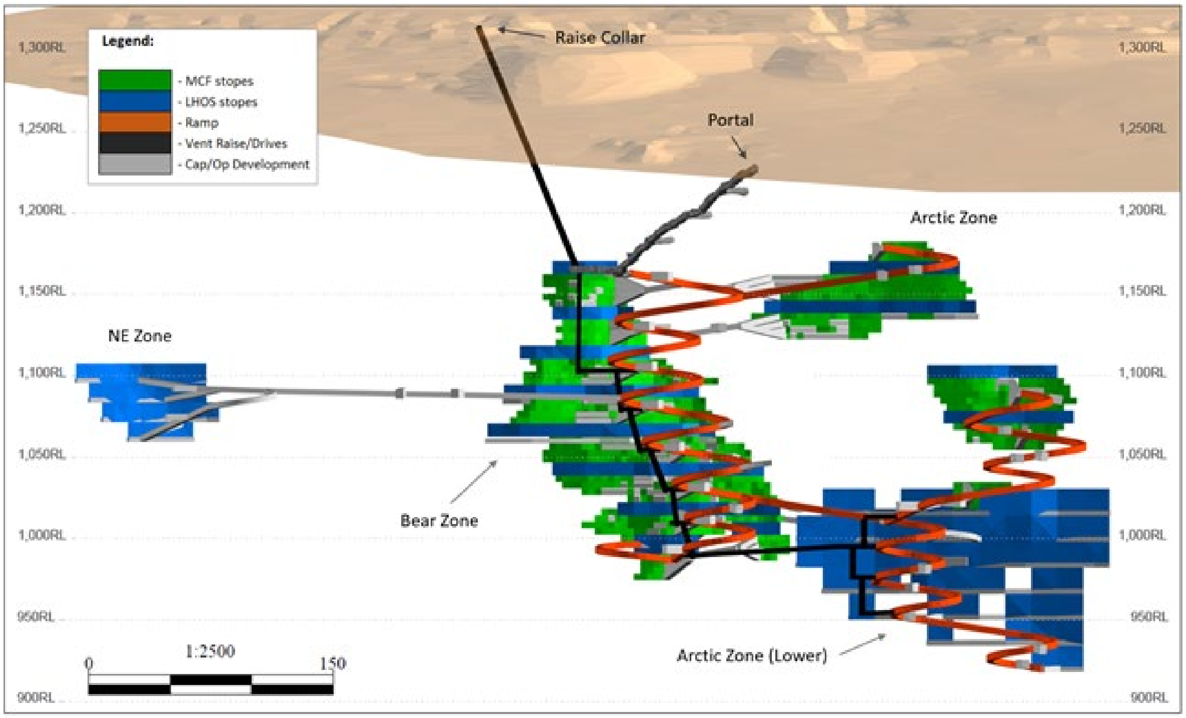


Figure 5-10: Bermingham Mine Isometric View (Looking SE)

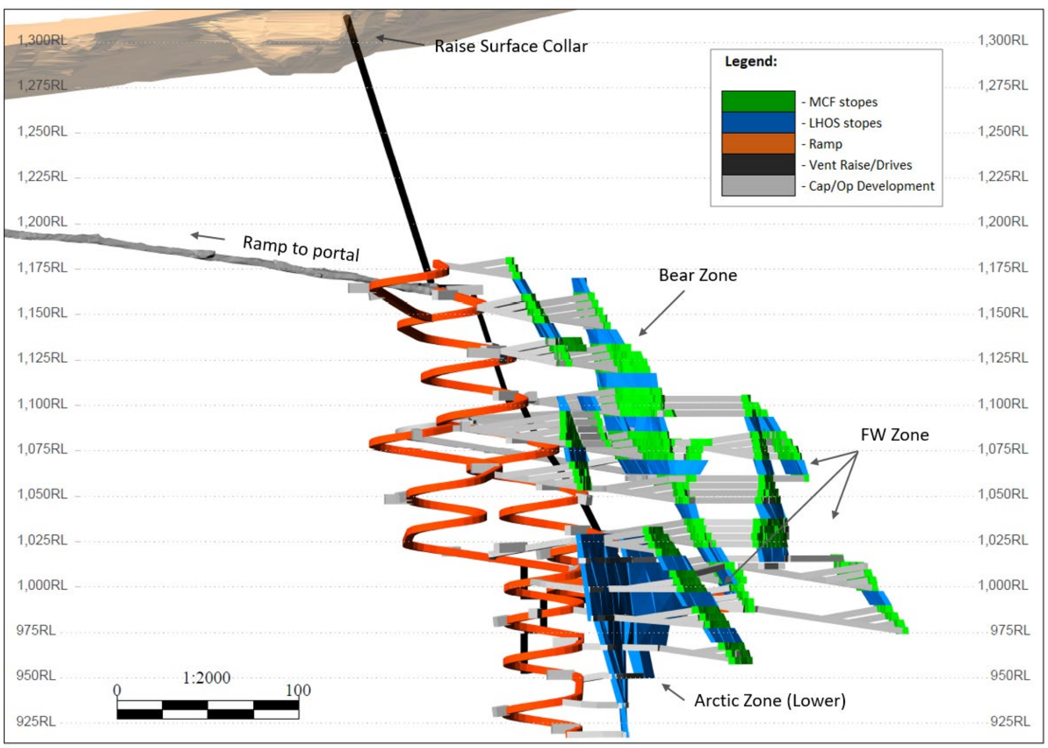


Figure 5-11: Bermingham Mine Isometric View Showing FW Zone (Looking NE)

Table 5-3: Bermingham Production Quantities

Bermingham	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Mill Feed Tonnes	630,173	25,220	67,984	103,692	118,977	119,965	119,849	74,487
Waste Tonnes	410,652	51,676	78,945	94,268	91,132	52,921	35,626	6,083
Total Tonnes	1,040,825	76,896	146,929	197,960	210,109	172,886	155,475	80,570
MCF Tonnes	300,344	25,220	58,645	79,528	57,253	49,274	15,280	15,144
LHOS Tonnes	329,829	0	9339	24164	61723	70690	104570	59343
MCF Backfill Tonnes	233,934	18,166	51,590	54,544	67,827	14,451	13,708	13,648
LHOS Backfill Tonnes	141,768	0	0	0	0	33825	70162	37781
Total Backfill Tonnes	375,702	18,166	51,590	54,544	67,827	48,276	83,870	51,429
Ag (g/t)	899	1,218	1,450	1,011	806	645	844	775
Au (g/t)	0.13	0.10	0.16	0.14	0.13	0.12	0.13	0.11
Pb (%)	2.26	3.02	3.05	2.69	1.78	1.54	2.37	2.47
Zn (%)	1.30	1.74	1.44	1.20	1.27	1.08	1.49	1.24
Ag (Oz)	18,208,749	987,745	3,170,200	3,369,385	3,084,887	2,489,370	3,250,464	1,856,697
Au (Oz)	2,626	82	359	470	486	474	499	256
Pb (lbs)	31,448,248	1,678,736	4,565,451	6,149,704	4,667,661	4,082,547	6,255,292	4,048,857
Zn (lbs)	18,022,151	967,463	2,162,914	2,745,546	3,331,304	2,845,857	3,928,589	2,040,478
Development (m)	9,707	1,247	1,846	2,200	2,257	1,251	741	166

Notes:

1. Development (m) are lateral and vertical metres.
2. Mill feed tonnes are all Probable Mineral Reserves.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
4. Tonnages are diluted and recovered.
5. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounce.

5.5 GROUND CONTROL

Ground support requirements are based on the expected ground conditions, interpreted from geotechnical assessments from the underground exploration declines and geotechnical logging of surface and underground core samples. Third party review on the geotechnical aspects regarding the Keno Hill Silver District has been used to develop site specific ground support standards for both development and production stoping. Specific ground control management plans are included in this document for Bermingham and Flame & Moth in Appendix B and Appendix C respectively. The following summarizes information used for mine design, which in turn guides operational requirements. Site wide procedures are summarized in Section 7.1 and Section 7.2 of this Plan.

Support classes have been determined from the geotechnical domains (see Section 5.2.1), and the geometry of different types of development. Table 5-4 shows the correlation between the domains presented in Section 5.2.1, and ground support classes.

Table 5-4: Ground Support Classes

Ground Support Class	Rock Mass Quality	Domain
Class I	Fair to Good	Domain 1 – Quartzite
Class II	Poor to Fair	Domain 2 – Schist; Domain 3 – Mineralization
Class III	Extremely Poor to Poor	Domain 4 - Fault

In general, the infrastructure is considered to be open for the long-term situation, and support has been designed accordingly. The infrastructure has been designed to avoid areas with potential poor ground conditions; in some situations, this is unavoidable, and support will be increased to provide long term stability.

Ground support standards were designed based on the strength requirements for each ground control class, which are determined to be representative of the likely rock mass conditions. Ground control strategy includes the use of both passive and active ground control components. The general approach to the application of ground support elements is on Table 5-5. In general, poorer ground just receives more ground support elements either in the form tighter spacing between bolts, adding new control element, or increasing the coverage. Intersections also received enhanced support to stabilize wider opening spans.

Table 5-5: Ground Control Strategy

Ground Class	Rock bolts	Mesh	Shotcrete	Coverage	Intersections
Class I	Yes	Yes	No	Back and walls	Extra rock bolts
Class II	Tighter bolt spacing	Yes	No	Back and walls, to sill	Extra rock bolts Shotcrete
Class III	Tighter bolt spacing	Yes	Yes	Back and walls, to sill	Not recommended

Specific ground support requirements are subject to change as more knowledge of the ground is acquired throughout the development of the district. To account for these changes, Alexco's Ground Control Management Plan is to be reviewed yearly.

A more detailed discussion of the rock mass quality designations, the ground support design methods and calculations and the numerical analyses used in design are contained in the site Technical Report (Alexco, 2021) and supporting prefeasibility studies. Also refer to Alexco's Ground Control Management Plan (Appendix B and Appendix C) for more detail on the current specific ground support standards for each outlined ground support class.

5.6 PRODUCTION SCHEDULE

The overall production schedule is based on operating the Keno District Mill to its nameplate capacity of 400 tpd. The mine production schedule includes concurrent operation of Flame & Moth and Bermingham with occasionally a third mine in development phase. The operations were sequenced to maximize Net Present Value and to minimize the number of operations concurrently active whilst satisfying the mill throughput targets.

The Bellekeno reserves will be mined out by Q4 of 2021. In parallel, development will be advanced in the Bermingham and Flame & Moth deposits to prepare them for full production by Q1 2022. Over the project life, Bermingham and Flame & Moth will be the two main ore sources to the mill.

Table 5-6: Plant Feed

Mine	Diluted Tonnes	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
Bellekeno	16,206	981	-	14.82	6.72
Bermingham	630,173	899	0.13	2.26	1.30
Flame & Moth	721,322	672	0.49	2.69	6.21
Total Plant Feed	1,367,701	780	0.31	2.64	3.95
Notes:					
1. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content. 2. All Diluted tonnes are probable Mineral Reserves. 3. Tonnages are diluted and recovered. 4. Tonnage and grade measurements are in metric units.					

5.7 DEVELOPMENT AND PRODUCTION CYCLES

Waste and ore development consist of common mining cycle activities including drilling, blasting, mucking, ground support and backfill. Each of these activities are further described. At each level access, a standardized and typical level access layout is established. Figure 5-12 shows a typical level access that will be developed at both Flame & Moth and Bermingham.

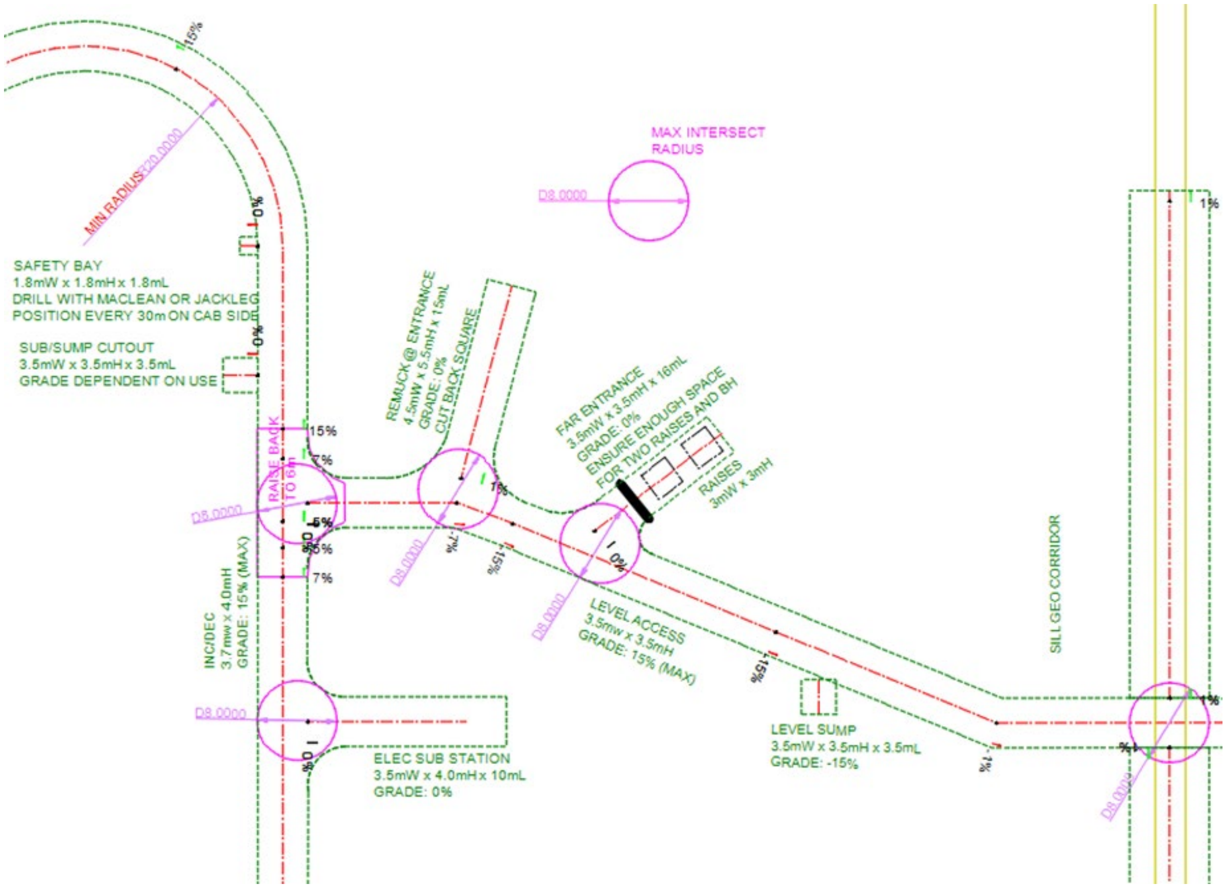


Figure 5-12 Typical Level Access Design

5.7.1 DRILLING PROCEDURES

A twin boom jumbo will be dedicated to the development of ramps, level accesses and sill development in the cut and fill stopes. A second single boom jumbo will be used in the smaller cut and fill headings and as a back-up for when the main jumbos are down for maintenance and servicing. In addition, jacklegs may be required if the vein geometry dictates.

Standard operating procedures are in place for drilling, loading, and blasting to ensure a safe work environment for all Alexco personnel, contractors, and visitors. These procedures follow all legislative requirements as set out in the Yukon OH&S Regulations and include but are not limited to the following.

5.7.1.1 Face Preparation and Drilling:

1. Before drilling on any face, the back and walls must be made safe by scaling, bolting or by other means of support as required;
2. The face must be properly washed with water;
3. The driller must thoroughly examine the face for misfires, cutoff holes, and remnants of blast holes (bootlegs). All remnants of blasted holes must be washed out and marked with paint;
4. Any hole, regardless of length is to be treated as a misfire where:
    a. The hole cannot be inspected;
    b. The toe of the hole is not visible; and
    c. Any explosive products and components remain in the hole.
5. Lifters must be dug out, washed, marked and flagged with lifter tubes;
6. All faces in abandoned headings must be examined, washed, marked up, dated and signed;
7. Drilling must not be carried out within 160mm (6 in.) of a hole that has been previously blasted, or an intact portion of a blasted hole;
8. Drilling must not be done within one meter (3.3 ft.) of any hole containing explosives;
9. Ensure that the face is marked up according to engineering or geology standards. This includes line, grade, drill pattern, and any special geological mark-ups;
10. Check scale the face as required as drilling proceeds;
11. The new cut must be rotated a minimum of one foot from the old cut;
12. Drilling and loading shall not be carried out concurrently on the same face; and
13. All holes must be collared and drilled wet.

5.7.2 BLASTING PROCEDURES

5.7.2.1 Loading and Blasting:

1. The face and work area must be checked for hazards before loading begins. Drilling operations may have loosened rock at the face. Check scale for loose before beginning loading operations. This is to ensure workers and explosives are not struck with loose rock. Falling rock could injure worker, cut nonel shock tube causing misfire, or in rare instances trigger a premature detonation wile loading;
2. All equipment not required for loading is to be removed from the working area;
3. "LOADED FACE" and "DO NOT ENTER" sign shall be hung across access to the drift before any holes are loaded;
4. Blow all holes clean to remove water and rock fragments, standing clear of the holes when blowing out to ensure nobody is hit by rock fragments;
5. Bring only the required number of detonators to the face;
6. Leave proper amount of collar in each hole;

7. Follow mine specifications for loading perimeter holes for perimeter control;
8. Stick powder shall be loaded into blast holes using a loading stick of non-sparking material;
9. When ground conditions allow for the use of ANFO, the following steps are to be carried out:
    - Inspect ANFO loader to ensure that no rock fragments are blocking the ejector;
    - Connect air hoses to the loader and use whip checks on connections. Before turning on the air make sure all valves are in the off position;
    - Only properly maintained anti-static hose shall be used when loading pneumatically;
    - The loader must be grounded to remove static electricity; and
    - Ensure you have control of the loading hose at all times.
10. When priming explosives, only a non-sparking tool can be used to punch a hole in a cartridge, such as a powder punch;
11. Load holes from the top of the face and work your way down;
12. All holes must be loaded before hooking up the nonels to the B-Line;
13. Connect the nonels to the B-Line;
14. The blaster in charge will string the electric cap to the lead wire after running out the shunted lead wire from the central blast line or blasting box to the face. Test the lead wire to ensure there is no voltage in the line;
15. Attach the electric detonator to the detonating cord. Lastly, tie in the lead wire to the blast line or blasting box and check the continuity of the circuit;
16. Place proper signage at entrance to drift, warning personnel of a loaded round;
17. When loading is complete, return all unused explosives to the proper storage magazines and make required entries into the log books; and
18. Remove material from the area prior to blasting. Ensure equipment is parked in an area where it will not be damaged by fly-rock or the concussion of the blast.

5.7.3 BLASTING MATERIALS

5.7.3.1 Emulsion

Emulsion explosives will be considered for wet conditions. Emulsion is water resistant and can be blended with ANFO for a product that is better suited to variable weather conditions. Emulsion will be provided from off-site and received as cartridges in 25 kg boxes.

5.7.3.2 Nitroglycerine Dynamite

Dynamite combines nitroglycerin with adsorbents and stabilizers, rendering it safe to use but retaining the powerful explosive properties of nitroglycerin. Pre-packaged explosives will be kept on-site for selective blast requirements, such as fragmenting boulders and removal of high spots on the ramp floor.

5.7.3.3 Non-Electric Detonators

Detonators will be non-electric and tied in with detonator cord. In this case, the initiation system is composed of a series of shock tubes connected to detonation devices. The shock tubes transmit shock waves to the non-electric detonators to initiate the blast. Non-electric Detonating System include the surface delay and in-hole detonator assembly, trunk-line assembly and lead-in line. The use of non-electric systems eliminates the danger of premature detonation owing to radio frequency energy or stray static electricity (e.g., wind, low humidity, plastic liners are sources of static electricity). Such systems can be used under all weather conditions, provide accurate surface and in-hole timing, and can be used in conjunction with lead-in line shock tube and detonating cord.

5.7.3.4 Electric Detonators

If warranted electronic detonation may also be considered to increase the accuracy of firing times and programmable detonation, if desired. The precision timing provided by electronic detonators may allow for a more uniform muck pile when conducting controlled pit blasting in different rock units. A more uniform muck pile will reduce processing costs and losses associated with the presence of oversized material and fines.

5.7.3.5 Detonator Cord

Detonating cord is a thin, flexible plastic tube filled with penta erythritol tetranitrate (PETN). Detonating cord may be used by the mine development contractor as a high speed fuse capable of detonating multiple charges almost simultaneously. This may be used to initiate pre-splitting blasts or for detonating large boulders simultaneously with the blast.

5.7.4 EXPLOSIVES QUANTITIES

The anticipated explosives required depends on how many and which mines Alexco has in operation. Estimated consumption for stick product is 1.15 kg/t (2.3 lb/ton). Actual explosive quantities will vary depending on breakage effectiveness, rock type, rock hardness, explosives cost versus crushing costs, and overall refinements to mining operations.

5.7.5 EXPLOSIVES STORAGE AND HANDLING

Explosives will be used for development and production in all mines across the Keno Hill District. Explosives and accessories will be delivered to site by truck. Explosives are trucked to the site and stored in approved magazines. The explosive magazines are located away from any other infrastructure meeting the distance requirements under the Explosives Act and Regulations and the Quantity Distance Principles – User's Manual from the Explosives Regulatory Division (*https://www.nrcan.gc.ca/explosives/resources/standards/9963*).

Four licensed magazines, constructed according to regulations, are located on site (i.e., two for explosives storage and two for detonator storage). The layout of the magazines follows explosive regulations and as such the following infrastructure are provided:

- Pre-constructed detonator magazine for detonators and shock tubes; and
- Powder magazine for boosters and cartridges.

Additional powder and detonator magazine will be constructed underground during pre-production using existing remucks off the main decline. The explosive magazines are barricaded with rock berms and constructed 80 m apart allowing for 20,000 kg of storage in total in accordance with distance requirements for explosives storage facilities as specified in the Federal Explosives Act and Regulations and the Quantity Distance Principles – User's Manual from the Explosives Regulatory Division *(https:// www.nrcan.gc.ca/ explosives/resources/standards/9963).*

All onsite handling, including operation of explosives magazine, will be completed by the mine development contractor. This qualified person will be required to use equipment designed for the handling and transport of such materials. And safe handling practices will apply to the handling and transport of explosives waste to the disposal site.

5.7.6 AUTHORIZED ACCESS

A Key Control Plan will be developed by the designated mine development contractor based on the federal requirement (*https://www.nrcan.gc.ca/explosives/resources/guidelines/13961).* In order to minimize any unauthorized access to the explosive magazines and storage areas, the plan will describe how security will be maintained and how access to explosives and raw material will be controlled. It will include the following:

- Every key to the magazine will be numbered;
- A person may only have possession of a key to the magazine if they are named in the plan;
- The number of people named in the will must not exceed the number necessary for the operation of the magazine;
- The lock on the magazine must be of a type for which keys can be obtained only from the lock's manufacturer or a certified locksmith designated by the manufacturer;
- If a key is lost or stolen, the lock must be immediately replaced; and
- Each key must be kept in a locked and secure location when it is not in the possession of a person named in the plan.

Access to the magazine and explosives storage areas will be restricted and only authorized personnel will be permitted to enter these areas. A register for the list of authorized personnel will be developed, and a daily sign-in/out log for persons entering the magazine will be maintained. The following are the type of personnel who will be permitted to enter the magazine and explosive storage areas:

- Appointed blasters;
- Mine development contractor employees (i.e., personnel required for explosive delivery and personnel involved in site maintenance);
- Blasting assistants;
- Security guards (external area only, no magazine access);
- Mine Manager;
- Mine Superintendent(s);
- Mines or explosives inspectors; and
- RCMP.

5.8 MUCKING

Mucking will be accomplished using 3.5 yd^3 and 2 yd^3 Load Haul Dump Loaders (LHD) to meet production targets. One 3.5 yd^3 LHD will be dedicated to waste production from the ramp and level accesses. Waste rock will be hauled from the face to a remuck bay (constructed every 150 m along the primary ramp) and then reloaded into 15 tonne trucks at the remuck bay. Depending on the backfill cycle, waste rock may be directly hauled from the development face to a backfilled stope. A 2 yd^3 LHD will be used for mineralized ore mucking from the face to an ore remuck located at the intersection of each level access.

5.9 BACKFILLING

Backfill materials consisting of development waste rock (N-AML and P-AML) and dry filtered tailings will be placed into empty stopes by Load Haul Dump (LHD) or 15-tonne trucks. The mix of these materials was determined based on geotechnical requirements and characteristics of backfill materials available (Minefill, 2021). Backfill mix design will also aim to minimize the surface environmental impact while optimizing the most efficient and cost-effective back filling sequence.

Based on the planned stopes geometry, the required strength for each stope, with a factor of safety of 1.3, was determined. Material samples sourced from Flame & Moth and the mill were sent to an independent laboratory to determine the optimal mix design to achieve the required strength. Refer to the Laboratory Test Report (Minefill, 2021) for detailed required strengths, mix design for each type of placement, and curing times.

Cemented backfill with the same cement by weight will be used in longhole and cut and fill stopes, except for the first lift of cut and fill stopes, where cement contentment will be higher. The cement, rock and water will be mixed by LHD bucket in a small sump-like cut out near the empty stope. Cement will be transported underground in bulk bags.

For cut and fill stopes, the backfill will be pushed up tight to the back using an LHD equipped with a rammer jammer. For long hole stopes, the backfill procedures vary depending on stoping methods. For conventional downhole stopes, the backfill will be placed by dumping rockfill or cemented rockfill from the top access using LHDs or underground trucks. For pillar recovery (up hole stopes), no backfill is necessary. They will be filled with P-AML and N-AML as needed for waste management.

Where sill pillars are required, a cemented fill will be used to provide a stable back to mine up to from beneath. Extraction of the vein from the final lift requires that the pillar is self-supporting and maintains integrity while the heading is active. The quality and the placement of the fill are both important factors in this application. These materials should be placed into headings as tight to the back as possible. An increased cement content will be required to provide the required strength of the pillar. In areas where additional caution is required during final lift extraction, the lift will be mined using up-holes and remote mucking.

Careful preparation of the excavation where cemented fill is to be placed will be required, including blasting beyond the vein contacts to provide a clean, rough surface for the fill to hang on. The floor should be cleaned prior to placement to prevent material falling from the back following mining. An appropriate lead time should be provided to allow set-up and cure for the cemented fill. Standard quality control procedures (e.g., unconfined compressive strength and slump tests) should be completed during batching and following placement of cemented tailing fill materials.

Quality assurance and quality control (QA/QC) procedures are in place to ensure backfill procedures are appropriate for short and long-term stability requirements.

5.10 MINE EQUIPMENT

All mobile equipment will be diesel-powered rubber-tired equipment owned by Alexco. Alexco currently owns or leases the majority of the underground equipment with the rest to be purchased over the mine life to supplement the fleet. Bellekeno, Bermingham, and Flame & Moth mine will all use similar size equipment while the Lucky Queen mine will use smaller scale units.

5.11 VENTILATION

A ventilation model was designed based on the mine plan for the Bermingham and Flame & Moth mines. Primary fans, auxiliary fans, and heating units will be reused from the Bellekeno mine as well as shared between other operations to reduce total capital expenditure.

5.11.1 MINE AIR HEATING

Alexco will need to heat the ambient air to +2°C in all three mines. Direct fire propane heaters have been purchased for Flame & Moth and Bermingham mines. Figure 5-13 displays the average temperature at the nearest weather station of Mayo, Yukon Territory, Canada.

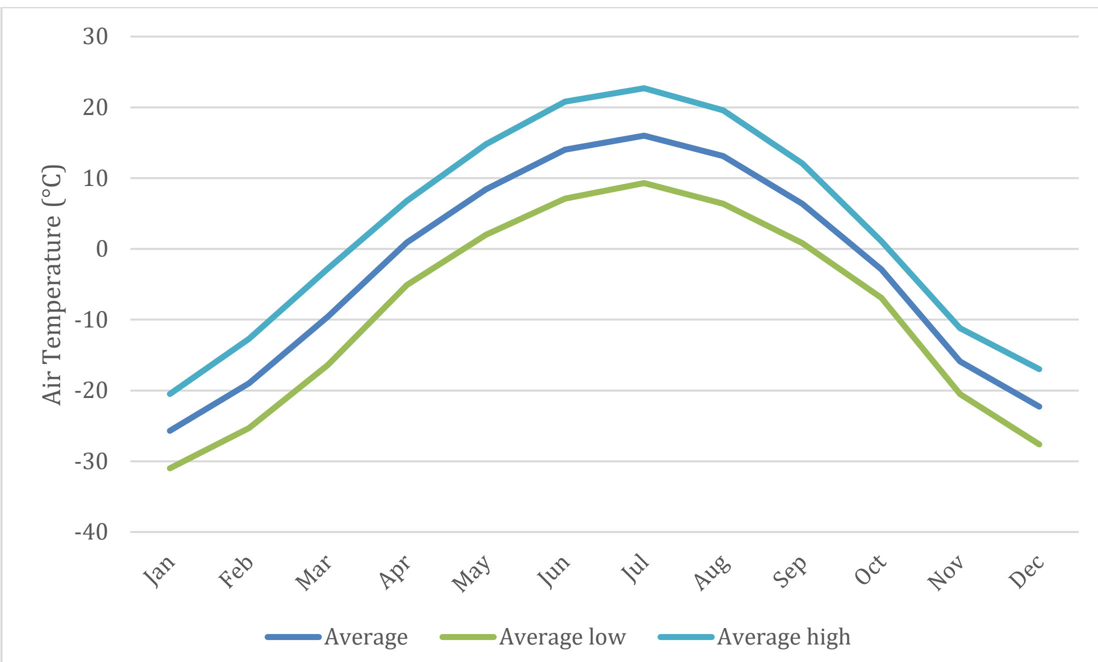


Figure 5-13: Mayo, Yukon Territory, Monthly Average Temperature

5.11.2 VENTILATION MODELLING

A ventilation model for each mine was developed based on the proposed designs for Bermingham and Flame & Moth. No model was created for the Bellekeno mine because the primary ventilation circuit is already in place and only auxiliary ventilation will be required. This was used to determine operability and estimate the required primary fan duties required at different stages of the mine life. Key modelling considerations include:

- Expected diesel equipment fleets for each mine are based on the mine plan;
- Utilization of availability for equipment is based on the mine plan;
- Associated friction factors and resistances are based on excavation methodology and accepted industry design values;
- A leakage allowance of 20% of total airflow demand;
- Early development work in both the main and ventilation declines will have fans located well outside each portal, a minimum of 50 m away, to limit the possibility of recirculation;
- Airways will maintain an air velocity of at least 0.5 m/s to remove contaminants and maintain an appropriate temperature in the mine; and
- Working areas will be limited to an air velocity of 4 m/s and travel ways to a maximum of 6 m/s to maintain a safe and healthy environment.

5.11.3 FLAME & MOTH MINE

The proposed layout for the Flame & Moth underground project will support a relatively simple positive pressure ventilation circuit. A primary fan located underground will force the heated air down the intake raises to each respective ventilation drives that intercept the intake raise on the working levels. Each production level will be ventilated using an auxiliary fan that draws fresh air from the intake raise and directs the air through ducting to the working area before it returns to the main ramp to exhaust out the portal. Each of these ventilation drive connections must be regulated to ensure only the desired volume is permitted to flow.

One primary surface intake raise is planned to support the Lightning and Christal zones and the Christal Zone will utilize a dedicated ventilation drive to connect to the fresh air system. Figure 5-14 shows the basic primary airflow through the fully developed mine.

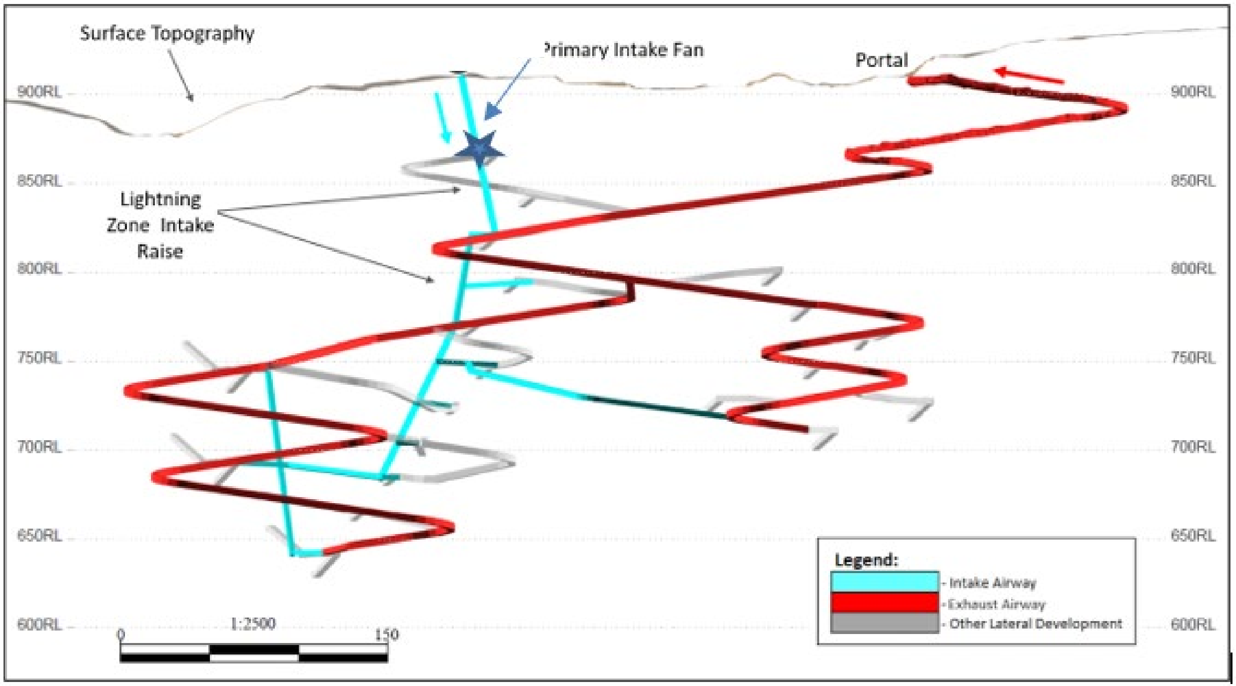


Figure 5-14: Flame & Moth Primary Airflow Schematic

5.11.4 BERMINGHAM MINE

The proposed layout for the Bermingham underground project will support a relatively simple positive pressure ventilation circuit. A primary fan located on the surface will force the heated air down the intake raises to each respective ventilation drives that intercept the intake raise on the working levels. Each production level will be ventilated using an auxiliary fan that draws fresh air from the intake raise and directs the air through ducting to the working area before if returns to the main ramp to exhaust out the portal. Each of these ventilation drive connections must be regulated to ensure only the desired volume is permitted to flow. Figure 5-15 shows the basic primary airflow through the Bermingham mine.

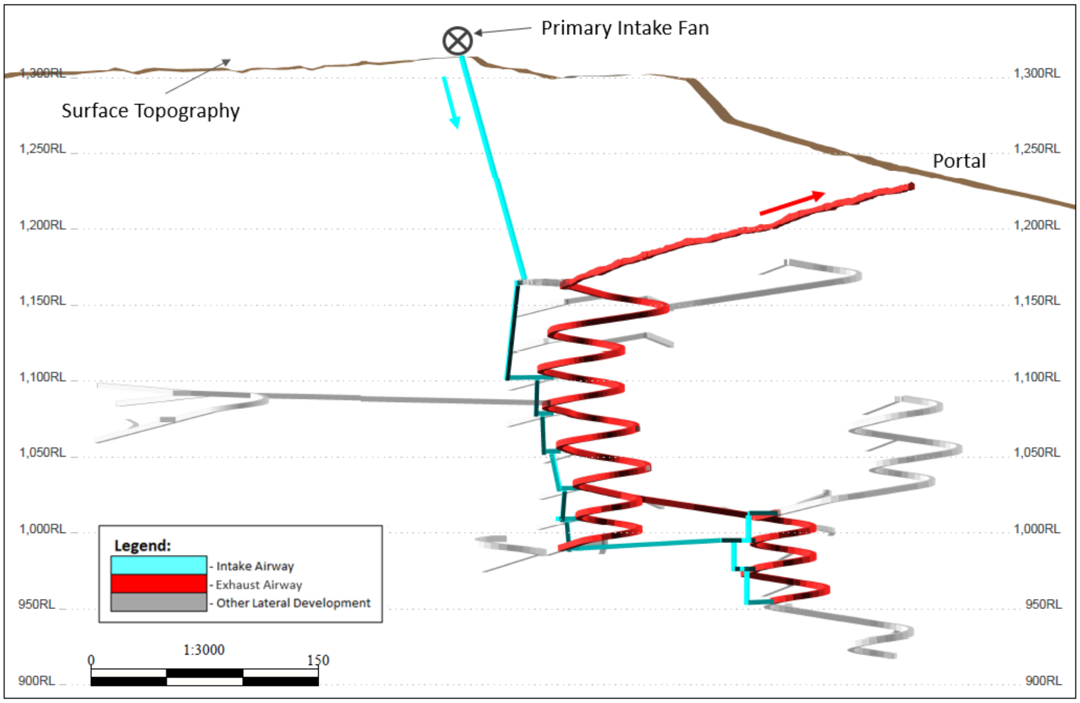


Figure 5-15: Bermingham Mine Primary Airflow Schematic

5.11.5 ESCAPEWAYS

The mine escapeways for both Bermingham and Flame & Moth are constructed within the respective ventilation raises. The escapeways at the Bermingham and Flame & Moth mines will be installed in a manner that is compliant with Regulation 15.49 as outlined in the *Yukon Occupational Health and Safety Act*. The detailed engineering design showing the ladder and landing configuration are provided in Appendix A of this document. These pages are extracted from the comprehensive construction drawing package retained on site by the Mine Technical Services.

5.12 MINE SERVICES

5.12.1 COMPRESSED AIR

Compressed air will be supplied underground through 2" diameter HDPE pipes. In the case of the Flame & Moth mine, the compressed air will be supplied by a fixed air compressor associated with the mill. In the case of the other three mines, compressed air will be supplied by an air compressor located at the portal.

5.12.2 DEWATERING

Bermingham and Flame & Moth mines are both expected to have higher groundwater flows.

For all the mines, the dewatering strategy is to use electric submersible pumps to collect water from sumps near the active mining areas and pump it in stages to the dirty water sump located on the ramp. The dirty water will decant to a clean water sump where a clean water pump will pump the water to the surface for recycling or treatment and eventual discharge.

5.12.3 ELECTRICAL

All the mines will be connected to the site and Yukon Energy hydro power grid. Primary power transmission underground will be 4,160 V to mobile power centres located in strategic locations underground where voltages will be stepped down to 600V for final distribution.

Single line electrical diagrams for Flame & Moth and Bermingham are shown in Figure 5-16 and 5-17.

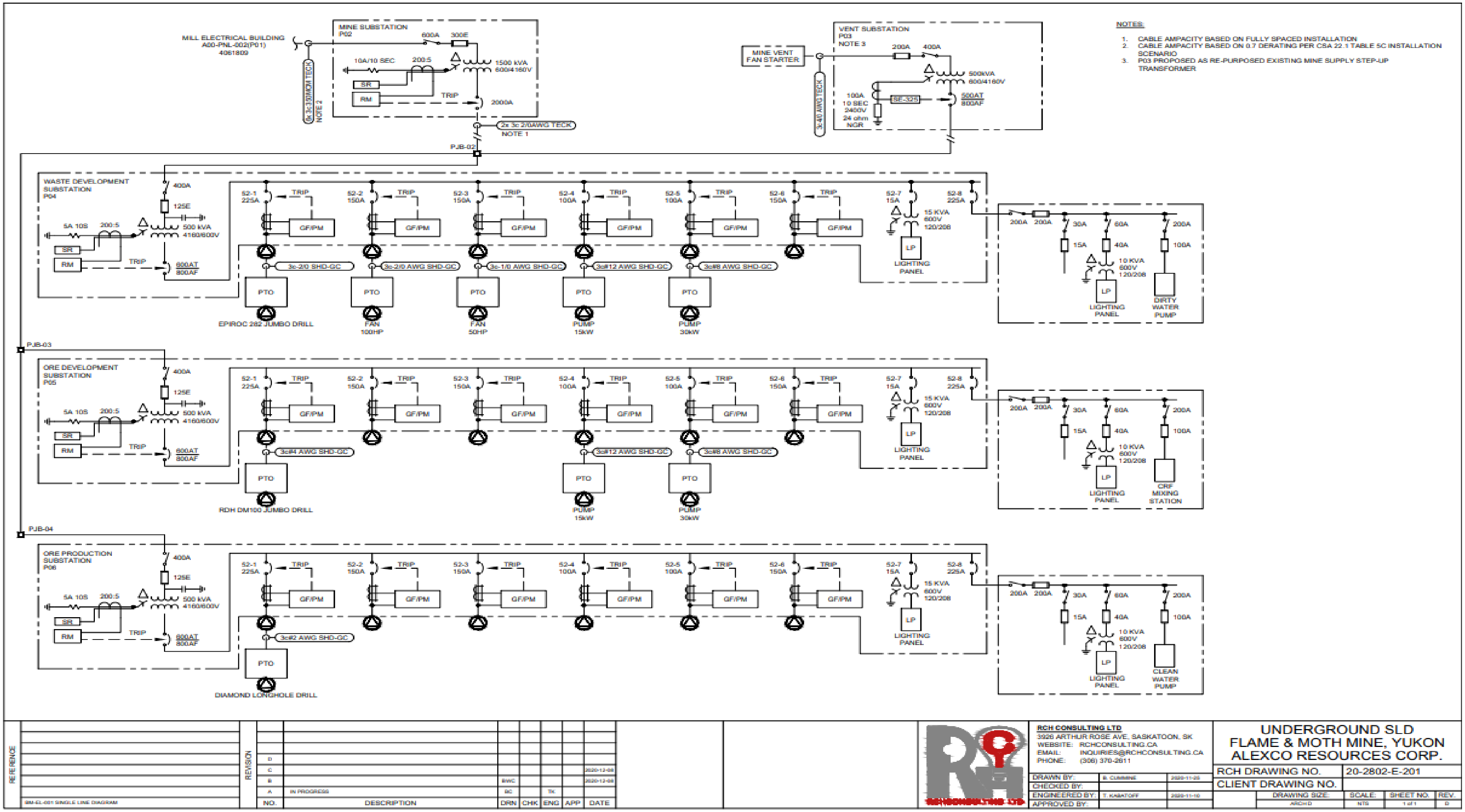


Figure 5-16: Flame & Moth Single Line Electrical Diagram

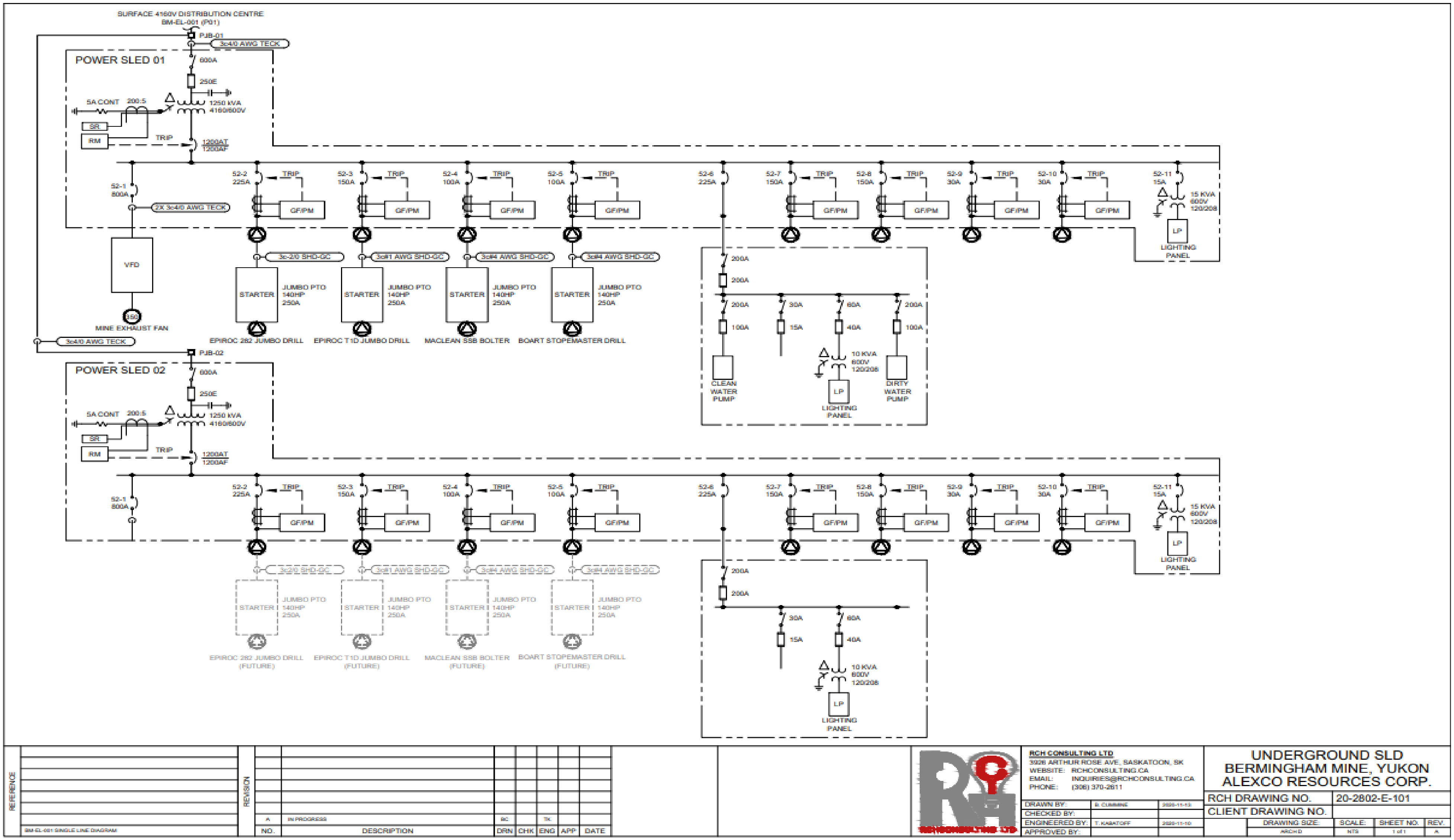


Figure 5-17: Bermingham Mine Single Line Electrical Diagram

5.12.4 FUEL

Fuel used on site includes diesel, gasoline, propane, and heating oil. Fuel and petroleum products will be delivered to site by truck. They will be held in Envirotanks to supply fuel for site services, personnel transportation, mine development, and production operations.

5.12.5 MAINTENANCE FACILITIES

For both Flame & Moth and Bermingham mines, the maintenance department will have a fuel/lube truck, a mechanic's service truck, a tractor, and access to a scissor lift and a boom truck.

In addition to the mobile equipment, the mine maintenance department will be responsible for the stationary equipment consisting of air compressors, main ventilation fans, propane air heaters, underground electrical distribution system, and main dewatering pumps.

Most of the mobile equipment maintenance will be performed in a surface shop, which will be constructed near each of the mine portals.

6. SITE INFRASTRUCTURE AND SERVICES

6.1 FLAME & MOTH MINE SURFACE INFRASTRUCTURE

The Flame & Moth mine uses existing infrastructure at the District mill site. Existing infrastructure consists of geology and engineering office trailers, maintenance shop and fuel storage facility as can be seen on Figure 6-1.

Additional infrastructure in the area of the Flame & Moth mine portal, northeast of the mill building, consists of:

1. a shop, a miners' office trailer and miners' dry facility;
2. cold storage structure;
3. ore and waste handling/storing facility;
4. electrical power distribution;
5. portal ventilation fan and heater;
6. air compressors; and
7. settling pond for mine water discharge, with clarified water supplying the underground mine.

An aerial photo with Flame and Moth infrastructure and portal location is provided in Figure 6-2.

The fresh air raise collar location is planned south of the crusher and coarse vein material stockpile. The main ventilation fans and mine air heater will be located at the raise collar.

N-AML waste rock generated from development and mining at Flame & Moth will be deposited around the mill area to create extensions to laydown and storage areas. P-AML waste rock will be deposited in a temporary P-AML waste rock storage facility constructed nearby before being backfilled underground.

The site receives explosives deliveries by truck and the explosives will initially be stored in licensed and permitted surface magazines. A second powder and detonator magazine will be constructed underground during pre-production using existing remucks off the main decline.

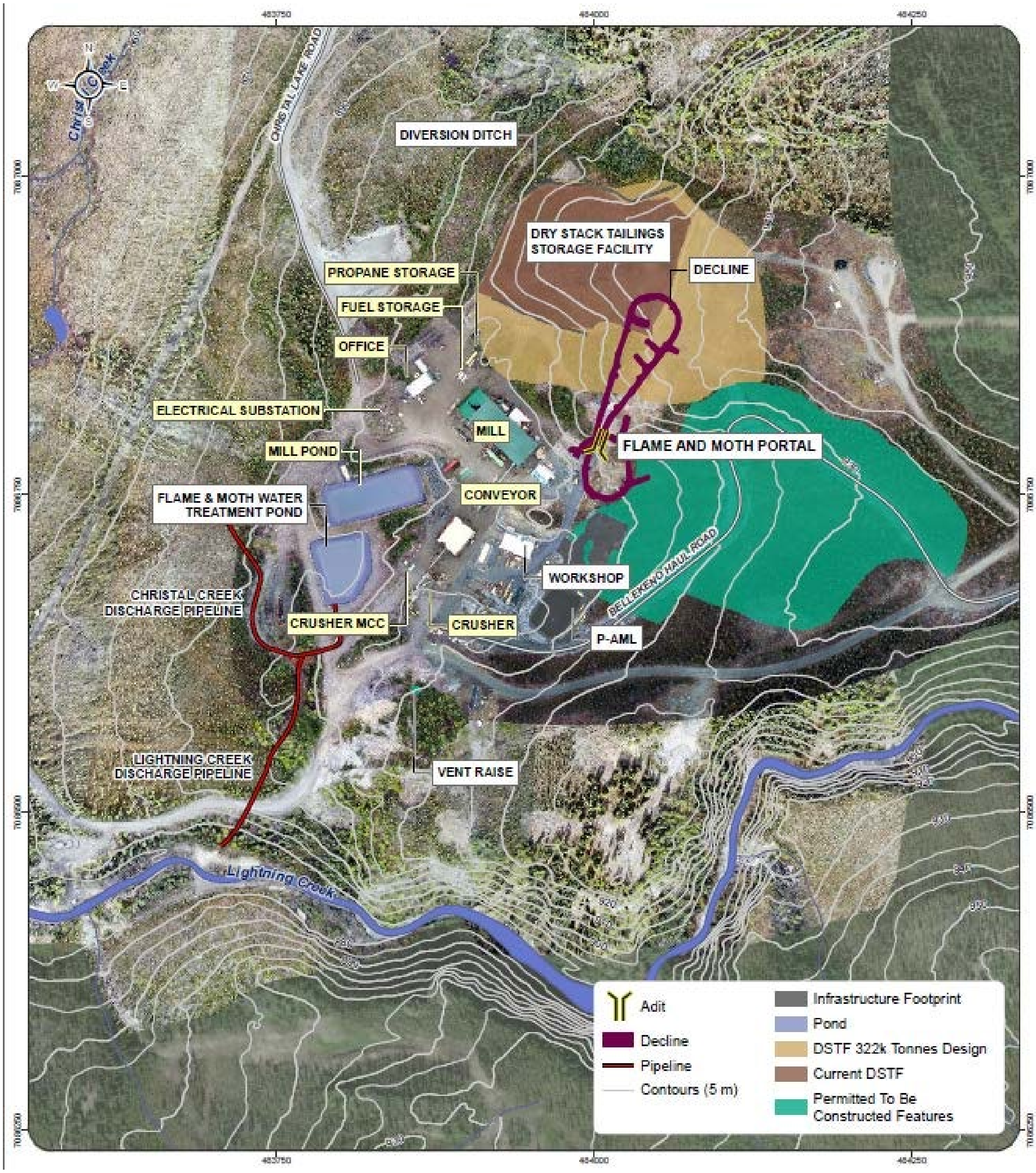


Figure 6-1: Site Layout at the Flame & Moth Portal

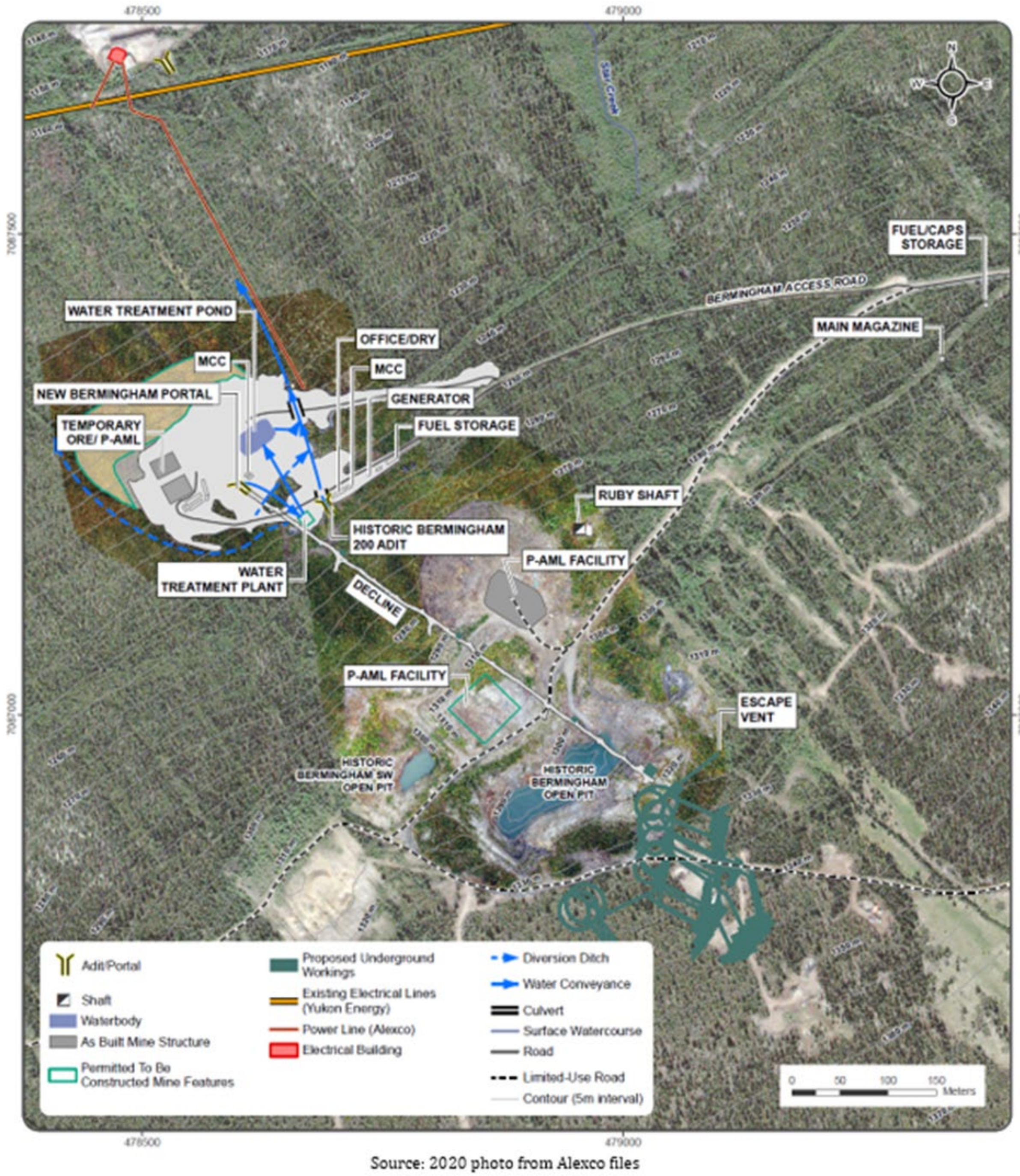
6.2 BERMINGHAM MINE SURFACE INFRASTRUCTURE

Various surface facilities and infrastructure were constructed in 2017 and 2020 to support the new Bermingham underground exploration decline including include a dedicated mine office, dry/lunch room, maintenance shop, water treatment plant, diesel power generation, fuel storage tanks and laydown yard. The surface infrastructure that was constructed for the advanced exploration decline will remain in place and be expanded to facilitate development and production at the new Bermingham mine. A lined water management pond already constructed will be used as part of the water treatment system required during active mine operations.

An aerial photo with Bermingham infrastructure and portal location is provided in Figure 6-2.

Non-acid metal leaching (N-AML) waste rock generated from development and mining at Bermingham will be deposited in a new N-AML waste rock disposal area, which will be built as an extension to the current waste rock disposal area at Bermingham. P-AML waste rock is expected to be deposited in a P-AML waste rock storage facility constructed nearby.

The site receives explosives deliveries by truck and the explosives will initially be stored in licensed and permitted surface magazines. A second powder and detonator magazine will be constructed underground during pre-production using existing remucks off the main decline.



Source: 2020 photo from Alexco files

Figure 6-2: Site Layout at the Bermingham Portal

6.3 ELECTRICAL POWER

The Project is supplied with electrical power from a hydroelectric plant near Mayo and connection to the Yukon wide electrical grid.

The Mayo hydro facility was expanded in 2011 which increased generation capacity from 5 megawatts to 15 megawatts. The power distribution grid was also upgraded from Pelly Crossing to Stewart Crossing during the same time. Recently the power distribution line from Mayo to McQuesten was completed in early 2021 to replace the 65 year old transmission line and to add system protection equipment.

A new 69 kV/4.16 kV 3 MVA substation was installed to deliver power to the mill facility, Flame & Moth, and associated infrastructure.

Alexco owns several substations in the area, including the Elsa substation, the Onek substation, and the Bellekeno 625 portal substation. Alexco also owns the transmission line connecting the latter two. Power for the Bellekeno and Flame & Moth mines is now provided exclusively by the YEC electrical distribution system.

Power for the camp is supplied from the local grid that runs through Elsa to Keno City.

Electrical power for Bermingham was initially provided by diesel-powered generators and was transferred over to YEC grid power in Q2 2021. A transmission line (via surface teck cable) connects to the Yukon grid near the Calumet Road to the site.

6.4 AREA HAUL ROAD SYSTEM

Alexco has constructed a series of access and haul roads to route mine traffic around the Keno City community. All traffic between Elsa and the mill facility and/or the Bellekeno mine is routed along the Christal Lake Road, and subsequently the Bellekeno haul road. The Bermingham mine traffic will use the Bermingham access road, Calumet Road, and a short section of the Duncan Creek Road (~3 km) between the mill and the Bermingham mine.

Heavy truck traffic from Lucky Queen will be routed along the Keno City bypass road to/from the Bellekeno haul road. The bypass road is approximately 2.1 km long and six to nine meters wide as per Yukon Workers' Compensation Health and Safety Board regulations and the identified haul road type.

6.5 MILL FACILITY

The current facilities at the District mill facility include mine and mill offices, male and female dry facilities, an assay lab, first aid facilities, and the mill, warehouse and DSTF complex. The mine geology and engineering office buildings from Bellekeno were moved to the mill area serve as a central administration office for all future mining operations.

A metallurgical and assay laboratory conducts all basic testwork to monitor and improve the process flowsheet metallurgy and efficiency, and to support environmental monitoring. The assay laboratory was constructed as a pre-packaged unit consisting of two retrofitted 40 ft shipping containers converted into laboratory modules, which are located adjacent to the mill building. The laboratory is equipped with the necessary analytical instruments to provide all routine assays for the mine, plant, and environmental quality control monitoring.

The equipment included allows for the preparation and analysis of approximately 80 samples per 12-hour shift. Standard analysis includes acid digestion of samples followed by analysis on an atomic absorption spectrometer.

6.6 FLAT CREEK CAMP FACILITIES

The currently licensed Flat Creek camp facilities include a trailer camp, kitchen facility, site sign-in/reception, and a dry. The Flat Creek camp has a total capacity of 123 permanent beds. There are four refurbished houses located nearby the townsite of Elsa with a total of 28 rooms, and an additional 20 rooms available in a bunkhouse next to the houses, which is used primarily for seasonal surface exploration programs. Two of the three bunkhouse complexes at Flat Creek camp were replaced with new units in Q4 2020. The entire capacity of the camp facilities is 171 rooms.

Alexco is licensed to withdraw water from Flat Creek and an existing groundwater well for domestic use. A water treatment facility located within the Flat Creek camp consists of 5,000 L of storage, a water softener, UV treatment, and chlorination. Alexco has two sewage disposal permits at Elsa; one for the Flat Creek camp and one for the houses. Waste water is treated in septic tanks and released via drain fields.

Commercial Dump Permits #81-012 and #81-067 are currently held from YG Environment in accordance with the *Environment Act*, Solid Waste Regulations, as well as the *Public Health and Safety Act*. The permits were renewed effective January 1, 2017 and will continue to be used in support of mine operations. In compliance with this permit, upgrades to the location of solid waste disposal included upgrades to the electric bear fence and the addition of a cattle guard to prevent animals from entering the facility. Kitchen refuse is incinerated in a diesel incinerator located in Elsa.

6.7 SUPPORT FACILITIES

The administrative offices and first aid facilities are currently based in a complex located adjacent to the District mill. A mobile equipment maintenance facility is located near the District mill and a new warehouse is located adjacent to the District mill complex.

6.8 OFF SITE FACILITIES

The Project is located in central Yukon, near the community of Keno City. Keno Hill is approximately 460 highway kilometers or approximately 5 hours by road north of Whitehorse via Yukon Highway 2 and Highway 11. Road access to the project is maintained year round by the Yukon Government, Department of Highways. The site operations are supported from an existing office and administrative building located in Whitehorse.

The nearest airport with scheduled service is the Mayo Airport, located approximately 55 kilometers south of the Project. Charter flights between Whitehorse and Mayo also utilize this airport.

7. SITE WIDE PROCEDURES

7.1 GROUND CONTROL

The procedures for ground control management are documented in the AKHM Ground Control Management Plans for both Flame & Moth and Bermingham and are included here as Appendix B and Appendix C. The following summarizes the key elements of that plan based on the design description and geotechnical characterization discussed in Section 5.2 and ground control design discussed in Section 5.5.

The key aspects of the ground control management plan are:

- Mining practices in and around the deposit will need to be cautious and excavation size and overbreak limited as much as possible to maintain the stability of the excavations;
- Ground control components must be installed according to best practice, manufacturer's standards, and GCMP standards and the management plans. Special attention must be given for bit size and rockbolt diameter match, and the physical integrity of ground support components (corrosion, bolt length, screen gauge and shape, bit wear);
- Ongoing field observations of ground support installations, including at least a quarterly inspection of all ground support installed;
- Pull-test will be performed to determine anchor strengths of various ground support rockbolts. These tests will be performed on each type of rockbolts and in each type of ground condition, testing a total of 1% of bolts installed;
- Any suspected ground concerns will be monitored by the appropriate instrumentation including field observations, ground movement monitors (GMMs), extensometers and/or other methods deemed appropriate. A dedicated ground control log book will be located at each mine and ground control concerns tracked and documented on an on-going basis; and
- Ground Control Management Plan will be reviewed yearly.

7.2 BACKFILLING

The procedures for backfilling and backfill quality control/quality assurance are documented in site specific standard operating procedures. The following summarizes the key elements of these procedures based on the design description and geotechnical characterization discussed in Section 5.9.

The key aspects of the backfilling plan are:

- All stopes requiring backfill will be filled as soon as practicable after the extraction of ore is complete. A backfill log is to be maintained to track fill schedules and determine schedule bottlenecks;
- An engineering design by a competent P. Eng. will be maintained to ensure that the backfill properties are met;
- A monthly and cumulative fill placement record will be maintained to track the filled and void space throughout the mine. This will provide the mine planning engineers the required information to ensure that the rate of backfilling is consistent with overall targets and ore production requirements;

- Specifications for fill material size, % solids, % cement, etc. are to be tracked and regularly reviewed to ensure consistency with the fill design;
- Strength of the cemented rock fill will be constantly assessed. Cemented rock fill samples will be collected and sent to an independent laboratory for 7 day and 28 day UCS testing. Slump tests will be conducted on site. A set of samples will be collected once per lift of material placed;
- Tight filling will be used for minimizing stope failures and to prevent hangingwall and crown failures from propagating and to maximize the confinement of the fill in the stope; and
- Appropriate safety measures will be established and maintained through standard operating procedures and JHA's to ensure employee safety where waste is actively dumped into the edge of an open stope. Examples of these safety measures include site specific standard operating procedures, site evidence of engineered backstops, additional lighting, training, and employee competency assessment.

7.3 QUALITY MANAGEMENT REQUIREMENTS

The Quality Assurance (QA) and Quality Control (QC) procedures for ground control and backfilling are common across all site mining operations.

The design basis and technical specifications on which QC and QA are planned are documented in the site Ground Control Management Plan (see Appendix B and Appendix C).

7.4 WORKER HEALTH & SAFETY

Alexco has an established program for achieving and maintaining a healthy and safe workplace. Operations at the Keno Hill Silver District fall under the jurisdiction of the *Occupational Health & Safety Act* and Regulations administered by the Yukon Workers' Compensation Health & Safety Board. Alexco has an approved Health & Safety Policy and a detailed safety management plan and procedures. All documents govern the course of operations at all mining operations in the district. The safety record at the Keno Hill Silver District is exemplary with over 500,000 manhours worked without a lost time accident as of year end 2020.

All personnel and Contractors will meet the standards outlined in the Occupational Health and Safety Legislation, Mine Safety Rules, and Regulations of the Worker's Compensation Board. The site induction process governs the initial training on required safety procedures and protocols.

There is a fully qualified Mine Rescue Team on site. There are emergency first aid responders providing 24-hour service. There are multiple first aid rooms and an ambulance on the site as well. The Emergency Response Plan has been submitted to YG under separate cover.

Alexco policy requires that all employees (including contractors' employees) have pre-employment medical examinations including a drug and alcohol test. All employees will be fully equipped with the proper personal protective equipment standard for working underground, taking into consideration hazards caused by noise level, air born particulates, and confined work space.

All new employees will have a site wide and safety orientation and another orientation of the underground work site prior to commencing work. Regular safety meetings with supervisors, safety officer and employees are mandated. Any changes in procedures, equipment, or hazards require immediate notification to employees.

Underground contractors, Alexco personnel and others will have to comply with the Yukon Occupational Health & Safety (OH&S) regulations, in addition to Alexco and contractor's in-house standards.

A Safety Coordinator/Officer specific for the underground operation will ensure all workers are orientated to all aspects of the work site including hazard identification, protective equipment requirements and that medical and health requirements are followed according to legislation. That position is also charged with ensuring continued training and skill development for all personnel.

7.5 SITE ORGANIZATION CHART AND MINE OPERATIONS RESPONSIBILITIES

The mine department organization chart is shown in Figure 7-1. Daily operations are the responsibility of the Manager of Mining or qualified designate. The mine plan is the responsibility of the Chief Mine Engineer and Operations Manager. Any changes to the mine plan including ground control or backfilling must be approved in writing by the Operations Manager or designate. Mine safety is a line responsibility, supported by the AKHM Health and Safety Department, the Manager of which reports to the General Manager.

Alexco is using a combination of contractor for development and employees for production. Some specialty tasks such as diamond drilling and Alimak raising will be contracted out.

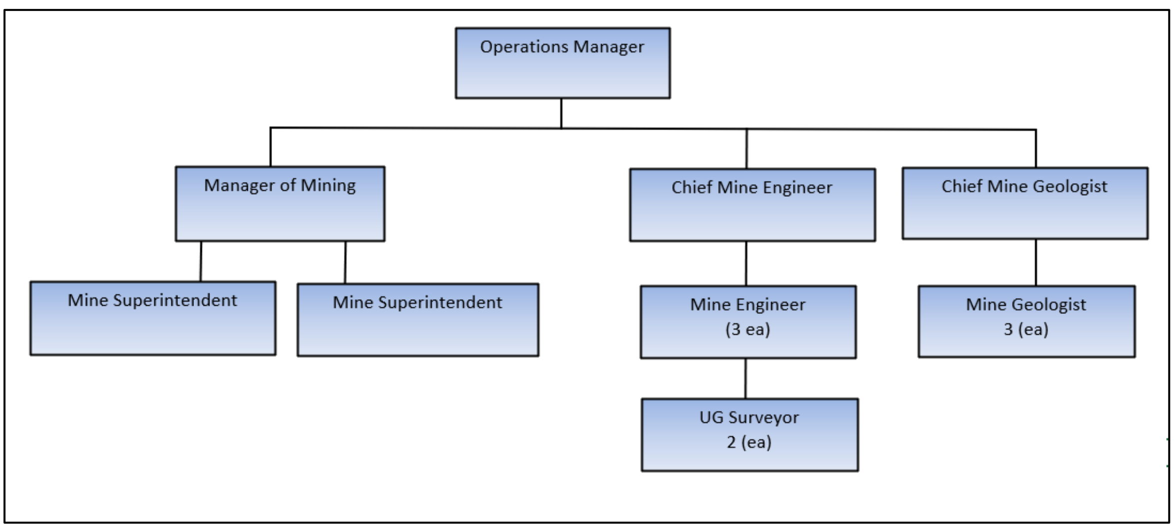


Figure 7-1: Mine Operations Management Organization Chart

7.6 REPORTING

There are both internal and external reporting requirements. Internal reporting requirements include but are not limited to:

- Mill and Mine Month End Report;
- Mine Mill Daily Report; and
- Weekly Mine Ventilation Report.

External reporting requirements include but are not limited to:

- Annual report under QML 0009;
- Monthly reporting under Water Licence QZ018-044; and
- Annual reporting as required by licence

8. REFERENCES

Alexco Resource Corp. (Alexco). 2021. NI 43-101 *Technical Report on Updated Mineral Resource and Reserve Estimate of the Keno Hill Silver District.* File: MP-8959 Alexco Technical Report Apr 2021. Prepared by Mining Plus Canada.

Boyle, R.W. 1965. *Geology, Geochemistry, and Origin of the Lead-zinc-silver Deposits of the Keno Hill-Galena Hill Area, Yukon Territory*. Geol. Surv. Canada, Bull. 111, p. 302.

Cathro, R. J. 2006. *The History and Geology of the Keno Hill Silver Camp, Yukon Territory*. Geoscience Canada, v 33, No. 3, P. 103-134 (2006)

Jacobs Engineering (Jacobs). 2019. *Pre-feasibility Study Geotechnical Engineering*. Report prepared Alexco Resource Corp. March 21, 2019.

McOnie, A. and Read, P.B. 2009. *Stratigraphy, Structure and Exploration Opportunities Sourdough, Galena and part of Keno Hills, Keno Hill Mining Camp, Central Yukon*. Internal Report Alexco Resource Corp.

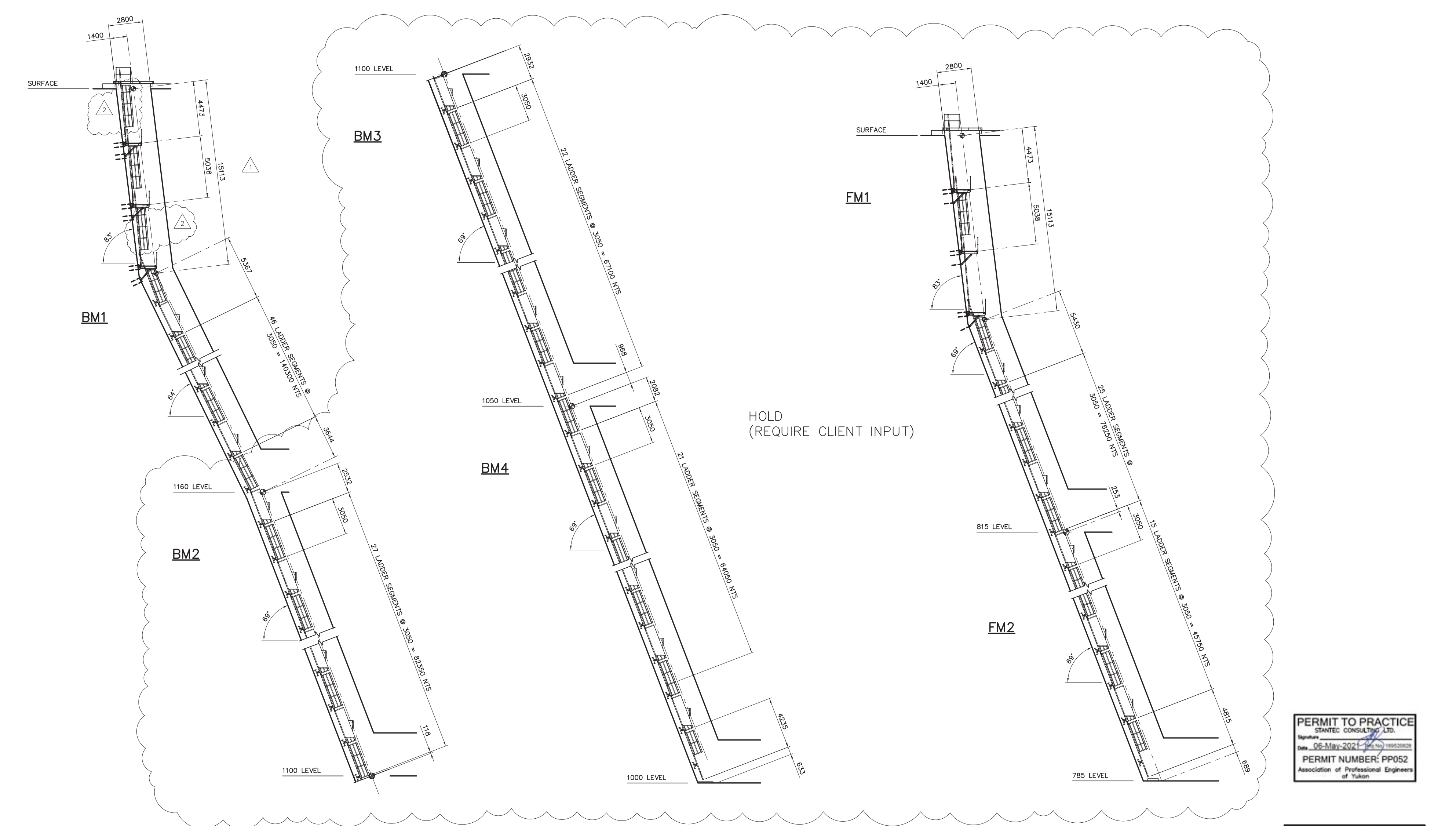
Minefill Services (Minefill). 2021. Lab Test Report Number 20004-TWR-8002 Issued January 27, 2021, internal document prepared for AKHM, prepared by Minefill Services, Dr. D. Stone.

United Keno Hill Mining Ltd. (UKHM). 1997. Unpublished report on "Mineral Resources and Mineable Ore Reserves" dated March 9, 1997.

APPENDIX A

MINE ESCAPEWAY DETAILED DESIGN DRAWINGS

(EXTRACTS FROM DETAILED ENGINEERING AND CONSTRUCTION PACKAGES)



ELEVATION – BERMINGHAM RAISE (BM)

ELEVATION – FLAME AND MOTH RAISE (FM)

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FOR CONSTRUCTION

Rev.	Description	Date	By	Chk	Eng.	Appd.
2	ADDED HOOPED LADDERS	2021-05-07	GC	DJG	RJV	MD
1	REVISED BM1 & PUT HOLD ON RAISE	2021-04-23	GC	DJG	RJV	MD
0	FOR CONSTRUCTION	2021-01-07	GC	DJG	RV	MD

Ref. Dwg. No. Ref. Dwg. Title

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YUKON TERRITORY

File Name: 551-S001-2.dwg GC Dwn. Chkd. RV Dsgn. 2020.10.26 YY.MM.DD

Title
LADDERWAY - 2.8x2.8 RAISE
GENERAL ARRANGEMENT
LONG ITUDINAL ELEVATIONS

Project No. 169520628 Scale 1:150

Drawing No. 551-S001 Revision 2

Dwg Updated: 5/6/2021 3:22:22 PM By: gcrocker

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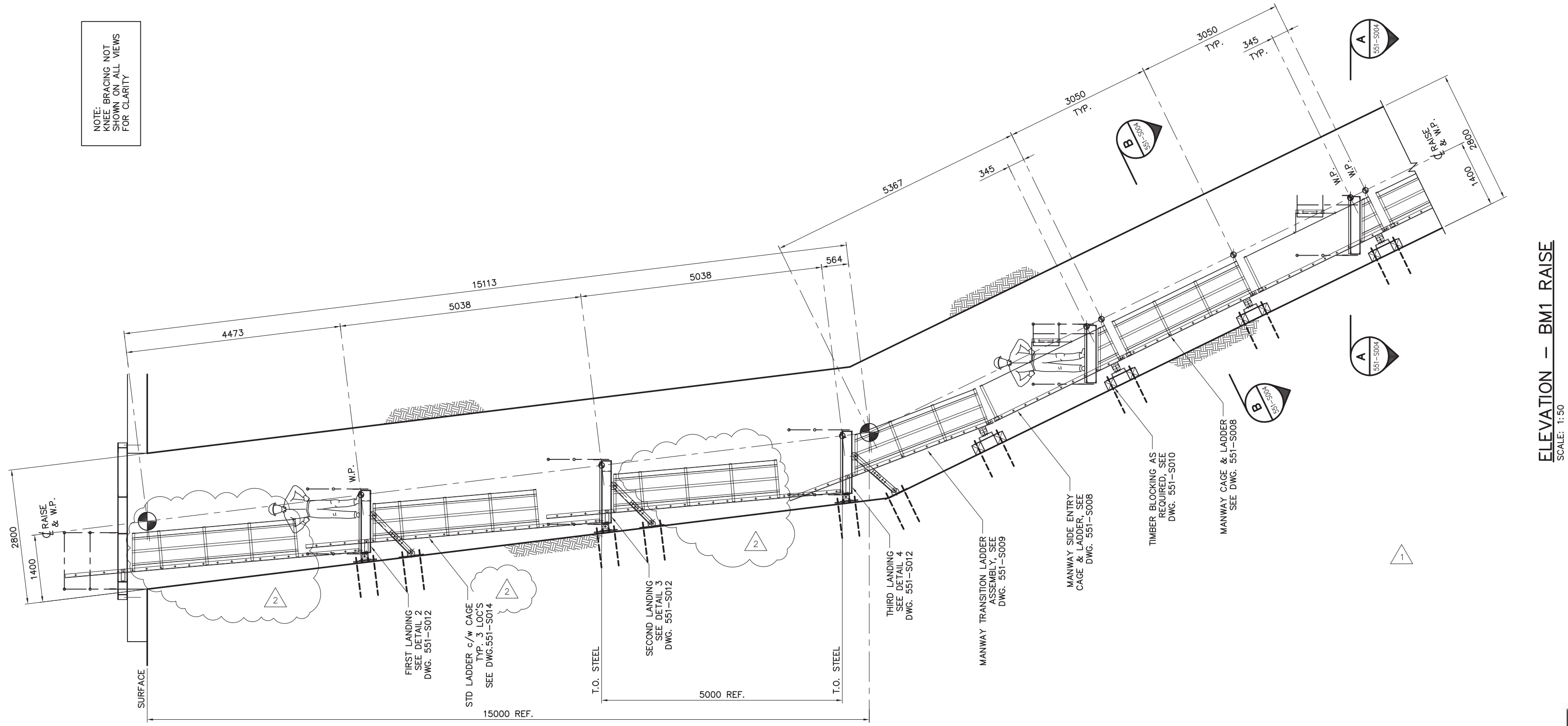

ELEVATION – BM1 RAISE
SCALE: 1:50

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Ref. Dwg. No.	Ref. Dwg. Title
551-S002A	G.A. - BERMINGHAM; ELEVATIONS

Rev.	Description	Date	By	Chk	Eng.	Appd.
2	ADDED HOOPED LADDER	2021-05-07	GC	DJG	RJV	MD
1	MOVED BM2, 3 & 4 TO 551-S002A	2021-04-23	GC	DJG	RJV	MD
0	FOR CONSTRUCTION	2021-01-07	GC	DJG	RV	MD

ORIGINAL SHEET - A1

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File Name: 551-S002-2.dwg

Dwn.	Chkd.	Dsgn.	YY.MM.DD
GC		RV	2020.10.26

Title
LADDERWAY - 2.8x2.8 RAISE
G.A. - BERMINGHAM BM1 RAISE
ELEVATION

Project No. 169520628
Scale AS NOTED

Drawing No. 551-S002
Revision 2

Dwg Updated: 5/6/2021 3:10:38 PM By: gcrocker

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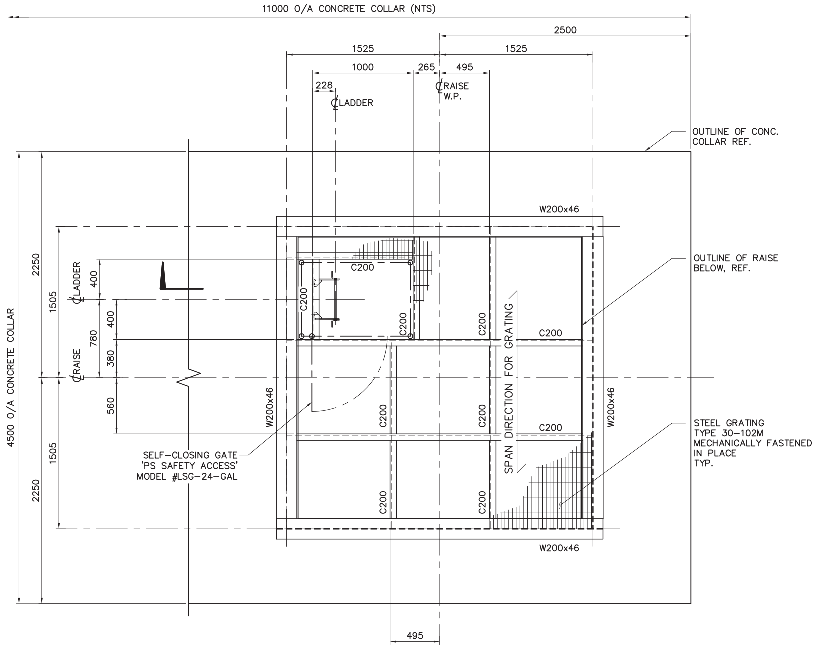

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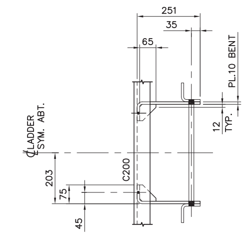
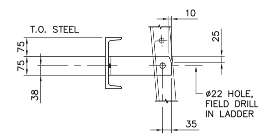
RAISE COLLAR PLATFORM
SCALE 1:25
MAT'L: C200 – C200x21 UNO

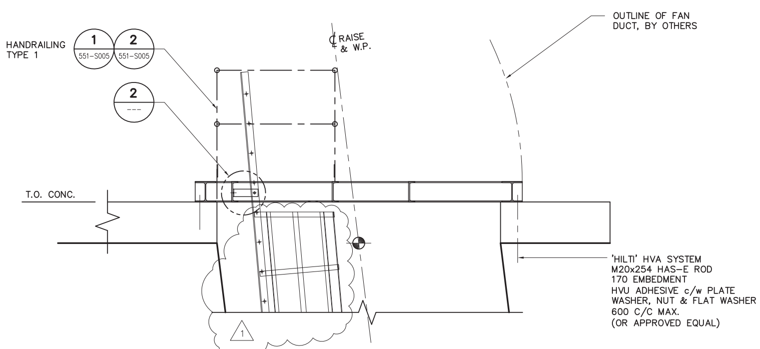


PLAN



ELEVATION

2 DETAIL
SCALE 1:10
551-S012



A SECTION
SCALE 1:25

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2. FOR GENERAL FABRICATION NOTES SEE DWG 551-S005
3. FOR HANDRAIL NOTES & DETAILS SEE DWG 551-S005

Rev.	Description	Date	By	Chk	Eng.	Appd.
1	SAFETY CAGE ADDED	2021-05-07	GC	DJG	RJV	MD
0	FOR CONSTRUCTION	2021-04-23	GC	DJG	RJV	MD

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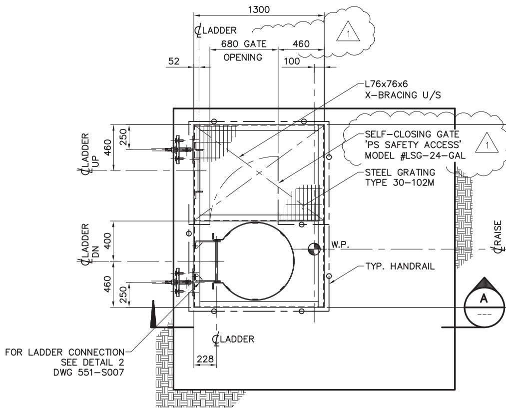
File Name: 551-S007-1.dwg GC Dwn. Chkd. RV Dsgn. 2020.10.26 YY.MM.DD

Title
LADDERWAY - 2.8x2.8 RAISE
COLLAR - STEEL
PLAN, SECTION & DETAIL

Project No. 169520628
Scale AS NOTED
Drawing No. 551-S007
Revision 1

ORIGINAL SHEET - A1
Dwg Updated: 5/7/2021 9:11:44 AM By: gcrocker



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SCALE 1:25
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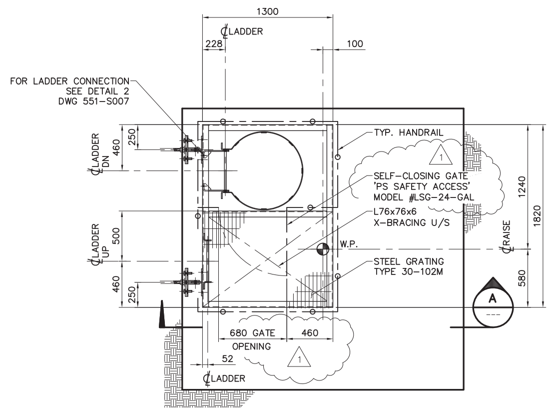
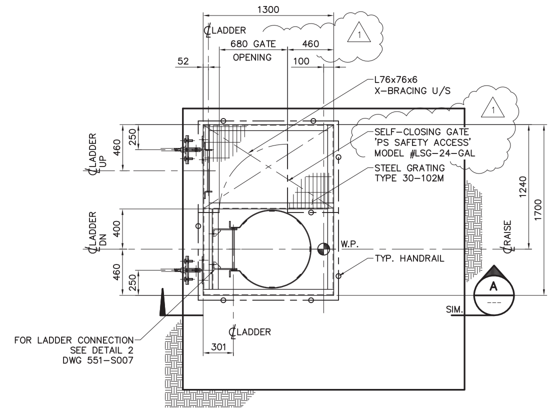
FRAME: C200x21 UNO



3 DETAIL
551-S002
PLATFORM
SCALE 1:25
ELEV.: T.O. STEEL
FRAME: C200x21 UNO



4 DETAIL
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PLATFORM
SCALE 1:25
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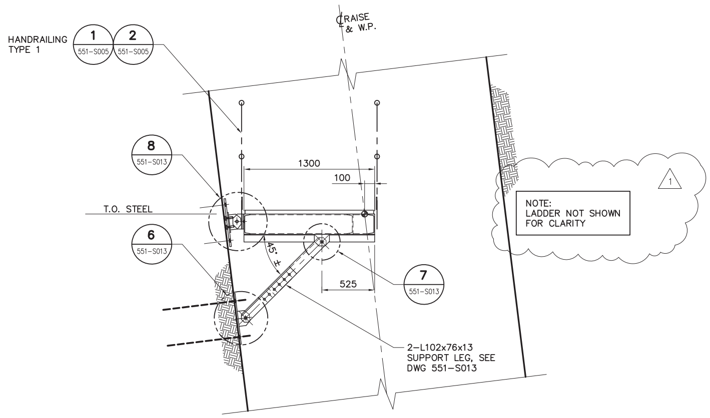
FRAME: C200x21 UNO



A SECTION
SCALE 1:25

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2. FOR GENERAL FABRICATION NOTES SEE DWG 551-S005
3. FOR HANDRAIL NOTES & DETAILS SEE DWG 551-S005

Rev.	Description	Date	By	Chk	Eng.	Appd.
1	REMOVED HATCHES & ADDED GATES	2021-05-07	GC	DJG	RJV	MD
0	FOR CONSTRUCTION	2021-04-23	GC	DJG	RJV	MD

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File Name: 551-S012-1.dwg | GC Dwn. | Chkd. | RV Dsgn. | 2020.10.26 YY.MM.DD

Title
LADDERWAY - 2.8x2.8 RAISE
PLATFORM - STEEL
SECTIONS & DETAILS

Project No. 169520628
Scale AS NOTED
Drawing No. 551-S012
Revision 1

Dwg Updated: 5/7/2021 9:01:53 AM By: gcrocker

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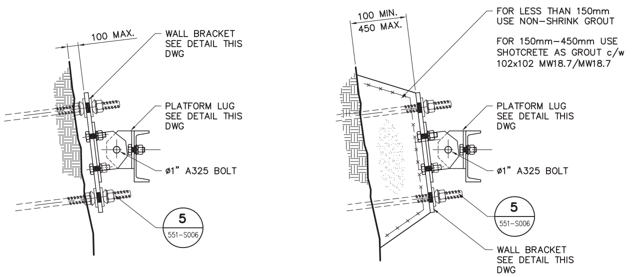
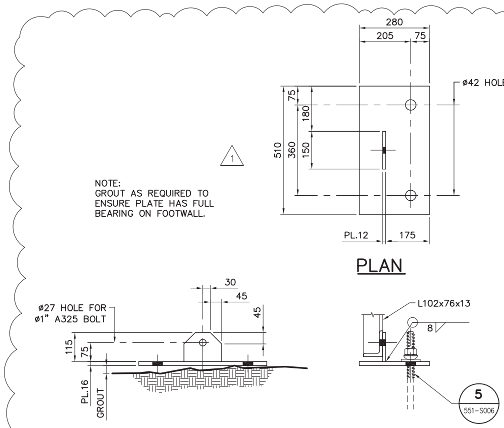


NORMAL OVERBREAK

LARGE OVERBREAK

8 DETAIL

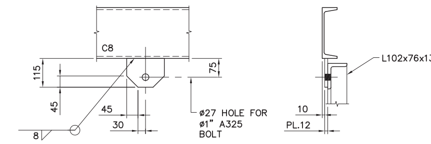
SCALE 1:10



SIDE VIEW

ELEVATION

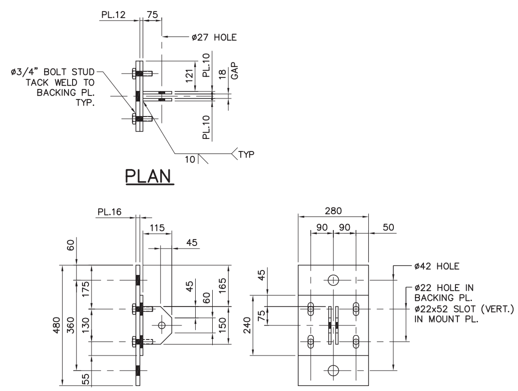
6 DETAIL
551-S012
SCALE 1:10



SIDE VIEW

ELEVATION

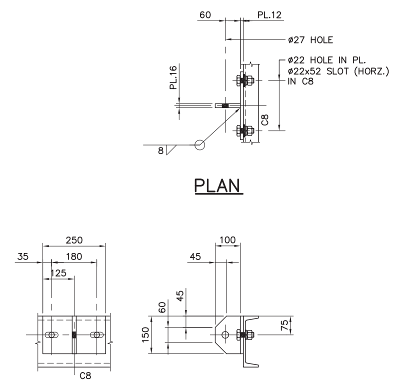
7 DETAIL
551-S012
SCALE 1:10



SIDE VIEW

ELEVATION

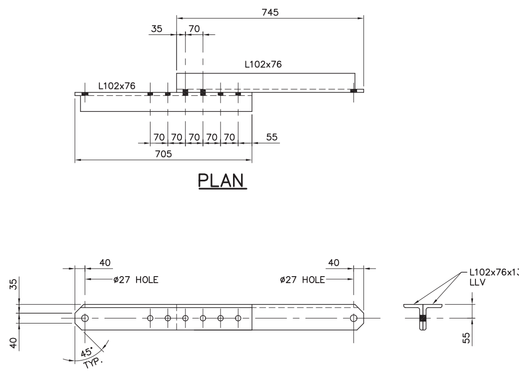
DETAIL – WALL BRACKET
SCALE 1:10



ELEVATION

SIDE VIEW

DETAIL – PLATFORM LUG
SCALE 1:10



ELEVATION

END VIEW

DETAIL – SUPPORT LEG
SCALE 1:10

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3. FOR HANDRAIL NOTES & DETAILS SEE DWG 551–S005

Rev.	Description	Date	By	Chk	Eng.	Appd.
1	MODIFIED DETAIL 6	2021-05-07	GC	DJG	RJV	MD
0	FOR CONSTRUCTION	2021-04-23	GC	DJG	RJV	MD

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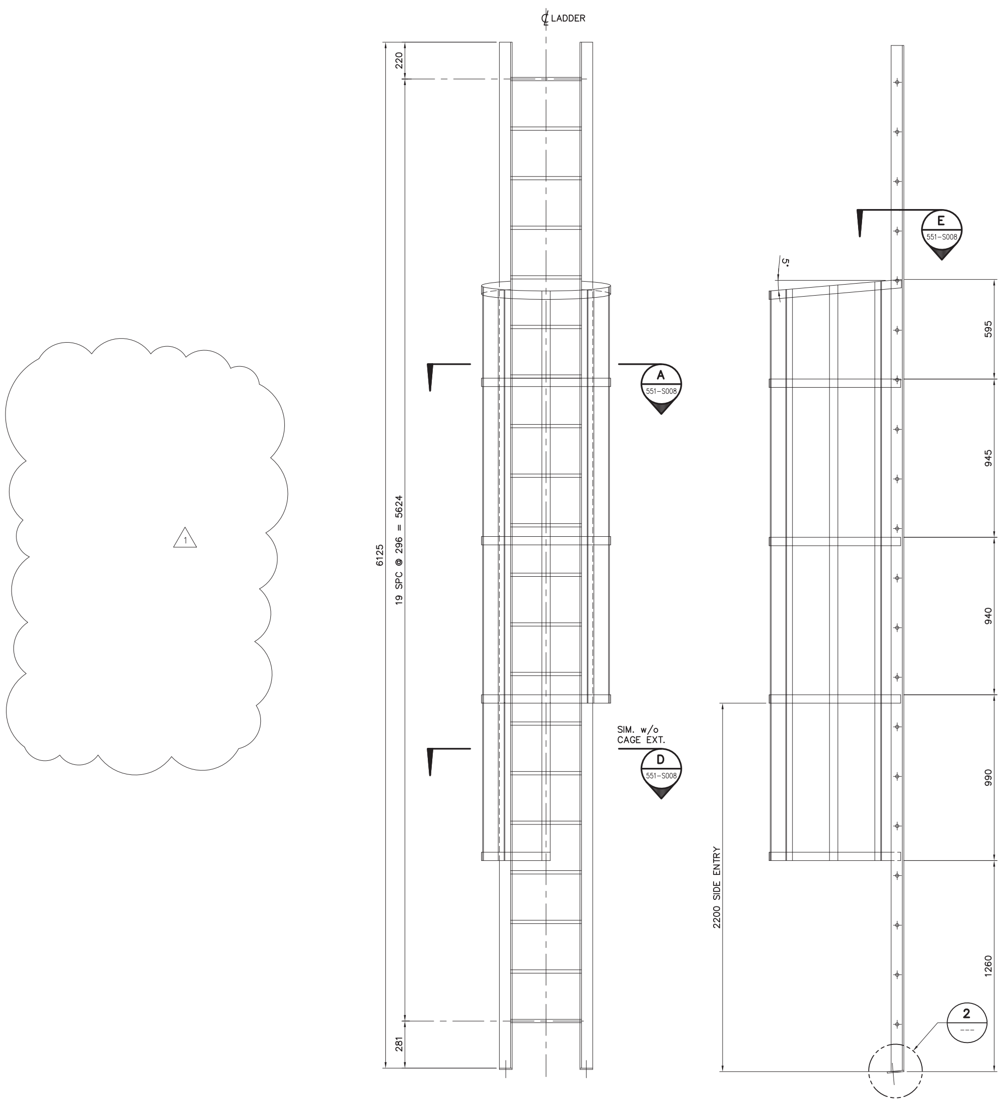
File Name: 551-S013-1.dwg | GC Dwn. | Chkd. | RV Dsgn. | 2020.10.26 YY.MM.DD

Title
LADDERWAY - 2.8x2.8 RAISE
PLATFORM - STEEL
MISC. DETAILS & SECTION

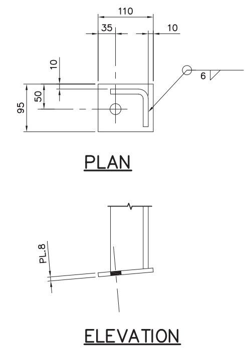
Project No. 169520628
Scale AS NOTED
Drawing No. 551-S013
Revision 1

ORIGINAL SHEET - A1

Dwg Updated: 5/6/2021 3:13:18 PM By: gcrocker



STD. LADDER c/w CAGE
SCALE 1:15
MID DECK: AS SHOWN
UPPER & LOWER DECK: OPP. HAND
REF.: 551-S002



2 DETAIL
SCALE 1:5

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2. FOR GENERAL FABRICATION NOTES SEE DWG 551-S005

Rev.	Description	Date	By	Chk	Eng.	Appd.
1	REMOVED LADDER & REVISED LADDER QUANTITY	2021-05-07	GC	DJG	RJV	MD
0	FOR CONSTRUCTION	2021-04-23	GC	DJG	RJV	MD

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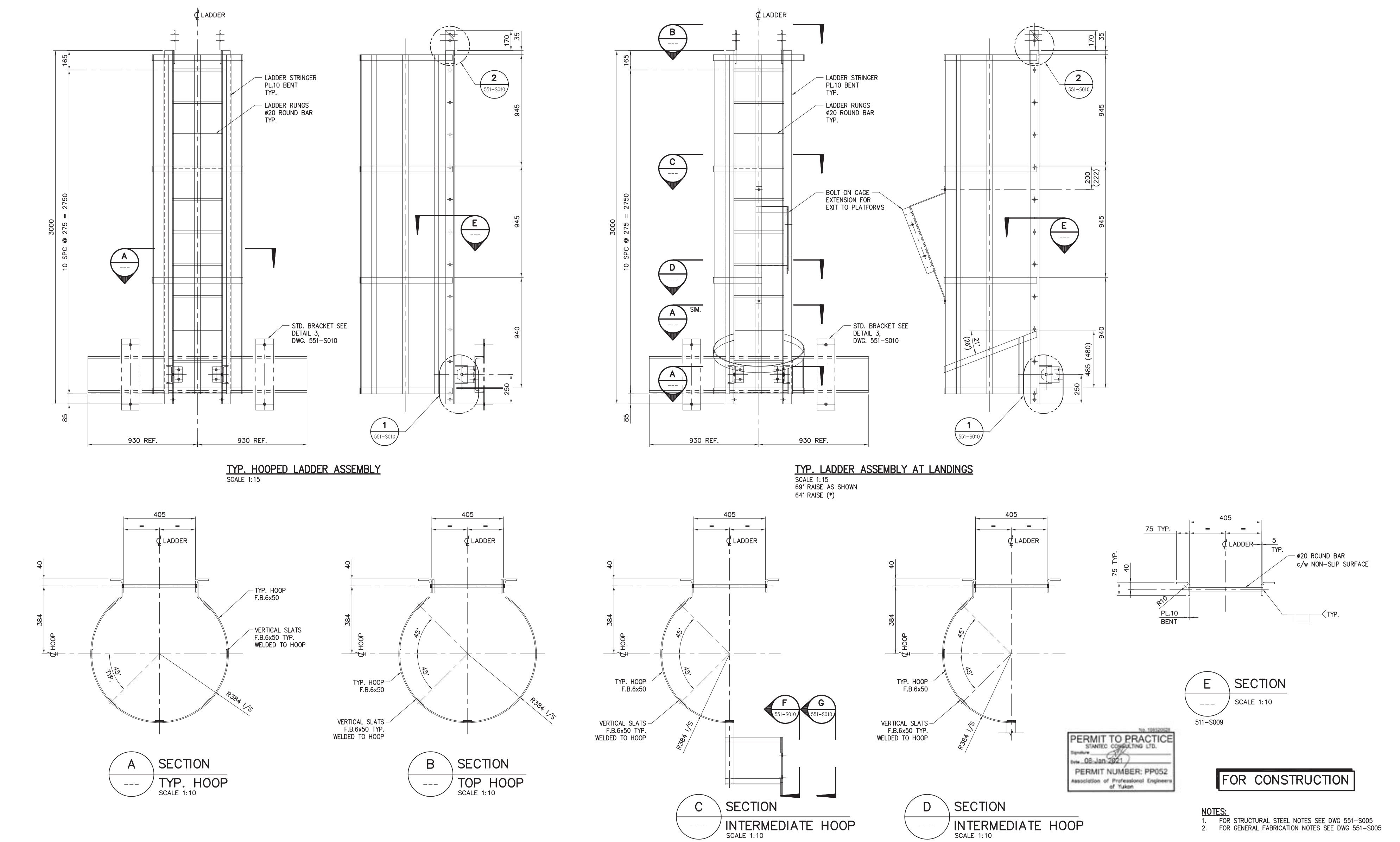
File Name: 551-S014-1.dwg GC 2020.10.26
Dwn. Chkd. Dsgn. YY.MM.DD

Title
LADDERWAY - 2.8x2.8 RAISE
LADDER & SAFETY CAGE - STEEL
ELEVATIONS & SECTIONS

Project No. 169520628
Scale AS NOTED
Drawing No. 551-S014
Revision 1

ORIGINAL SHEET - A1
Dwg Updated: 5/6/2021 3:15:14 PM By: gcrocker



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2. FOR GENERAL FABRICATION NOTES SEE DWG 551-S005

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YUKON TERRITORY

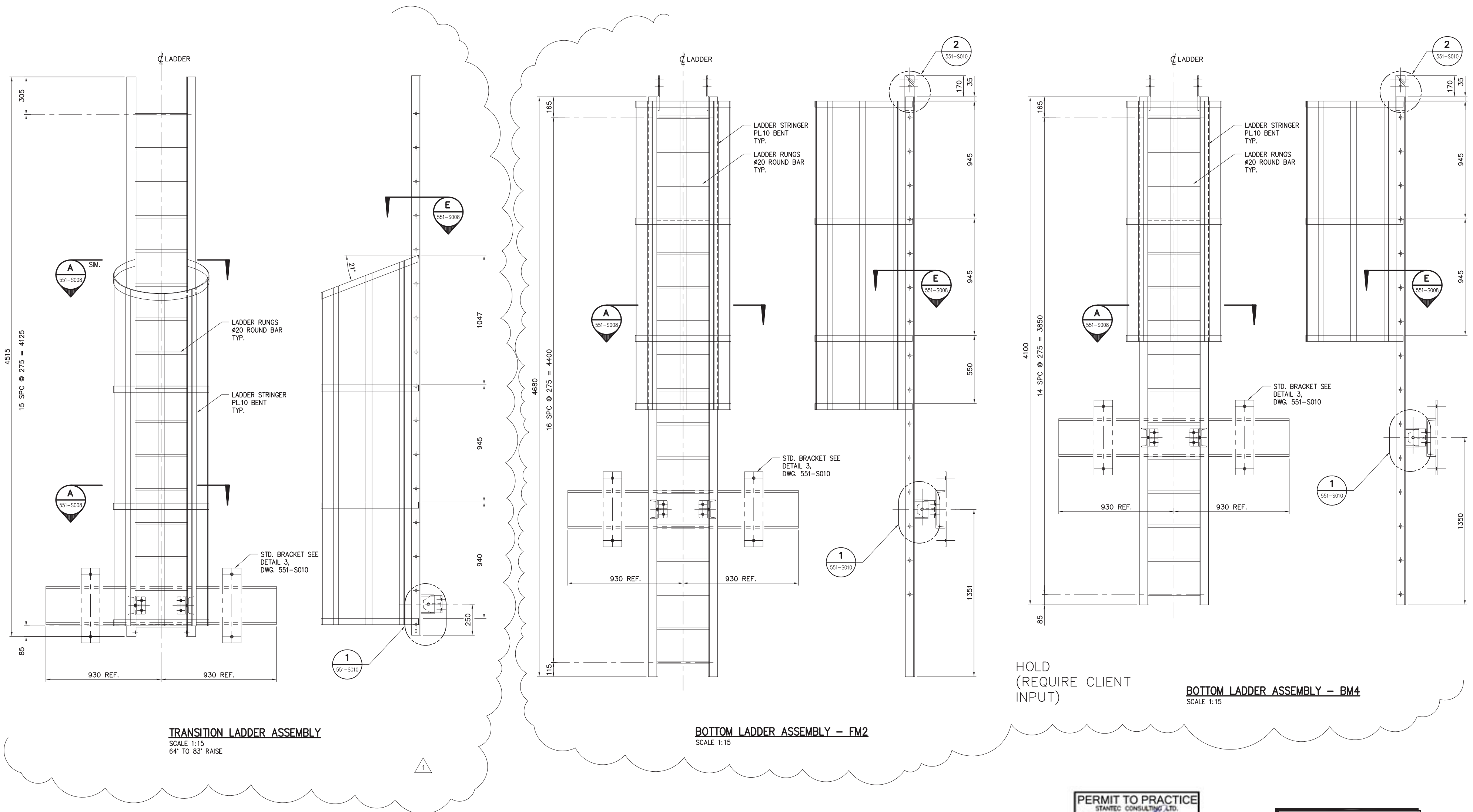
File Name: 551-S008-0.dwg | GC Dwn. | Chkd. | RV Dsgn. | 2020.10.26 YY.MM.DD

Title
LADDERWAY - 2.8x2.8 RAISE
LADDER & SAFETY CAGE - STEEL
ELEVATIONS & SECTIONS

Project No. 169520628
Scale AS NOTED
Drawing No. 551-S008
Revision 0

ORIGINAL SHEET - A1

Dwg Updated: 1/6/2021 3:48:51 PM By: gcrocker



TRANSITION LADDER ASSEMBLY
SCALE 1:15
64° TO 83° RAISE

BOTTOM LADDER ASSEMBLY – FM2
SCALE 1:15

BOTTOM LADDER ASSEMBLY – BM4
SCALE 1:15

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2. FOR GENERAL FABRICATION NOTES SEE DWG 551-S005

Rev.	Description	Date	By	Chk	Eng.	Appd.
1	LADDER DETAIL ADDED & CHANGED HOLD ITEMS	2021-04-23	GC	DJG	RJV	MD
0	FOR CONSTRUCTION	2021-01-07	GC	DJG	RV	MD

Ref. Dwg. No. Ref. Dwg. Title

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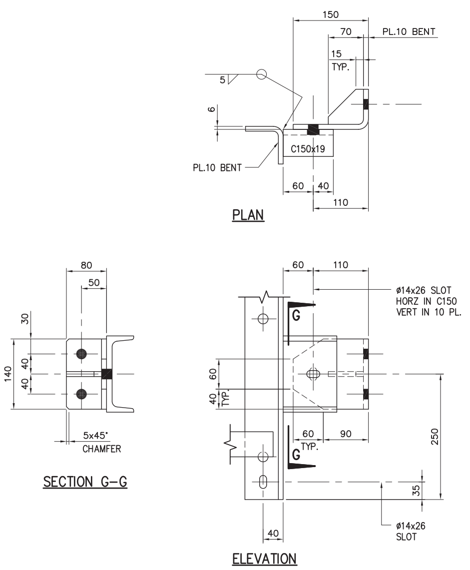
YUKON TERRITORY

File Name: 551-S009-1.dwg | GC | | | 2020.10.26
Dwn. Chkd. Dsgn. YY.MM.DD

Title
LADDERWAY - 2.8x2.8 RAISE
LADDER & SAFETY CAGE - STEEL
ELEVATIONS & SECTIONS

Project No. 169520628
Scale AS NOTED
Drawing No. 551-S009
Revision 1

ORIGINAL SHEET - A1
Dwg Updated: 4/23/2021 10:44:18 AM By: gcrocker



1 DETAIL
551-S008
551-S009
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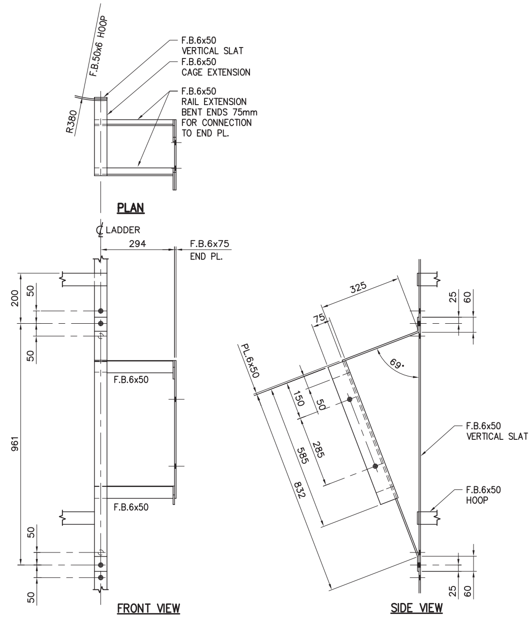
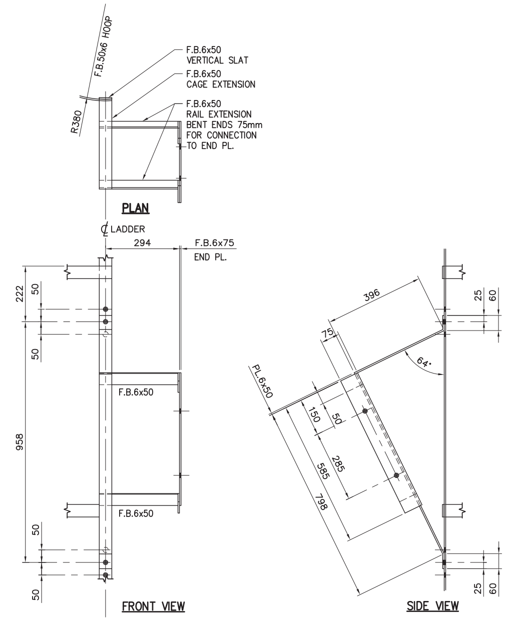
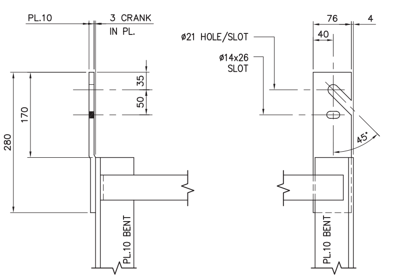


F SECTION
551-S008
CAGE EXTENSION
SCALE: 1:10
69° RAISE



G SECTION
551-S008
CAGE EXTENSION
SCALE: 1:10
64° RAISE



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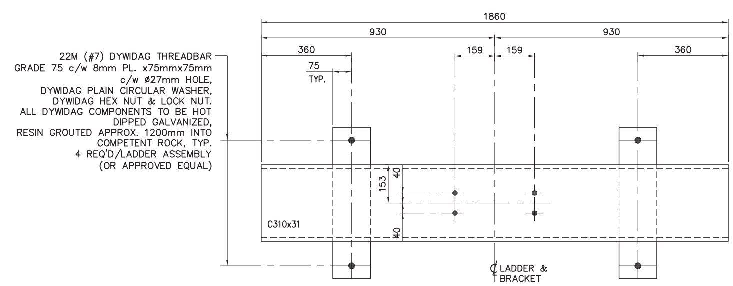

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551-S009
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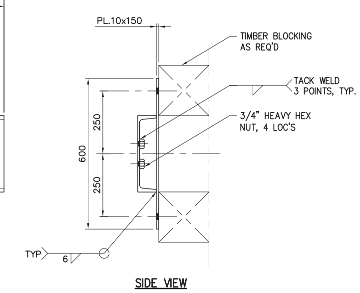

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Rev.	Description	Date	By	Chk	Eng.	Appd.
0	FOR CONSTRUCTION	2021-01-07	GC	DJG	RV	MD

Permit-Seal

PROFESSIONAL YUKON R.J. VANDER WEL TERRITORY ENGINEER

Stantec

200 - 147 McIntyer St West
North Bay, Ontario P1B 2Y5
www.stantec.com

The Contractor shall verify and be responsible for all dimensions. DO NOT scale the drawing - any errors or omissions shall be reported to Stantec without delay.

Client/Project
PROCON GROUP

ALEXCO RESOURCE
RAISE LADDERWAY
YUKON TERRITORY

File Name: 551-S010-0.dwg	GC		RV	2020.10.26
	Dwn.	Chkd.	Dsgn.	YY.MM.DD

Title
LADDERWAY - 2.8x2.8 RAISE
LADDER & SAFETY CAGE - STEEL
SECTIONS & DETAILS

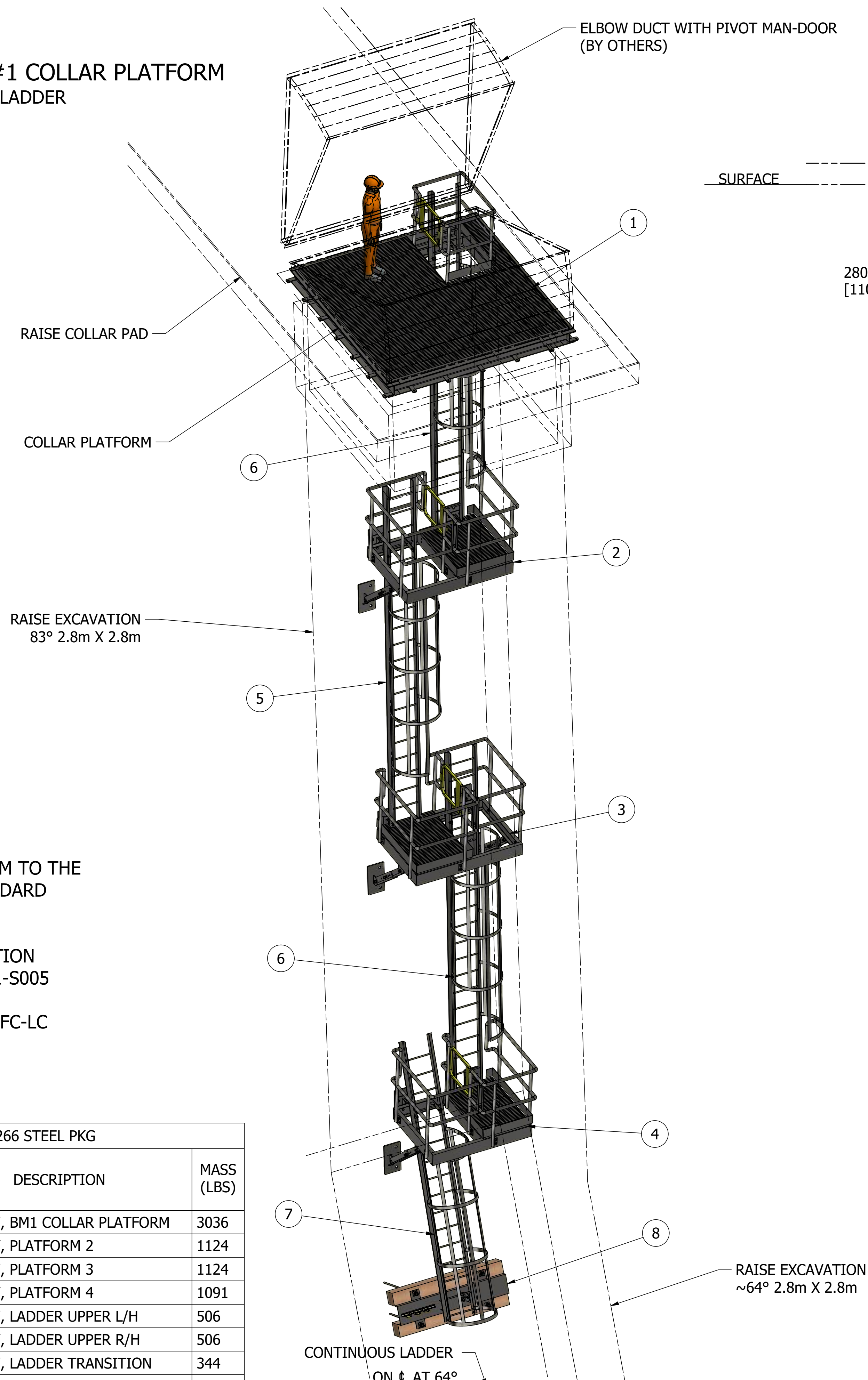
Project No. 169520628
Scale AS NOTED
Drawing No. 551-S010
Revision 0

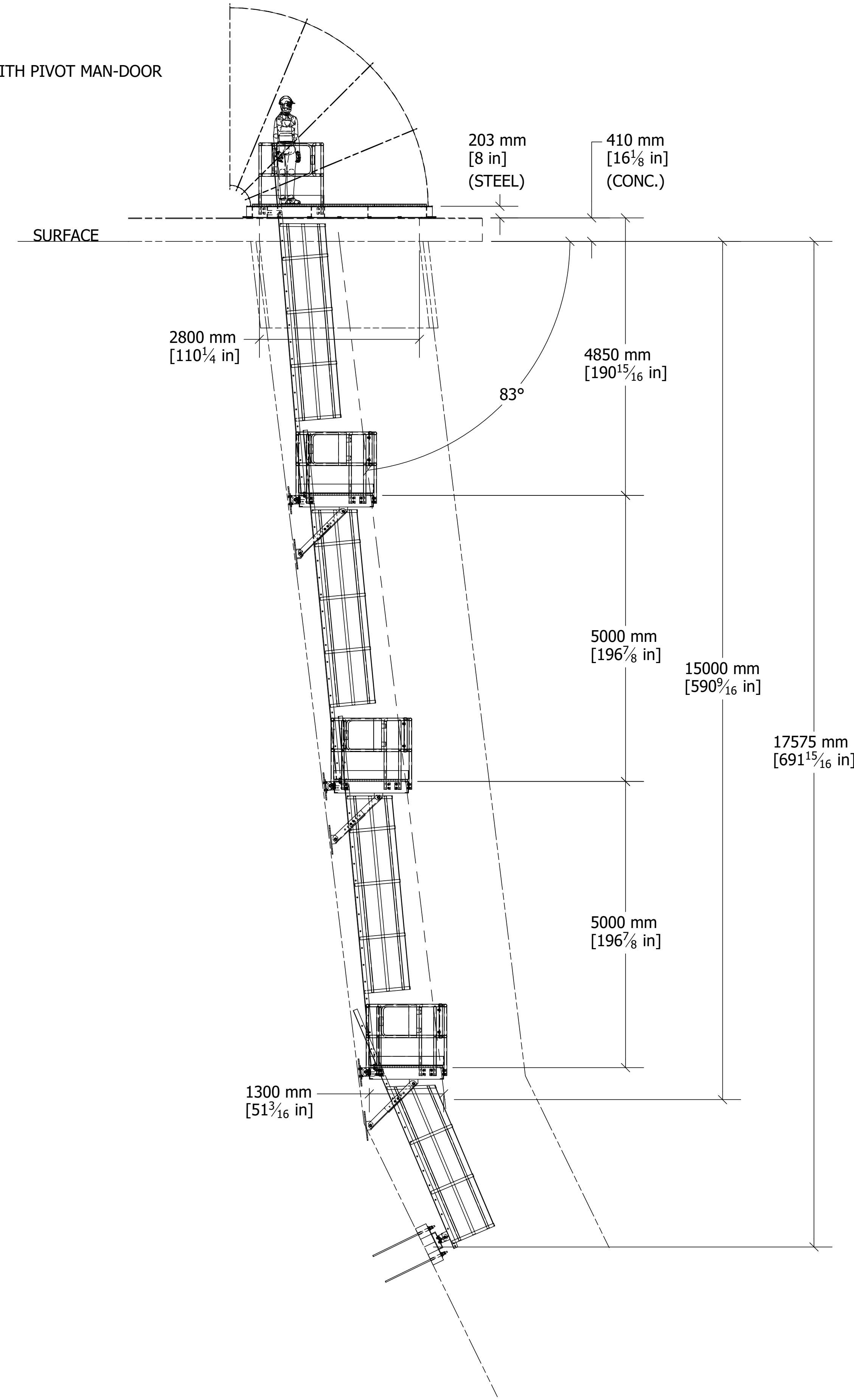
Dwg Updated: 1/6/2021 3:48:18 PM By: gcrocker

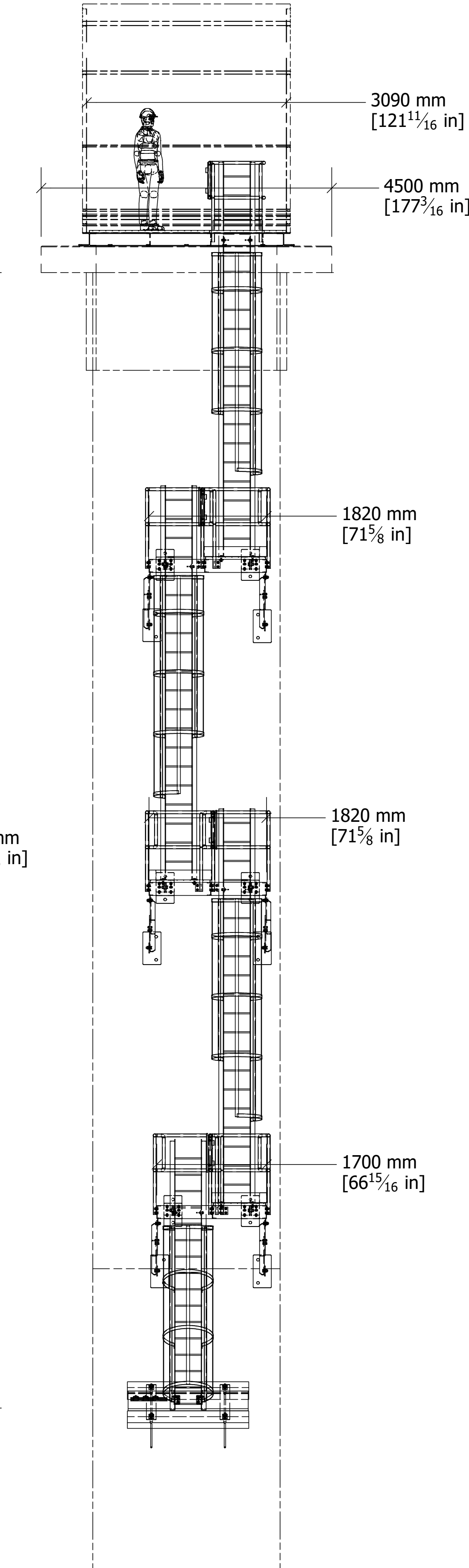
843-4008266-10

ESCAPEWAY STEEL
BIRMINGHAM SEGMENT #1 COLLAR PLATFORM

~18M INCLUDING TRANSITION LADDER
TOTAL WEIGHT ~8500 LBS







NOTES:

1.) ALL WELDING SHALL CONFORM TO THE REQUIREMENTS OF CSA STANDARD W59 (LATEST EDITION)

2.) REFER TO GENERAL FABRICATION NOTES ON STANTEC DWG 551-S005

3.) REFER TO WELD PROCEDURE FC-LC

PARTS LIST - 843-4008266 STEEL PKG

ITEM	ITEM QTY	PART NUMBER	DESCRIPTION	MASS (LBS)
1	1	4008266-AS1	ASSEMBLY, BM1 COLLAR PLATFORM	3036
2	1	4008266-AS2	ASSEMBLY, PLATFORM 2	1124
3	1	4008266-AS3	ASSEMBLY, PLATFORM 3	1124
4	1	4008266-AS4	ASSEMBLY, PLATFORM 4	1091
5	1	4008266-AS5	ASSEMBLY, LADDER UPPER L/H	506
6	2	4008266-AS6	ASSEMBLY, LADDER UPPER R/H	506
7	1	4008266-AS7	ASSEMBLY, LADDER TRANSITION	344
8	1	4007290-AS4B	SEE DWG 843-4007290-AS4-20	369

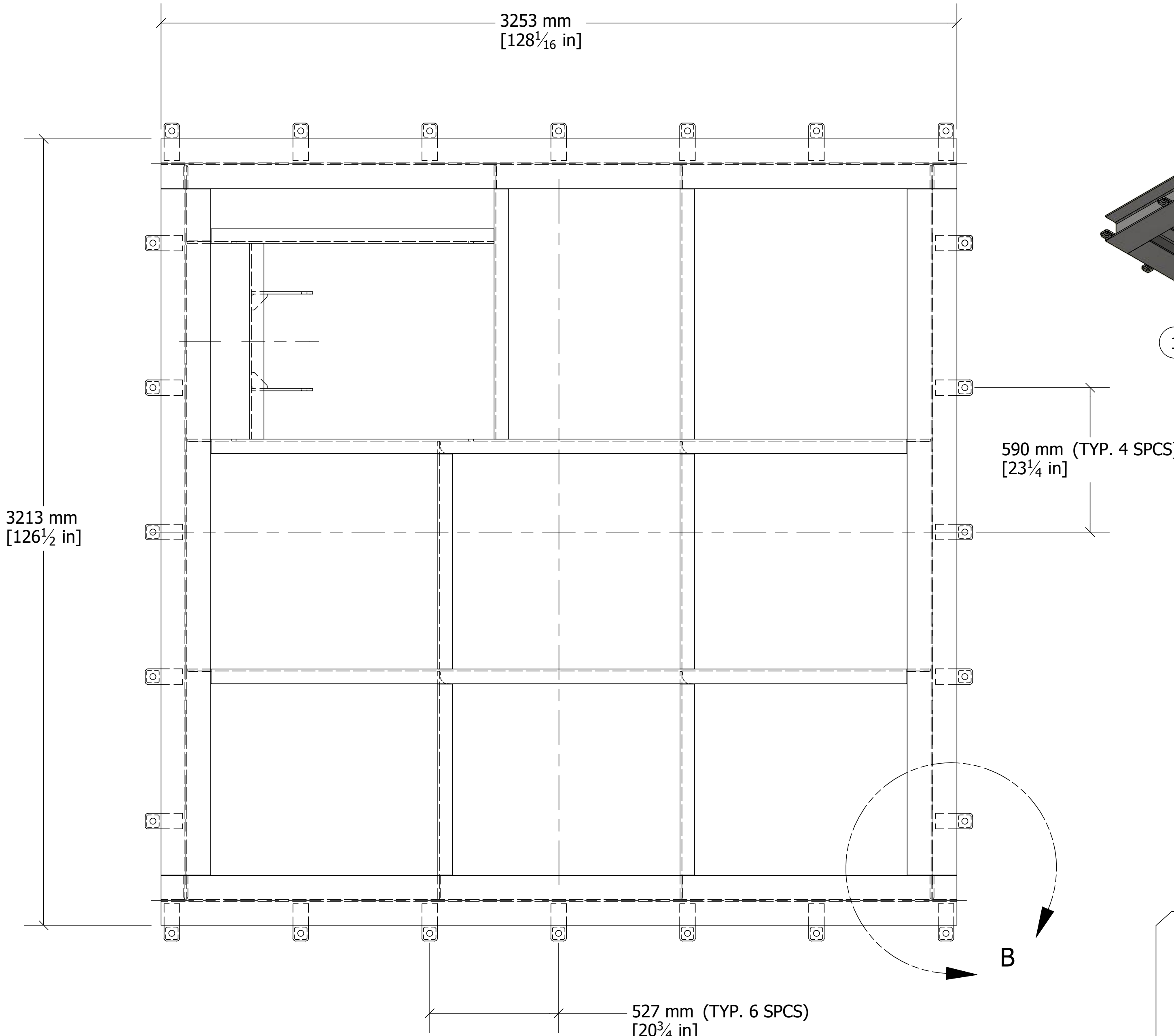
FOR CONSTRUCTION

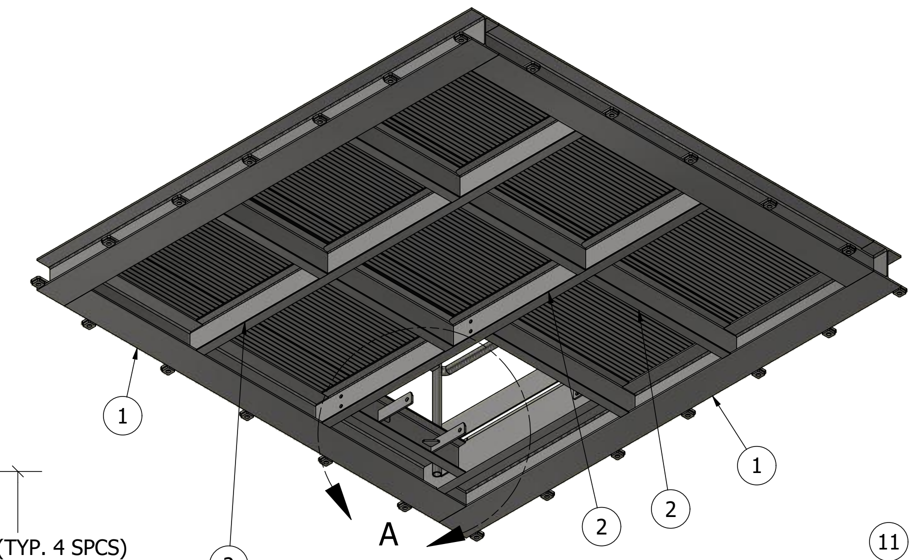
ALEXCO
KENO PROJECT 569

Procon MINING & TUNNELLING LTD.
2203 - 8th St., NISKU, ALBERTA

SIZE B | DWG # 843-4008266-10 | TITLE FABRICATION DETAILS - ESCAPEWAY STEEL BM1

DRAWN BY I.LAING | CHECKED BY
SCALE 1:100 | SHEET 1 of 43
REV 0 | DATE 2021-05-20





ITEM	ITEM QTY	PART NUMBER	DESCRIPTION
1	4	W 8x31	BEAM, W8X31
2	10	C 8 x 13.75	CHANNEL, C8X13.75
3	25	4008266-AS13	WELDMENT, COLLAR PLATFORM CONC. MNT.
4	1	4008266-BG1	BAR GRATING (1-1/2" x 3/16" - W19-4)
5	1	4008266-BG2	BAR GRATING (1-1/2" x 3/16" - W19-4)
6	1	4008266-BG3	BAR GRATING (1-1/2" x 3/16" - W19-4)
7	1	4008266-BG4	BAR GRATING (1-1/2" x 3/16" - W19-4)
8	1	4008266-BG5	BAR GRATING (1-1/2" x 3/16" - W19-4)
9	1	4008266-BG6	BAR GRATING (1-1/2" x 3/16" - W19-4)
10	2	4008266-PL9	PLATE, UPPER LADDER MNT.
11	2	4008266-PL10	PLATE, LADDER UPPER MNT. GUSSET
12	1	LSG-24-PCY	GATE KIT, SELF-CLOSING
13	1	4008266-AS12	WELDMENT, GUARDRAIL 1

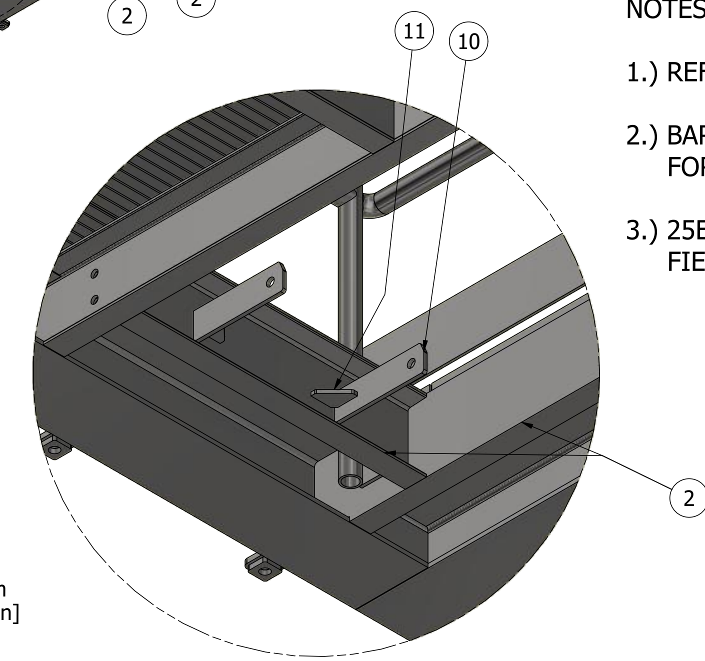
PARTS LIST

NOTES:

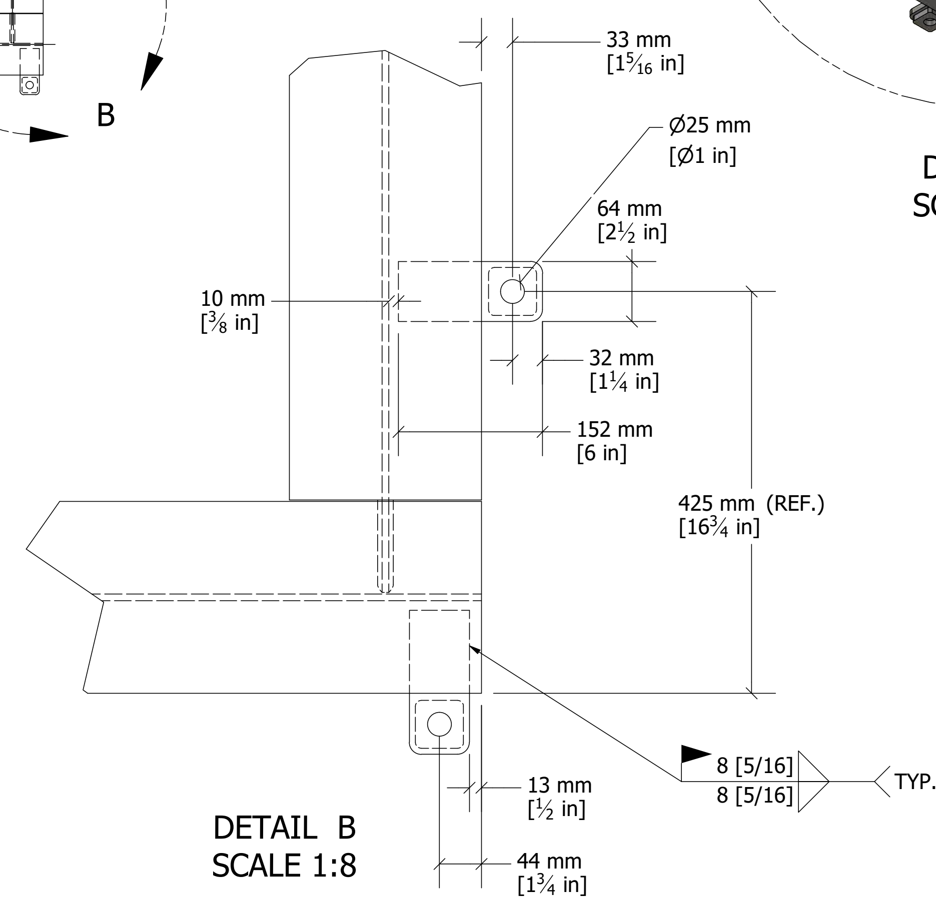
1.) REFER TO STANTEC DWG 551-S007 & NOTES ON 551-S005

2.) BAR GRATING SHIPPED LOOSE AS PRE-CUT, FULLY BANDED FOR FIELD MECHANICAL MOUNTING.

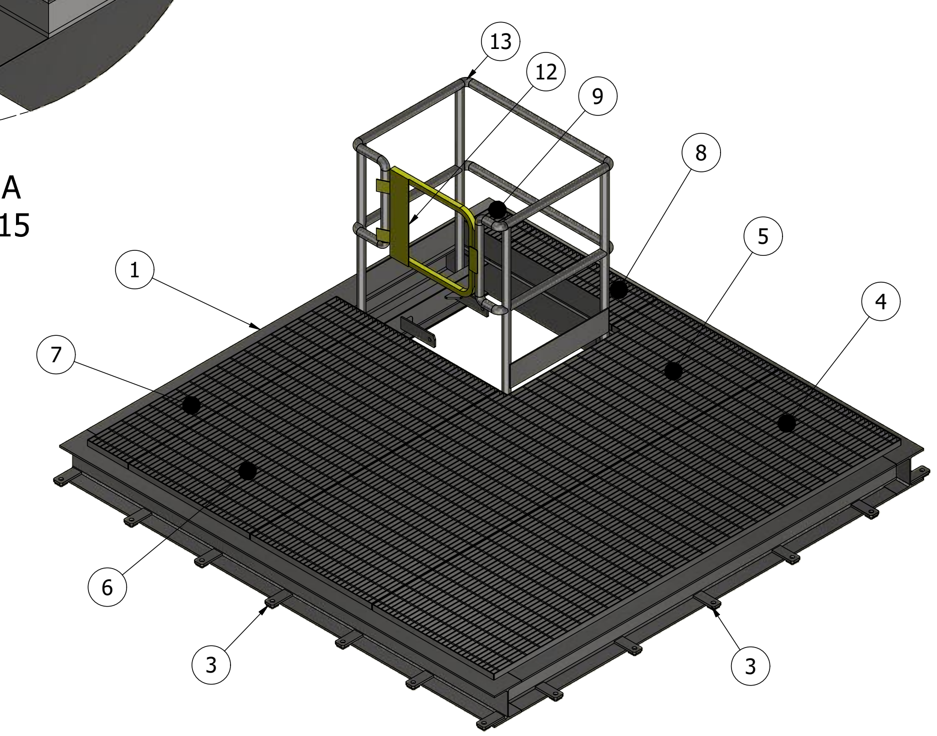
3.) 25EA. MOUNTING CLIPS [ITEM3] SHIPPED LOOSE FOR FIELD WELDING.



DETAIL A
SCALE 1:15



DETAIL B
SCALE 1:8

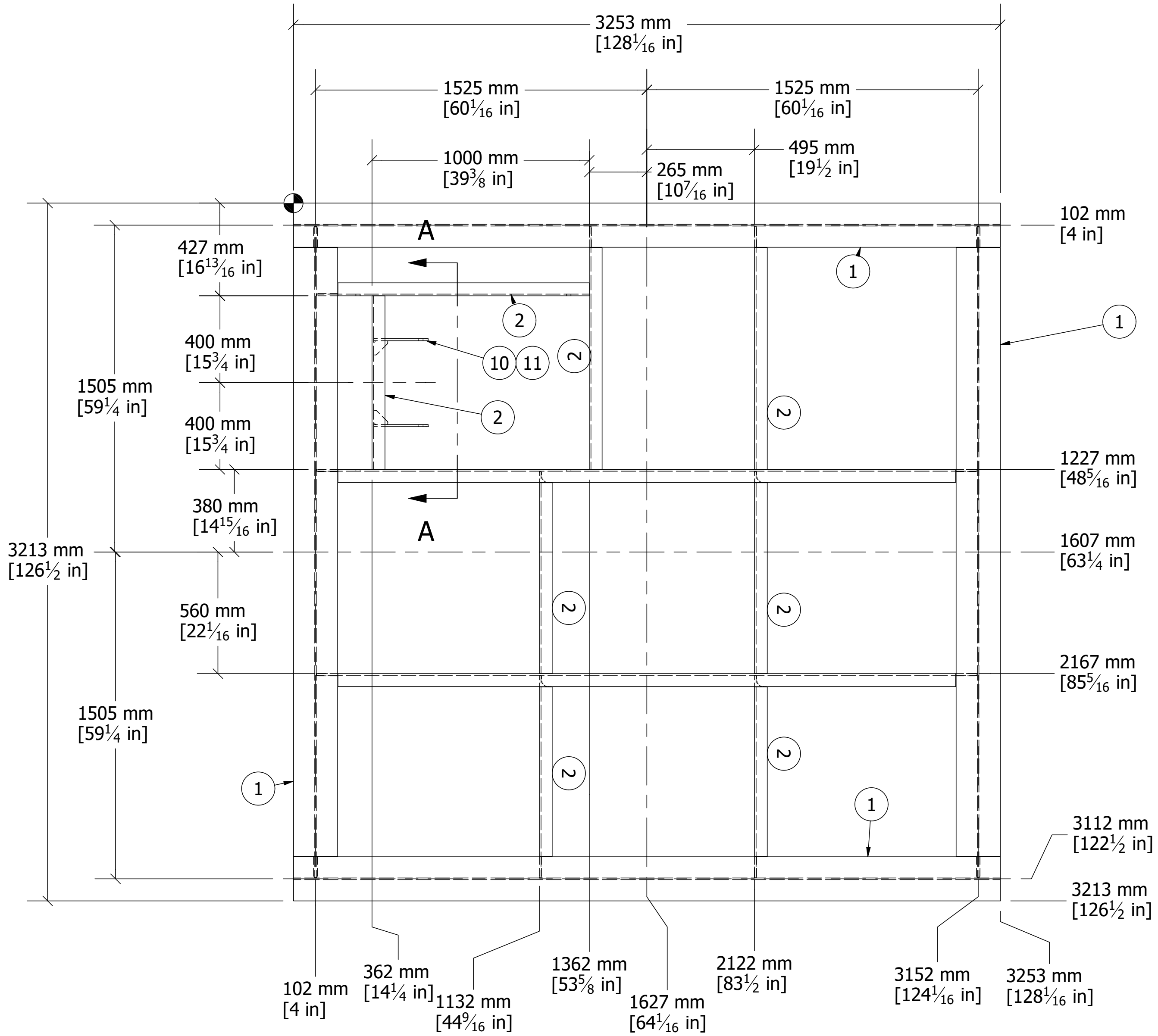


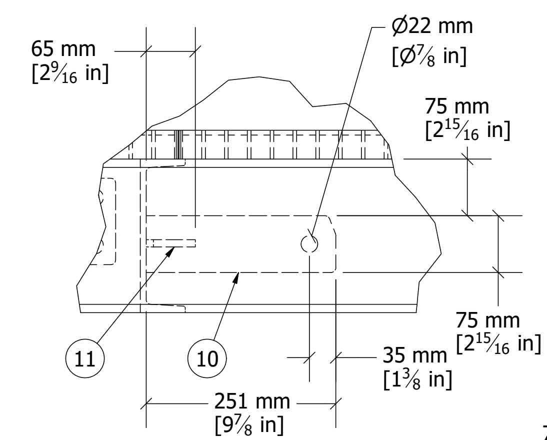
4008266-AS1

ASSEMBLY, BM1 COLLAR PLATFORM
MOUNTING CLIP DETAILS
TOTAL WEIGHT C/W BAR GRATING AND GUARDRAIL ~3031 LBS
1EA. REQ'D

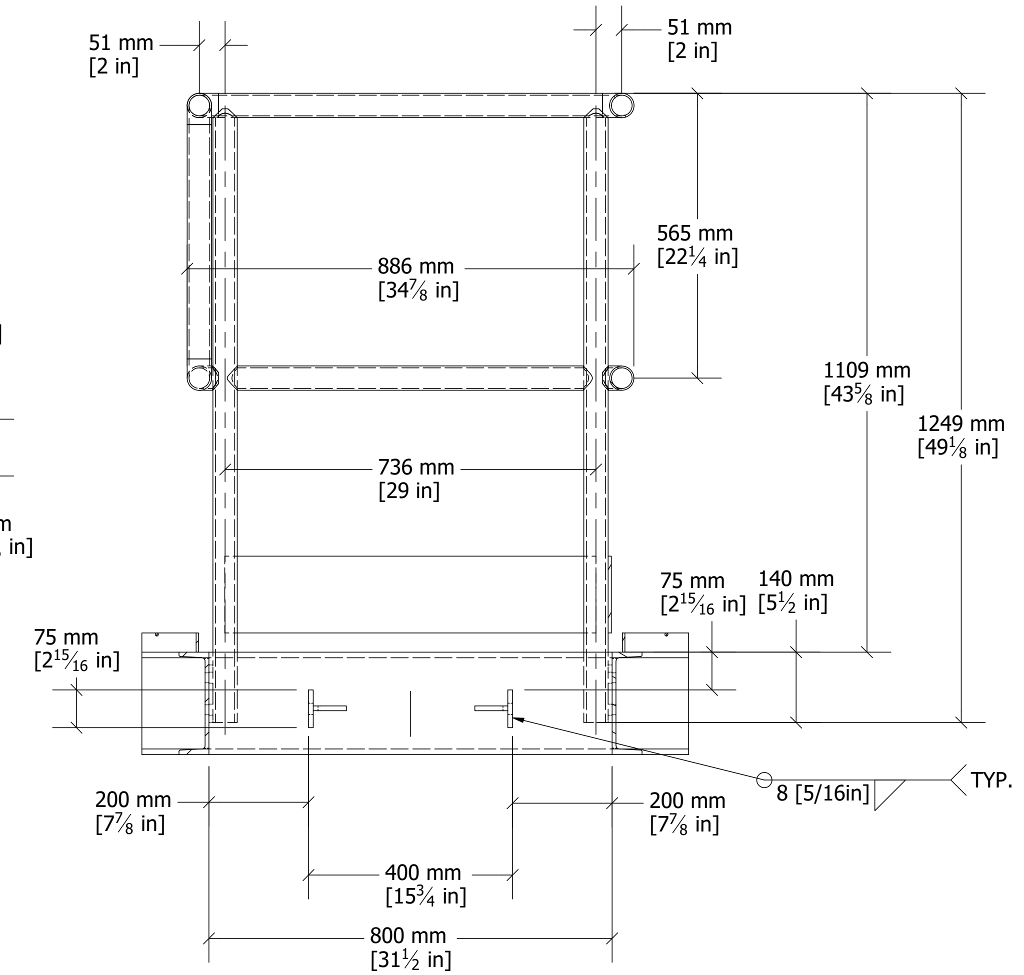
FOR CONSTRUCTION

Procon MINING & TUNNELLING LTD. 2203 - 8th St., NISKU, ALBERTA		DRAWN BY I.LAING	CHECKED BY
SIZE B	DWG # 843-4008266-10	SCALE 1:25	SHEET 2 of 43
TITLE ASSEMBLY, BM1 COLLAR PLATFORM		REV 0	DATE 2021-05-20

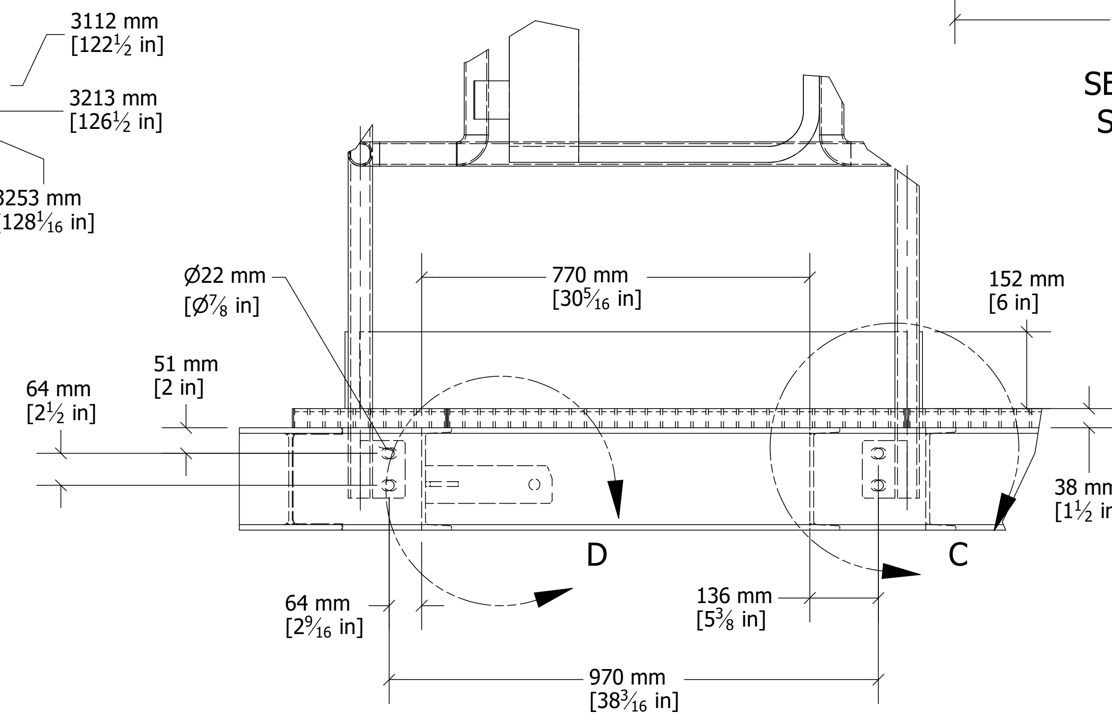




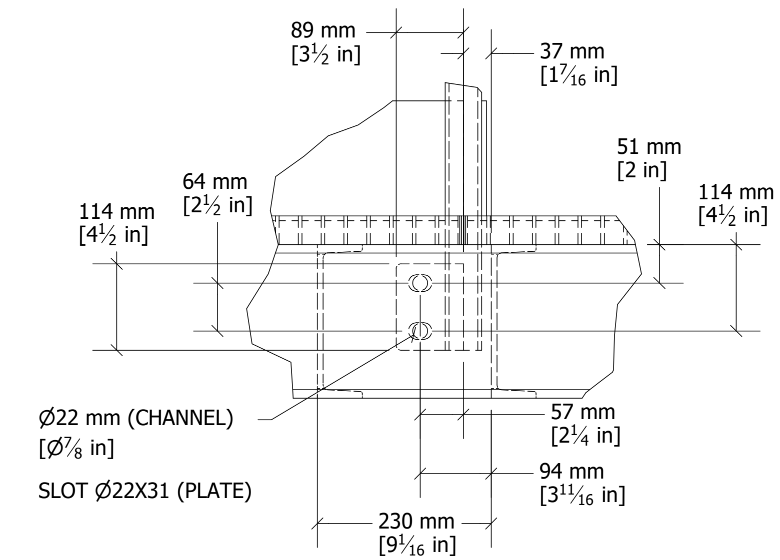
DETAIL D
SCALE 1:10



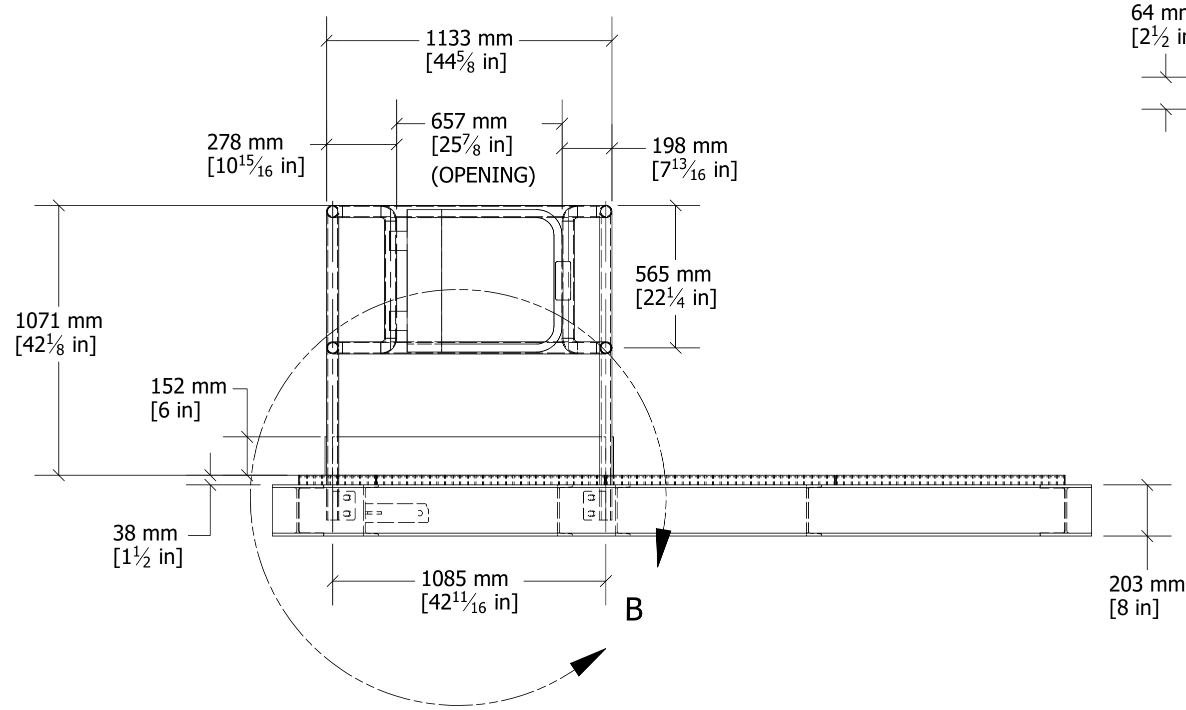
SECTION A-A
SCALE 1:15



DETAIL B
SCALE 1:15



DETAIL C
SCALE 1:10



4008266-AS1

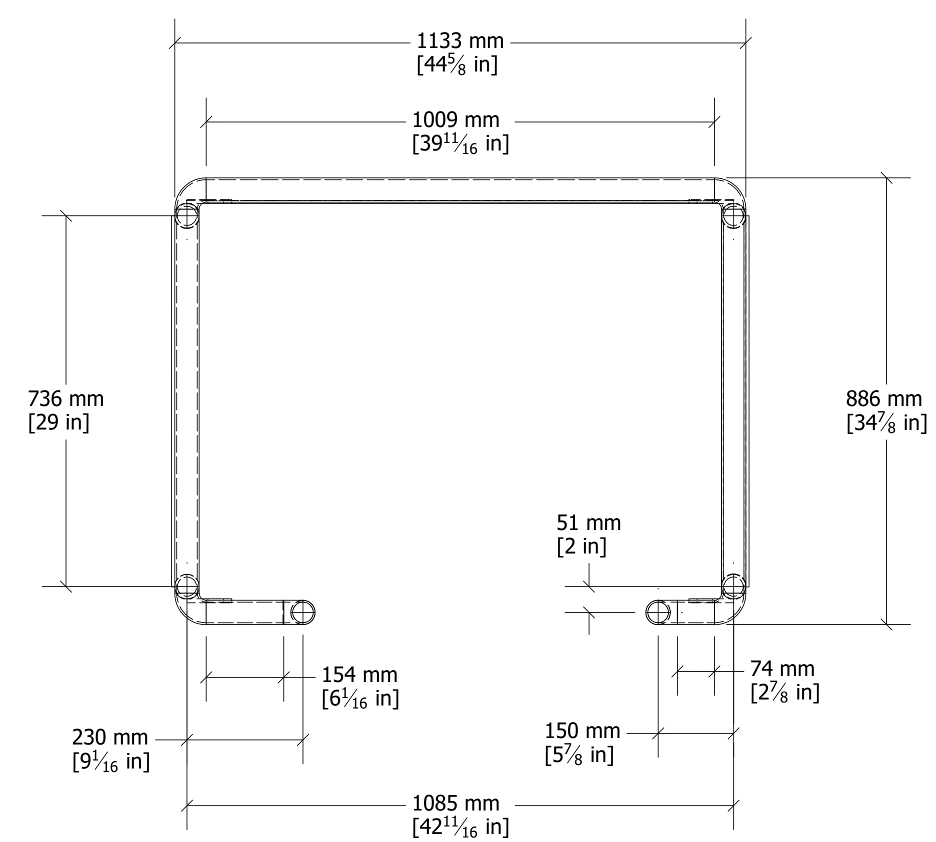
ASSEMBLY, BM1 COLLAR PLATFORM
FRAMEWORK, LADDER MOUNTING AND
GUARDRAIL W/SELF-CLOSING GATE KIT

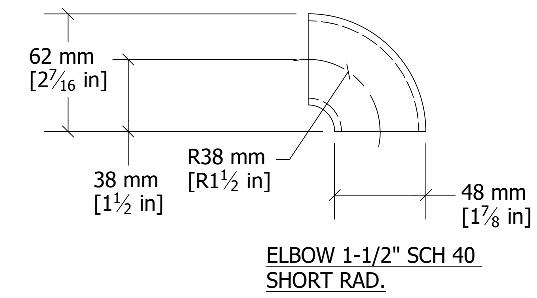
FOR CONSTRUCTION

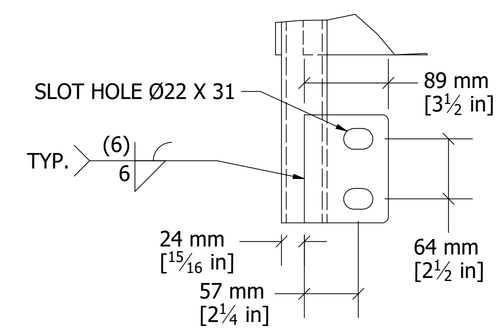
Procon MINING & TUNNELLING LTD. 2203 - 8th St., NISKU, ALBERTA		DRAWN BY I.LAING	CHECKED BY
SIZE B	DWG # 843-4008266-10	SCALE 1:30	SHEET 3 of 43
TITLE ASSEMBLY, BM1 COLLAR PLATFORM		REV 00	DATE 2021-05-20

PARTS LIST

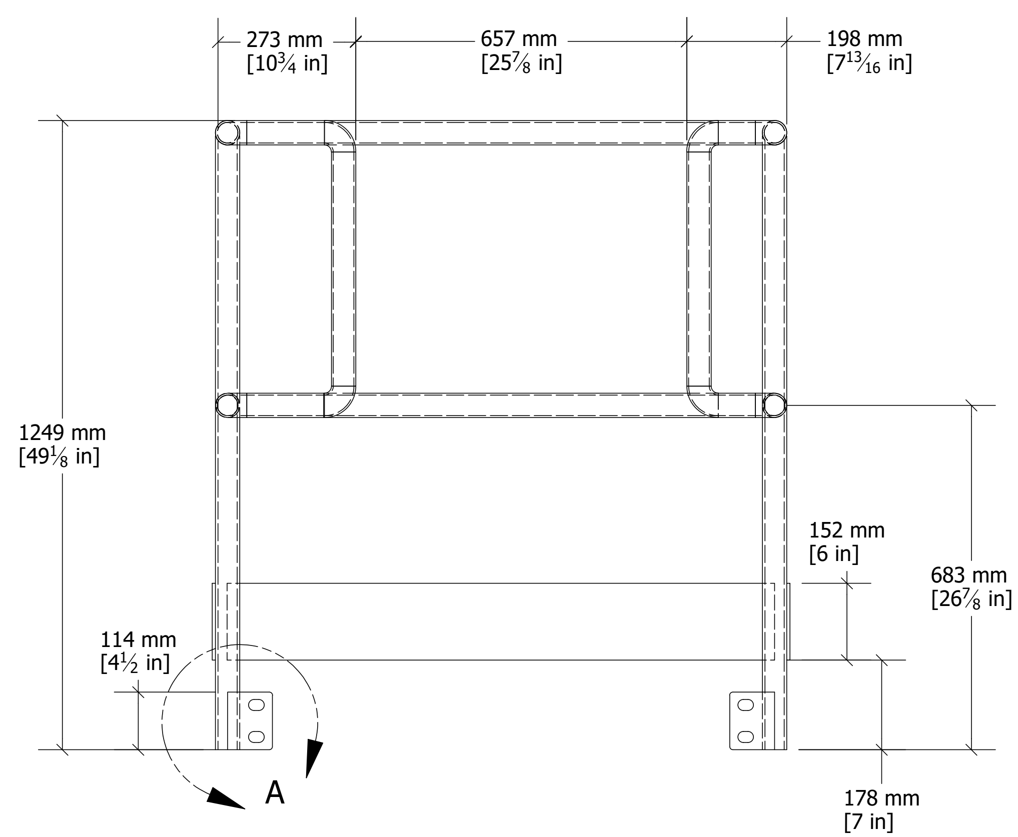
ITEM	ITEM QTY	ITEM LENGTH(IN)	PART NUMBER	THICKNESS	DESCRIPTION	MATERIAL
1	4		4008266-PL1	0.313 in	PLATE, MOUNTING	STEEL, 300W
2	2	29	FLAT 6x1/4 - 29		FLAT BAR, 6 X 1/4" X 29" LG.	STEEL, 300W
3	1	42.75	FLAT 6x1/4 - 42.75		FLAT BAR, 6 X 1/4" X 42.75" LG.	STEEL, 300W
4	12		ELBOW 90S - 1 1/2		ELBOW, 1-1/2" SCH 40 SHORT RAD.	STEEL, A53
5	1	2.89	PIPE 1 1/2-2.8946, PSW		PIPE, 1-1/2" SCH 40	STEEL, A53
6	1	2.9	PIPE 1 1/2-2.8964, PSW		PIPE, 1-1/2" SCH 40	STEEL, A53
7	2	6	PIPE 1 1/2-6, PSW		PIPE, 1-1/2" SCH 40	STEEL, A53
8	2	18.3	PIPE 1 1/2-18.3, PSW		PIPE, 1-1/2" SCH 40	STEEL, A53
9	2	28.97	PIPE 1 1/2-28.9711, PSW		PIPE, 1-1/2" SCH 40	STEEL, A53
10	2	30	PIPE 1 1/2-30, PSW		PIPE, 1-1/2" SCH 40	STEEL, A53
11	2	39.71	PIPE 1 1/2-39.7064, PSW		PIPE, 1-1/2" SCH 40	STEEL, A53
12	4	48.19	PIPE 1 1/2-48.19, PES		PIPE, 1-1/2" SCH 80	STEEL, A53

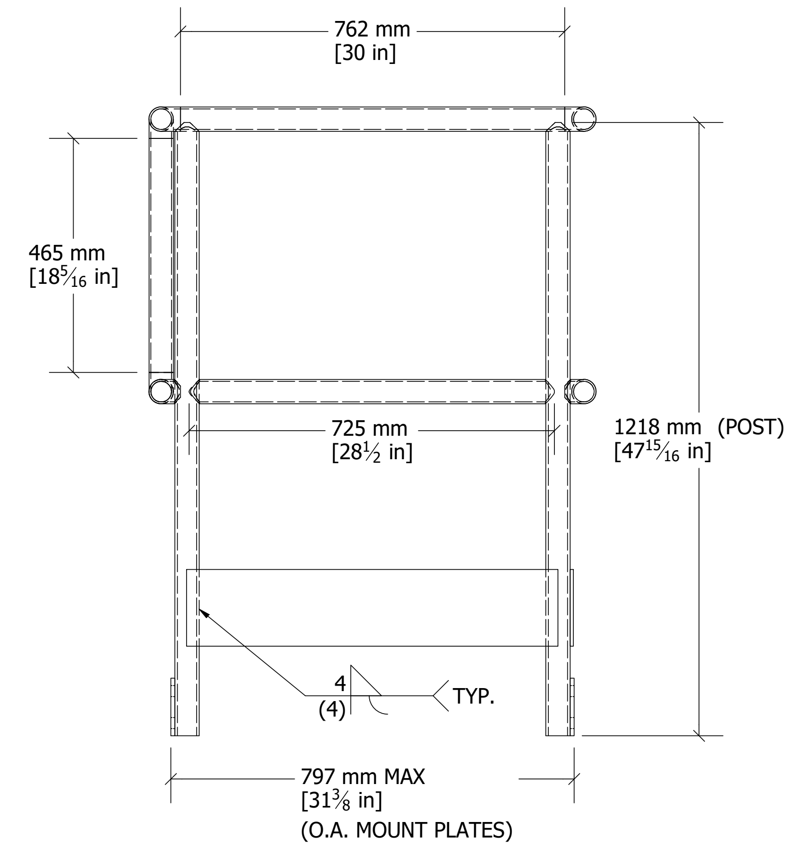


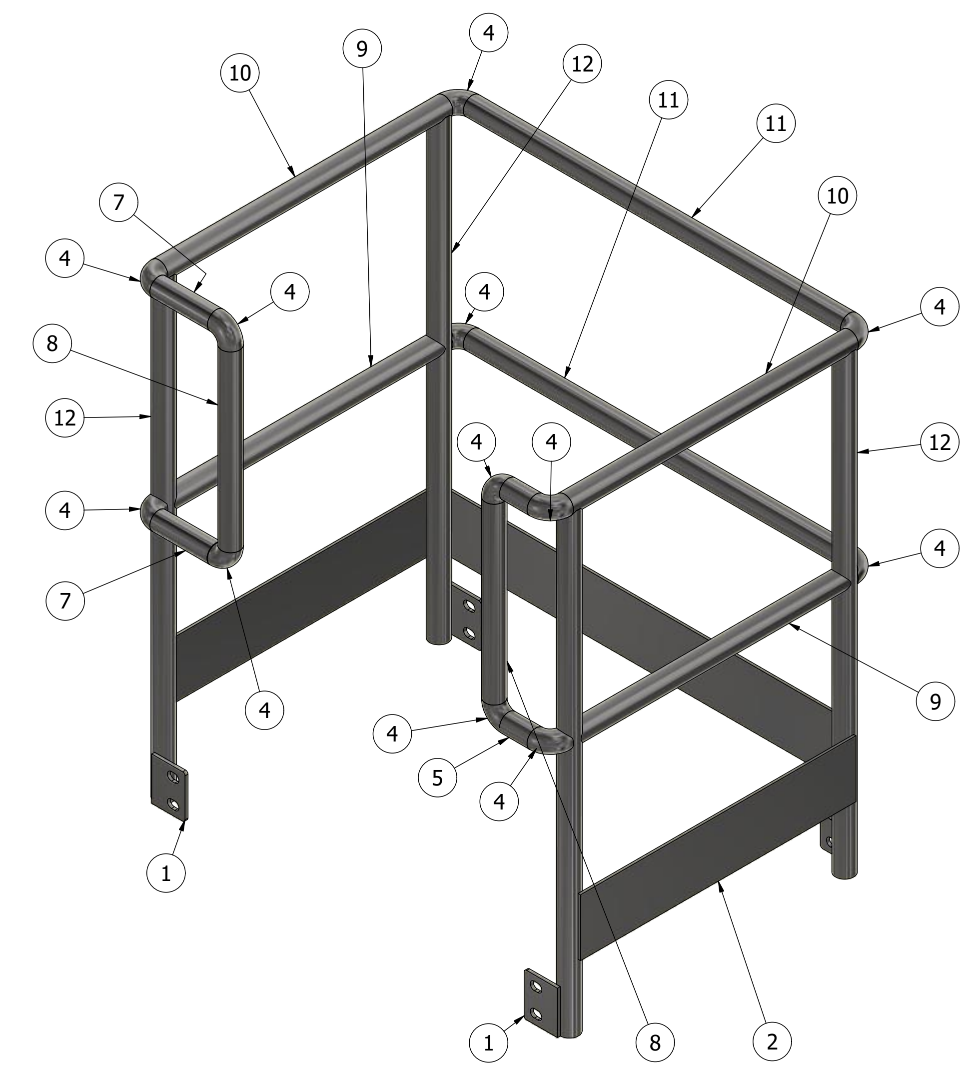




DETAIL A
SCALE 1:8







NOTES:

1.) REFER TO HANDRAIL NOTES DWG 551-S005.

2.) ADD VENTS FOR GALVANIZING

4008266-AS12

WELDMENT, GUARDRAIL 1
FABRICATION DETAIL
TOTAL WEIGHT ~169 LBS
1EA. REQ'D

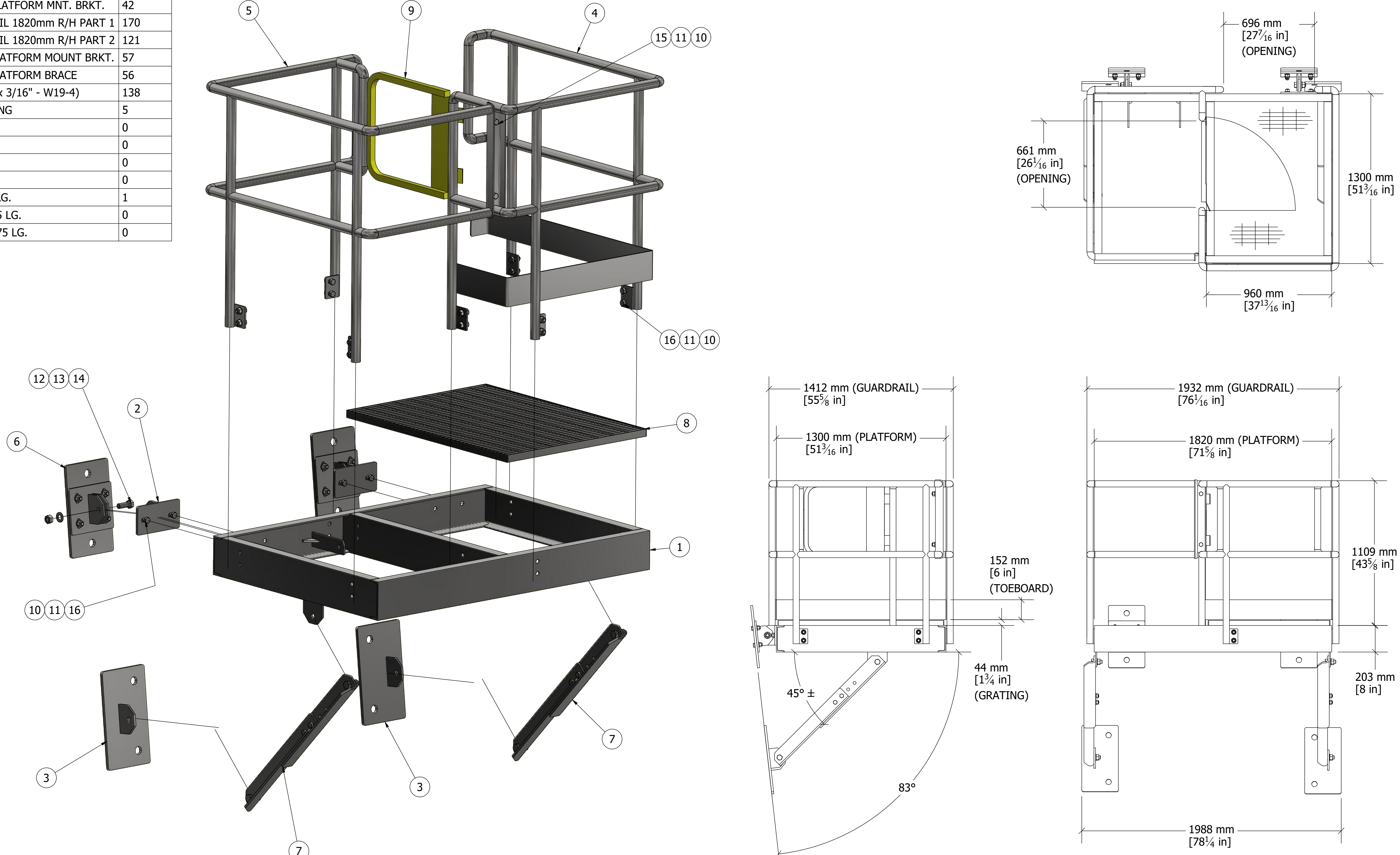
FOR CONSTRUCTION

Procon MINING & TUNNELLING LTD.
2203 - 8th St., NISKU, ALBERTA

SIZE B	DWG # 843-4008266-10	TITLE WELDMENT, GUARDRAIL 1	
DRAWN BY I.LAING	CHECKED BY	SCALE 1:15	SHEET 4 of 43
REV 0	DATE 2021-05-20		

PARTS LIST

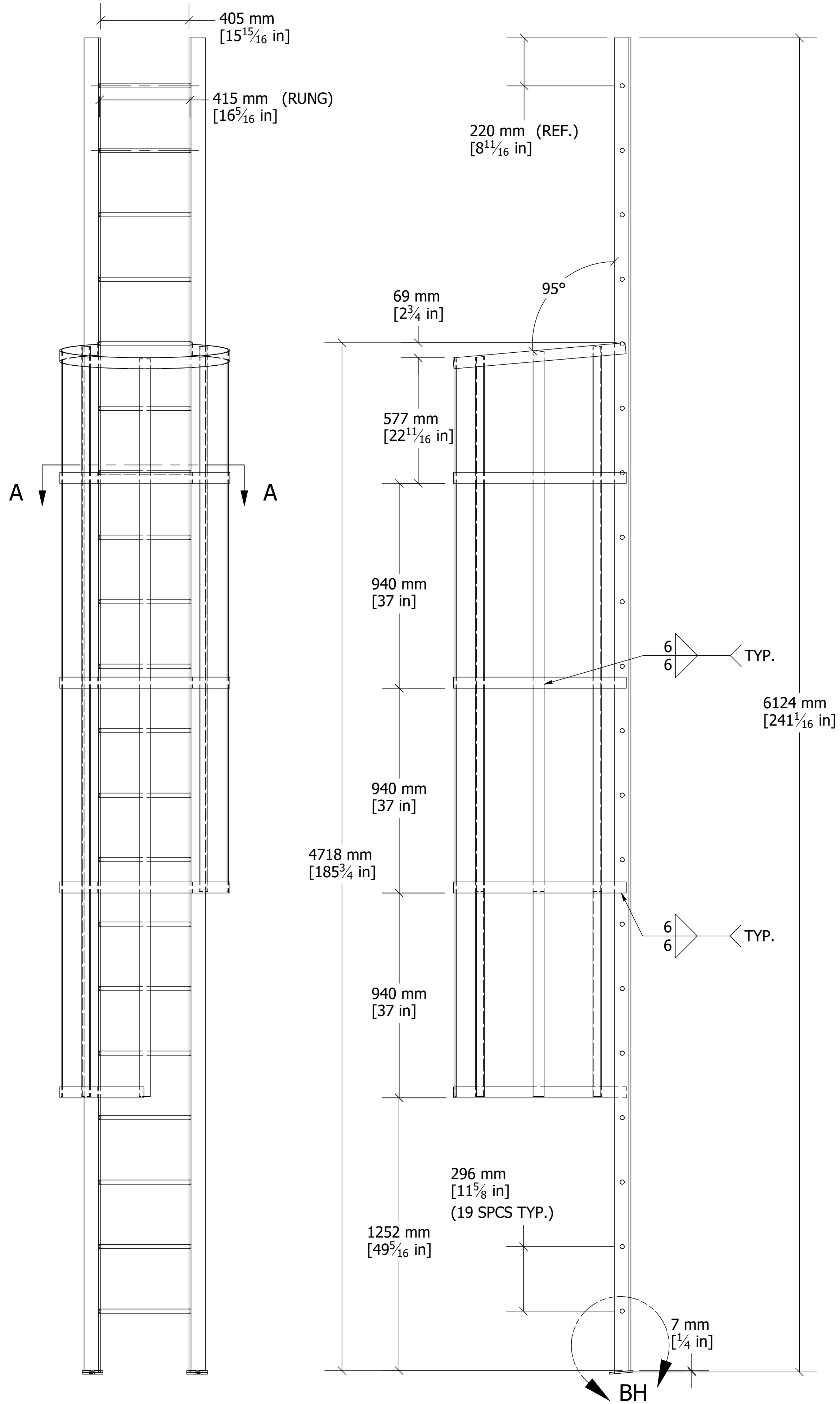
ITEM	QTY	PART NUMBER	DESCRIPTION	MASS (LBS)
1	1	4008266-AS21	WELDMENT, PLATFORM 2	342
2	2	4008266-AS22	WELDMENT, PLATFORM FRAME MNT.	12
3	2	4008266-AS23	WELDMENT, LOWER PLATFORM MNT. BRKT.	42
4	1	4008266-AS24	WELDMENT, GUARDRAIL 1820mm R/H PART 1	170
5	1	4008266-AS25	WELDMENT, GUARDRAIL 1820mm R/H PART 2	121
6	2	4008266-BB1	BOLTED ASSEMBLY, PLATFORM MOUNT BRKT.	57
7	2	4008266-BB2	BOLTED ASSEMBLY, PLATFORM BRACE	56
8	1	4008266-BG7	BAR GRATING (1-1/2" x 3/16" - W19-4)	138
9	1	LSG-24-PCY	GATE KIT, SELF-CLOSING	5
10	20		FLATWASHER, 3/4	0
11	20		HEX NUT, 3/4-10 UNC	0
12	2		FLATWASHER, 1	0
13	2		HEX NUT, 1-8 UNC	0
14	2		BOLT, 1-8 UNC - 2.75 LG.	1
15	2		BOLT, 3/4-10 UNC - 1.5 LG.	0
16	18		BOLT, 3/4-10 UNC - 1.75 LG.	0

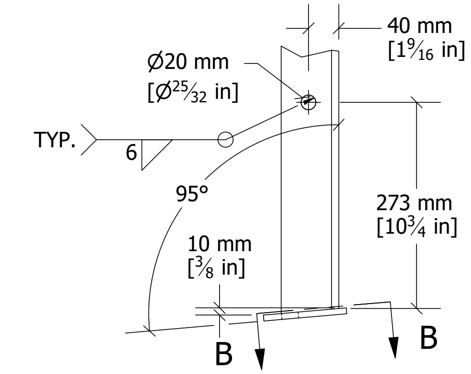


4008266-AS2 - (SHOWN)
4008266-AS3 - (OPPOSITE)

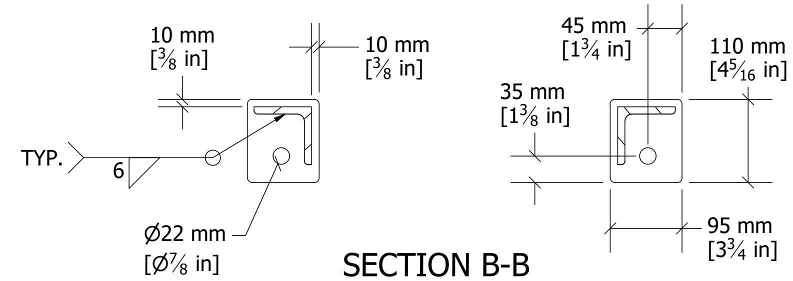
ASSEMBLY, PLATFORM 2 & 3
1EA. REQ'D PLATFORM 2
1EA. REQ'D PLATFORM 3

Procon MINING & TUNNELLING LTD. 2203 - 8th St., NISKU, ALBERTA			DRAWN BY I.LAING	CHECKED BY
			SCALE 1:30	SHEET 6 of 43
SIZE B	DWG # 843-4008266-10	TITLE ASSEMBLY, PLATFORM 2	REV 0	DATE 2021-05-20

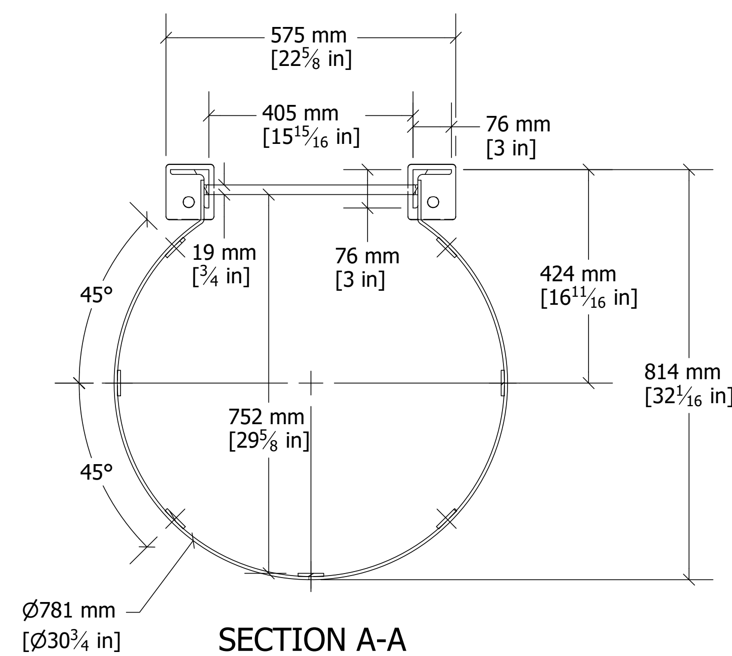




DETAIL BH
LADDER BASE
SCALE 1 / 10



SECTION B-B
SCALE 1 / 10



SECTION A-A
SCALE 1:15

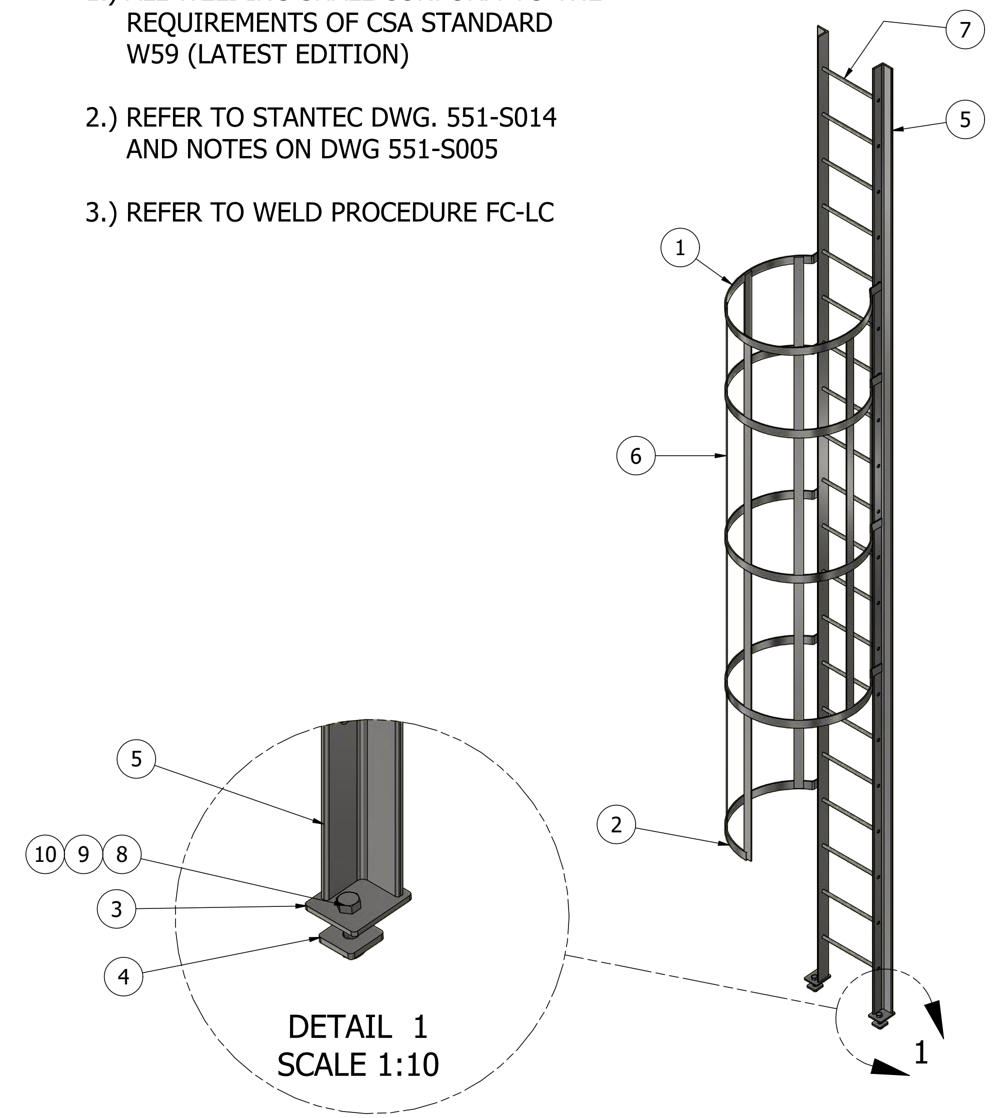
PARTS LIST - ASSEMBLY						
ITEM	ITEM QTY	TOTAL QTY (IN)	PART NUMBER	DESCRIPTION	THICKNESS	MATERIAL
1	4		4007290-PR1	FORMED BARSTOCK, LADDER HOOP	.250 in	Steel, 300W
2	1		4007290-PR2	FORMED BARSTOCK, LADDER HOOP HALF	.250 in	Steel, 300W
3	2		4008266-PL19	PLATE, LADDER FOOTPAD	.313 in	Steel, 300W
4	2		4008266-PL20	PLATE, LADDER MNT. *SHIPPED LOOSE*	.375 in	Steel, 300W
5	1	482.28	ANGLE L 3 x 3 x 3/8	ANGLE, 3" X 3" X 3/8"		Steel, 350W
6	7	830.19	FLAT 2x1/4	FLAT STOCK, 2" X 1/4"		Steel, 300W
7	20	326.77	RND ROD 3/4	LADDER RUNG - 3/4" DIA.		Steel, 300W
8	2			BOLT, 3/4" UNC X 3.25" LG.		A325
9	2			FLATWASHER, 3/4"		A325
10	2			NUT, 3/4" UNC		A325

NOTES:

1.) ALL WELDING SHALL CONFORM TO THE REQUIREMENTS OF CSA STANDARD W59 (LATEST EDITION)

2.) REFER TO STANTEC DWG. 551-S014 AND NOTES ON DWG 551-S005

3.) REFER TO WELD PROCEDURE FC-LC



DETAIL 1
SCALE 1:10

4008266-AS5 - (SHOWN L/H)
4008266-AS6 - (OPPOSITE R/H)

ASSEMBLY, LADDER UPPER

TOTAL WEIGHT ~506 LBS
1EA. REQ'D L/H, 2EA. REQ'D R/H

FOR CONSTRUCTION

Procon MINING & TUNNELLING LTD.
2203 - 8th St., NISKU, ALBERTA

SIZE B	DWG # 843-4008266-10	TITLE ASSEMBLY, LADDER UPPER L/H	DRAWN BY I.LAING / CHECKED BY
			SCALE 1:25 / SHEET 18 of 43
			REV 0 / DATE 2021-05-20

PARTS LIST - ASSEMBLY

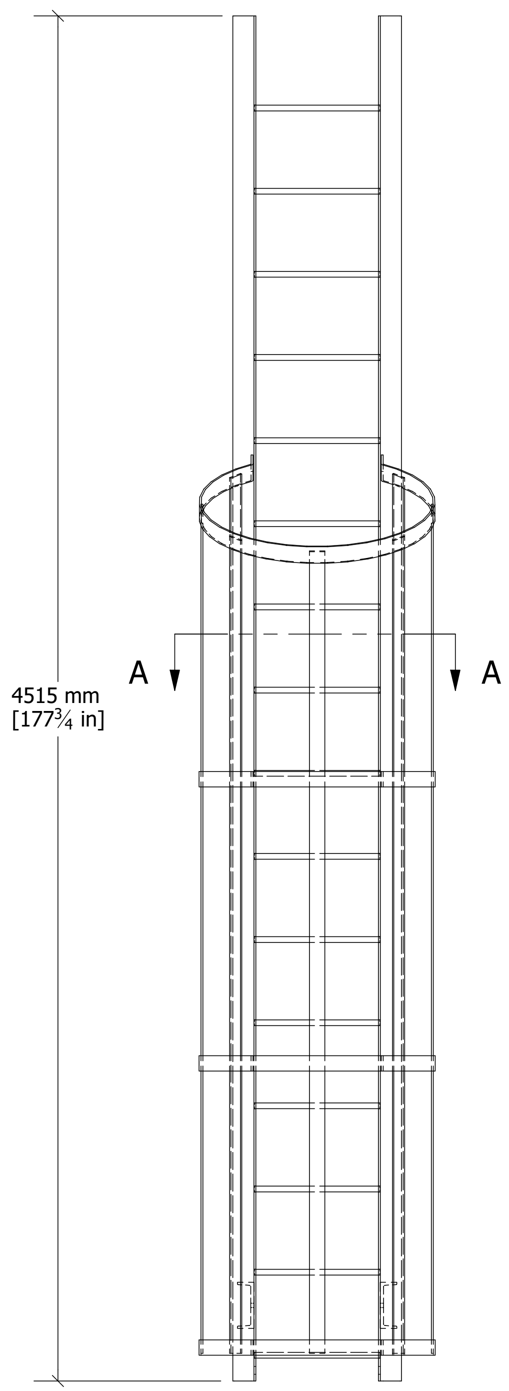
ITEM	ITEM QTY	TOTAL QTY (IN)	PART NUMBER	DESCRIPTION	THICKNESS	MATERIAL
1	3		4007290-PR1	FORMED BARSTOCK, LADDER HOOP	.250 in	Steel, 300W
2	1		4008266-PR5	FORMED PLATE, LADDER HOOP ANGLED	.250 in	Steel, 300W
3	2	7.874	4007290-CH02	CHANNEL, LADDER MNT.		Steel, 300W
4	1	355.512	ANGLE L 3 x 3 x 3/8	ANGLE, 3" X 3" X 3/8"		Steel, 350W
5	7	767.86	FLAT 2x1/4	FLAT STOCK, 2" X 1/4"		Steel, 300W
6	16	261.42	RND ROD 3/4	ROUND STOCK, 3/4"		Steel, 300W

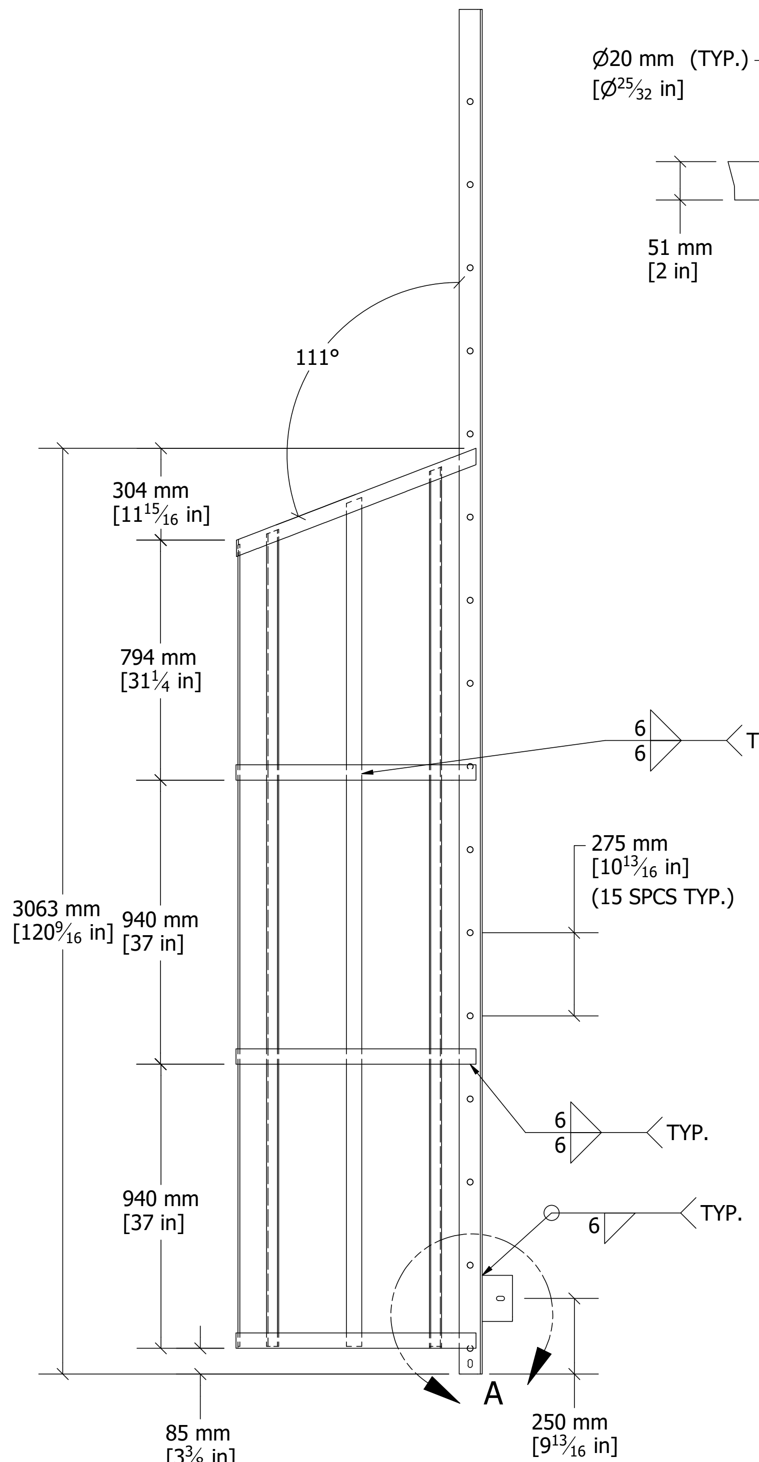
NOTES:

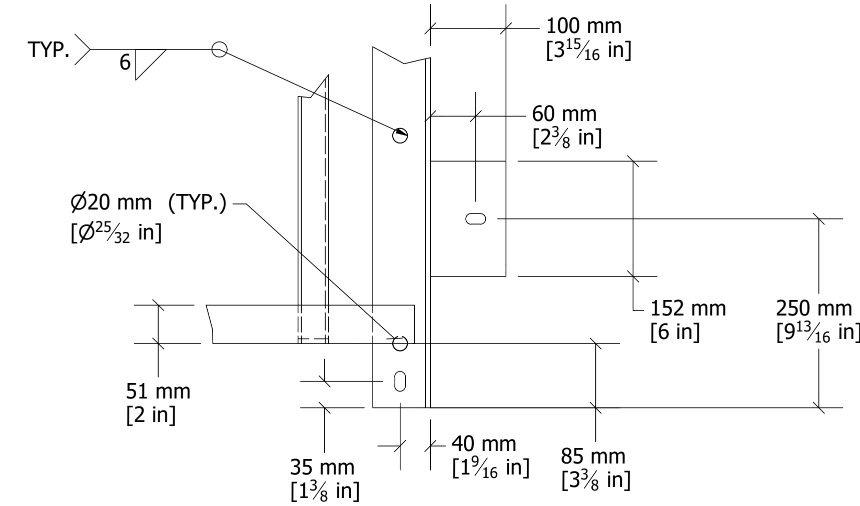
1.) ALL WELDING SHALL CONFORM TO THE REQUIREMENTS OF CSA STANDARD W59 (LATEST EDITION)

2.) REFER TO STANTEC DWG. 551-S008 AND NOTES ON DWG 551-S005

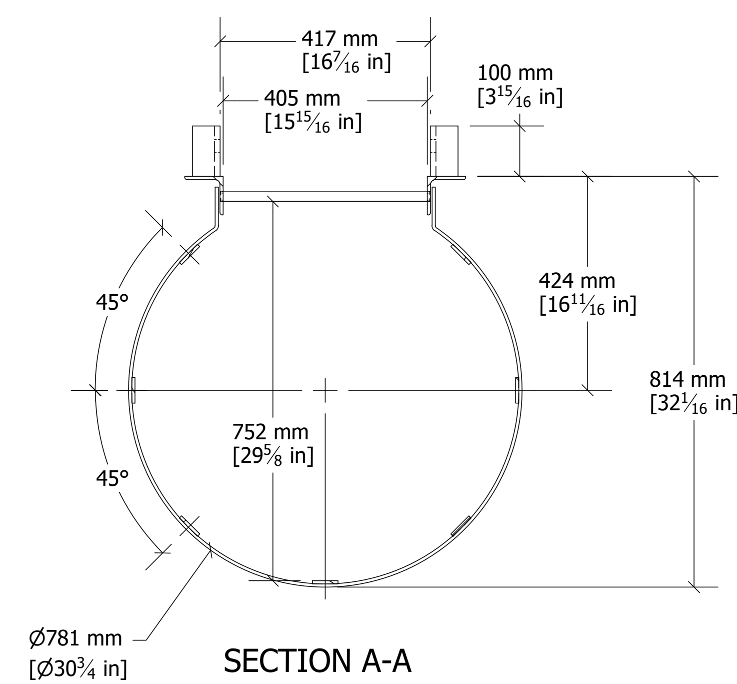
3.) REFER TO WELD PROCEDURE FC-LC



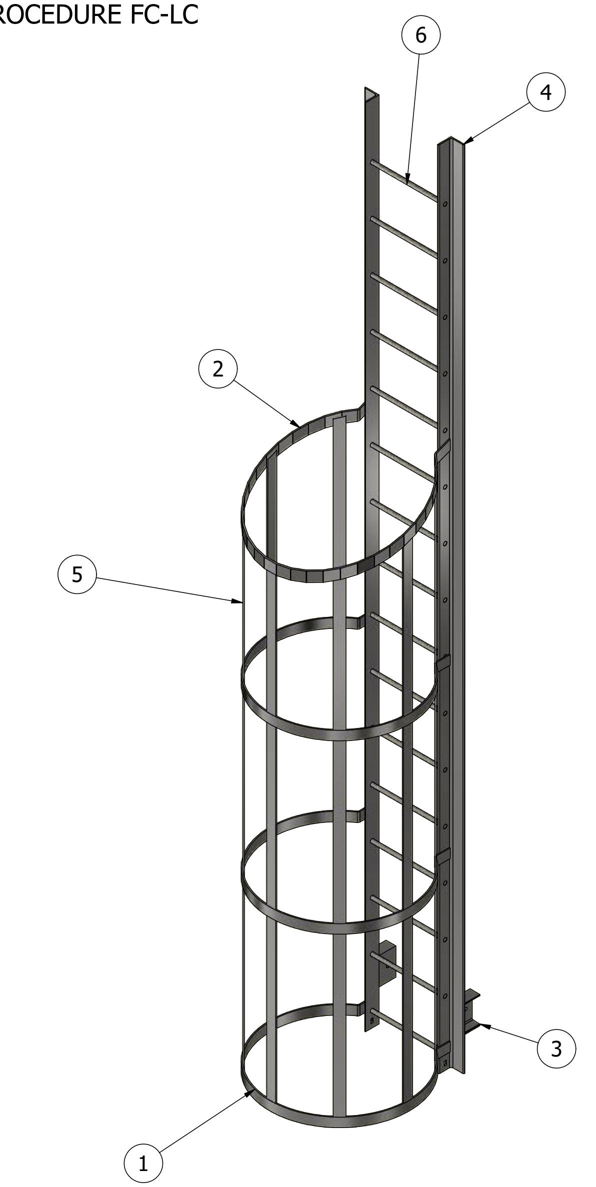




DETAIL A
SCALE 1 / 10



SECTION A-A
SCALE 1:15



4008266-AS7

ASSEMBLY, LADDER TRANSITION

TOTAL WEIGHT ~344 LBS
1EA. REQ'D

FOR CONSTRUCTION

Procon MINING & TUNNELLING LTD.
2203 - 8th St., NISKU, ALBERTA

SIZE	DWG #	TITLE	DRAWN BY	CHECKED BY
B	843-4008266-10	ASSEMBLY, LADDER TRANSITION	I.LAING	
			SCALE 1:25	SHEET 19 of 43
			REV 0	DATE 2021-05-20

APPENDIX B

KENO HILL SILVER DISTRICT, BERMINGHAM DEPOSIT

GROUND CONTROL MANAGEMENT PLAN

Keno Hill Silver District

Bermingham Deposit

GROUND CONTROL MANAGEMENT PLAN

February 9, 2021

Report No. 001-2021

Alexco Resources Keno Hill Silver District
Bermingham Deposit
Ground Control Management Plan (GCMP)

First Issue Date: February 9, 2021
Last Modification: February 9, 2021
Final Issue Date:

Version Control

Rev Number	Issue Date	Description & Location of Revision Made	Signatures		
			Originator	Checked	Approved
0	Feb. 9 2021	First Draft	W.S		
1					
2					
3					
4					
5					

GEOTECHNICAL ENGINEER AUTHORISATION

Authorized

-- ----------------------------

Woo Shin, Ph.D, P.Eng — Date

Rock Mechanics Specialist

MINE MANAGER AUTHORISATION

Authorized

-- ----------------------------

Wayne Zigarlick — Date

Mine General Manager

MINE ENGINEER AUTHORISATION

Authorized

-- ----------------------------

Neil Chambers, P.Eng — Date

Chief Mine Engineer

MINE OPERATION AUTHORISATION

Authorized

-- ----------------------------

Date

Mine Superintendent

Contents

LIST OF TABLES

LIST OF FIGURES

1. GROUND CONTROL MANAGEMENT GUIDELINE

1.1 Strategy

The focus of ground control strategy is the provision of enhanced resources for the collection and utilization of geotechnical information for integration into mine planning and design functions. This will be accomplished by:

- Providing robust geotechnical resources at Alexco Keno Hill Bermingham mine site;
- Improving and standardizing the Ground Control Management Plan (GCMP) in use at each of Alexco Keno Hill mine operations;
- Effectively implementing the GCMP at each of Alexco Keno Hill mine operations;
- Developing a structured geotechnical training program, including a ground condition awareness and risk analysis training program; and,

Encouraging a good corporate attitude towards the sponsor and funding of innovative geotechnical research and development.

An effective ground control management strategy for Alexco Keno Hill Silver's operating mine is aimed at qualifying and reducing the geotechnical 'risk' in planning and operation at these mine sites.

1.2 Scope

The scope of this plan is specific to the Alexco Keno Hill Silver Bermingham operation and is based on understanding of ground control principals and the geological, geotechnical and mining conditions that apply at the time of the current division.

The GCMP,

- Applied to all underground mine personnel, contractors and visitors who have stated duties under the GCMP;
- Takes effect from the date of issue and is not retrospective;
- Form the basis for training content and specifies requirements for training and competency under the GCMP;
- Outlines the responsibilities and roles of individuals under the GCMP;
- Specifies the Ground Support Rules, requirements for development and production;
- Details the Trigger Action Response Plan (TARP) for both the development and extraction processes; and,
- Does not address controlled or uncontrolled movement of ground resulting in subsidence of ground.

1.3 Detailed Process and Procedures

1.3.1 Mine Design Process

The design of openings, ground support, or pillars should be undertaken in a systemic manner take into general account;

Geological Factors

- Distribution of regional structure
- Distribution of rock types
- Groundwater conditions

Geotechnical Factors

- Back, floor and wall geology and geotechnical parameters
- Known or predicted geological structure and rock defects
- Rock strength parameters (uniaxial compressive strength, cohesion and friction angle)
- In-situ stress
- Expected change in stress accordance with development and extraction sequence
- Ground response from monitoring

Mining Factors

- Excavation dimensions
- Mining methods and sequencing
- Required use of excavation
- Ground support equipment and constraints
- Required life of area or excavation

1.3.2 Ground Control Process

No extraction or development shall take place unless the area has been assessed and an appropriate support system designed, documented and authorized by the Alexco Keno Hill Mine Manager. The GCMP is enacted by following the Ground Control Management Procedures, as listed below,

Table 1.1 Outline of Ground Control Management Procedure

Activities	Summary of Activity
Geotechnical Data Collection	Collection of relevant geological and geotechnical data for characterization of the ground condition.
Modeling, Analysis and Design	Use of the sound geotechnical engineering principles to design excavations (development and production) which are fit for their intended use. Where important, the sequencing of the excavations may be described and any adjustments to the proposed excavation design and/or sequence re-evaluated.

Excavation Performance Monitoring	Ensuring excavations are mined to appropriate dimensions and are properly supported. Where appropriate equipment and/or procedures should be used to monitor conditions.
Remedial Measures	Determination of appropriate, effective techniques for post failure treatment to regain control of excavations as necessary. This includes, but is not limited to, the rehabilitation of failed or old mining areas and ground support, and back analysis of failures, if appropriate.
Producing the GCMP	Incorporating the above into a clear and concise document that can be used as a guide for managing ground conditions. The document should explain the philosophy of the ground control system and list any assumptions used in the design. The plan should be able to be read by third parties to quickly gain an understanding of the principle aspects of ground control at the mine and the procedures and/or processes in place for managing these aspects.

1.4 Objectives

The objectives of the GCMP are as follows;

- Reduce the risk of uncontrolled ground failure
- Contribute to the development and maintenance of a safe working environment
- Contribute to efficient extraction of ore reserves

The objectives are achieved through;

- Identification of hazardous areas and assess associated risks
- Design and implementation of appropriate ground control system
- Communicating known hazards to the workforce in advance of both development and production
- Design and implementation of systems to detect and control change (TARP)
- Design and implementation of procedures associated with ground control including a Standard Operation Procedure (SOP) for installation or ground support
- Providing clear and unambiguous definitions of roles and responsibilities for individuals working under the plan
- Internal and external auditing to assess the effectiveness and degree of compliance with the GCMP and assist in identifying improvement requirements

1.5 Requirements

Human Resources, Equipment, Training, Materials and Systems

Mining operation should provide for sufficient resourcing to implement and maintain a ground control strategy. The key human resource needed to achieve this aim is competent full-time site based geotechnical engineer or a combination of site based personnel and external resources. Other people appointed in the roles listed in Section 3.1, also need adequate training to meet their requirements covered in the GCMP.

Aside from the basic requirement of fulfilling regulatory standards, all equipment used for ground control must be appropriate for its intended use.

Personnel performing ground control tasks must be adequately trained and deemed competent in the correct use of ground control equipment and materials. As such, the mine should provide resources to document the specifications and develop Standard Operating Procedures (SOP's) where necessary or appropriate. It is then the responsibility of the mine to train and access operations in the use of these SOPs.

Data Collection Techniques and Risk Assessment

The collection of suitable, high quality data is the basis for building a solid ground control strategy. Time should be spent determining what type of data can and should be collected for use in the efforts of ground control.

The concept of risk is an integral part of the ground control strategy, such that mitigation of risk to personnel and equipment is routinely considered. System of ground control management should be thought with the practice of assignment the highest practicable level of hazard control whenever possible.

1.6 Ground Control Definitions

Nominal: Refers to an approximate dimension of a drift utilizing the same support requirements.

Primary Support: Ground Support Anchors (GSA) used in conjunction with wire mesh. Accepted bolts of types are 1.8m and 2.4m fully grouted rebar bolts, expandable friction bolts (Swellex) or split tube friction bolts (split set). Accepted wire mesh type are either galvanized chain link mesh or welded wire panels. Welded wire mesh is the preferred type of mesh to use with shotcrete.

Secondary Support: Bolts longer than 2.4m in length. Accepted bolt types are 24 tonnes expandable friction bolts (Super Swellex or Connectable bolt) and cable bolts.

Short Term Drift: Anticipated working life of less than 2 years. In these areas, corrosion protection is not typically necessary. Regular friction bolts are acceptable for the installation. This includes uncoated Swellex bolts.

Long Term Drift: The corrosion protection is required where the working life is anticipated longer than 2 years, depending on the ground water condition. Regular friction bolt and screen may be applicable for these area in dry condition under approval from ground control engineer. Regular friction bolts for long term drift in wet condition should re-bolt under inspection by ground control engineer within 2 years after installation following full-out test results. Installation and quality control program of uncoated bolts for long term drift need to be reviewed and approved by Geotechnical Engineer or designated Engineer under Chief Mine Engineer's supervision.

2. DEVELOPMENT OF GROUND CONTROL MANAGEMENT PLAN

2.1 Development of Formal GCMP

Keno Hill operations, which include Bermingham shall conduct site based risk assessments to support the development of GCMP and related activities. Risk assessments generally includes but are not limited to the following key consideration:

- Geotechnical assessment and monitoring
- Ground stability, surface subsidence and potential in-rush (air, mud, bodies of water, etc.)
- Material and equipment selection criteria
- Identification of required standard operation or work procedures (based on consequences)
- Workforce training and competency levels
- Mining methods and operations planning criteria (including excavation size and sequence)
- Significant changes in opening plans or ground conditions
- Ground condition monitoring methods (focused on earliest possible detection)
- Emergency response planning.

2.2 Content of the GCMP

Generally, the following information may be included within the GCMP:

- A process of technical mine planning
- Technical competency requirements of personnel and resources involved in the management of ground control (including inspection) and analysis of technical data
- The technical data utilized in modeling, design, excavation and support methods
- Procedures to allow person to work in conditions where the hazards have been identified, formally assessed and controlled, standard operating procedures for work in such areas have been produced

- Methods, materials certification, and quality criteria for stability enhancement such as rock fixtures, plates, backfill, barricades, shotcrete, cribbing, wire mesh, etc.
- Corrective action for removal of loose or unconsolidated materials
- The ongoing inspection processes for ground control conditions which specify corrective action and emergency procedures. Rock mass conditions should be monitored for all departures from normal
- Specifications of monitoring equipment for type, location and frequency of data collection and review.

2.3 Consistent of Systemic Approach

The GCMP presents a systematic approach to allow the reader to understand the important aspects of ground control for the mine. Factual information should be clearly separable from any inferred or analytical judgements proposed in the document. These should be a logical flow from data collection, analysis and design to monitoring and back analysis work.

Given that rock is a dynamic material and mining is a dynamic process, the geotechnical engineer must usually make a general assumption about the property of a given rock mass for design purpose at a given time in the mining process. Reliance is then placed on an “observational approach” to monitor the effectiveness of the design and the appropriateness of the design assumptions. The concept of the observational approach was first described by Terzaghi and Peck in 1967 and can be outlined as follows:

- Decide on some sort of initial mine layout
- Begin mining
- Monitor the rock mass response – normally visual or by monitoring equipment
- Redesign based on the observed field conditions – model calibration.

Using this iterative process, the geotechnical engineer builds a case for the reliability of their assumptions over time and in doing so becomes more confident in predicting better safety and accuracy of technical design.

Detailed ground control management system and underground mine and mine planning system based on the concept suggested Terzaghi and Peck (1967) are shown in Figure 2.1 and 2.2

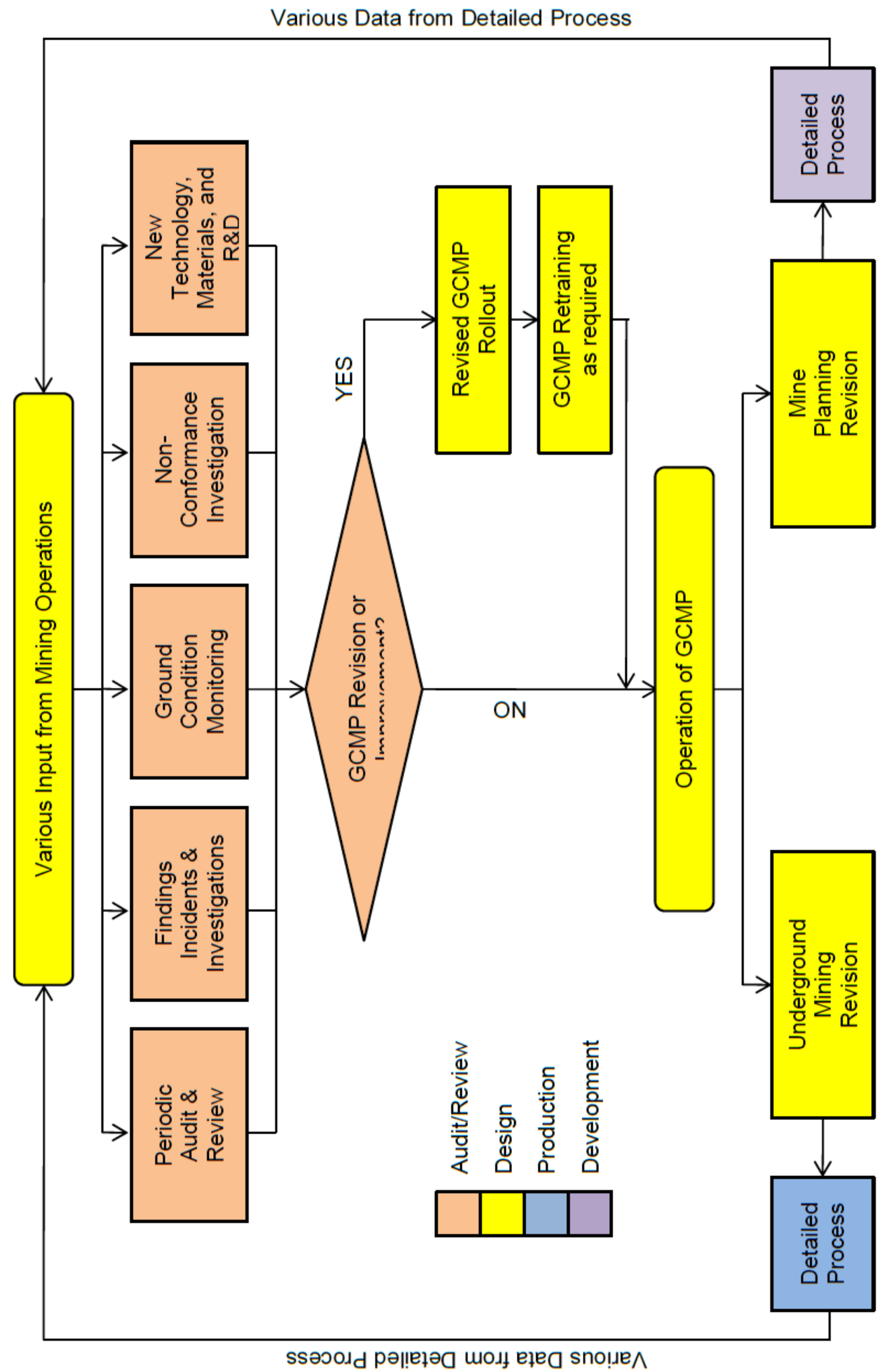


Figure 2.1. Ground Control Management System

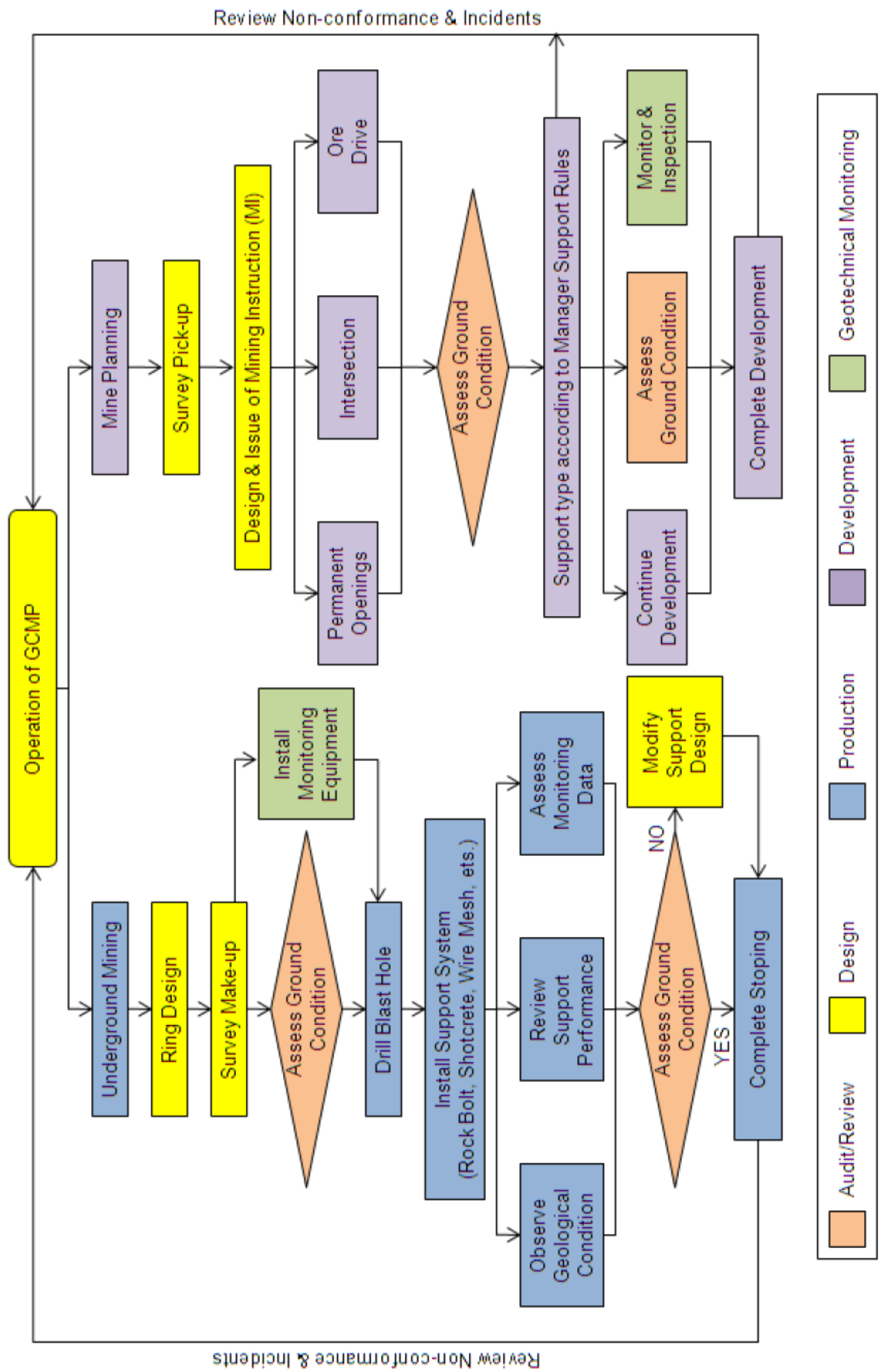


Figure 2.2. Detailed Underground Mining and Mine Planning system

2.4 Risk Assessment

The main focus of the Keno Hill Bermingham Mine Ground Control Management Plan is to facilitate early recognition and timely control of ground control hazards by the underground workforce. It is recognized that not all hazards are predictable and accurately defined in advance of mining by such methods as exploration, geological evaluation and therefore the GCMP must remain responsive to ground conditions and mining variations to reduce the risks to an acceptable level.

2.4.1 Hazard Identification

The key hazard associated with underground development in regard to ground control is rock fall due to;

- Geological structure
- Over-excavation
- Groundwater
- Ground movement
- Stress change
- Drill and blast techniques

Geological Structure

Geological structures include normal faults, strike slip faults and folds. These can have an adverse impact on conditions primary through weakening the rock mass conditions and creating unstable wedges in the back and walls.

Over-ecavation

Increasing the span or heights over the specified dimensions can have an adverse impact because;

- The capacity of the ground to support itself may be exceeded
- By increasing the size of the potential wedge over the capacity of the ground support elements.

Groundwater

Ground water in the general back or walls can have an adverse impact on ground control. Water can weaken the immediate ground or reduce the integrity of ground support, particularly cement based support element such as shotcrete and grout. It can have a lubricating effect on slip and joints.

Water can be from;

- Natural source along with discontinuity
- Exploration drill holes

Ground Movement

Ground movement is a result of post mining relaxation or change in local conditions. Ground movement can be monitored for underground mine with various instruments, from relatively simple disto-meter and Ground Movement Monitor (GMM) to multi-point extensometers. Change in rate of movement may mean that the primary or secondary support design may need to be supplemented or access to that area restricted.

Stress Change

Changes in ground stress can lead to loading ground support and possible failure. At Keno Hill underground this is not likely to occur around all underground openings including main ramp, production drift and long hole stope access drift areas but may become apparent in development at depth. Indicators of stress may include flattering or buckling or rock bolt plates, straining of cable plates, bird caging of secondary support tendons, spalling of shotcrete and unusual popping sound caused by rock burst.

Unusual roof noise: audible cracking, squeaking or “banging” observed in the backs or walls generally indicate that the ground is “working”. This is a sign of ground instability which can lead to loss of control and ground failure. To date this has not been reported at Keno Hill underground. Because this noises associate with major faults, immediate notice by miners and special remedial action were required for this case.

Drill and Blast Techniques

Drill and blast is the one major variable that can be controlled. Ground control can be enhanced by ensuring that drilling is to design and the appropriate explosives and numbers are used when firing development headings. Drill and blasting techniques should limit collateral damage to host rock surrounding the excavation.

2.4.2 Likelihood and Consequence of Occurrence of the Risk

The likelihood of occurrence can be based on both past experiences and judgements; it must be clearly stated which,

In some circumstances the likelihood of a potential failure may be quantified from failure record in Keno Hill Ground Control Risk Assessment Report (Appendix-A). The report should be used to record all back and/or wall failures that occurred in any supported ground. A failure that requires an Incident Report shall be recorded in the Keno Hill Mine Incident Investigation Report.

2.4.3 Risk Assessment Report

The risk associated with ground related and other identified hazards are estimated by considering the “Consequence, Exposure and Probability of the Hazard”. During the daily and weekly meetings risk shall be reviewed and if required highlighted so that appropriate action can be taken.

2.4.4 Trigger Action Response Plan

The aim of a Triger Action Response Plan (TARP) is to ensure a response to changed ground conditions at an early stage. The TARP for use in Keno Hill Bermingham mine is shown in Appendix-B. From the empirical guideline and numerical study, 11 different types of ground supporting regimes are recommended for the Keno Hill BM UG mine depending on ground condition, life time of openings and development geometry conditions. The TARP provides a list of indicators, observable at operator level that can be used to guide the selection of the appropriate Support Type as defined by the Ground Support Standards (see Section 5.4)

The key indicators are;

- Rock qualification (GSI)
- Contact orientations between FW/HW of Fault zone
- Presence and condition of the geotechnical structures

In addition of the Geotechnical Engineer may dictate extra support based on geotechnical monitoring or visual inspections.

The Geotechnical Engineer or Supervisor will conduct an inspection of the area in the event the ground Support Type is changed.

2.5 Ground Support Installation Guidelines

The designed support shall be installed to established standard Bermingham mine operating procedures and as outlined in Keno Hill Ground Support Rules and TARP's.

Operators shall observe the ground conditions and monitor effectiveness of ground support installation (e.g. drilling rates, water loss/gain, bolting problems, voids etc) and report any unusual conditions and action the TARP's. The operators shall only use approved (UG Mine Manager or Supervisor) installation equipment and support hardware.

The requirements for ground support installation are listed below;

2.5.1 Split Tube Friction Bolt

- Bit gauge is critical for this type of bolts, and the hole size should be monitored to maintain the inside-diameter (ID) between 35 to 38 mm. Holes should be drilled 150 mm longer than the bolt to ensure pressure on the plate when installed.
- If split sets are the primary ground support in an area, secondary support will also may be required if the excavation is a long-term excavation under inspection by geotechnical engineer.
- The use of drive-time tests is also useful.

- Pull tests should be conducted routinely and can be done in the current headings. It should be ensured that a large portion of the testing be undertaken in the excavation back, as this area has the highest risk of poor support performance related failure.
- Report all split set defects to the technical department so that they can follow up with manufacturer on quality control.

2.5.2 Expendable Friction Bolt – Standard and Super

- Bit gauge is less important for optimum performance. Use a 32 mm to 43 mm bit with standard bolt (12 ton) installation and a 43 mm to 52 mm bit for super bolt (24 ton). Pull testing has demonstrated that 38 mm diameter bits give optimum anchorage for standard bolt. Undersized or oversized bits will reduce the anchorage capacity of friction bolts.
- It is important that the bolt is pressurized to the recommended 300 bar. Using a pressure less than the recommended value will reduce the anchorage capacity.
- It is important that the bolt is held at the 300 bar of pressure for a full 6 seconds as per the manufacturer's recommendation. Failure to hold the pressure for this length of time could reduce the anchorage of the bolt. This is a function of the bolt and not the pump and so the guideline should be followed regardless of the pump power.
- Re-pressurizing a friction bolt can give an indication if the bolt has been damaged in the cases where the damage would cause it to leak and not hold pressure (for example the bolt was sheared off).
- Pull tests should be conducted routinely.
- Report all friction bolt defects to the technical department so that they can follow up with manufacturer on quality control.

2.5.3 Cable Bolt

- Cable bolt should have an interrupted lay at approximately 0.6 m to 0.9 m centers to provide anchorage along the length of the bolt. Garford pattern cables are normally used.
- Cable bolts should be fully grouted with cement grout using a grout tube and bleeder tube. The grout tube (20 mm) always terminates at the lowest point on the bolt (the collar in an up hole and the toe of the hole for a hole drilled angled down). Bleeder tubes (10 -12 mm) always terminate at the highest point on a bolt (the toe on an up hole or the collar for a hole drilled angled down). Tubes should never extend past the fish hook anchor of the cable to prevent them from being bent back and kinked.
- The quality of the grout is important to the effectiveness of cable bolt support. The water/cement ratio should be approximately 0.35 to 0.4. Grout that is too thin will reduce bolt strength. Grout that is too thick will make proper grouting difficult.

- The use of grout additives, such as BASF's "Flow Cable", should be considered in order to optimize the grout' characteristics.
- Bolts are grouted until return of grout is achieved from the bleeder tube. Both tubes are pinched off to keep them grout filled. Empty bleeder tubes create voids along the cable and reduce its anchorage.
- If plates are used they should be installed no sooner than 48 hours after grouting, to ensure that the grout has had time to cure.

2.5.4 Rebar Bolt

- Two types of resin capsules have been recommended: the first is of a quick set type that will set within 30 seconds after mixing. The second type is a slower set that activates after about two to five minutes. The fast set capsules are to be installed at the end of the hole followed by the slower set cartridges. This practice allows the bolts to be pre-tensioned prior to the setting of the full column resin (with approximately 1 ton of pre-tension for each 20 to 25 foot-pounds of torque). This process clamps the rock mass together and then secures the bolt.
- Mixing instructions should be adhered to, paying special attention to the number of revolutions (of the jackleg or stopper) while the bolt is being spun for mixing, and the hole-gauge.
- Resin needs to be within its shelf life (the expiration date is marked on the end of the box) and in good condition. Damaged or stale resin must be disposed of, unused, in an environmentally appropriate site-specific manner.
- Pull testing should be done on a regular basis. If testing indicates that the bolts have not been installed properly, then they should be individually checked and new bolts installed in the immediate vicinity to replace them. If subsequent testing shows that the bolt installation remains sub-standard, the issue needs to be escalated, and the miner's supervisor needs to become involved with corrective action. All testing needs to be documented.
- No more than 100 mm of thread (sometimes referred to as the tail) should be allowed to stick out of the hole beyond the collar. If the tail is longer than this, then new bolt should be installed immediately adjacent to the bolt in question.

2.5.5 Wire Mesh

- Mesh must be 100mm x 100mm welded mesh.
- Mesh may be pinned with friction bolts, but all other bolts must be the prescribed type and at correct bolt spacing and ring spacing.

- Adjacent sheets of mesh must overlap by 3 squares with the bolt pinning them together in the middle (second) row of overlap.
- Always advance your wire from supported ground. When working with jackleg do not drill holes beyond the next row of bolts to be installed – the “one hole, one bolt, policy”.
- As far as practicable once installed mesh must be pushed to fit shape of the excavation to guard against voids forming behind the shotcrete once it is applied.

2.5.6 Shotcrete

- All Headings are to be hydro scaled prior to shotcrete application to ensure any loose material is washed away and to remove excess dust, both of which contribute to shotcrete fallouts.
- All shotcrete applied to headings will be as per the prescribed mix design.
- Shotcrete thicknesses must be comply with the relevant Ground Support Type currently applicable to that specific heading;
- All headings are considered non-entry for a period of 1 hour after shotcreting to allow the shotcrete to achieve 1MPa, which is the industry standard for shotcrete re-entry strengths;
- Where mesh is not applied fiber reinforced shotcrete as per the prescribed mix design will be used;
- Where shotcrete is unavailable for any reason all development shall use mesh for the relevant Ground Support Type.
- Where ground conditions dictate fiber reinforced shotcrete will be applied before installing mesh with shotcrete then being sprayed over the mesh.

3. ROLES AND RESPONSIBILITY

The Mine General Manager or designated personnel has the overall responsibility for implementation, review and revision of the GCMP and is the only official who may authorize the GCMP, its review and revisions.

The Ken Hill Underground Mine technical team, in conjunction with operation staff, will determine the appropriate levels of development support, monitoring and hazard response for all headings and stopes.

3.1 Ground Control Management Responsibilities

Relevant personnel (employees, staff, contractors and visitors) entering Keno Hill Silver District Operation should be made aware of and take note of their responsibilities under the Keno Hill Underground GCMP, relevant regulations and implied duty of care.

The Keno Hill mine GCMP defines the specific responsibilities of key personnel in terms of the Bermingham underground mining process.

Mine Manager / Chief Engineer

- Ensure the requirements of the GCMP are compiled with
- Shall approve and sign all Managers Support Rules
- Shall oversee and drive the GCMP and ensure the GCMP and TARP are audited annually
- Appoint and ensure that the necessary resources are provided to manage the GCMP
- Ensure budgets are sufficient to provide for adequate geological/geotechnical understanding of the mining environment
- Provide guidance and input as required

Mine Superintendent

- Ensure the requirements of the GCMP are compiled with
- Ensure sufficient materials are on site to implement the Ground Support Rules
- Ensure clear communication of the GCMP to all Cementation contracting personnel
- Shall communicate operational deficiencies and improvements in the GCMP to relevant technical support personnel
- Ensure channels of communication are open for the operators to make suggestions regarding the GCMP
- Provide guidance and input ground support as required

Mine/Geotechnical Engineer

- Ensure that GCMP is taken into account in mine design
- Arrange the annual internal and external auditing of the GCMP
- Provide guidance and input to ground control as required
- Responsible for ground support in the mine
- Provide geotechnical input into the ground control management process at Bermingham Mine

- Undertake regular inspections of their work areas, specifically back and wall support, making reports of any non-conformance or deterioration
- Facilitate the design of the various Support Types, in terms of Ground Support Rules
- Ensure that required testing of support performance is carried out
- Manage the installation, reading and interpretation of monitoring equipment and ensure findings are communicated to management in a timely manner
- Ensure ongoing monitoring occurs of the ground control and geotechnical/geological environmental
- Determine and communicate trigger levels and TARP

Geologist

- Shall gather data and information, in so far as it relate to geological and geotechnical parameters and record that information in face mapping, line mapping and database
- Report areas of concern to the Geotechnical Engineer, Supervisor or other relevant staff
- Provide advice on any geological issues as they relate to ground support
- Shall ensure that the geological model is updated and ensure that the geology and structure indicated on the plans is correct

Shift Boss/Supervisor

- Ensure that those people under their charge who have responsibilities under the GCMP understand and perform those duties
- Contribute to the design and implementation of the various Support Types
- Communicate minutes and outcomes of all meetings to all mining crews
- Undertake inspections of the backs and walls or the mine and ground support
- Ensure crews are reporting all unusual visual observations, ground noise or ground (control) related events on their plods or end of shift reports
- Ensure that the appropriate changes in support hardware are made in accordance with the Underground Inspection Memo, TARP's and other instructions
- Quality control: ensure Shift Supervisors and Operators are aware of and conduct necessary QC checks on installed ground support.

Operators

- Develop headings and install support in accordance with the Ground Support rules

- Verbally report any changes or anomalies in ground conditions or support behavior to the Shift Boss / Supervisors
- Install monitoring tools as instructed
- Quality Control: ensure the necessary QC checks on installed ground support are conducted in a timely manner

Geotechnical Consultant

- Provide advice on any geotechnical issues raised by the Mine Manager, Chief Engineer, Mine/Geotechnical Engineer or other technical support team
- Periodically review and manage change / update of the GCMP

3.2 Other Key Personnel

Mine Surveyor

- Shall report to the Mine Engineer, Shift Bass/Supervisor and Mine/Geotechnical Engineer any development or intersection that exceeds design dimensions
- Survey the locations of all types of monitoring instruments and boreholes drilled through the mine and record

Safety and Training Officer

- Assist with the development of training modules that address the GCMP in conjunction with the Geotechnical Engineer
- Develop and maintain a comprehensive training and assessment plan and maintain records of any training and assessment conducted in compliance with the GCMP

3.3 Temporary Delegation of Responsibilities

The Keno Hill mine system of mining on 24 hours per day, 7 days a week basis (with personnel requiring rostered time off), requires particular attention when considering available personnel. Where staffs are absent or unavailable, it is the responsibility of individuals to provide clear and unambiguous delegation of their authority to appropriate proxy. Such delegation should be made in writing (including e-mail) and will include details of;

- Contact details for the proxy
- Duration of delegation
- Any potential limitations of duty with respect to the proxy
- Resource authorization of the proxy
- Any specific instructions to the proxy

4. GEOTECHNICAL DATA ASSESSMENT

4.1 Geological Domains and Structural Feature Sets

The Bermingham deposit is a narrow vein divided by three ore zones. The ore zones in this deposit are NE zone, Bear zone and Artic zone from north east to south west as shown in Figure 4.1. Five major faults were also observed within this deposit and average dip/dip direction for each fault zone are summarized in Table 4.1. NE zone is located underneath Ruby fault. Bear zone and Artic zone are located between Ruby fault and Cross fault, Cross fault and Cross fault and Mastiff fault. Super fault intercept upper part of Artic zone and Hangingwall fault is just to the south of all mineralization but does not intercept the ore veins within the Bermingham deposit.

The BM strat-views in HW and FW looking NE and SE respectively were also provided by Alexco and shown in Figure 4.2 Overall ground condition and rock formation of BM deposit is similar with other deposits in Kino Hill district and rock mass properties for BM deposit is assumed same as FM deposit due to lack of geotechnical logging and test results.

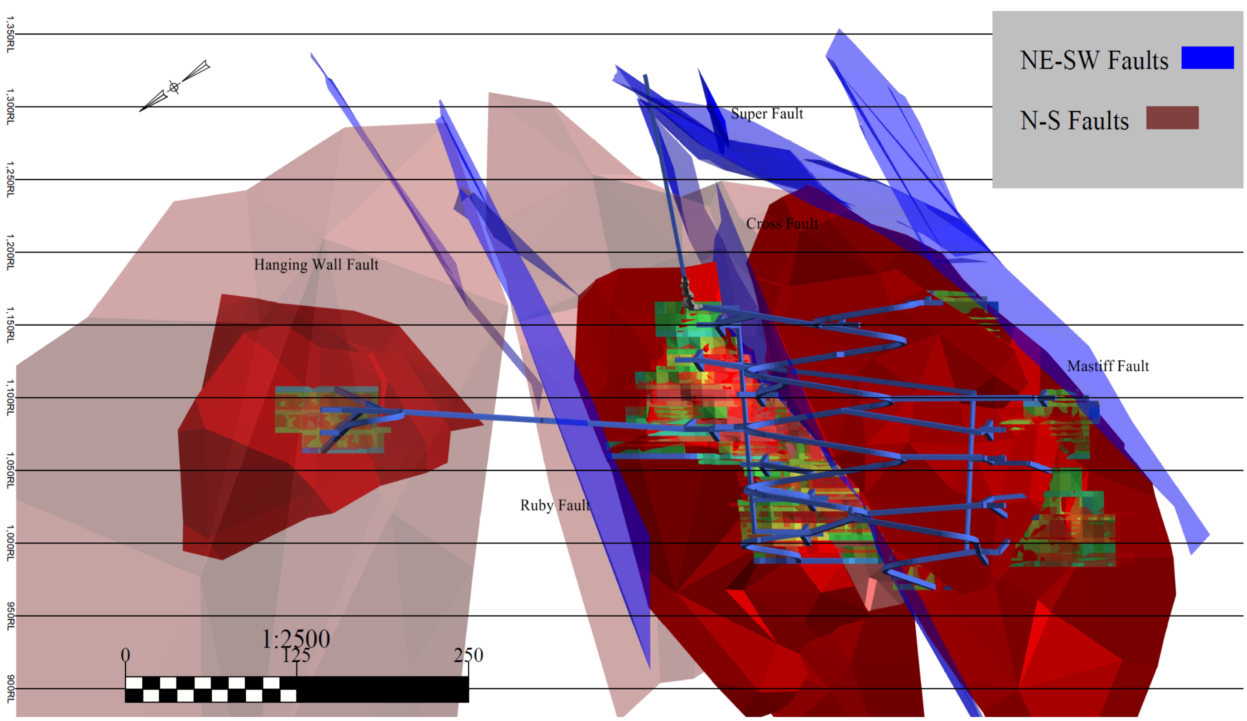


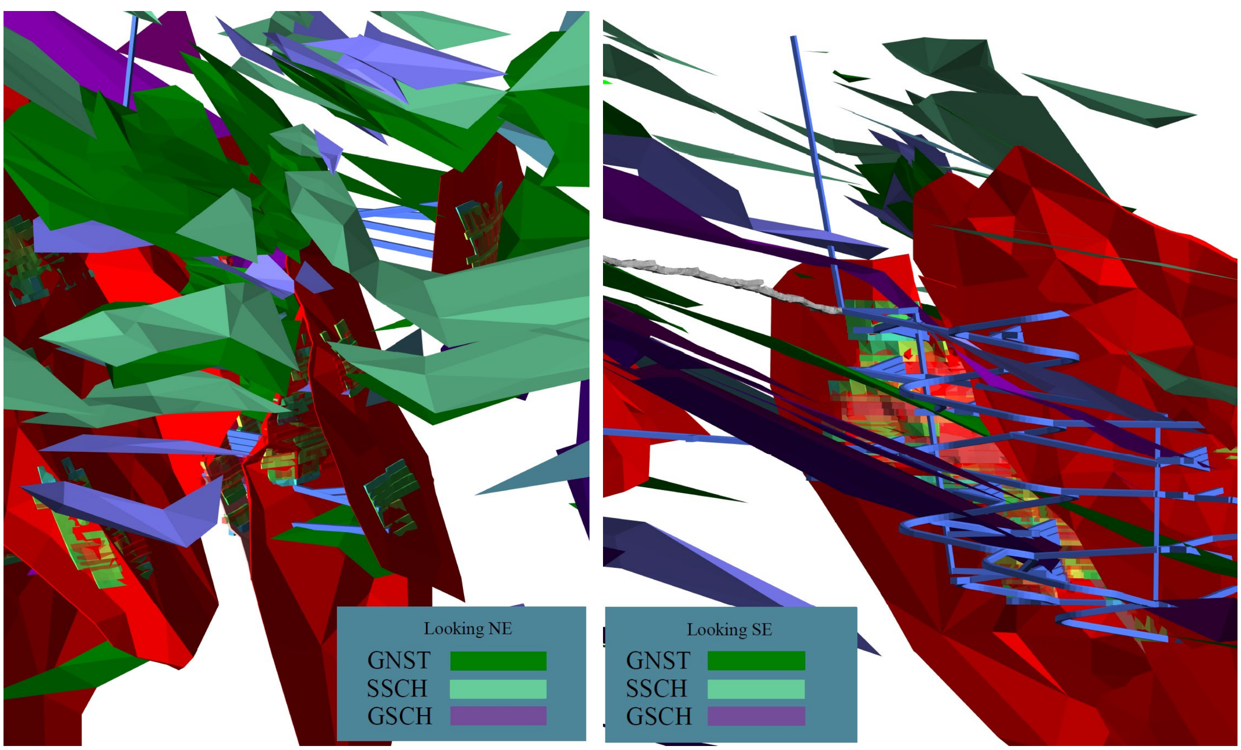
Figure 4.1 Longitudinal section view (looking south-east) of Bermingham

Table 4.1 Average Strike and Dip/Dip Direction of major faults in Bermingham deposit

Fault ID	Avg. Strike	Avg. Dip	Avg. Dip Dir.
Artic Fault	120	-60	210
Mastiff Fault	131	-52	221
Ruby B Fault	124	-68	214
Hangingwall Fault	0	-60	90
Super Fault	133	-25	223

To understand the ground conditions at the Keno Hill BM mine, geological domains were identified for each deposit. Preliminary geotechnical parameters were assessed using major lithology units as identified by Alexco geology team. Geotechnical domains are outlined below on which geotechnical designs have been based:

- Quartzite domain – waste development
- Schist domain – waste development
- Faults domain – waste and production development
- Ore Vein domain – production development



(a) HW (b) FW

Figure 4.2 Strat-vies in HW and FW

4.2 Rock Mass Classification

Rock mass classification was conducted using the Norwegian Geotechnical Institute's tunneling quality index (the NGI Q-system), as proposed by Barton et al. (1974), where Q value is determined from the following relationship,

$$Q = \frac{RQD}{J_n} \cdot \frac{J_r}{J_a} \cdot \frac{J_w}{SRF}$$

Where,

RQD: Rock Quality Designation

Jn: Joint set number

Jr: Joint roughness number

Ja: Joint alteration number

Jw: Joint water reduction factor

SRF: Stress Reduction Factor

Table 4.2 Rock Quality Categories by Q-System (Barton et al, 1974)

Q	< 0.1	0.1 - 1	1 - 4	4 - 10	10 - 40	40 - 100	100 <
Description	Extremely Poor	Very Poor	Poor	Fair	Good	Very Good	Extremely Good

Table 4.3 Bermingham rock mass classification by SRK

Domain	RQD (%)	IRS (MPa)	RMR	**Q**
Quartzite	70 - 90	90 – 150	55 – 65	**3.4 – 10.3**
Schist	50 – 90	20 – 50	40 – 55	**0.6 – 3.4**
Fault	30 - 60	20 - 40	30 - 45	**0.2 – 1.1**

SRK (2016) determined Q value using data collected from drill core at Flame and Moth, correlated to the condition of drill core and underground observations from previous Bellekeno mine. The final rock mass classifications have been engineered based on the anticipated ground conditions and SRK recommended the geotechnical parameters based on rock classification for each domain should be reviewed and adjusted during initial mining to optimize design, and to reflect the actual ground conditions encountered. Table 4.3 presents the estimated rock mass classification by SRK.

Jacobs also developed basic descriptive statistics and histograms for each geotechnical domain to better understand the statistical variability and character within each data set. This information was used to identify representative values for each Q input value. In the case of Jw and SRF, site experience, assumed far-field stress conditions, and typical depth of mining were applied. The Q value estimations for domains in Bermingham are summarized in Table 4.4

Table 4.4 Summary of NGI Q value for Bermingham deposit

Input	Quartzite		Schist		Faults		Ore Vein	
RQD	Mean (drill core)	40	Mean (drill core)	25	Mean (drill core)	20	Mean (drill core)	50
Jn	2 Joint sets	4	2 Joint sets	4	2 Joint sets	4	2 Joint sets	4
Jr	Undulating, smooth	2	Undulating, smooth	2	Undulating, smooth	2	Undulating, smooth	2
Ja	Non-softening, fine	3	Non-softening, fine	3	Non-softening, medium	3	Non-softening, medium	3
Jw	Dry (minor inflow)	1	Wet (drips/rain)	0.7	Wet (drips/rain)	0.7	Dry (minor inflow)	1
SRF	Low stress	2.5	Low stress	2.5	Low stress	2.5	Low stress	2.5
Q		**2.7**		**1.3**		**0.9**		**3.3**

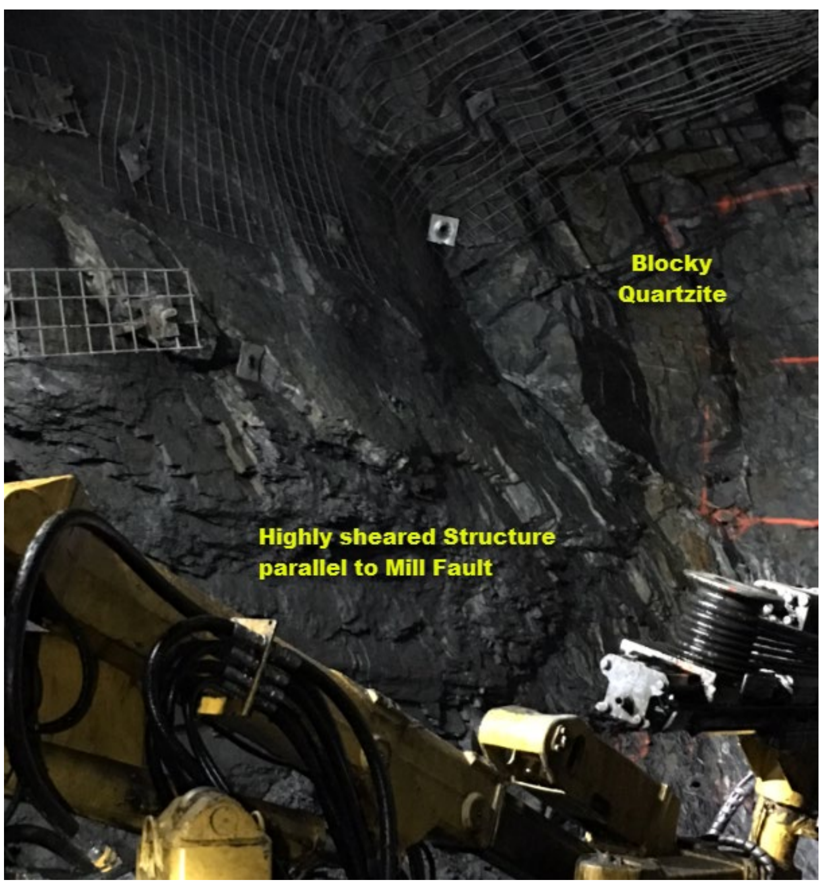


Figure 4.3 Ground condition of first remuck drift in main decline ramp from Flame and Moth

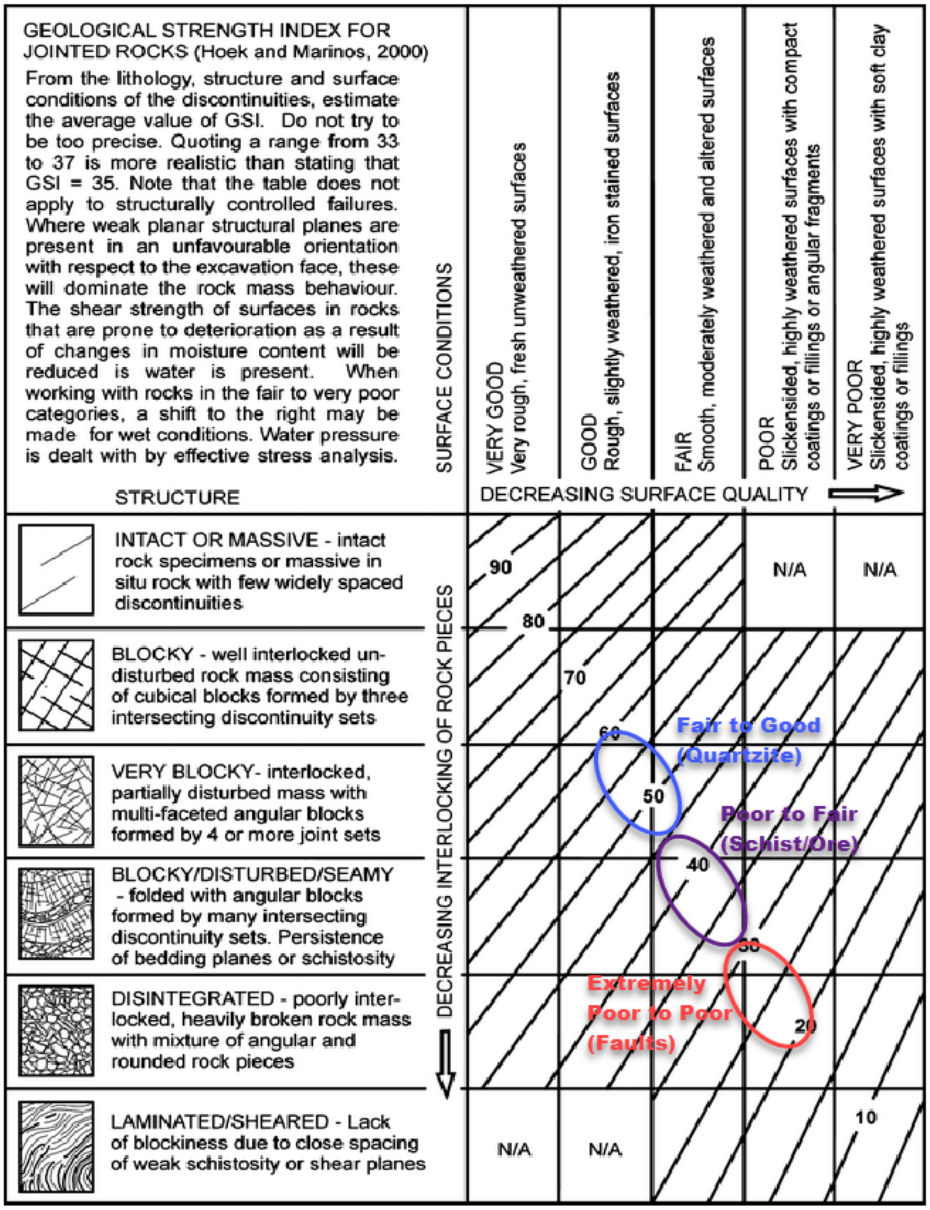


Figure 4.4 Estimated range of representative rock mass for geotechnical domains

Table 4.5 Summary of applied rock mass classification for Bermingham deposit

Rock Mass Quality	Domain	GSI			Q
		Structure	Surface	Value	
Fair to Good	Quartzite	Blocky Very Blocky	Rough Smooth	**45 -60**	**2.0 – 6.0**
Poor to Fair	Schist Ore Vein	Very Blocky Seamy	Smooth Weathered	**30 – 45**	**0.3 – 2.0**
Ext. Poor to Poor	Faults	Disintegrated Foliated	Slickensided	**20 - 30**	**0.05 – 0.3**

4.3 Rock Mass Properties

Rock mass strength criteria and material properties were estimated for each domain using geotechnical data to conduct numerical and empirical assessment. The Hoek-Brown failure criterion was applied, which requires the GSI rock mass classification scheme to be initially assessed.

Hoek and Brown (1980a, 1980b) proposed a method for obtaining estimates of the strength of jointed rock masses, based upon an assessment of the interlocking of rock blocks and the condition of the surface between these blocks. The Hoek-Brown criterion for all geotechnical domains was estimated using the approach outlined by Hoek et al (2003).

Table 4.6 Applied rock mass properties for the ground support analyses

Rock Mass Properties		Quartzite	Schist/Vein	Faults/Vein
Intact Rock Strength, UCS (MPa)		50	45	25
Geological Strength Index, GSI		50	40	25
Young's Modulus, E_i (GPa)		75	50	7.5
Disturbance Factor, D		0.3	0.3	0.2
Hoek-Brown Constant	m_b	1.5	1.0	0.8
	a	0.5	0.5	0.5
	s	0.002	0.0006	0.0001
Rock Mass Modulus, Em (GPa)		50	40	5
Poisson's Ratio, v		0.3	0.3	0.3

5. GROUND SUPPORT DESIGN – MEN ENTRY OPENINGS

5.1 Opening Dimensions

Men entry design span for main ramp and production drifts have been reviewed based on the critical span curve presented by Ouchi et al. (2004) as shown in Figure 5.1. From this work, the back span for openings in fair to good ground which is mostly in quartzite, some schist and ore vein domains was ranged from 5 m to 9 m. However, maximum critical span in poor ground, faults and some ore vein domain, is limited less than 4 m, which means immediate ground support such as pre-spraying of shotcrete before installing primary ground support by pattern bolting with screen. Possible span of heading in extremely poor ground is less than 2.5m which lies on the boundary between unstable and potentially unstable back condition and, if wider than the critical span heading is required in this low rock mass quality ground, pre-ground support method, spilling and/or grouted pore-poling, may will be required.

To mine the full mineralized width using C&F mine method in central Lightening and upper Christal deposits, wide drift with retreat slashed (up to 7 m wide ore body) or backfill with side drift (ore body width ranging from 7 m to 10 m) will be required depend on ground condition. For the production drifts with wide span near the surface, the use of shotcrete girder structure and/or artificial pillar support can be further evaluated to increase opening span, stability and recovery.

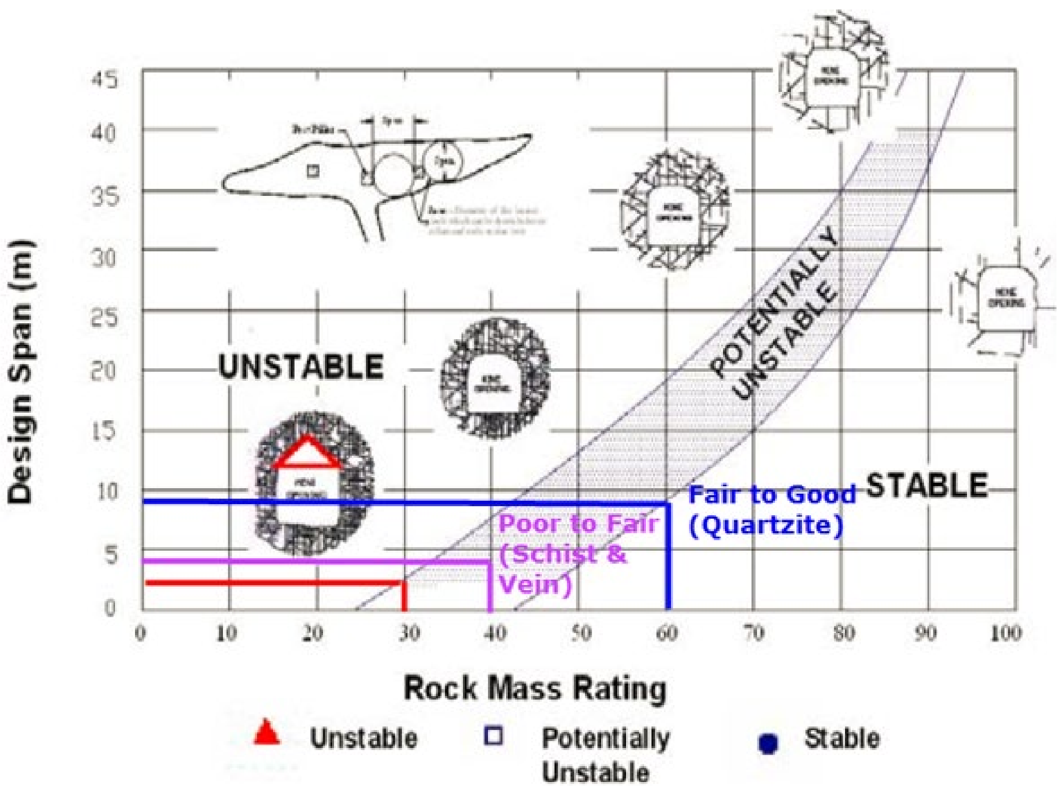


Figure 5.1 Recommended design span from Ouchi critical span curve

The planned opening dimensions that are to be used to access and mine ore bodies, for which support will be required, are given in Table 5.1

Table 5.1 Planned dimensions of men entry openings

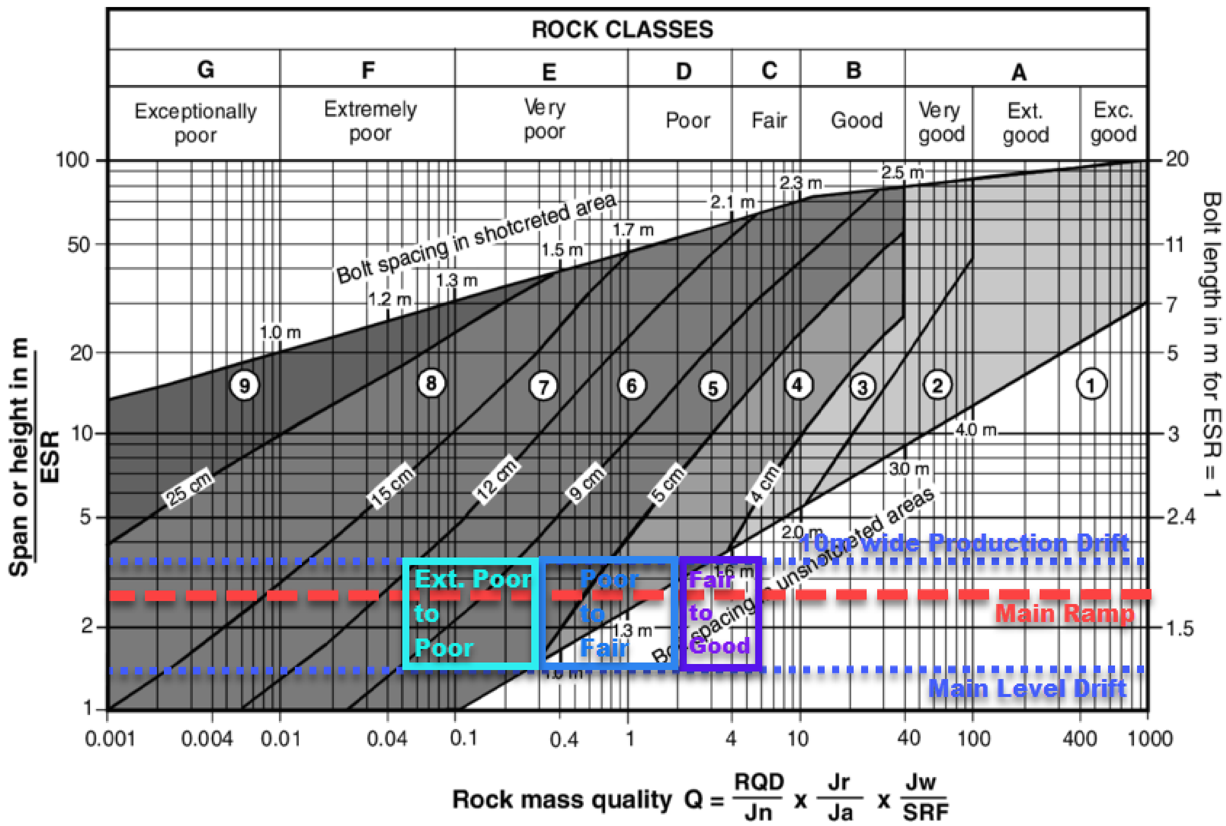
Opening Development	Dimension (W x H, m)
Main Ramp	4.2 x 4.2
Level Access Drift	3.5 x 4.0
Production Drift	3.5 ~ 10.0 x 4.0
Take Down Back (TDB) retreat Drift underneath Backfill	5.5 x 6.0
Raise	2.8 x 2.8 or D = 3.0

5.2 Support Requirements from Empirical Q Support Guideline

The ground support guidelines for main drifts (blue) and main ramp (red) are plotted in Figure 5.2. The value of Excavation Support Ratio (ESR) in the chart is relate to the intended use of the excavation and to the degree of security which is demanded of the support system installed to maintain the stability of the excavation for the planned stand-up time. For Bermingham mine, two broad categories of excavation are supported: a) Long term infrastructure, main ramp, for which the ESR values is 1.6 and b) short term mining excavations for which the suggested ESR value is 3 (Table 5.2).

Table 5.2. The value of ESR related to the intended use of the excavation and to the degree of security which is demanded of the support system installed to maintain the stability of the excavation. (Barton et al, 1974)

	Excavation Category	ESR
A	**Temporary mine openings**	**3 - 5**
B	**Permanent mine openings, water tunnels for hydro power (excluding high pressure penstocks), pilot tunnels, drifts and heading for excavations**	**1.6**
C	Storage rooms, water treatment plants, minor road and railway tunnels, civil defense chambers, portal intersections.	1.3
D	Power stations, major road and railway tunnels, civil defense chambers, portal intersections.	1.0
E	Underground nuclear power stations, railway stations, sports and public facilities, factories	0.8



REINFORCEMENT CATEGORIES;

1. Unsupported.
2. Spot bolting (SB).
3. Systematic bolting (B).
4. Systematic bolting with 40-100 mm unreinforced shotcrete
5. Fiber reinforced shotcrete, 50-90 mm, and bolting.
6. Fiber reinforced shotcrete, 90-120 mm, and bolting.
7. Fiber reinforced shotcrete, 120-150 mm, and bolting.
8. Fiber reinforced shotcrete, >150 mm, with reinforced ribs of shotcrete and bolting.
9. Cast concrete lining.

Figure 5.2. Estimated ground support requirements for temporary mine drifts and permanent infrastructure openings based on the empirical Q-support guideline.

According to Barton chart, ground support category for most openings in fair to good ground and some short-term openings in poor to fair fall into category 1 which means openings can stand-up without supports. However, long-term openings such as main ramp in poor ground or all openings in faults zone area need to apply proper ground support in timely manner.

Barton et al (1974) also provide additional information on rock bolt length, maximum span of rock bolt. According to Barton et al, the length, L, of rock bolts can be estimated from the excavation width (B) and ESR value, and rock bolt span can be calculated using Q-value and ESR. Both empirical correlations and ground support patterns for different ground conditions using empirical methods are summarized in Table 5.3.

Table 5.3 Ground support estimation for men entry openings using empirical correlation suggested by Barton et al. (1974)

			Support Category	Rock bolt length (m) $L = 2 + 0.15B / ESR$	Bolt spacing (m) $S = 2 \times ESR \times Q^{0.4}$
Main Ramp (ESR = 1.6)	B = 4.2m	Q = 4	(1)		5.6
		Q = 1	(4), (5)	2.4	3.2
		Q = 0.1	(6)		1.3
Level Access (ESR = 3.0)	B = 3.5m	Q = 4	(1)		10.4
		Q = 1	(1)	2.2	6.0
		Q = 0.1	(5)		2.4
Wide Ore Drift	B = 7.0m	Q = 4	(1), (4)	2.4	10.4

5.3 Stand-up Time Analysis

The stand-up time of unsupported spans is one of the fundamental issues in mine development. The Bieniawski diagram (Figure 5.3) shows the relationship between the unsupported span and stand-up time of an excavation with reference to its rock mass quality. The basic relationship that governs stand-up time is:

- For a given rock mass quality, a stand-up time decrease as the unsupported roof span become wider, and
- For a given roof span, a stand-up time decrease as the rock mass quality becomes poorer.

Using data collected from Bermingham mine, stand-up time for two different roof span in three different ground conditions were estimated based on the Bieniawski diagram as shown in Figure 5.3.

The stand-up time of Openings with 3.7 m span and 7.0 m span in fair rock mass (GSI =50) can be assumed 20 days and 4 days respectively. The other cases, face of main ramp or regular level drift in extremely poor fault zone (GSI =25) would be stand-up less than an hour according to Bieniawski. This chart can be apply for the delay time of ground support for current developing faces and this time does not means stand-up time for whole mine drift.

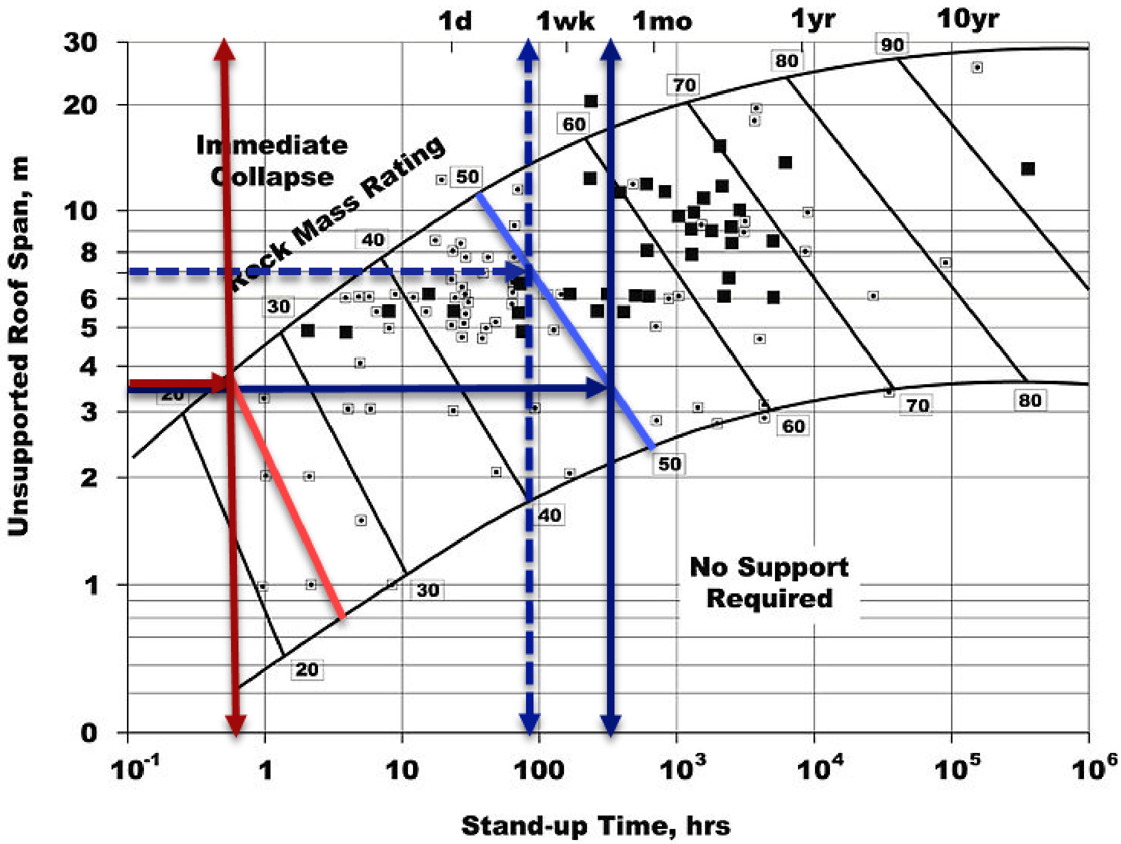


Figure 5.3 Relationship between stand-up time, roof span and RMR after Bieniawski (1989)

5.4 Ground Support Standards for Men Entry Openings

Ground support standards for men entry openings, such as main decline ramp, level access drift, vertical ventilation/escape raise and possible wide production drift in different ground conditions are recommended in this section. Although some decline ramp and main access drift in fair to good ground are able to stand-up relatively long period without ground support based on Barton's empirical Q support chart and Bieniawski's stand-up chart, minimum ground support using 1.8m to 2.4m long rock bolts for the back and walls are required to prevent possible wedge failure or unconsolidated back and wall sloughing caused by blasting damage.

Split sets or Swellex (expandable friction bolt) are preferred as a primary support element for production drift with relatively short term of opening period to reduce cycle-time of ground support installation. As decline ramp need to keep open longer period, fully grouted rebar bolts can be recommended for the ramp support. Minimum ground support standards for Bermingham mine underground men entry openings are summarized in Table 5.4 and detailed ground support regimes for each opening are shown in Appendix – B.

Table 5.4 Ground support standards for Bermingham men entry openings

Type	Ground Condition	Ground Support Standards (Bolt space)	
Decline Ramp (permanent openings)			
Ramp – I	Fair to Good (45 < GSI < 60)	Back	1.8m Rebar (1.2m x 1.2m)
		Wall	1.8m Split set (1.2m x 1.2m), 1.8m from sill
Ramp – II	Poor to Fair (30 < GSI < 45)	Back	2.4m Rebar (1.2m x 1.2m), SC as req.
		Wall	1.8m Split set (1.2m x 1.2m), 1.2m from sill
Ramp – III	Ext. Poor (GSI < 30)	Back	2.4m Rebar (0.8m x 0.8m), 2” SC, Spilling as req.
		Wall	2.4m Rebar (0.8m x 0.8m), 2” SC, Spilling as req.
Main Access Drift (opening less than 3 years)			
MD – I	Fair to Good (45 < GSI < 60)	Back	1.8m Swellex (1.2m x 1.2m)
		Wall	1.8m Split set (1.2m x 1.2m), 1.8m from sill
MD – II	Poor to Fair (30 < GSI < 45)	Back	2.4m Swellex (1.2m x 1.2m), SC as req.
		Wall	1.8m Split set (1.2m x 1.2m), 1.2m from sill
MD – III	Ext. Poor (GSI < 30)	Back	2.4m Swellex (0.8m x 0.8m), 2” SC, Spilling as req.
		Wall	2.4m Swellex (0.8m x 0.8m), 2” SC, Spilling as req.
Wide Production Retreat Drift (3.5 m ~ 7.0 m)			
WD – I	Fair to Good (45 < GSI < 60)	Back	MD-I + 3.6m Connectable (2.4m x 2.4m)
		Wall	MD- I
WD – II	Poor to Fair (30 < GSI < 45)	Back	MD-II + 3.6m Connectable (1.8m x 1.8m), 2” SC as req.
		Wall	MD-II
Remuck			
RMK – I	Fair to Good (45 < GSI < 60)	Back	2.4m Rebar (1.2m x 1.2m)
		Wall	2.4m Rebar (1.2m x 1.2m), 1.2m from sill
RMK - II	Poor to Fair (30 < GSI < 45)	Back	2.4m Rebar (0.8m x 0.8m)
		Wall	2.4m Rebar (0.8m x 0.8m), 1.2m from sill
Raise			
SR – I CR – I	Fair to Good (45 < GSI < 60)	Face	1.2m Rebar (1.2m x 1.2m)
		Wall	1.2m Rebar (1.2m x 1.2m)
SR – II CR – II	Poor to Fair (30 < GSI < 45)	Face	1.2m Rebar (0.8m x 0.8m), 2” SC as Req.
		Wall	1.2m Rebar (0.8m x 0.8m), 2” SC as Req.
Intersection			
IS – I	Fair to Good (45 < GSI < 60)	Back	Ramp/MD-I + 3.6m Connectable (2.4m x 2.4m)
		Pillar	3 rows of strap with 1.8m Split set
IS - II	Poor to Fair (30 < GSI < 45)	Back	Ramp/MD-II+ 3.6m Connectable (1.8mx1.8m), 2” SC
		Pillar	Screen + 3 rows of strap with 1.8m Split set, 2” SC

No intersection and Wide ore drift in extremely poor ground

The Ground Support Standards form the basis for all ground support and are to be installed according to specification. It is the responsibility of the operator to report and deviation to the standard and the reason for it.

In advance geotechnical ground conditions (e.g. fair to good ground, presence of structures, expected corrosion), the Ground Support Standards shall be review and additional support recommendations will be made by Geotechnical Engineer or designated personnel. The Ground Support Standards cannot be reduced without recommendation by Geotechnical Engineer and approved by Mine Manager.

The Ground Support Standard will be revised as experience is gained upon excavation of the mine. The support regimes employed at Bermingham mine are composed of main ramp, main access drift, ore extraction drift and intersection (Appendix – B). Intersections pose a higher risk for ground instability than normal development due to the large spans and repeated blasting damage. Specific regimes for intersection area also have been formulated to support the increased span both horizontally as well as vertically.

- 4-way intersections are to be avoid as much as possible
- Intersection in extremely poor aground must be relocated
- Over-excavation should be minimized

The Ground Support Standards specify the ground support required in all development

- There are 9 basic support types and 2 intersection support types depending on ground conditions and development geometry.
- No Ground Support Standard was recommended for wide ore extraction drift, remuck and intersection in extremely poor ground. Special mine and support plans need to develop for these activities in such ground condition.

The Trigger Action Response Plan (TARP) as summarized in Appendix - C specifies the circumstances under which a change in support type is to occur.

- The TARP provides a description of ground condition indicators which, where observed separately or individually may indicate a change in Support Standard for individual headings

Copies of the GCMP shall be kept in the Shift Supervisor’s office and Mine Superintendent’s office, Engineering Main office and the crew lineup meeting room. The Ground Support Standards and TARP should be prominently displayed. The Supervisor shall ensure that all Shift Supervisors responsible for ground support during development are familiar with the GCMP, Ground Support Standards and TARP.

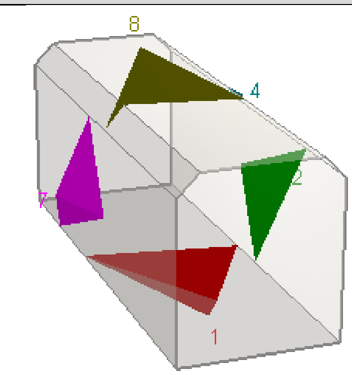
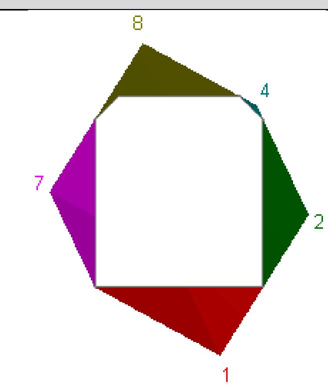
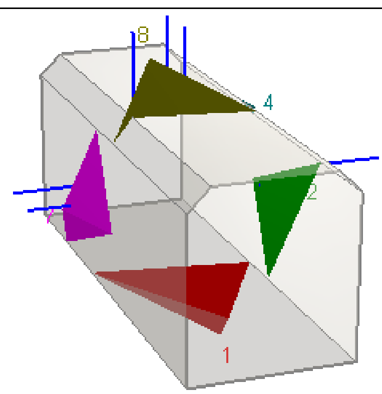
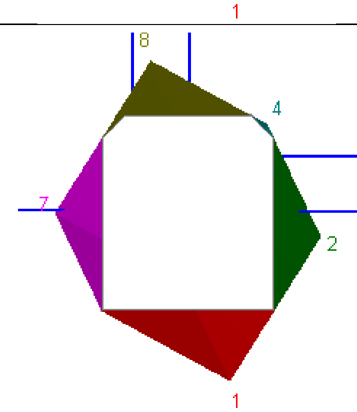
5.5 Kinematic Wedge Stability Analysis

Three most prominent sets of joint including major joint set parallel to Mill Faults were identified during site visit and these joint sets were used in the kinematic analysis to identify the potential for wedge failure around the stopes and development in the proposed underground excavations. (The RocScience program UNWEDGE was used for the analysis). For the analysis, it was assumed that the joint sets are ubiquitous, continuous and planar, and as such does not take into consideration joint spacing and persistence. This usually results in a lower factor of safety and a more conservative assessment of the excavation geometry.

Value for cohesion and tensile strength were set to zero for both the foliation and joints in the analysis, and the field stress was set to 1 MPa lithostatic, to prevent the formation of unrealistic high aspect wedge in the analysis.

A wedge analysis for the man entry excavations was conducted using opening size of 3.7m wide by 4.2 m high. The model was conducted without inclusion of support; and in instances where unstable blocks were identified, ground support was added to the model and re-evaluate the factor of safety for the wedge failure. The ground support recommended in Section 5.4 provides enough support pressure to prevent the wedge generated from falling out of the back (Table 5.5).

Table 5.5 Kinematic wedge analysis results for the main drift with 3.7 mW × 4.2 mH dimension

Support	Perspective	Front	FoS
Before Support			Lower Right wedge [2] FS: 15.584 Weight: 0.043 MN Upper Right wedge [4] FS: 28.085 Weight: 0.000 MN Lower Left wedge [7] FS: 9.615 Weight: 0.043 MN Roof wedge [8] FS: 0.000 Weight: 0.038 MN
After Support			Lower Right wedge [2] FS: 21.307 Weight: 0.043 MN Upper Right wedge [4] FS: 28.085 Weight: 0.000 MN Lower Left wedge [7] FS: 14.657 Weight: 0.043 MN Roof wedge [8] FS: 7.955 Weight: 0.038 MN

5.6 Verification of Applied Support Standard

5.6.1 Assessment of Damaged/Disturbed Zone around Openings

Damaged/Disturbed Zone(DZ) around two different dimensions openings, B = 3.7m and 7.0m, in different ground condition were estimated by numerical parametric study. Using elasto-plastic model in RS2 two-dimensional numerical analysis package, the depth of DZ around openings can be estimated from Strength Factor (SF) because If the Strength Factor is less than 1, this indicates that the stress in the material exceeds the material strength (i.e. the material would fail, if a plasticity analysis were carried out). From the work it is indicated that the ratio between wedge height (H_w) and opening width (B) changes relate to opening width (B) and ground condition as shown in Table 5.6 and 5.7. For the main ramp and drift 0.3B, 0.4B and 0.6B can be assumed as a possible failure depth for the opening in different ground condition respectively (Table 5.6). 0.35B and 0.45B can be considered as a DZ for the wide ore extraction drift with 7 m of width in fair to good and poor to fair ground (Table 5.7).

Table 5.6 Damaged/Disturbed zone at the back of main ramp openings

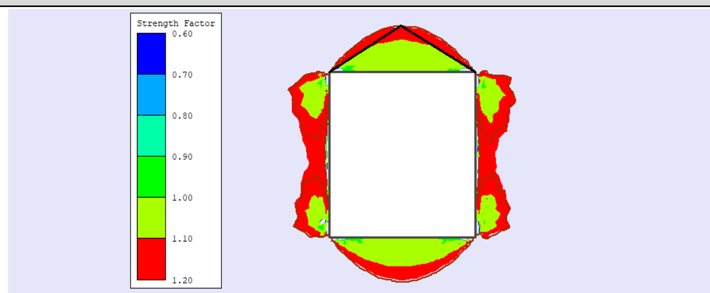
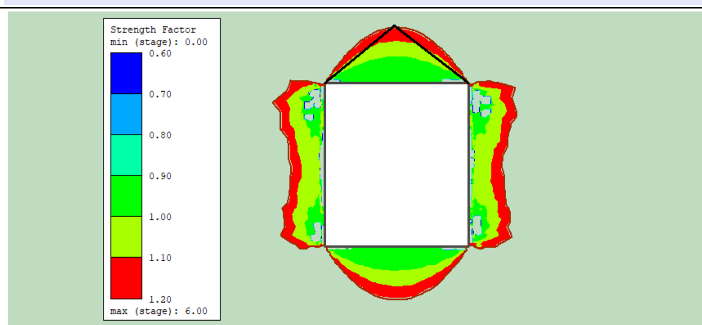
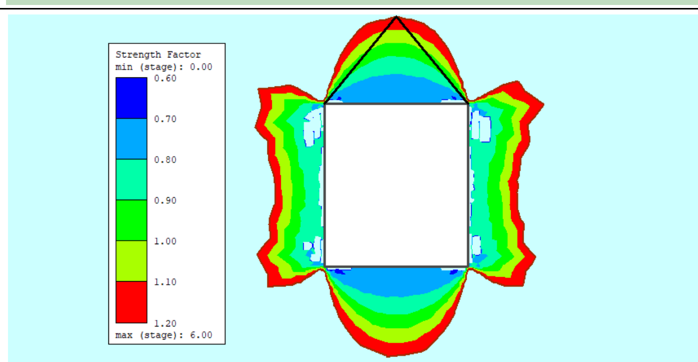
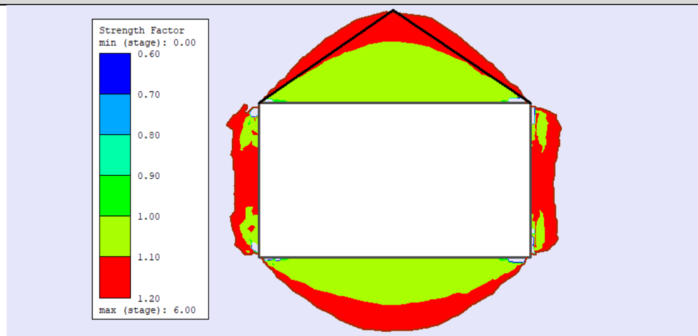
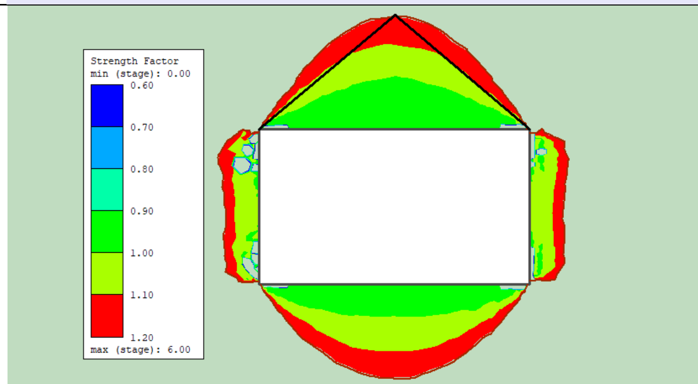
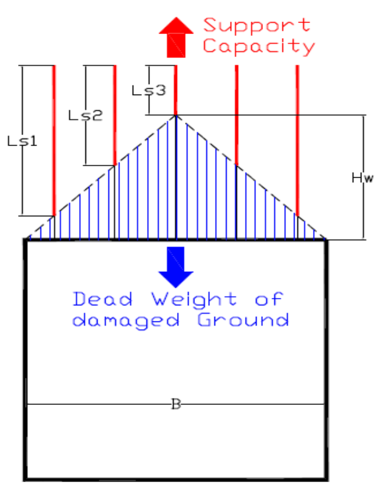
Ground	Strength factor and damaged	DZ/B
Fair to Good		B = 3.7m DZ = 1.1m **DZ/B = 0.3**
Poor to Fair		B = 3.7m DZ = 1.5m **DZ/B = 0.4**
Ext. Poor to Poor		B = 3.7m DZ = 2.2m **DZ/B = 0.6**

Table 5.7 Damaged/Disturbed zone at the back of 7m wide ore extraction drift

Ground	Strength factor and damaged	DZ/B
Fair to Good		B = 7.0m DZ = 2.45m **DZ/B = 0.35**
Poor to Fair		B = 3.7m DZ = 3.1m **DZ/B = 0.45**

5.6.2 Dead Weight Analysis

Safety factors for all support patterns associate with ground conditions and opening dimensions were estimated by Dead Weight analysis. Outline of Dead Weight analysis is illustrated in Figure 5.4. Safety factor is the capacity of rock bolts installed at the back against weight of failed wedge block. The weight of wedge can be calculated by opening width and failure depth, capacity of rock bolts should be estimated using the installed length beyond the wedge.



Dead Weigh
$= 1/2 \times \Upsilon_{rock\ mass}$ (t/m^3) $\times B \times DZ \times$ Bolt Space
Support Capacity
$=$ Bond strength $\times (2L_{s1}/2 + 2L_{s2} + L_{s3})$

<u>FS = Support Capacity / Dead Weight</u>

FS > 1.5 : permanent opening (Ramp)
FS > 1.2 : temporary opening (Main Drift)

Figure 5.4 Factor of Safety from Dead Weight analysis

Factor of Safety (FoS) for three different support patterns with 3.7 m and 3.5 m wide heading in different conditions ground were estimated using Dead Weight analysis (Appendix – D) and results of the analysis were summarized in Table 5.8. Minimum 1.5 of FoS is required for decline ramp support as a permanent opening and FoS of 1.2 is considered for minimum FoS for main drift as a temporary production opening. According to long life of opening, 1.8 m and 2.4 m long fully grouted rebar were recommended for the back support of decline ramp. As a primary support for temporary opening, 1.8 m split set and 2.4m regular swellex can be recommended depend upon ground condition. 3.6 m long connectable super swellex and pre/post shotcrete also need to apply for openings in extremely poor ground. Pre-support with spills may requires as an additional ground support in extremely poor ground because of less than an hour of stand-up time (Figure 5.3).

Table 5.8 Factor of Safety (FoS) from Dead Weight analysis (Appendix – D)

Opening Type	Ground Condition	Ground Support (Spacing) **Factor of Safety**		
Decline Ramp B = 4.2m	Fair to Good (45 < GSI < 60)	1.8m Split Set (1.2m x 1.2m) **FoS = 1.5**	1.8m Rebar (1.2m x 1.2m) **FoS = 2.8**	2.4m Rebar (1.2m x 1.2m) **FoS = 3.6**
	Poor to Fair (30 < GSI < 45)	1.8m Split Set (1.2m x 1.2m) **FoS = 0.6**	1.8m Rebar (1.2m x 1.2m) **FoS = 1.5**	2.4m Rebar (1.2m x 1.2m) **FoS = 2.3**
	Extremely Poor (GSI < 30)	2.4m Split Set (1.2m x 1.2m) **FoS = 0.7**	2.4m Rebar (1.2m x 1.2m) **FoS = 1.1**	2.4m Rebar (0.8m x 0.8m) **FoS = 3.2**
Main Drift B = 3.5m	Fair to Good (45 < GSI < 60)	1.8m Split Set (1.2m x 1.2m) **FoS = 1.3**	1.8m Swellex (1.2m x 1.2m) **FoS = 2.0**	2.4m Swellex (1.2m x 1.2m) **FoS = 2.7**
	Poor to Fair (30 < GSI < 45)	1.8m Split Set (1.2m x 1.2m) **FoS = 0.8**	1.8m Swellex (1.2m x 1.2m) **FoS = 1.3**	2.4m Swellex (1.2m x 1.2m) **FoS = 2.0**
	Extremely Poor (GSI < 30)	2.4m Split Set (1.2m x 1.2m) **FoS = 0.6**	2.4m Swellex (1.2m x 1.2m) **FoS = 0.9**	2.4m Swellex (0.8m x 0.8m) **FoS = 2.1**
Production Drift B = 7.0m	Fair to Good (45 < GSI < 60)	3.6m Connect. (2.4m x 2.4m) **FoS = 0.9**	3.6m Connect. (1.8m x 1.8m) **FoS = 1.4**	
	Poor to Fair (30 < GSI < 45)	3.6m Connect. (2.4m x 2.4m) **FoS = 0.9**	3.6m Connect. (1.8m x 1.8m) **FoS = 1.3**	

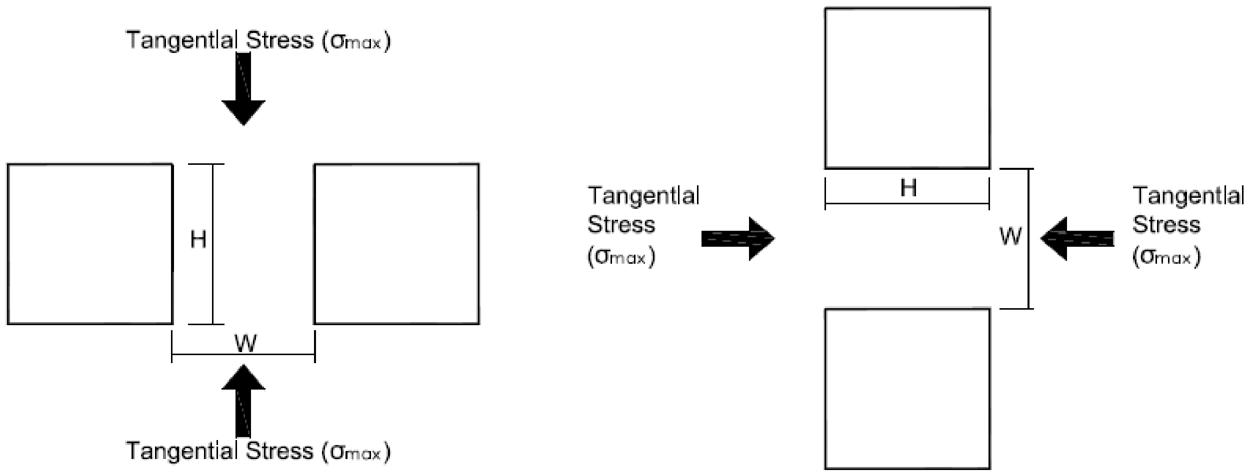


Figure 5.5 Dimension of rib and sill pillar

5.7 Pillar Design for Men Entry Openings

5.7.1 Pillar Geometry

Pillars are usually designed to be rectangular or square shapes in both plan and section. The design of pillars relates the strength to pillar shape. Figure 5.5 illustrates the pillar dimension. It is important to note that the pillar height is defined relative to the direction of the maximum stress. For example, for sill pillar, the pillar height is actually in the horizontal direction as the maximum pillar stress will be in the horizontal (Figure 5.5).

5.7.2 Pillar Failure Modes

There are three modes of pillar failure which are commonly observed underground: (1) structurally controlled failure; (2) stress induced progressive failure; and (3) pillar burst,

Structurally controlled failure

Most rock masses contain pre-existing failure plane (discontinuities) known as joints, faults, etc. Structurally controlled failure occurs when the pillars are oriented unfavorably with respect to the discontinuities present within the rock mass. Failure of these planes is usually in the form of shear movement along the plane. This type of failure is often observed as corners of pillars coming off along wall defined planes.

Progressive failure

The second mode of failure is termed stress-induced progressive failure. This is observed as slabs spalling off the walls of the pillars. The progressive spalling mode of failure, otherwise known as “hour-glassing”, is generally observed in squat pillars where the skin of the pillar which has little confinement and high tangential stresses causes cracking and slab formation parallel to the direction of the major principal stress in the pillars.

Kaiser et al (1996) suggested that the first stage of stress-induced failure was the 'hour-glass' effect commonly observed in hard rock pillar failure (Figure 5.6). They suggested that the failed material should be turned 'baggage' because if unsupported it simply forms detached slabs. The extent of this spalling failure could be predicted by,

$$\Delta = \frac{h\ (or\ W)}{2 \cdot sin\alpha} \cdot \left(1 - \frac{sin\alpha}{tan\alpha}\right)$$

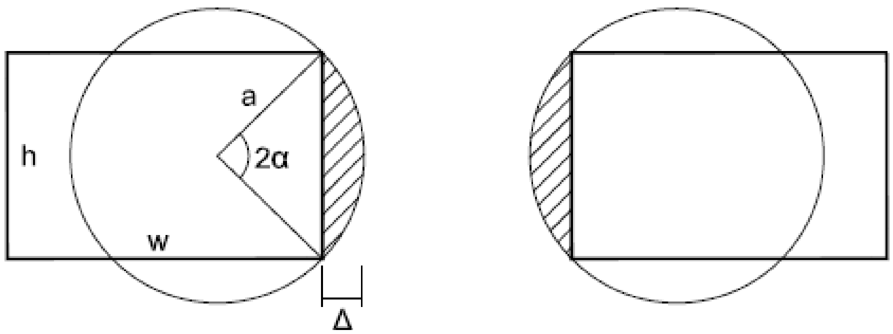


Figure 5.6 Definition of baggage (after Kaiser, McCreath and Tannant, 1996)

Initially the core of the pillar remains intact after spalling failure, because it is still confined and, hence, the pillar still remains most of its load carrying capacity. As spalling occurs, the stresses flowing through the pillar are redistributed to the intact pillar rock. The loss of the slabs relaxes the confinement on the adjacent intact core rock in the pillar and further damage then occurs to the newly exposed pillar wall surfaces (Figure 5.7). If this type of progressive failure is allowed to propagate too far, then the intact core of the pillar can reach a critical cross-sectional area and fail.

If the loads around an opening were sufficient to cause additional stress-induced failure (Figure 5.7), the depth of the failure could be approximated by the linear relationship given by (Martin, 1990),

$$\frac{df}{a} = 1.34 \cdot \frac{\sigma_{max}}{\sigma_c} - 0.57$$

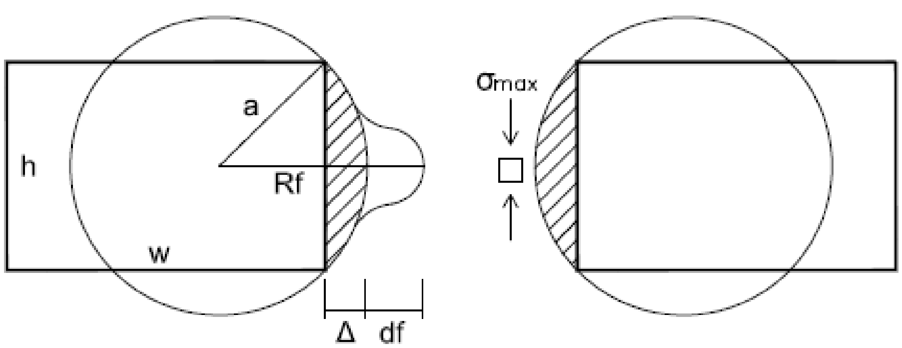


Figure 5.7 Depth of stress-induced failure (after Kaiser, McCreath and Tannant, 1996)

It is important to note that the above equation the stress-induced failure propagates when the maximum tangential stress exceeds approximately one third of uniaxial compressive strength.

Pillar Bursting

The third mode of failure encountered in pillar is pillar burst. This mode of failure is usually encountered when the following two constraints are satisfied: (1) The stress in the pilaar must exceed the strength; and (2) the local mine stiffness must be less than that of the pillar.

Based on the work by Martin (1990) and Kaiser et al (1996) when the pillar stress exceeds 1/3 of the uniaxial compressive strength of the rock the first constraint is generally satisfied. Once the strength of the pillar exceeded, the violence of the failure is governed by stiffness of the surrounding mine environment. If the local mine stiffness is high compared to the post-peak stiffness of the pillar, then the failure will be nonviolent (stress-induced progressive failure mode). However, if the local mine stiffness is low, less than that of the pillar, then the failure will be violent as more energy is put into the failing pillar.

5.7.3 Pillar Design

Pillar stability analyses against stress-induced progressive failure were conducted for Bermingham mine pillar design because structurally controlled failure can be controlled by additional spot bolting during regular basis geotechnical inspection and possibility of pillar bursting in this mine is low according to mine stiffness and given low in situ stress condition.

Table 5.9 Maximum stresses, extent of damaged depth in pillars

Opening Dimension (3.5mW x 4.0mH)		Pillar Width, Wp (m)				
		3.0	4.0	5.0	6.0	7.0
σ_{max}/σ_c	Fair to Poor (45<GSI<60)	Failed	1.0	0.96	0.84	0.77
	Poor to Fair (30<GSI<45)	Failed	Failed	1.0	1.0	0.85
	Extremly Poor (GSI<30)	Failed	Failed	Failed	Failed	1.0
DZ $\Delta + df$ (m)	Fair to Poor (45<GSI<60)	-	2.84	2.79	2.62	2.53
	Poor to Fair (30<GSI<45)	-	-	2.84	2.84	2.64
	Extremly Poor (GSI<30)	-	-	-	-	2.84
DZ/Wp (%)	Fair to Poor (45<GSI<60)	-	**71**	**56**	44	36
	Poor to Fair (30<GSI<45)	-	-	**57**	**47**	38
	Extremly Poor (GSI<30)	-	-	-	-	**40**

Maximum tangential stresses in rib pillar area with 5 different pillar width from 3 m to 7 m were estimated using 2-dimensional numerical analyses (Phase2) for different conditions of ground and the results were summarized in Appendix – E. Damaged/disturbed depth (Δ + df) and percentage of failure area in rib pillars were assumed in accordance with maximum pillar stresses and pillar dimensions. Maximum stress, progressive stress-induced Excavation Damaged ⁄ Disturbed Depth (EDZ), and the percentage of damaged area in pillar (EDZ⁄Pillar width) from the analysis were summarized in Table 7.11 and Figure 7.9.

The results from numerical analysis (Appendix – E) on 3 m wide pillars shows stress induced failure propagate whole pillar area regardless of ground conditions. Pillar in fair to good condition ground with 4 m width shows less than 75% of damaged depth ratio to pillar width which means pillar will be stable with additional support and the ratio shows lower than 60% if the pillar width is wider than 5 m which noted that the pillar should be wider than 5.0m without additional support in fair to good ground. If ground condition of pillar location is poor to fair, pillar width must be wider than 5 m with additional support plan. However, according to this stability analysis, wider than 7 m of pillar width is required for the opening with 3.5 m wide by 4 m high dimension in extremely poor to poor ground and shotcrete to the pillar walls and displacement monitoring are strongly recommended.

6. OPTIMIZATION OF LONGHOLE STOPE DIMENSION

6.1 Stress Change surrounding Longhole Stope

To determine proper dimension of longhole stopes and mining sequence it is requested that understand stress path change caused by development of longhole mine. Failure is a result of rock mass relaxation and that is defined as a reduction in stress static parallel to wall excavation. Wedge failure occurs when the minor principal stress is below or equal to zero as shown in Figure 6.1 stress path A. The severity of sloughing (stress path B) also possible failure mode for the longhole stope and the failure is related directly to the rock tensile strength. However, rock mass has a self-supporting capacity depending on the material properties and geological structures.

6.2 Maximum Stope Strike Length

The widely used empirical tool for a maximum stope strike length is the stability graph method. The method is developed by Mathews et al (1981) and defined by Potvin (1988). The stability graph method associates the stability number to the hydraulic radius of a stope. The graph helps to access the stability of an opening according to the stope hydraulic radius. The stability number (N) can be calculated by the following equation,

$$N' = Q' \times A \times B \times C$$

Where, N: Stability number
Q': Modified NGI Q value with stress reduction factor
A: Stress factor – ratio of intact rock strength to applied stress
B: Joint orientation factor – relative orientation of dominant structure with respect to the excavation surface
C: Gravity factor – influence of gravity on the stability of the face being considered.

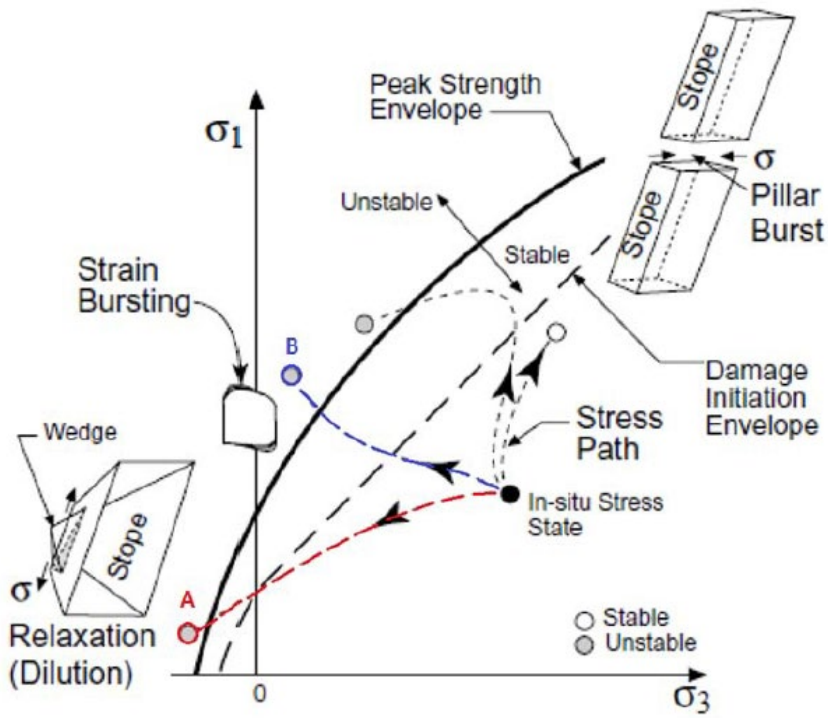


Figure 6.1. Possible stress path for a longhole stope (Martin et al, 1999)

Table 6.1. Stability Graph assumption for stope designs

Parameter	Value	Design Assumption
Q'	5 1 0.5	Fair to good ground (45 < GSI < 60) Poor to Fair ground (30 < GSI < 45) Extremely Poor to Poor ground (20 < GSI < 30)
A	0.5 (wall) 0.2 (back)	Assume induced stresses concentrate above and adjacent to back. Walls are generally destressed
B	0.3 (wall) 0.3 (back)	Conservative assumption based on structural variability in all domain
C	5.0 (wall) 3.0 (back)	Defined based on critical discontinuity set assuming horizontal structure in back and structure parallel to wall

Q’ values of 5, 1, and 0.5 were used in combination with A, B, and C inputs to calculate permissible stope strike length for 15 m high and 5 m wide stope in three different category of ground conditions (fair to good, poor to fair, poor). Input parameters for Stability Graph are summarized in Table 6.1 and recommendation of maximum stope strike lengths for each ground condition are shown in Table 6.2 and Figure 6.2.

Table 6.2. Maximum Stope Strike Length Recommendation

Ground Condition	Basic Stope Height / Width: 15 m / 5 m		
	N’	HR	Max. Strike Length (m)
Fair to Good (45 < GSI < 60)	Wall: 3.8 Back: 0.9	Wall: 4.5 Back: 5.2	**W: 20m with 15m high (unsupported)** B: over 100m with 5m wide (supported)
Poor to Fair (30 < GSI <45)	Wall: 0.8 Back: 0.2	Wall: 2.9 Back: 3.5	**W: 10m with 15m high (unsupported)** B: over 100m with 5m wide (supported)
Ext. Poor to Poor (20 < GSI < 30)	Wall: 0.4 Back: 0.12	Wall: 2.1 Back: 3.0	**W: 6m with 15m high (unsupported)** B: over 100m with 5m wide (supported)

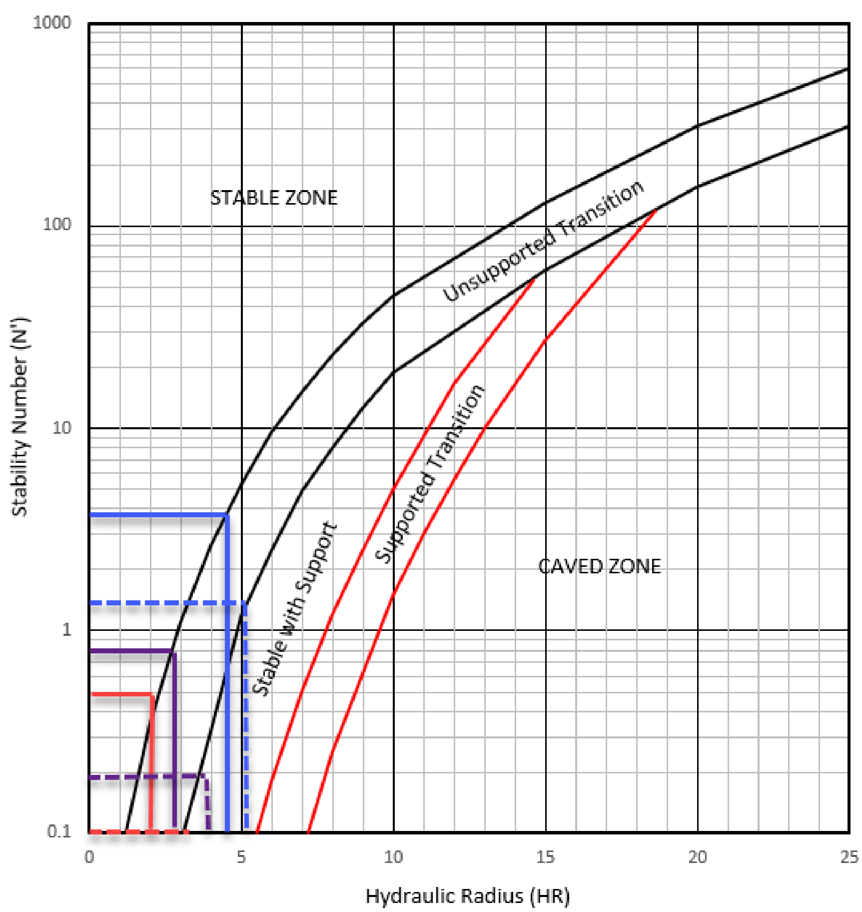


Figure 6.2. Stope Stability Plot for unsupported stopes (Potvin, 1988)

6.3 Ground Support for Longhole Stope

According to stope stability analysis, 20 m to 30 m of maximum stope strike length for 15m high stope can be recommendable depending on ground condition in poor to good ground. However, these stopes assume the use of proper rib and sill pillars that clamp the edges of the various conditions of stope surface. If an Avoca method, continuous stope development following backfill, is used as a extraction method, these stope strike lengths and stope heights will need to be reduced, as the Avoca fill does not provide the same level of support/stiffness as pillars. Stope heights would need to be limited 15 m. Previously recommended stope lengths have to include 5 to 10 m length of the previous stope because of the unconsolidated rock fill.

In this case of open stopping, the use of cable bolting to increase spans and lengths of stope has been used with varying degrees of success. Cable bolting to increase the potential dimension of stopes generally falls into two categories: pattern bolting across the full span or supporting the stope from cable bolting drifts located adjacent to the stopes, or targeted cable bolting to locally improve the rock mass.

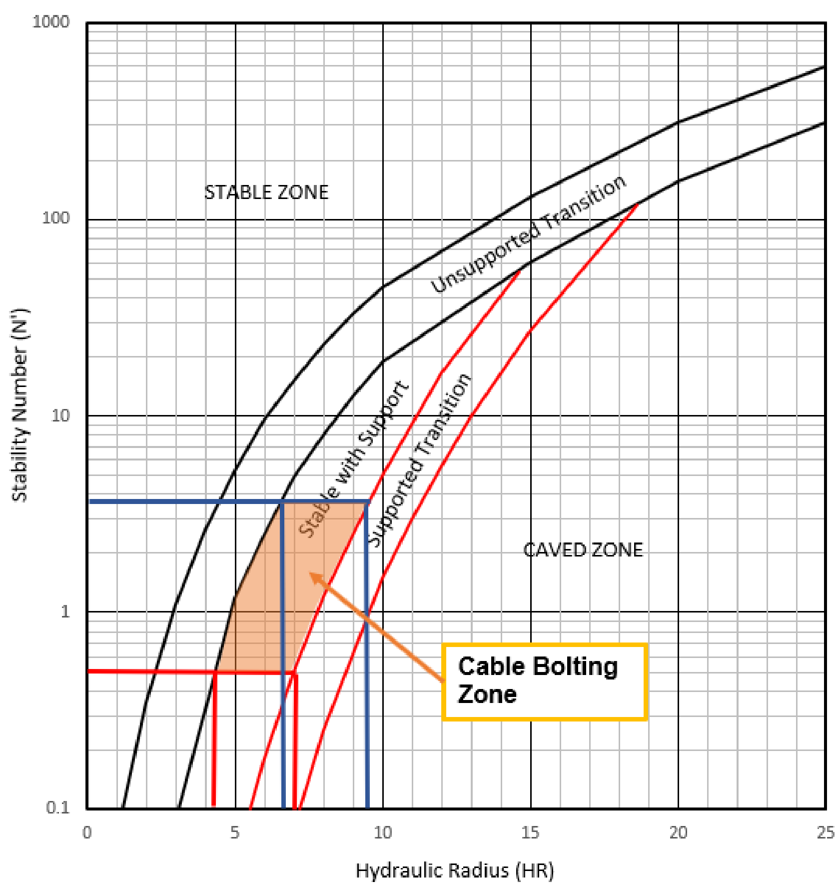


Figure 6.3 Cale bolt design zone for open stopes using modified stability graph method

Hutchinson and Diederichs (1996) discussed the work of Nickson (1992) where the use of localised high density cable bolting has allowed an increased total stope span (or height) by reducing the unsupported span. The theory behind this approach is that a block of reinforced rock on the stope perimeter has an effect similar to that of a pillar, and provides localised support. Smaller sub spans are then formed between these reinforced blocks allowing greater total spans to be opened up.

The blocks marked on Figure 6.3 represent the zones where cable bolting is considered appropriate for potential improvement to stope dimension. This shows that based on the range of N' values and calculated hydraulic radius values, some improvement to the stope dimension may be possible.

6.4 Stope Rib Pillar

If longhole stopes filled with uncemented fill, rib pillars between longhole stopes are required. The function of rib pillars is to ensure the stability of the longhole stopes and in particular the HW during mining and to keep the uncemented fill in the adjacent stope. There are two factors to consider when estimating stope pillar dimensions; the load applied to the pillar must be determined; and, the strength of the pillar to which a suitable safety factor is applied.

Pillar strength was estimated using the formula below from Potvin *et al* (1989). The strength of the pillar is a function of the pillar aspect ratio (the width to height ratio, W/H), the Unconfined Compressive Strength (UCS) of the intact rock, and a calibration factor to account for specific regional stress and rock conditions.

$$\text{Pillar Strength (in MPa)} = 0.42 * \text{UCS} * \text{W/H}$$

Upper and lower bound estimates of pillar strength and stress were calculated for the largest stope size of 15 m height and a 30 m strike span. The rib-size recommendations presented in Table 8.3 were calculated using tributary area theory, which incorporates; the calculated pillar strength, the in situ stresses, and an appropriate factor of safety (of around 1.2 in this case).

Table 6.3 Rib Pillar Recommendations

Back Width (m)		3.0	5.0	7.0
Rib Pillar Width (m)	45 < GSI < 60	5.5	7.0	9.5
	30 < GSI < 45	6.5	8.5	11.0

7. STOPE BACKFILL

7.1 Introduction

The use of cemented backfill is an increasingly important of underground mine operations and is becoming a standard practice for use in many cut & fill and longhole mine around the world. Cemented Rock Fill (CRF) can be considered as a primary backfill method for Bermingham mine to allow maximize pillar recovery of the narrow vein ore in longhole mine and optimize ground support for conventional underhand/overhand cut & fill mine area. The use of CRF not only provides ground support to the pillar and wall, but also helps prevent caving and roof falls, and minimize dilution of ore, which enhances productivity.

7.2 Backfill as a Ground Support and Ground Control Element

7.2.1 Backfill Target Design Criteria

Cemented backfill design criteria will be based on target backfill properties, which will be dependent on the backfill function, the mining system conditions, and other site-specific factors. Key target design criteria include;

- Geotechnical properties
- Distribution and placement criteria
- Environmental performance
- Socio-Economic performance

Where backfill is required for ground support or to provide a working floor, backfill strength is the primary geotechnical property. Backfill strength can be increased with the cement or other binding agents. A related geotechnical property pf backfill is liquidation potential, which is dependent on physical and mechanical properties of the tailing material but not major factor for CRF in Bermingham mine.

Distribution and placement criteria including system capacities and scheduling are based on the requirements of the mining system as well as the rheological properties of the material.

Environmental considerations have played a growing role in the determination of backfill target properties in recent years. Mining operations face increasing pressure to reduce and limit surface waste disposal of tailings. Target backfill functions and design criteria are critical to improving underground environmental health and safety working conditions also affects target backfill functions and design criteria.

Inevitably socio-economic performance of the backfill influences backfill design. As a primary resource, mining has a significant effect on the local, regional economy in terms of employment and income. The advancement of technology that contributed to the sustainability of environment will serve to enhance the continued economic viability of the mining industry. However, even though the technology may be available to meet the required geotechnical and environmental criteria and the logistical parameters for transportation and placement, the backfill system design will go no further if the cost is too high. The sophistication of backfill systems contributes to relatively high capital costs, but it is important that some of the less tangible cost benefits such as those relate to environmental factors and potential increased mining recoveries be accurately factored into trade-off studies comparing backfill to alternative backfill methods.

7.2.2 Strength of Backfill

As a target backfill property, the required strength of fill will depend on its intended function and site specific factors pertaining to rock mass quality. If the function of the fill is to provide a working floor, as in cyclical mining methods, the curing time must be short and the fill must provide early strength to support personnel and mechanized equipment. For delayed type backfilling, the backfill must achieve and maintain longer term stability and be capable of providing a free standing wall to enable pillar recovery and the mining of secondary stopes with minimal dilution.

Backfill strength can be greatly increased by the addition of binding agents. The most common biding agent used in backfills is Portland cement. Portland cement, containing lime, iron, silica and alumina components, sets and hardens in hydration reactions.

7.3 Design of Required Backfill Strength

7.3.1 Strength Design for Backfill Face Exposure

In order to maximize ore recovery, it is very common to return for mine pillar after primary ore recovery. While this is being done, large vertical heights of massive backfill may be exposed. For delayed backfill, as used in open stopping operations, the fill must be stable when free standing wall faces are exposed during pillar recovery. It is necessary that the fill has sufficient strength to remain free-standing during and after the process of pillar extraction by resisting the blast effect.

In the difficulty of numerical modeling, many mine engineers still rely on 2-dimensional limit equilibrium analyses along with calculated Factor of Safety (FoS) to determine fill expose stability. These analyses typically result in an over conservative estimate of the limiting strength which increase the cost of backfill operations. However, 2-dimensional and pseudo 3-dimensional empirical models have been developed to account for arching effects, cohesion and friction along sidewalls (Mitchell et al, 1982; Smith et al, 1982; Arioglu, 1984; Mitchell & Roettger, 1989; Chen & Jiao, 1991; Yu, 1992).

Narrow Exposed Fill Face

This design method accounts for arching effects on confined fill by adjacent side walls (Figure 7.1) using Terzaghi's vertical pressure model. Based on 2-dimensional finite element modeling, Askew et al (1978) proposed the following formula to determine the design fill compressive strength;

$$UCS_{design} = \frac{1.25 \cdot B}{2 \cdot K \cdot tan\varphi}\left(\gamma - \frac{2 \cdot c}{B}\right)\left[1 - exp\left(-\frac{2 \cdot H \cdot K \cdot tan\varphi}{B}\right)\right] \cdot FoS$$

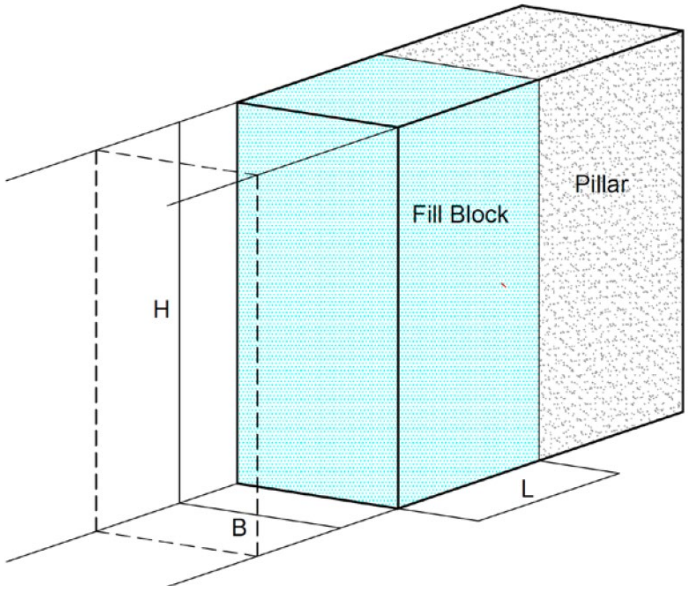


Figure 7.1 Narrowly exposed fill face mechanism

Exposed Friction Fill Face

This design refers to an exposed fill where both opposite sides of the fill are against stope walls (Figure 7.2). By assuming that there is shear resistance between the fill and stope walls due to the fill cohesion, the design UCS can be determined by the following relationship (Mitchell, 1982);

$$UCS_{design} = \frac{(\gamma \cdot B - 2 \cdot c)\left[H - \frac{1}{2} \cdot tan\left(45^{\circ} + \frac{\varphi}{2}\right)\right] \cdot sin\left(45^{\circ} + \frac{\varphi}{2}\right)}{B} \cdot FoS$$

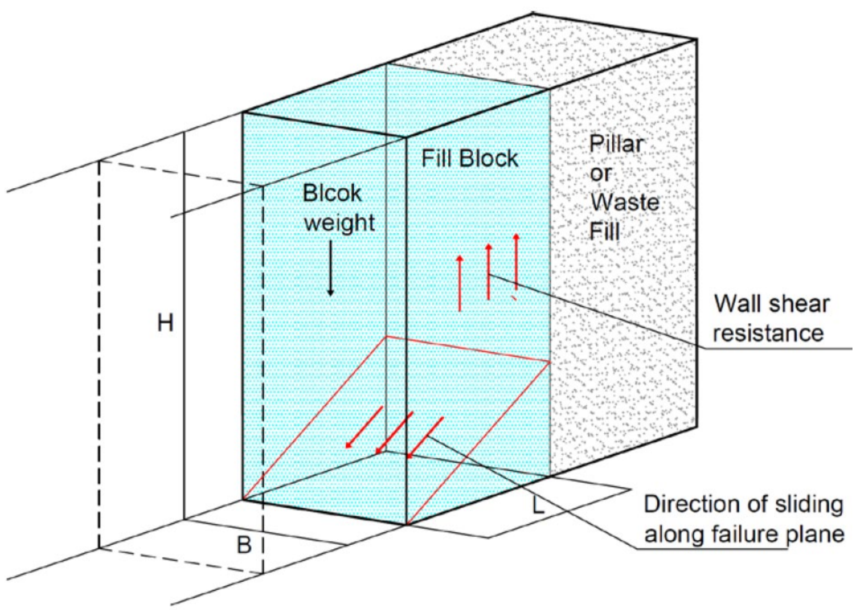


Figure 7.2 Confined block with shear resistance mechanism

Exposed Frictionless Fill Face

The compressive strength of backfill is mainly due to binding agents and any strength contributed from friction can be considered negligible for the long term (i.e. $\varphi = 0$). For a frictionless material (Figure 7.3), cohesion is assumed to be half of the UCS (c = UCS/2). Thus, the design UCS can be evaluated by the following relationship proposed by Mitchell et al (1982);

$$UCS_{design} = \frac{(\gamma \cdot B - 2 \cdot c)\left[H - \frac{L}{2}\right] \cdot \sin(45^{\circ})}{B} \cdot FoS$$

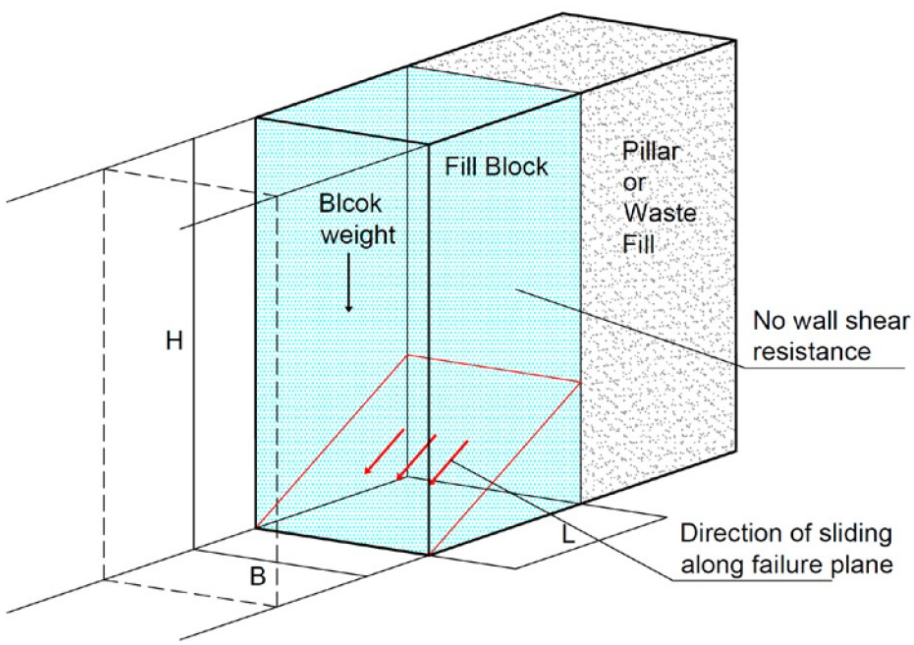


Figure 7.3 Confined block without shear resistance mechanism

Where,

B = width of stope

H = total height of filled stope

K = coefficient of fill pressure (K = 1/[1+2tan2(φ)])

C = cohesive strength of fill (kPa)

Φ = angle of internal friction of fill (°)

γ = bulk unit weight of the fill (kN/m^3)

FoS = Factor of Safety

Required Strength (UCS) for Fill Face Exposure

Required backfill strength for different width of fill face exposures were evaluated using three different relationship with different shear failure mechanism. The detailed evaluation results are shown in Appendix – G and required backfill strength for exposed fill face ranging from 4 m to 10 m are summarized in Table 7.1. The continuous longhole extraction for 4 m thick ore deposit need backfill with minimum 175 MPa of UCS but more than 400 MPa of backfill will required to place for 10 m wide longhole stope backfill.

Table 7.1 Required backfill strength for fill face exposure

Exposed Face Width (m)		4 m	5 m	6 m	8 m	10 m
Required Strength (MPa)	No Friction Shear	**175**	**218**	**260**	**343**	**422**
	Shear for Wall and Plane	157	184	208	249	281
	No Shear for Wall	158	185	209	250	282

7.3.2 Strength Design for Underhand Cut Stability

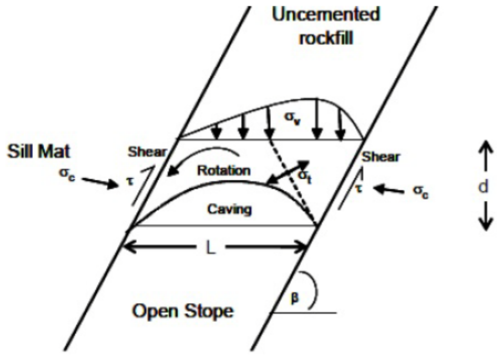
Failure modes of backfill for Underhand Developing

The methodology of span design under paste fill is complex because many different factors affect the overall stability, as shown in Figure 7.4 (a). The failure modes and combination thereof should be analyzed with respect to the cement paste properties, stope geometry, and other factors relate to filling practice, such as cold joints and gaps above not tightly filled.

For the underhand cut design, Factor of Safety (FoS) against four different types of failure mode can be estimated from limit equilibrium analysis summarized by Mitchell (1991) and illustrated in Figure 7.4 (b).

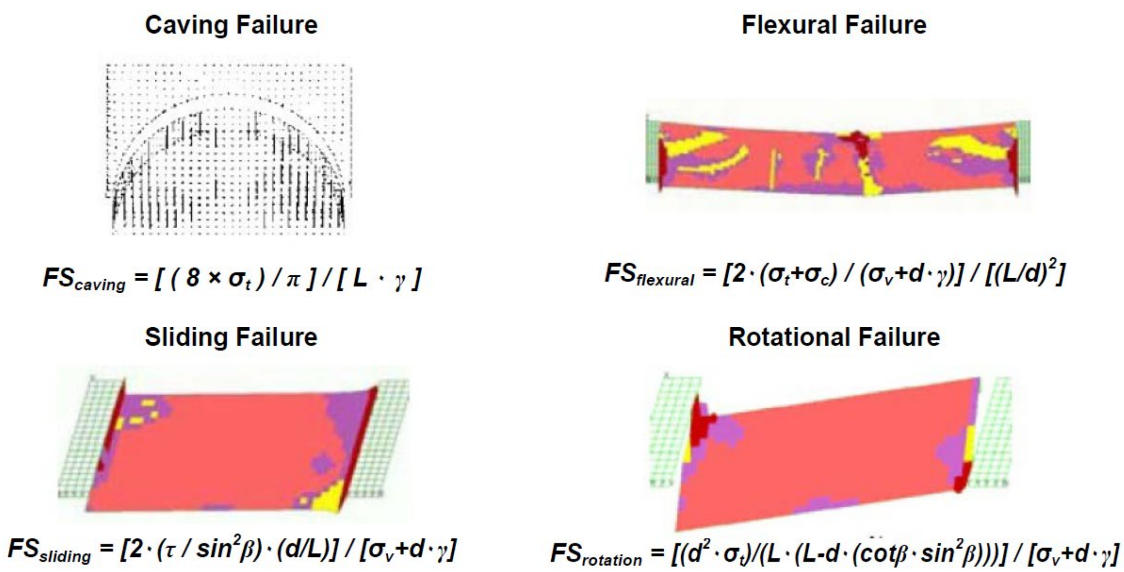
Caving failure would occur when the unsupported weight of backfilled sill material exceeds the tensile strength of the material. The caving is assumed to extend to a semi-circular arch shape defined by L/2 where L is the undercut span. This failure is assumed to be related only to the self-weight of the material, independent of external loadings. Other than the sill drive geometry, the assumed tensile strength of the material is the critical factor to consider in this analysis.

Flexural failure would occur when the moments due to bending of the sill mat under its self-weight plus the vertical stresses applied to the sill exceed the moment capacity of the sill material. Following this analysis, the tensile strength of the material and thickness of the sill would provide the main resistance to flexural instability.



L : Span of the underhand-cut stope
γ : Unit weight of paste fill
σ_t : Tensile strength of the cement fill
d : Thickness of paste sill
σ_c : Horizontal confinement (assumed zero – conservative)
σ_v : Vertical stress above paste sill (uncemented rockfill)
τ : Shear strength along fill and wall contact
β : Stope wall dip angle

(a) Schematic showing typical failure mode after Mitchell (1991)



(b) Limit equilibrium analysis of typical failure modes

Figure 7.4 Limit equilibrium criteria developed by Mitchell (figure from Pakalnis et al. 2005)

Sliding or shear failure along the sill mat abutments would occur when the weight of the backfill material, in combination with the vertical loads emplaced on the sill mat, exceed the shear strength of the paste material. For the assessment of UCS against sliding, shear strength (τ) is defined by initial failure strength of UCS test.

Rotational failure strongly depends on backfill thickness (d) as shown in Figure 7.4.

Minimum required backfill strength for underhand cut from 3 m to 10 m wide were estimated against four different failure modes and summarized in Table 7.2. 4.0 m of sill thickness and 1.5 of FoS were applied for the analysis to evaluate design backfill strength for underhand cut developments with Cut & Fill mine method. From the analysis results it is noted that minimum 540 KPa of backfill strength will be required for 3.5 m span underhand cut and the strength must achieve more than 1700 KPa for the production drift underneath 10 m wide backfill span (Table 7.2).

Table 7.2 Required backfill strength for underhand cut drives (FoS = 1.5)

Backfill Span (m)		3.5	4	5	6	8	10
Design Strength against Failure Modes (KPa)	Caving	540	615	765	920	1225	1530
	Flexural	35	100	150	210	375	590
	Sliding	310	410	510	615	820	1020
	Rotational	115	210	430	620	1100	1700

8. GROUND CONTROL PROGRAM - IMPLEMENTATION

8.1 Risk Assessment and Hazard Identification

Risk assessment and hazard identification involves the systematic examination of any activity, location or operational system. The risks and hazards are identified and the likelihood and potential consequences of an event are reviewed so that planned approaches to manage the risk exist. This GCMP should be re-assessed and updated by an authorized person or group on an annual basis, or before any major change is made to the mine design, method, or equipment used. It should be made available for examination, in conjunction with the mine design, on request by any relevant parties.

8.2 The Mines Act and Other References

This GCMP should be read and implemented within the context of the prevailing legislative framework (as defined by "The Mines Act"), industry-accepted best practice, and Health and Safety policies, guidelines, and targets, as amended from time-to-time.

Some of the most important sections of the Mines Act which deal specifically with ground control should be reiterated. And they deal with the *"examination of workings"* and the *"daily examination and report book"*:

Examination of Workings

- All active workings shall be examined by the certified shift boss or supervisor with assigned responsibility to ascertain that they are in a safe working condition, as often as the nature of the work necessitates.
- All persons working underground shall have their work areas inspected by a shift boss or supervisor at least twice per shift.

Daily Examination and Report Book

- The person making the examination shall record all unusual and hazardous conditions and corrective actions taken or proposed in a daily examination and report book and sign the report as a record of the conditions found. For underground mines the record shall include a report on each working place examined.
- The report shall be read and countersigned by the corresponding supervisor on the oncoming shift and the unusual and/or hazardous conditions discussed with the workers before they are permitted to resume operations in the areas indicated in the record.
- In addition, all mining personnel are responsible for recognizing poor ground conditions in active headings and notifying supervision so appropriate action can be taken.

The miner(s) assigned to specific work areas are responsible for examining and testing for loose ground. The miner(s) assigned to a specific work area shall examine and, where applicable, test ground conditions in areas where work is to be performed, prior to work commencing, after blasting, and as ground conditions warrant during the work shift.

8.3 Communication

A communication process that ensures a two-way flow of information between operations and mine management shall be fostered.

8.3.1 Communication Process

The process shall ensure that;

- Operators are provided with an understanding of expected conditions, anticipated support, mining procedures and any relevant changes in support design prior to implementation.
- Personnel are aware of typical warning signs which suggest that installed support may be inadequate and need review.
- Close communication exists between all members working under the GCMP.
- Management has an early opportunity to respond to unexpected mining conditions and/or support system behavior.

Communication channels may include;

- Geotechnical Daily Logging Book
- Start of roster meeting
- Underground inspections
- Daily/weekly planning meeting
- Support rules and drawings
- Plans and sections
- Shift reports
- Toolbox meeting

- Safety meetings
- TARP's and work procedures
- Tell tale and other monitoring forms
- Incident reports
- Inspection checklists

8.3.2 Non-conformance and Corrective Action

Treatment of non-conformances and corrective actions under the GCMP will be in accordance with the framework defined bellow;

- Identification and notification of non-conformances
- Documentation of non-conformances using the relevant Keno Hill Bermingham mine forms (Appendix-H)
- Identification of potential corrective actions that may be applied
- Determination of required corrective actions (taking into account impacts of change including potential additional hazards and effects on other operations)
- Allocation and recording of responsibilities and target dates for completion of corrective actions
- Monitoring and review of non-conformances and progress of completion of corrective actions (generally conducted at Monthly Planning Meetings, additionally as required or warranted)
- Record of completion and closure of corrective actions by responsible person
- Storage of records

The Geotechnical Engineer shall maintain a Ground Control Non-conformance register.

8.3.3 Identification of Non-conformances

Non-conformances may be identified through means including;

- Observations and inspections by Alexco Underground personnel, contractors, consultants and visitors
- Monitoring of ground control performances
- TARP
- Incidents and incident investigations
- Internal audits (including systematic and non-systematic audits by Alexco technical staffs, materials and equipment suppliers and routine inspection)
- External audit typically done by 3rd party consultants (including by the Mining Inspectorate and systematic periodic audit)

Non-conformances will be reviewed at the Monthly Planning Meetings.

8.3.4 Corrective Action

The adequacy and effectiveness of corrective actions, allocation of responsibility, target completion date and progress towards completion will be reviewed and adjusted as appropriate/required at the Weekly Planning Meetings.

8.4 Monitoring

8.4.1 Ground Inspections

Routine ground inspection needs to be conducted by miners, supervision and technical staff. Additionally, quality testing of ground support will be conducted to determine the effectiveness of installations in supporting the ground. Internal reviews of standards need to be conducted to ensure applicability of ground support standards to evolving conditions as the mine matures.

The routine ground inspections, which should be conducted on a daily basis, are part of the "workplace inspection" each miner should conduct prior to the commencement of work. The supervisors need to verify that the workplace inspection has been done by the workers, miners, and would also need to inspect headings themselves.

On a weekly basis, the main travel-ways and haulages need to be inspected by both supervisors and technical staff.

8.4.2 Ground Control Logbook

A single book and set of plans that provides a record of ground control related issues, falls-of-ground (FOG), incidents/accidents, remedial measures, etcetera, needs to be kept. It enables easy review during meetings and at times when a single repository of information and data is required – but mostly, it ensures that ground control is adequately addressed at all levels of the organization. The regularly updated plans can be posted in the start-of-shift meeting areas for reference and discussion.

8.4.3 Overbreak Measurement Program

Overbreak tolerances of 15% by volume are considered good in most operations. Within the vein it will be critical to limit the overbreak as much as possible to avoid increasing the excavation span. With best practices, drilling and loading overbreak and loosening of the ground surrounding the excavation can be minimized.

Mining faces under geological control should be clearly delineated by the mine geologists prior to the face mark-up and drilling. The face should be photographed to record the geologists" decisions before the rock-face is worked on for the advance.

8.4.4 Design Effectiveness

The overbreak evaluation program will provide effective feedback on the drilling and blasting practices. Other measurements should be considered to assess the GCMP. These measures should be ones that can be readily collected and are meaningful, for example; rehabilitation requirements, excavation deformation measurements, shotcrete cracking, accidents/incidents, FOG, fill dilution, and so on.

8.4.5 Ground Support Quality Assurance and Quality Control (QA/QC)

QC is a critical part of ground control – for which a stand-alone guideline will be developed and used within the context of pre-existing SOPs.

Major points covered in the QA/QC process, apart from the individual support members" quality and installation procedures, include tangible means to achieve solid ground control – and they are:

- Installed rebar rockbolts, Swellex and split-set friction anchors should be randomly tested to ensure consistent. Effective installation methods are practiced at all times in mining operations. Drill-bit sizes should be reviewed daily by the Shift Boss or equipment operator, to ensure that the required drill-hole size is achieved.
- Testing of support elements should be performed monthly. On these occasions, 1 % of total installed rebar and each of the various FSA"s (Friction Support Anchors) should be pulled from random sites in the mine. Over the first three months of mining, the quantity of support installation, support unit performance and excavation performance, should be evaluated on a weekly basis. This will enable the short-term assessment of the suitability of the proposed (and implemented) support units.
- If a new type of bolt is planned for use and/or ground conditions have changed, additional pull tests are required. These should be undertaken both in the back and in the sidewalls, in the range of ground conditions in which the bolts are being proposed to be used.
- Records of all tests should be documented and maintained. These reports should be distributed to appropriate personnel for review and submitted for remedial and/or corrective measures where required – in a way that reflects the urgency of the case in-hand.
- A documented bi-annual inspection should be instituted in which the corrosion of splits-sets (and other steel elements) are monitored during the life of the excavations.
- Rehabilitation should be completed in areas in which the support capacity (of the original support units) does not meet, or is unable to adequately support, the required life-of-opening expectation. Rehabilitation with rebar support elements should be done in these instances.

8.5 Review

8.5.1 Conforming to Regulatory Requirements

Regulatory requirements should be adhered to on all fronts, on a daily basis. Ground support materials employed at the mine should conform to the Canadian Standards Association specification, as detailed in "CAN/CSA-M430-90 (R2007) Roof and Rock Bolts, and Accessories". Daily workplace inspections should be carried out, and the main haulages and travel-ways should be inspected weekly (or more regularly if weaker ground conditions or excavation performance warrants it).

8.5.2 Examination of Ground Conditions

All underground workers should be trained in the examination, and testing, for loose or unsafe ground conditions. This should occur prior to work commencing, after blasting, and at any time during the work shift if ground conditions change.

Underground haulage and travel ways, surface area high walls, and banks adjoining travel ways need to be inspected weekly or more often if ground conditions change.

8.5.3 Re-evaluate Failure Modes and Update Risk Management Studies

As experience is gained in the mining of the access-excavations and the extraction of the ore deposit, the potential modes of failure, the ground control practices, and the mining approach should be re-evaluated. They should be adjusted to reflect the increased understanding of the rock mass and its behaviour. The re-evaluation may precipitate an amendment to the base assumptions upon which the ground control design was built which may, iteratively, affect the minimum ground control standards for those conditions. The re-evaluation should be conducted annually.

8.5.4 Peer Review of Standard Work Practices

Ground control implementation guidelines should be made available for discussion, review, and comment, by any person at any time. This dialog will ensure the applicability of the various work standards as they apply to the installation and performance of ground support at the mine. A formal peer review of the standard work practices should be conducted on an annual basis.

8.5.5 External Review of the GCMP

The mine should provide for an external audit of the GCMP to be conducted annually. The overall plan should be revised and-or amended on an as needed basis and as conditions change. Any of these revisions should be vetted and „signed-off" by a qualified geotechnical professional.

9. ONGOING DATA COLLECTION

Once mining commences, a formal geotechnical data acquisition program needs to be invoked that includes excavation mapping, geotechnical logging and excavation and support performance monitoring. From these processes, support designs, excavation and stope dimensions are modified to reflect any changes in ground conditions as the extraction of the deposit progresses.

9.1 Diamond Drill Core Logging

Geotechnical information collected from core logging forms the basis for recommendation of appropriate mining methods, ground support designs, and stable mining geometries prior to (and during ongoing) mining excavation. Without these data, accurate rock mass quality estimates are difficult which leads to either a high risk of failure (economic and-or mining excavation) or a very conservative design methodology. For the purposes of mine development and sustainment, core logging for engineering parameters should be done for at least the amount of core suggested in Table 9.1.

Table 9.1 Suggested percentage of cored bore holes geotechnical logging

Stage of Mine Development	Suggested Percentage Logged
Feasibility Study	100 %
Operating Mine	35 % - 75 %

Geotechnical Logging of Boreholes

Geotechnical logging of boreholes´ rock-core (for engineering parameters) should, where possible, be a representative sample of the FW, ore, and HW conditions (and country or host rocks). This enables a balanced view to be formed on the inherent variability in the ground conditions and allow recognition and delineation of discrete geotechnical domains.

Geotechnical Logging Code

The diamond-drilled rock-core should be logged for the following main parameters used in the calculation of RQD, Q", RMR, and other measures of rock mass quality or condition.

Basic Geotechnical Parameters

- Total Core Recovery (TCR)
- Magnetic Susceptibility
- Orientation Comment
- Rock Quality Designation (RQD)
- Orientations Offset
- Notes: Other Geotechnical Observations

Detailed Geotechnical Parameters

- Intact Rock Strength (IRS) Strong
- Intact Rock Strength (IRS) Weak
- Percent Weak IRS Material
- Total Discontinuity Features : All
- Total Foliations Total
- Open Joints Foliation
- Angle: Alpha Foliation
- Angle: Beta number of Joint sets
- Joint Set Definition: Core Axis Angle, Roughness, Alteration, Fill Comments, geotechnical observations

Structural Geology - General

- Location
- Description/Quality
- Total Joints
- Alpha, Beta, Gamma Angles

Core Orientation

- Depth
- Feature Type
- Roughness
- Alteration
- Fill
- Oriented Structural Features
- Confidence
- Notes: General structural observations

Point Load Test

- Location
- Core Size
- Test Diameter
- Foliation Orientation
- Guage Roughness
- Failure Mode
- Test Quality
- Comments: General Observations

9.2 Geotechnical Mapping

During the mine development cycle, recognized ground control concerns should be addressed immediately. This is achieved mostly by the operator knowledge base, engineering design work applied to obtain a required profile, and the inherent (and post-excavation) stability of the rock mass.

Before any cycle begins, however, the design process involved for development headings should consider a range of aspects, for example:

- Geotechnical mapping requirements (for the building of an accurate geotechnical model of the rock mass)
- Geological and geotechnical domains of the area and local rock mass
- Engineering design process used for the profile, drilling, explosive selection, charge up and sequencing of rounds
- Geotechnical methodology used, and assumptions used, for determining the ground reinforcement and support of the heading

- Type, method, and timing of the support and reinforcement installation
- Engineering practice as applied on the site‟s excavations (blasting, ground support)
- Steps undertaken if the ground conditions vary from expected conditions, and remedial measures available
- Operator observations of installed systems, and effectiveness
- Operator training, and commitment to following and improving procedures
- Established and well-used communication channels which effectively relate ground conditions and-or work quality to those in a position to quickly implement remedial action.

Geotechnical Mapping Requirements

Geotechnical mapping records features of the rock mass which may influence the stability of an excavation, in both the short and long-term. These factors include:

- Representative face and sidewall photography and sketch-maps of significant features Intact strength of the rock, both estimated and measured
- Orientation, spacing, persistence, roughness, aperture, infill-type and shear strength of the mapped discontinuities
- The visible effects of water on the discontinuities and intact rock.

The amount of detail (or „resolution‟ of survey and mapping) required, depends on a number of factors, which are all related to the ultimate use of these data:

- Rock mass structure and fabric
- Analysis method,, and the resolution of engineering application, for example; local (heading), regional (mine), and so on…
- Level of refinement, or the number of iterations, used in the analysis.

Types of Geotechnical Mapping

The various techniques used for structural mapping of a rock mass can be divided into three main categories:

- Spot and-or face-mapping;
- Lineal mapping, which is an effective „fast-sampling‟ method
- Window mapping at pre-determined or random exposures

The objective and ultimate use for the data, as well as the mapping method employed, dictate the required amount or sample density of the data to be collected. In situations where fault-structures are not obvious or easily discernible from the available rock exposure and/or rock cores, the sample data sets should aim to record sufficient data to readily discern the fault from within the background random or ordered discontinuity suites.

The choice of mapping method to use depends on the extent of the exposed rock face and the ultimate use to which the data will be applied. The advantage of window over lineal mapping is that it reduces the sampling bias due to discontinuity orientation, as well as requiring less rock-exposure for a statistically significant result. It also provides a better representation of the trace-length distribution. A disadvantage is that a measure of the spacing distance between two very widely spaced discontinuities may be under-represented in the data set. Notwithstanding these limitations, this type of mapping should be conducted for each stope in all mining zones.

9.3 Deformation Monitoring

If higher risk local situations or areas are noted, they should be monitored for signs of deformation, and the results used to assess the potential for instability. Measurement of sidewall and back displacement in stopes and drifts should be undertaken using industry standard geotechnical instrumentation technology wherever possible. Tunnel displacement monitoring stations should be installed in higher risk areas. This monitoring will facilitate the development of ground reaction curves, and in so doing, will enable the suitability analysis of the existing support systems.

10. INCIDENT INVESTIGATION

Following the occurrence of an incident related to uncontrolled ground movement, general priorities will be;

- Removal of personnel from positions of potential harm
- To eliminate hazards sufficiently to enable safe recovery or treatment of injured personnel
- Investigation, data collection and reporting
- Securing the back and walls
- Recovering equipment and resumption of development/production

Alexco Keno Hill Health and Safety guidelines provide guidance as to responsibilities, communications, reporting and other requirements for incident investigation (Appendix-A).

Incidents will be reviewed at Special Meetings, Safety Meetings and Monthly Planning Meetings.

10.1 Guideline for Incident Investigation

Appropriately experienced personnel will be used in incident investigation. Consideration should be given to whether external opinion or other particular skills are also required.

Records of all investigations, including associated analysis, conclusions, recommended actions and action completion will be maintained by the Geotechnical Engineer.

As relevant, ground control incident investigation may include;

- Inspection of the incident site
- Photography and sketches of the incident site
- Soliciting of verbal and written statements from personnel involved in the incident

- Soliciting of verbal and written statements from personnel associated with the incident (e.g. Supervisor, Shift Supervisors, Leading Hands, Operators)
- Compilation of a chronology of events
- Review of equipment and materials in use
- Assessment of compliance with the GCMP
- Review of data
- Review of design
- Back analysis
- Review of ground support design or operating practice
- Review the GCMP

10.2 Incident Statutory Reporting Requirements

Ground related incidents will be reported to the relevant authorities by the Mine Manager or designated personnel as required by the appropriate regulations.

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APPENDIX – A.

RISK EVALUTION TABLE

AND

ASSESSMENT MATRIX

Team-Based Risk Assessment – Consequence, Exposure & Probability Risk Evaluation Tables

	CONSEQUENCE						SEVERITY FACTOR
	Financial	Compliance	Reputation	Communities Impact	Health and Safety	Environment	
C1	>$100M one off or NPV, or >$40M annually	Potential jail terms for executives. Very high company fines. Operations suspended or severely reduced by authorities. Loss of water licence and/or forfeiture of land lease.	Extended and widespread international condemnation.	Total social breakdown, significant damage to highly valued cultural objects or structures Irreparable and prolonged impact	Multiple fatalities; multiple cases of fatal chronic disease	Massive widespread, irreversible environmental damage. Could close mine permanently	100
C2	$20M - $100M NPV, or $8M - $40M annually	Major regulatory breach; potential for severe fines and prosecutions; Multiple, serious litigation.	Serious public or media attention with international coverage Alexco CEO exposure	Very serious social impacts ; Irreparable and widespread	Single fatality, Quadriplegia, paraplegia; fatal chronic disease	Significant, local, irreversible impact; likely short-term mine closure	50
C3	$5M - $20M NPV, or $2M - $8M annually	Potential for significant prosecution and fines. Very serious litigation, including class action.	Serious national media, NGO attention and public concern. Product Group CEO exposure	Significant social impacts and/or damage to culturally significant objects.	Serious permanent disabling injury or disease eg. blindness	Potential prosecution/ conviction. Negative perception. Significant but reversible	25
C4	$1M - $5M NPV, or $400K - $2M annually	Major breach of regulation; Potential for major fines; Major litigation or major legal issue.	Significant adverse national media, public and NGO attention. Alexco Managing Director exposure	Ongoing social impacts and damage to culturally significant objects. Major non-compliance with PA's or SEMA. Mostly reparable	Serious disabling injury. (Rehabilitation required) Loss of an arm or leg. Noise induced hearing loss	Non or compromised compliance with environmental obligations; generally reversible impact	10
C5	$100K - $1M NPV, or $40K - $400K annually	Serious internal non-compliance; serious regulatory breach; prosecution with moderate fines; Potential for investigation or report to authority.	Attention from media and/or heightened concern by local community. Criticism by NGOs; DDMI General Manager exposure	Medium term social impacts on local community. Serious non-compliance with PA's. Mostly reparable	Loss of a finger, broken leg or arm, asthma (e.g. LTI >2 wks)	Serious degradation or harm to environment but reversible.	5
C6	$20K - $100 NPV, or $5K - $40K annually	Minor legal issue, minor infraction of regulation; no fines (warning), no litigation.	Minor adverse local public or media attention and complaints. Alexco Manager exposure	Minor impact to social structures. Minor non-compliance with PA's. Fully reparable	Medical treatment injuries or illness (e.g. MTI or LTI <2 wks)	Minor impact requiring regulatory reporting	1
C7	$5K - $20K NPV, or $2K - $5K annually	Minor non-compliance with internal policy.	Public concern restricted to local complaints. Alexco manager issue	Very minor impact. Fully reparable.	Minor medical/first aid treatment eg. Dust in eye (no MTI/LTI)	Nuisance only; minimal impact	0.5

2. EXPOSURE TO THE RISK		
LEVEL	**EXPOSURE DESCRIPTION**	**S.F**
E1	Continuous or several times per day or several employees once per day	**10**
E2	Approximately once per day	**6**
E3	Once per week to once per month	**3**
E4	Once per month to once per year	**2**
E5	Once a year to once every ten years	**1**
E6	Rarely, but it has been know to occur	**0.5**
E7	No exposure identified	**0.1**

3. PROBABILITY OF OCCURRENCE OF UNWANTED EVENT		
LEVEL	**PROBABILITY DESCRIPTION**	**S.F**
P1	Always 90% to 100%	**10**
P2	Frequent 51% to 90%	**9**
P3	Common: heard of it happening a number of times 30% to 50%	**5**
P4	Probable – Have heard of it happening 11% to 30%	**3**
P5	Possible – Could happen 6%to 10%	**1**
P6	Unlikely 1% to 5%	**0.5**
P7	Extremely Unlikely (less than 1%)	**0.1**

Risk Evaluation

Risk Rating = 1 **Consequence** x 2 **Exposure** x 3 **Probability**

DDMI Risk Rating	Risk Level	YZC Risk Determination	Action	Minimum Notification and Accountability
>3000	**Extreme**	Class V	Risks that significantly exceed the risk acceptance threshold and need urgent and immediate attention	President / COO
1501 - 3000	**Very High**	Class IV	Risks that exceed the risk acceptance threshold and require proactive management	General Manager / VP Responsible
501 - 1500	**High**	Class III	Risks that exceed the risk acceptance threshold and require proactive management	General Manager
101 - 500	**Moderate**	Class II	Risks that exceed the risk acceptance threshold and require review of controls and required mitigations.	Department Manager
0 -100	**Low**	Class I	Risks that are below the risk acceptance threshold and do not require active management	

Flame & Moth Underground Mine Project Risk Ranking Matrix for Job Hazard Analysis

		PROBABILITY					RISK ASSESSMENT CATEGORY
		A	B	C	D	E	
CONSEQUENCE	1	1	2	4	7	11	CRITICAL
	2	3	5	8	12	16	HIGH
	3	6	9	13	17	20	MODERATE
	4	10	14	18	21	23	LOW
	5	15	19	22	24	25	

Potential sequence and probability details

Potential **CONSEQUENCE** of the incident	
1	Could kill, permanently disable or cause very serious damage
2	Could cause serious injury (major LTI) or major damage
3	Could cause typical MTC / LTI or moderate damage
4	Could cause First Aid injury or minor damage
5	Could not cause injury or damage

PROBABILITY of this occurring again	
A	ALMOST CERTAIN to happen
B	LIKELY to happen at some point
C	MODERATE, POSSIBLE, it might happen
D	UNLIKELY, not likely to happen
E	RARE, practically impossible

Flame & Moth Project Risk Assessment

Green	Minimum impact – Work your plan
Blue	Some disruption – Re-evaluate the control measures in order to reduce the overall risk
Yellow	Unacceptable major disruption likely – Re-evaluate the control measures with the Supervisor. Determine lower risk options
Red	Unacceptable major disruption likely – Job sut down for re-evaluation with Superintendent and the job team

Keno Hill Flame and Moth Project Priority of Risk Controls

1. **Elimination** – Controlling the hazard at source
2. **Substitution** – Replacing one substance or activity with a less hazardous
3. **Engineering** – Installing guards on machinery
4. **Administration** – Policies and procedures for safe work practices
5. **Personal Protective Equipment** – Respirators, earplugs, etc.

Flame & Moth Ground Control Risk Assessment Form

Area/Location/Activity		
Unwanted Events/Potential Loss		
Cause/s		
Impacts		
Inherent Risk	Type of Loss	
	Consequence	
	Exposure	
	Probability	
	Risk Ranking	
	Risk Level	
Controls		
Contingency		
Residual Risk	Type of Loss/Benefit	
	Consequence	
	Exposure	
	Probability	
	Risk Ranking	
	Risk Level	
Recommendations/Actions		
Who		
When		

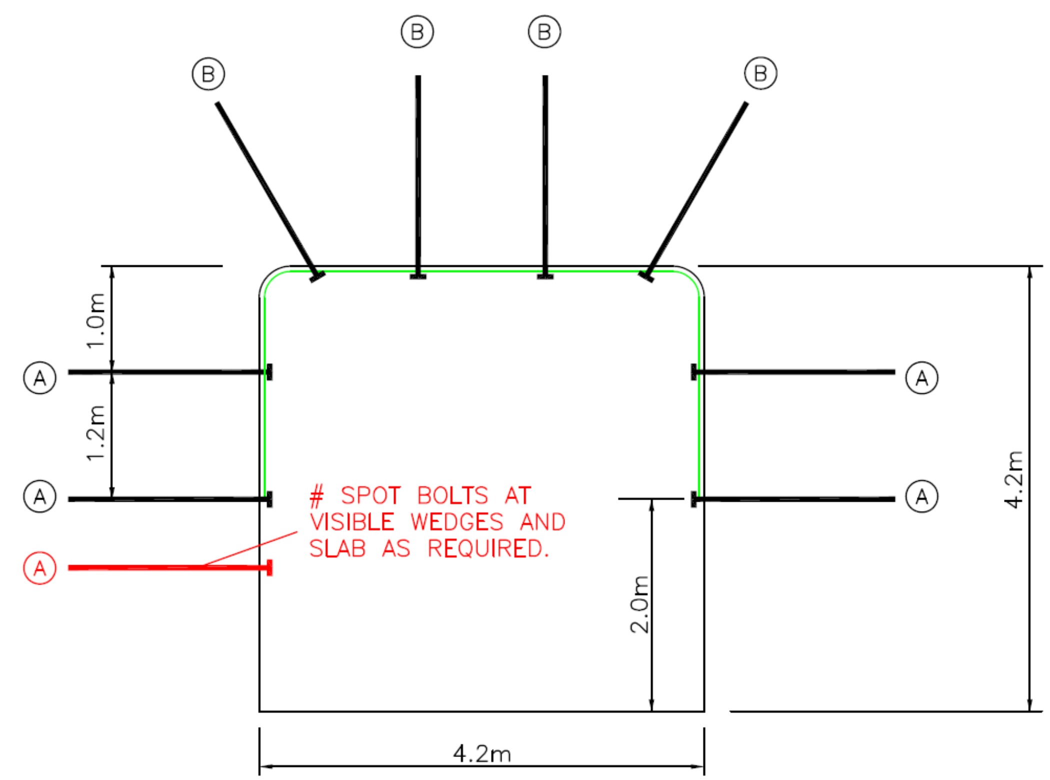
APPENDIX – B.

SPECIFICATION OF SUPPORT STANDARDS

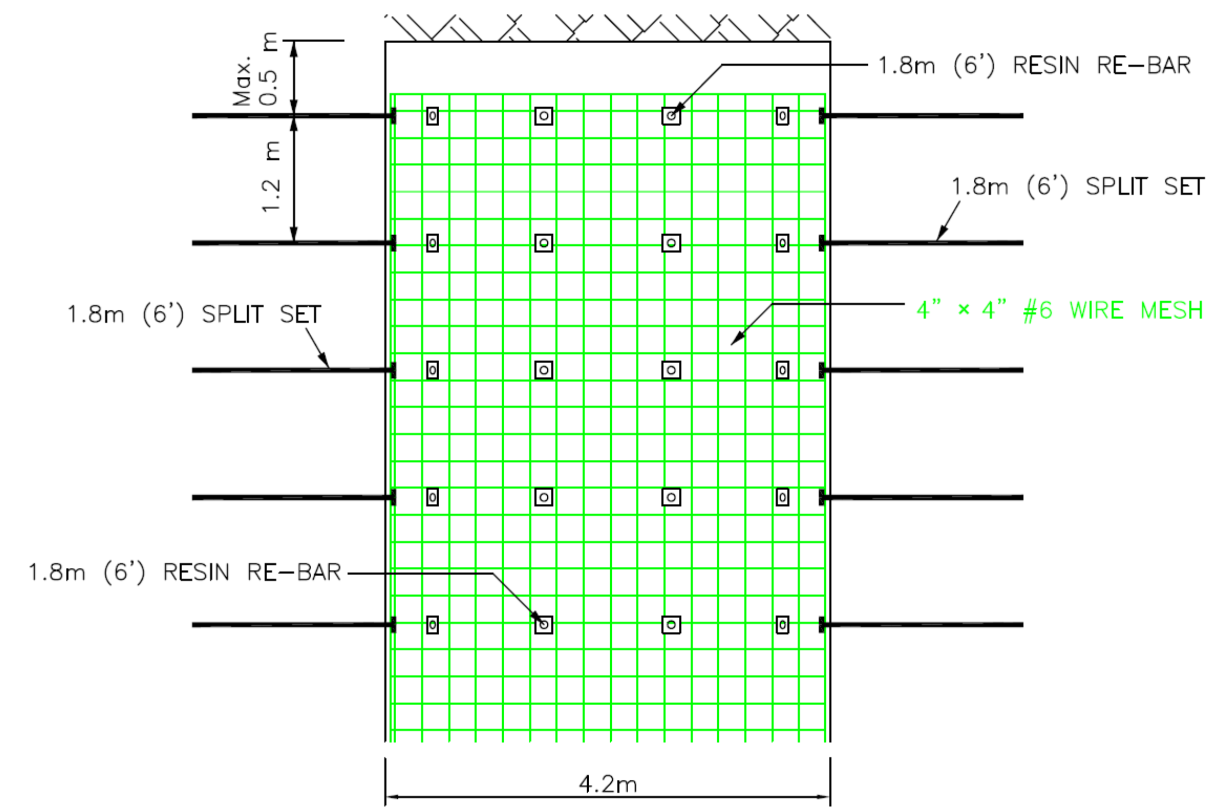
RAMP - I (4.2mW by 4.2mH)

Fair to Good Ground (60 > GSI > 45)

SECTION



PLAN



SUPPORT ELEMENTS

LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN RE–BAR	1.8m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE

No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN RE–BAR	1.8
C	12T SWELLEX	2.4
D	RESIN RE–BAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+–10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE N.T.S

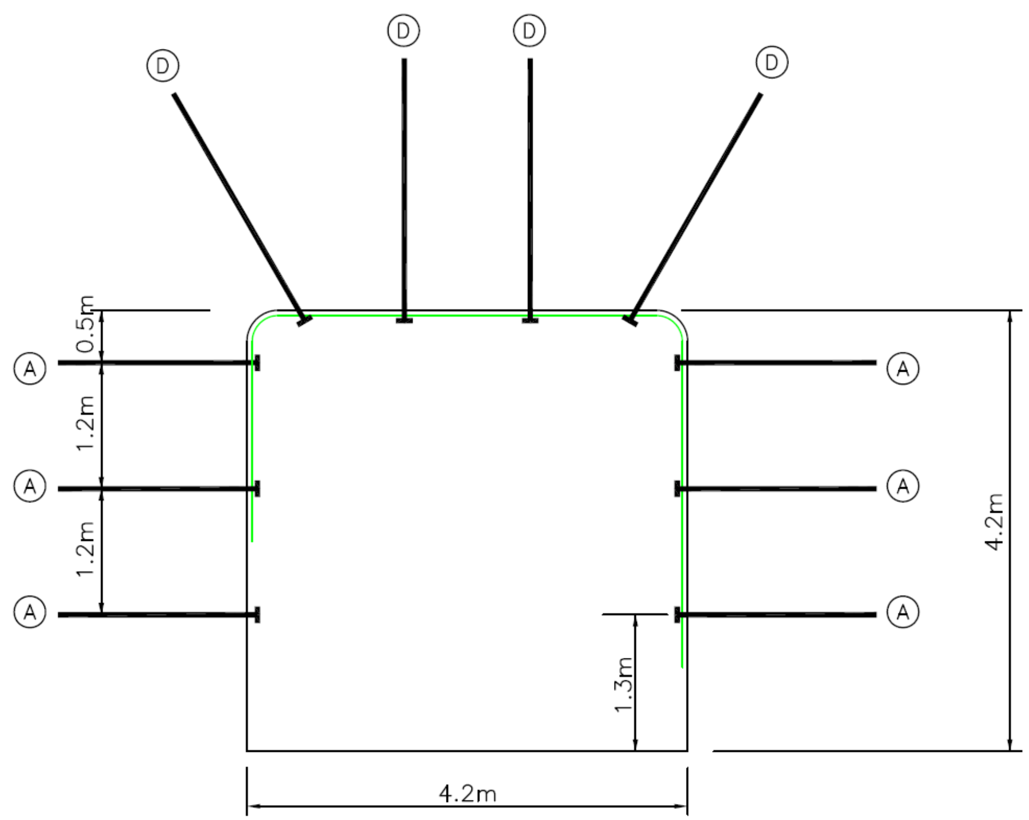
TYPE RAMP - I

FILE NAME GCMP SUPPORT STANDARDS REV. 001

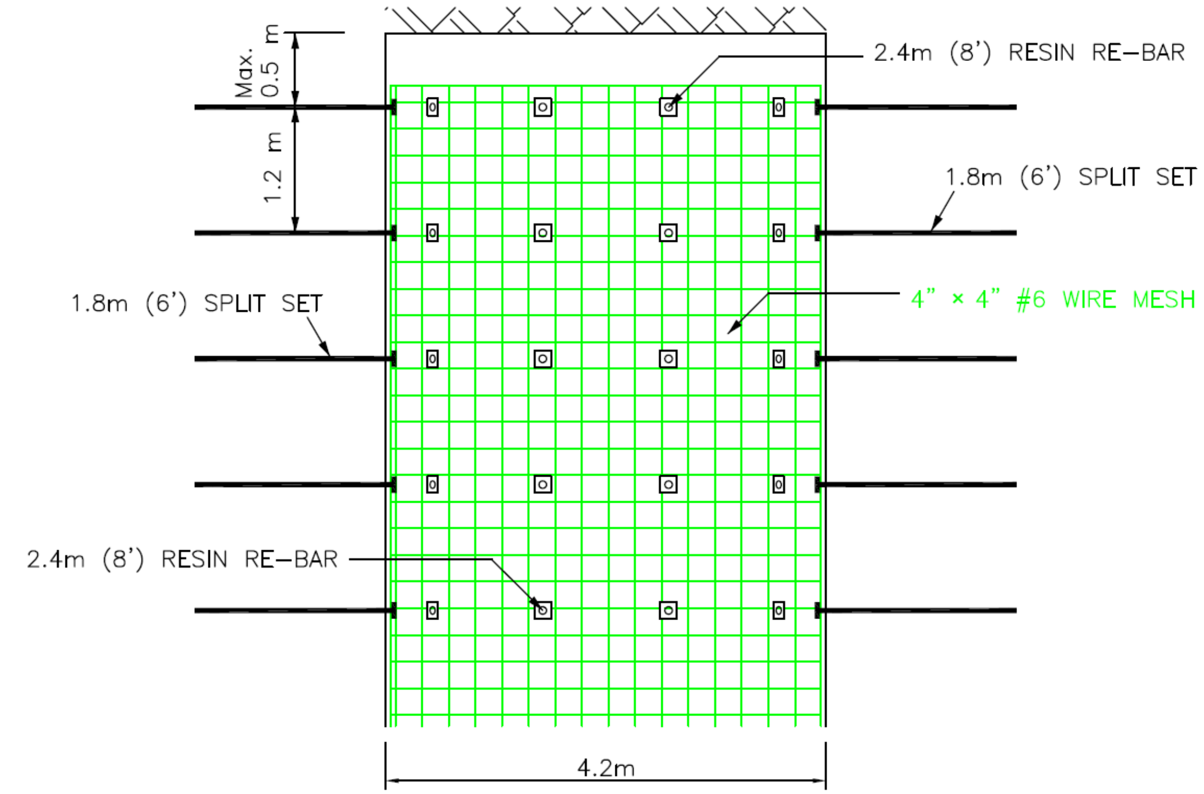
RAMP - II (4.2mW by 4.2mH)

Poor to Fair Ground (45 > GSI > 30)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN RE-BAR	2.4m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN RE-BAR	1.8
C	12T SWELLEX	2.4
D	RESIN RE-BAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE RAMP - II

FILE NAME: GCMP SUPPORT STANDARDS

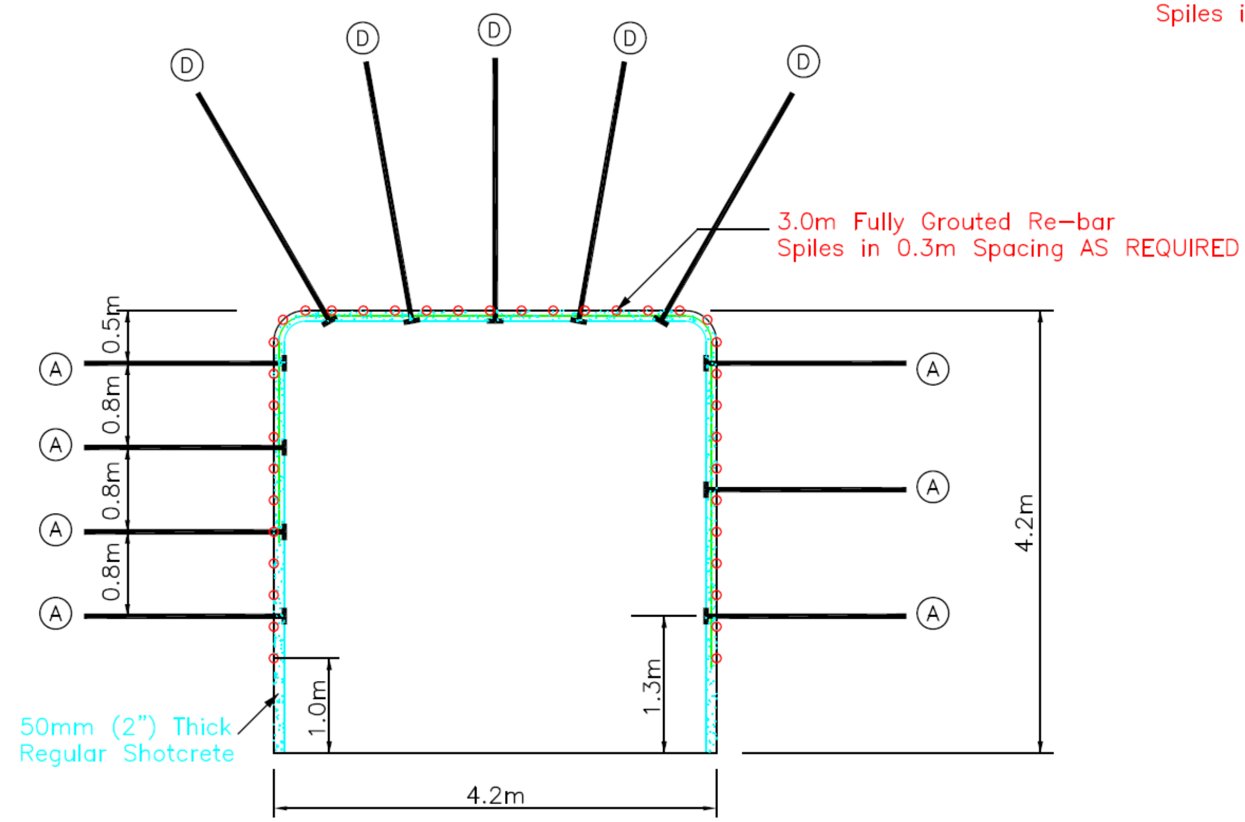
REV. 001

RAMP - III (4.2mW by 4.2mH)

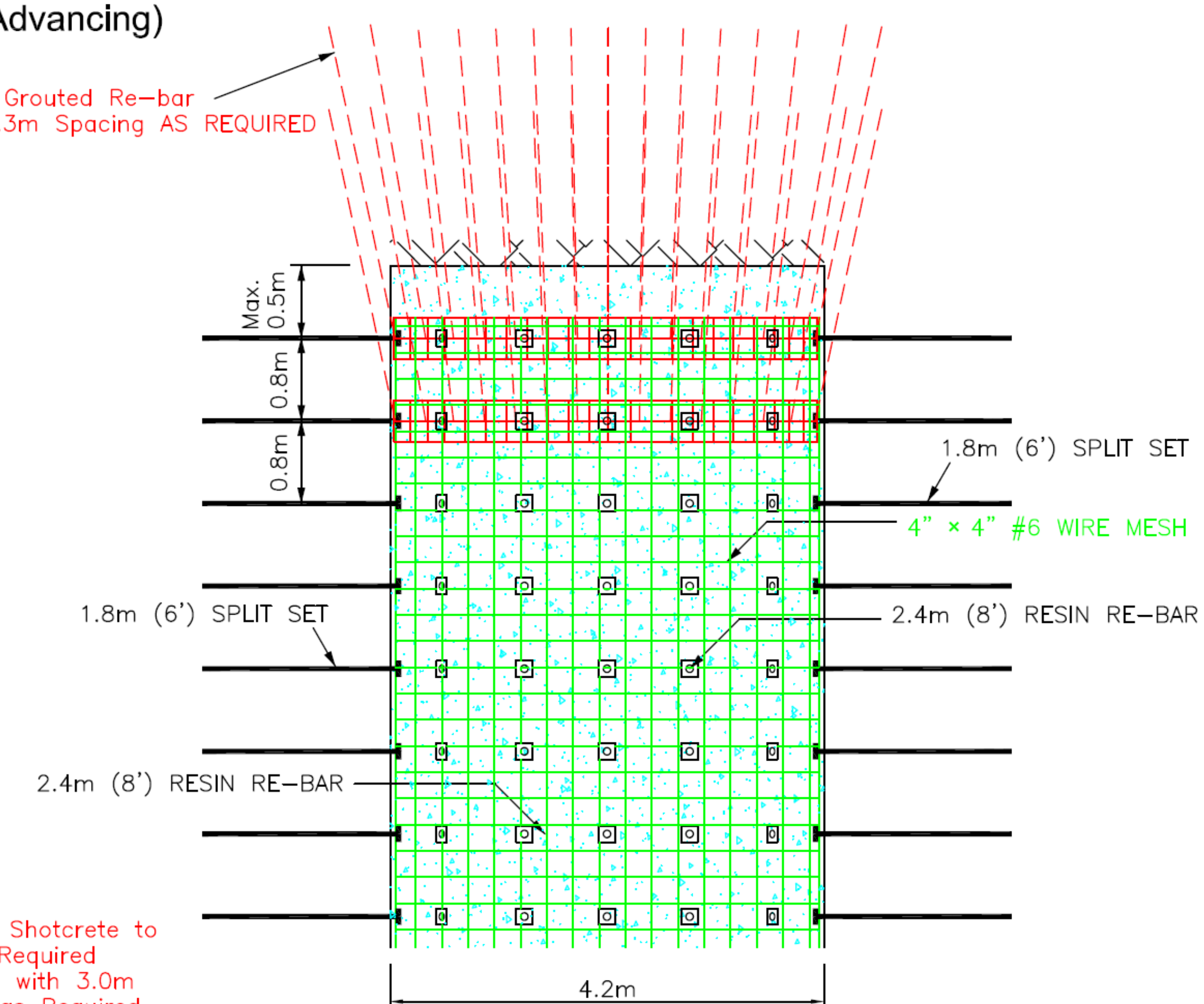
Extremely Poor Ground (GSI < 30)

(Short Round - Max. 2m Advancing)

SECTION



PLAN



– 25mm (1") Flash–Coat Shotcrete to Back and Shoulder as Required
– Spiling at 0.3m centers with 3.0m Grouted Re–bar Spiles as Required

SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN RE–BAR	2.4m	0.8mX0.8m	REG	25mm (PRE) 50mm (POST)	AS NOTED
WALLS	SPLIT SET	1.8m	0.8mX0.8m	REG	25mm (PRE) 50mm (POST)	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN RE–BAR	1.8
C	12T SWELLEX	2.4
D	RESIN RE–BAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+–10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:	
Checked By:	
Drawn By:	
SCALE:	N.T.S

TYPE RAMP - III

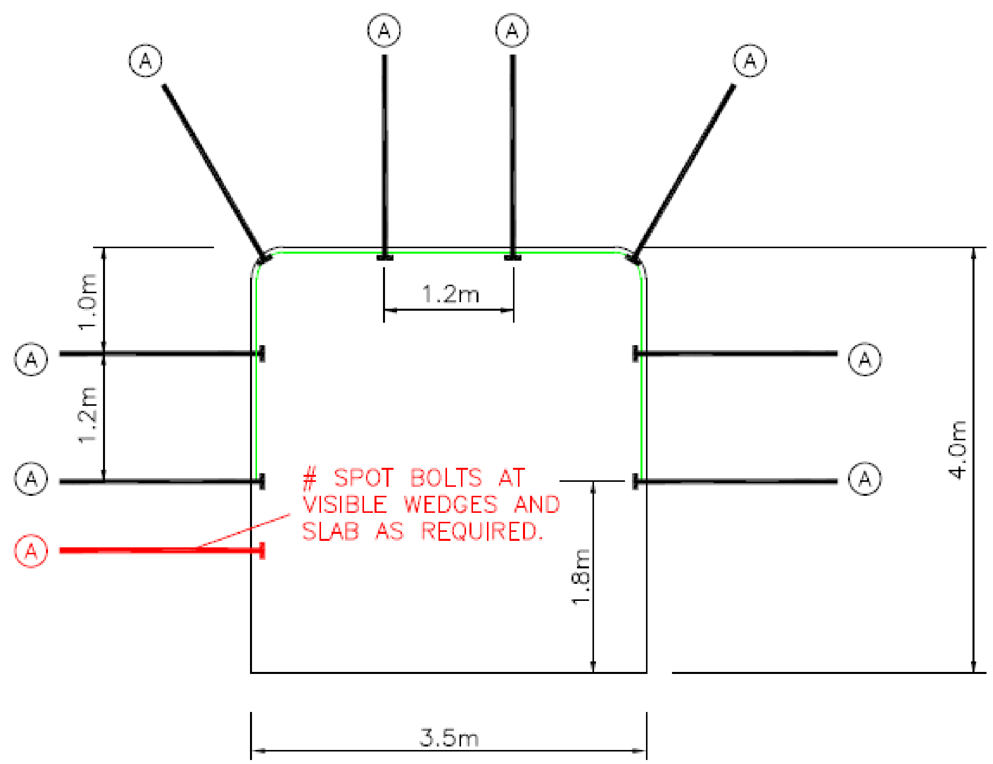
FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

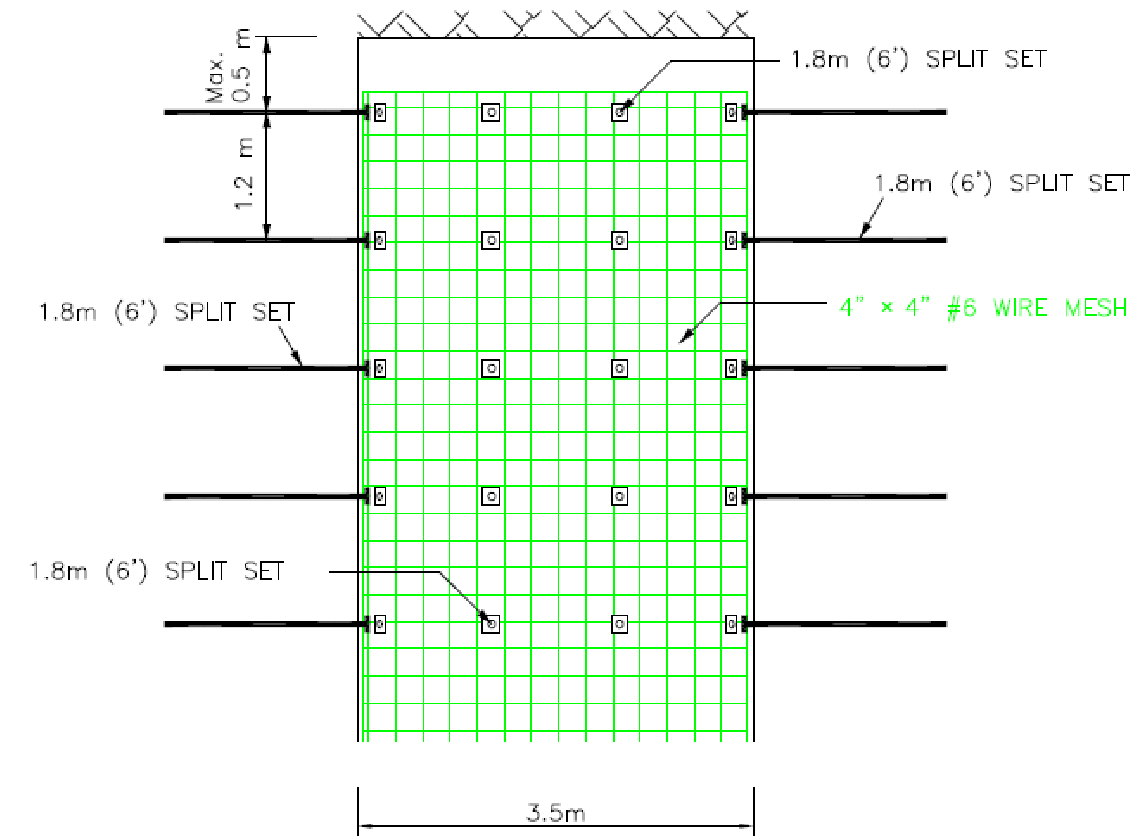
MD - I (3.5mW by 4.0mH)

Fair to Good Ground (45 < GSI < 60)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

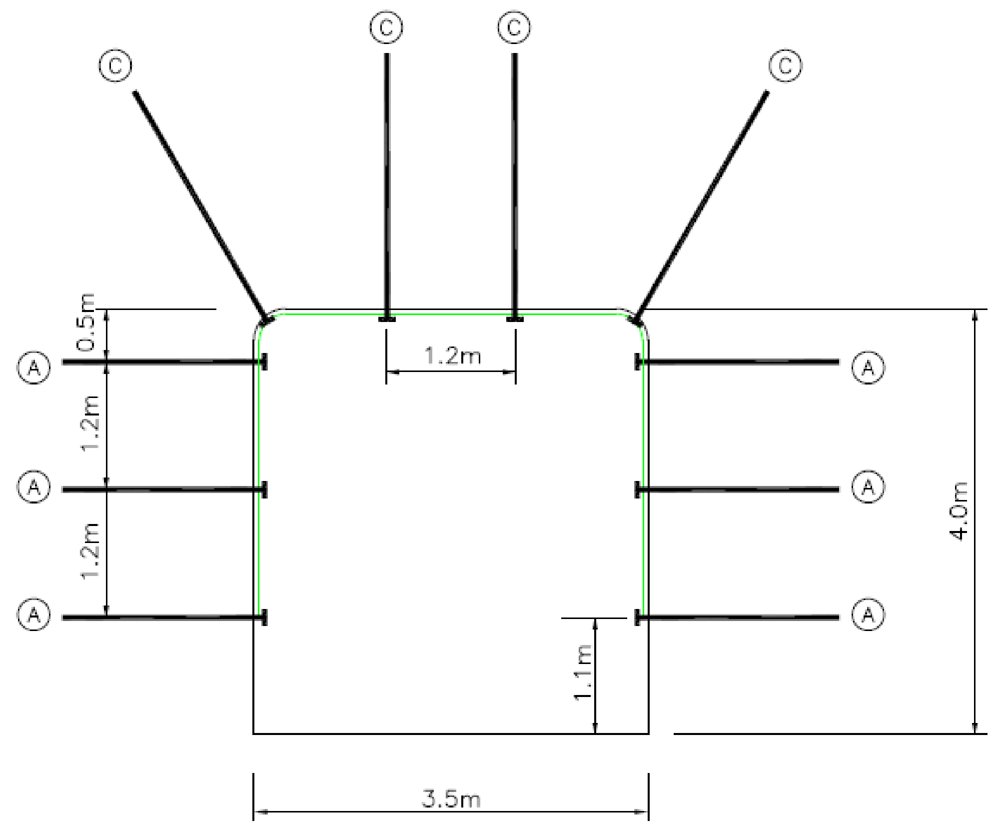
Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE MD - I

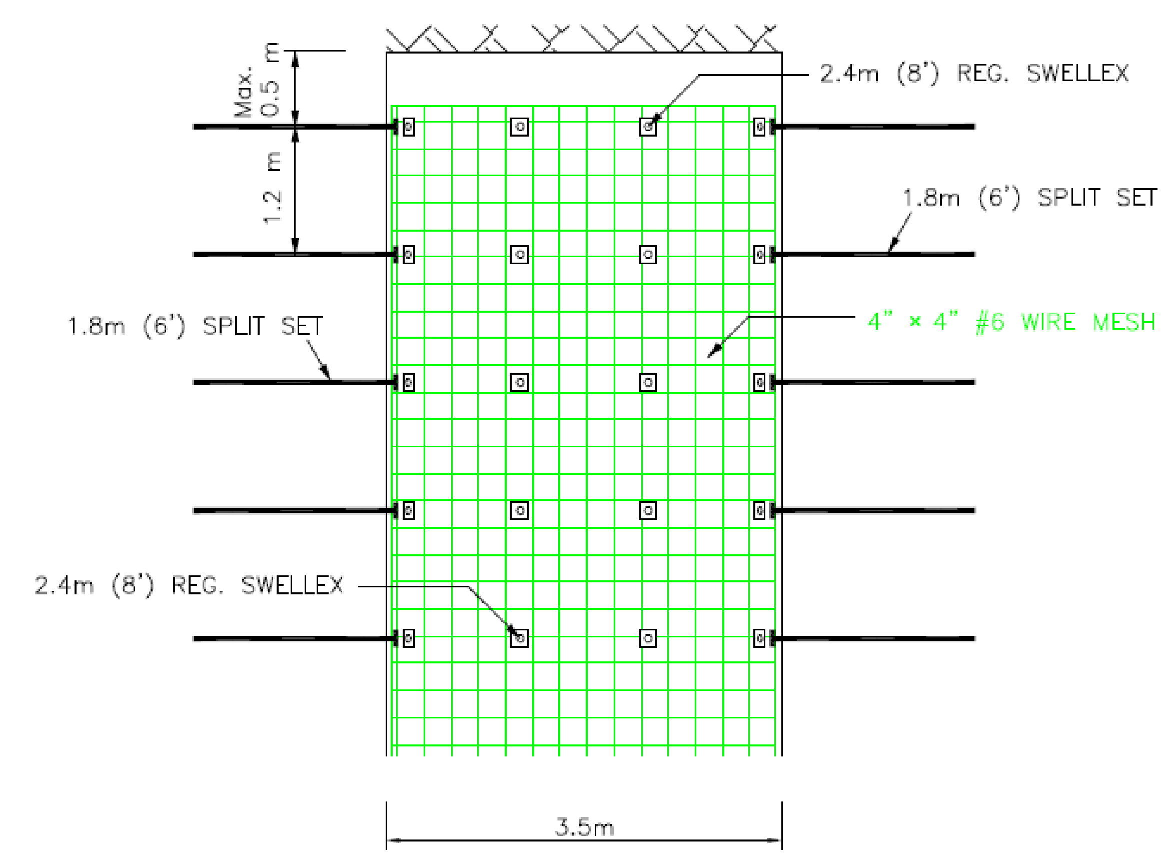
FILE NAME: GCMP SUPPORT STANDARDS
REV. 001

MD - II (3.5mW by 4.0mH)
Poor to Fair Ground (30 < GSI < 45)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	REG. SWELLEX	2.4m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+–10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE MD - II

FILE NAME: GCMP SUPPORT STANDARDS

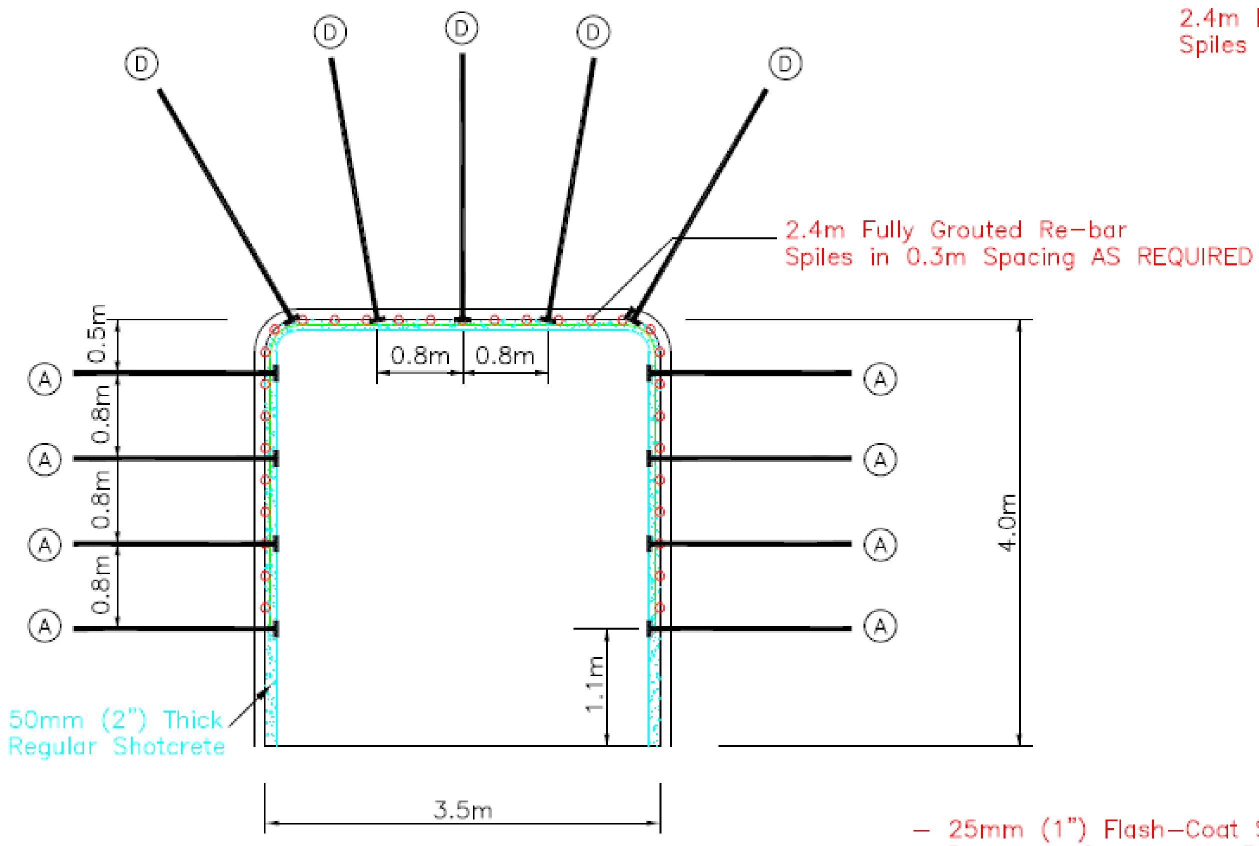
REV. 001

MD - III (3.5mW by 4.0mH)

Extremely Poor Ground (GSI < 30)

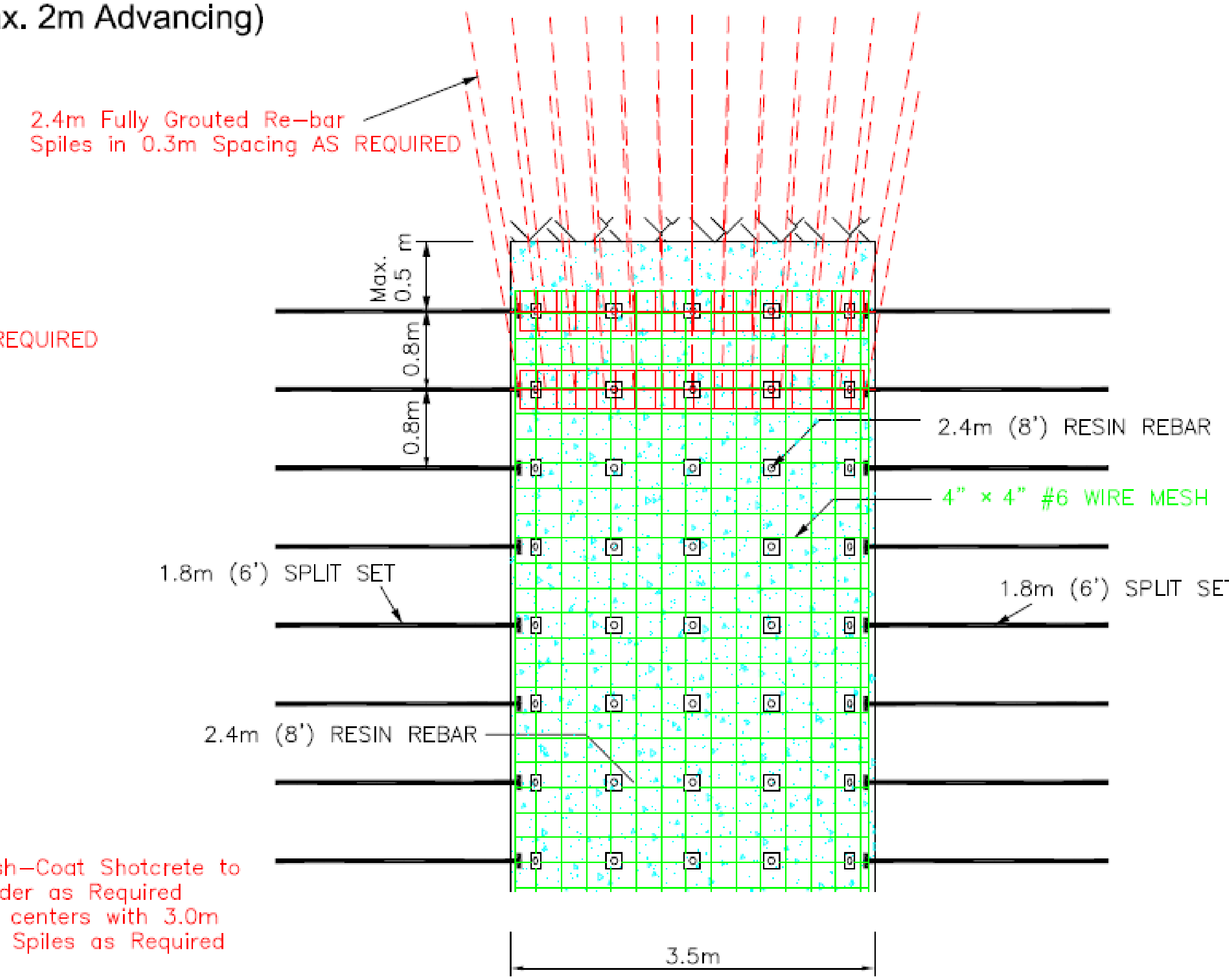
(Short Round - Max. 2m Advancing)

SECTION



- 25mm (1") Flash-Coat Shotcrete to Back and Shoulder as Required
- Spiling at 0.3m centers with 3.0m Grouted Re-bar Spiles as Required

PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	REG. SWELLEX	2.4m	0.8mX0.8m	REG	25mm (PRE) 50mm (POST)	AS NOTED
WALLS	SPLIT SET	1.8m	0.8mX0.8m	REG	25mm (PRE) 50mm (POST)	AS NOTED

SUPPORT BOLT TABLE

No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE MD - III

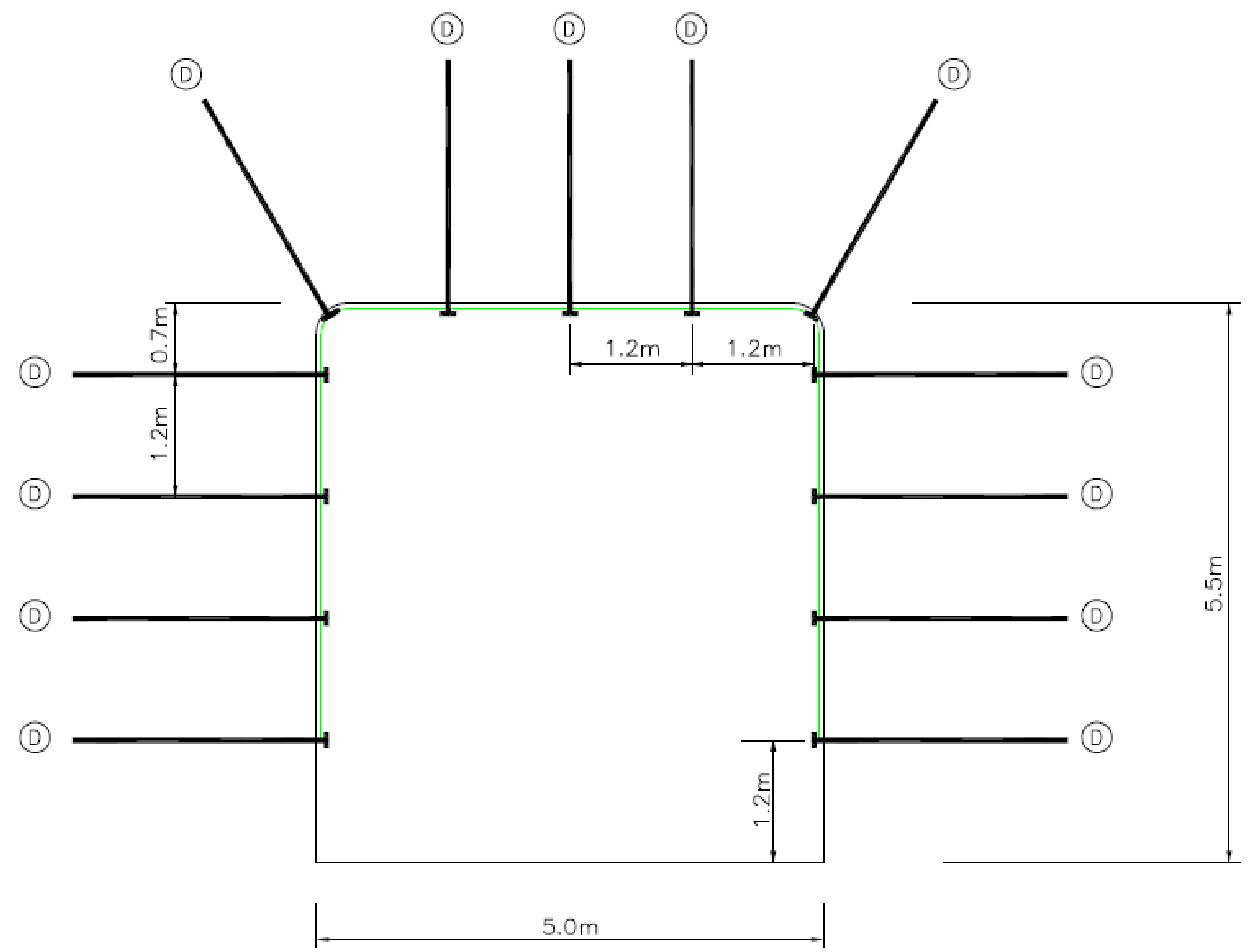
FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

RMK - I (5.0mW by 5.5mH)

Fair to Good Ground (45 < GSI < 60)

SECTION



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN REBAR	2.4m	1.2mX1.2m	–	–	AS NOTED
WALLS	RESIN REBAR	2.4m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE N.T.S

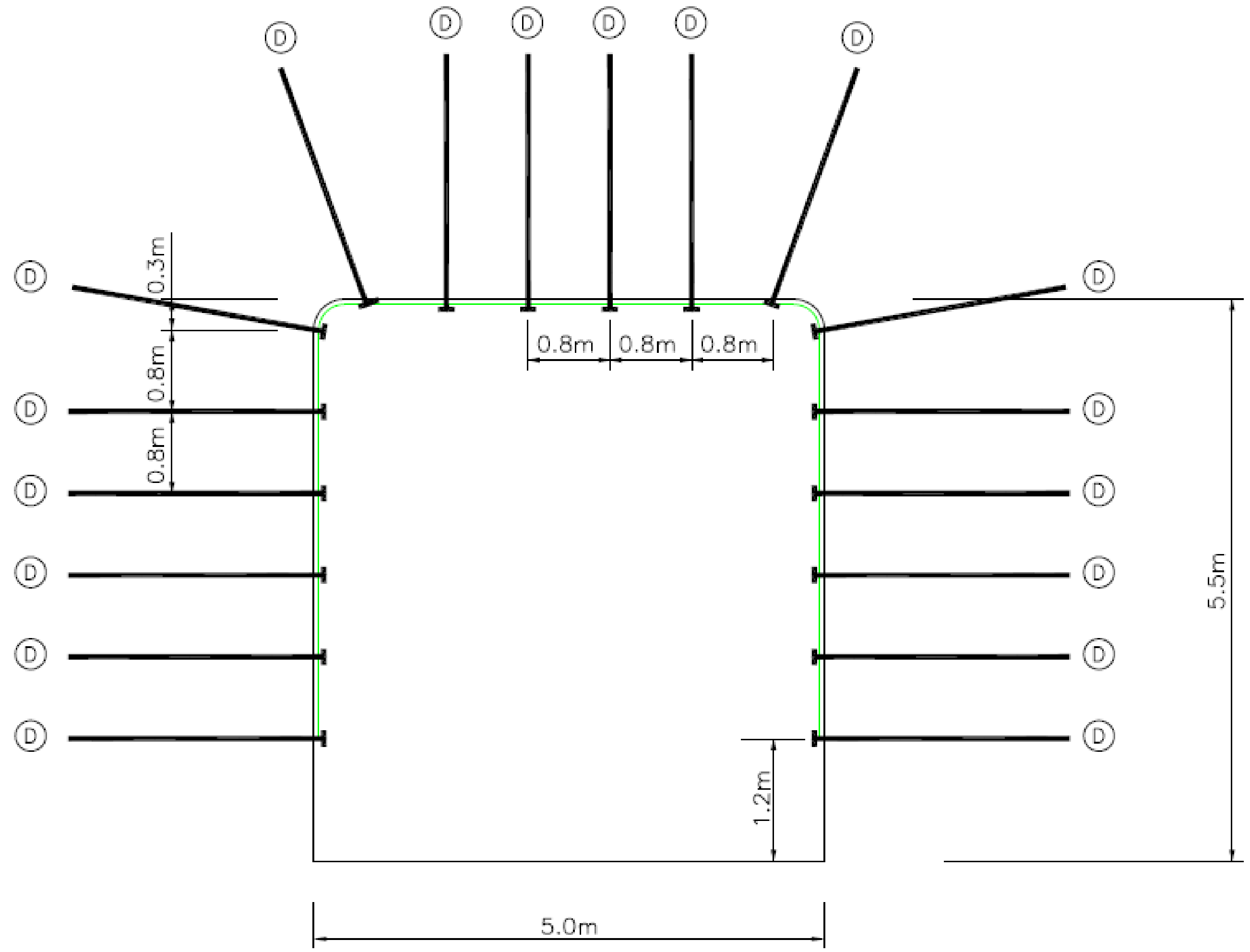
TYPE RMK - I

FILE NAME GCMP SUPPORT STANDARDS

REV. 001

RMK - I (5.0mW by 5.5mH)
Poor to Fair Ground (30 < GSI < 45)

SECTION



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN REBAR	2.4m	0.8mX0.8m	–	–	AS NOTED
WALLS	RESIN REBAR	2.4m	0.8mX0.8m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

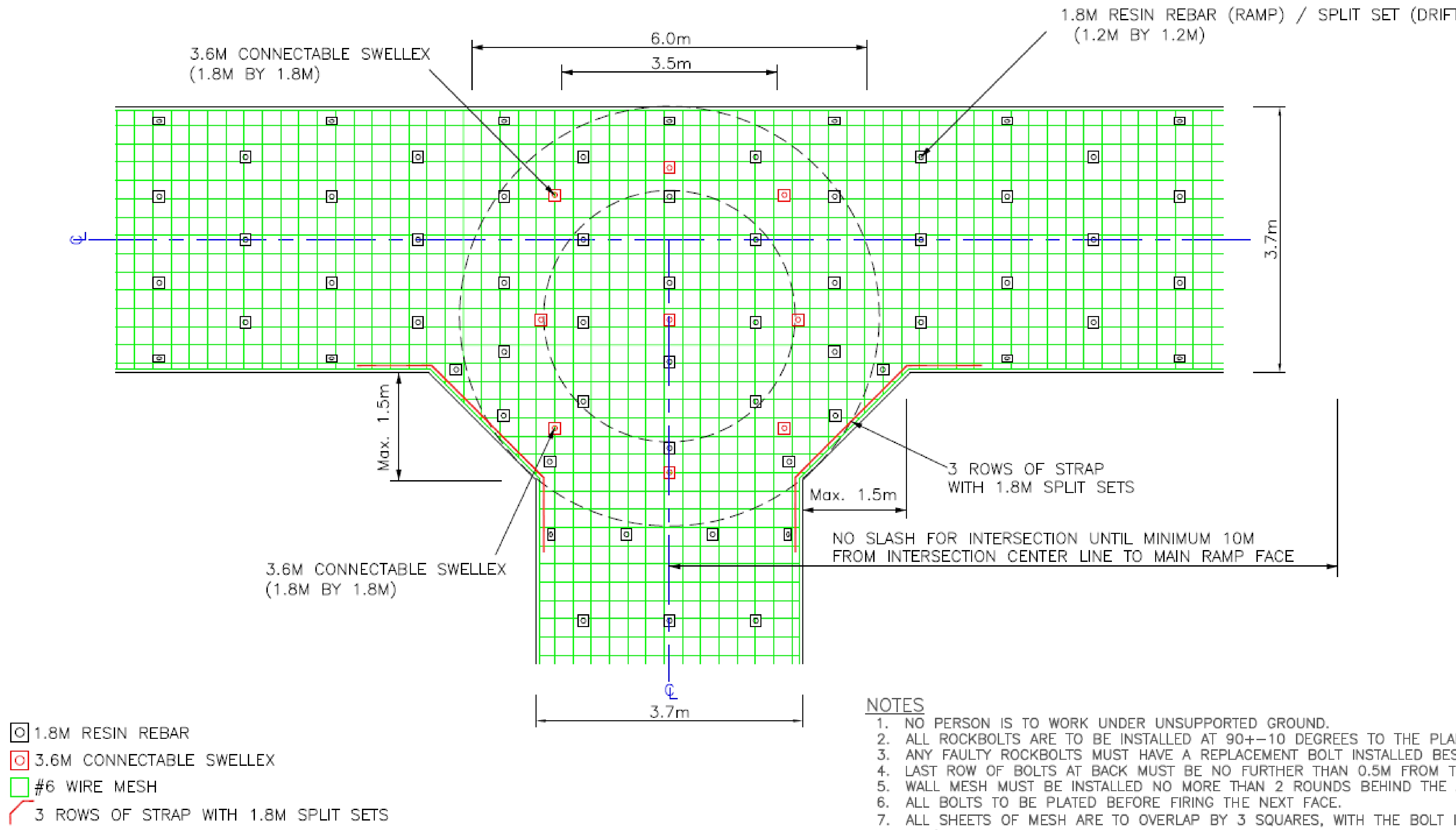
TYPE RMK - II

FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

IS - I

Fair to Good Ground (60 > GSI > 45)



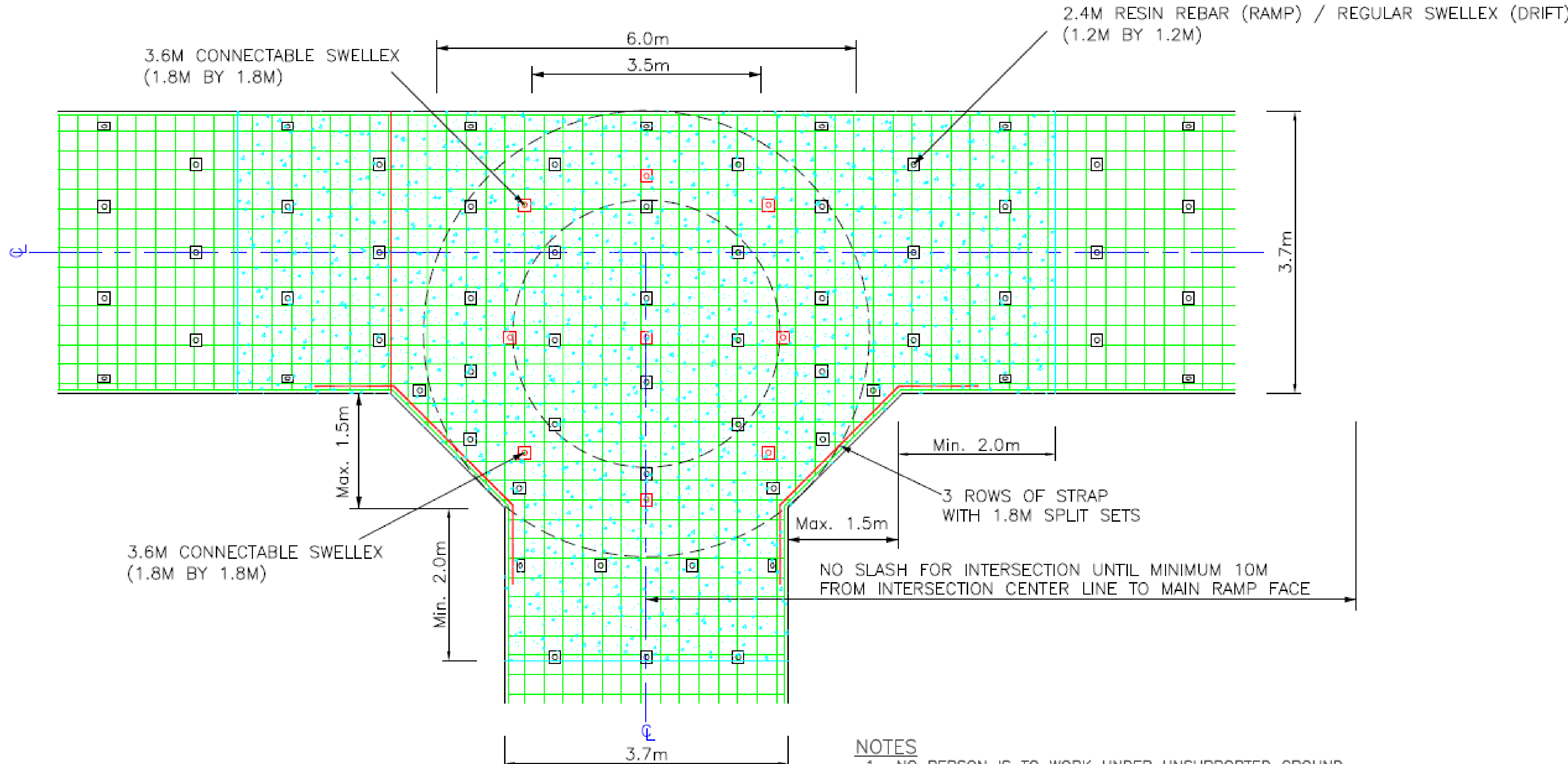
NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS	ALEXCO	Designed By: / Checked By: / Drawn By: / SCALE: N.T.S	TYPE IS - I / FILE NAME: GCMP SUPPORT STANDARDS / REV. 001

IS - II

Poor to Fair Ground (45 > GSI > 30)



- 2.4M RESIN REBAR (RAMP) / 2.4M REGULAR SWELLEX (DRIFT)
- 3.6M CONNECTABLE SWELLEX
- #6 WIRE MESH
- 3 ROWS OF STRAP WITH 1.8M SPLIT SETS
- 50mm REGULAR POST SHOTCRETE OR 50mm FIBER PRE SHOTCRETE AS REQUIRED

NOTES

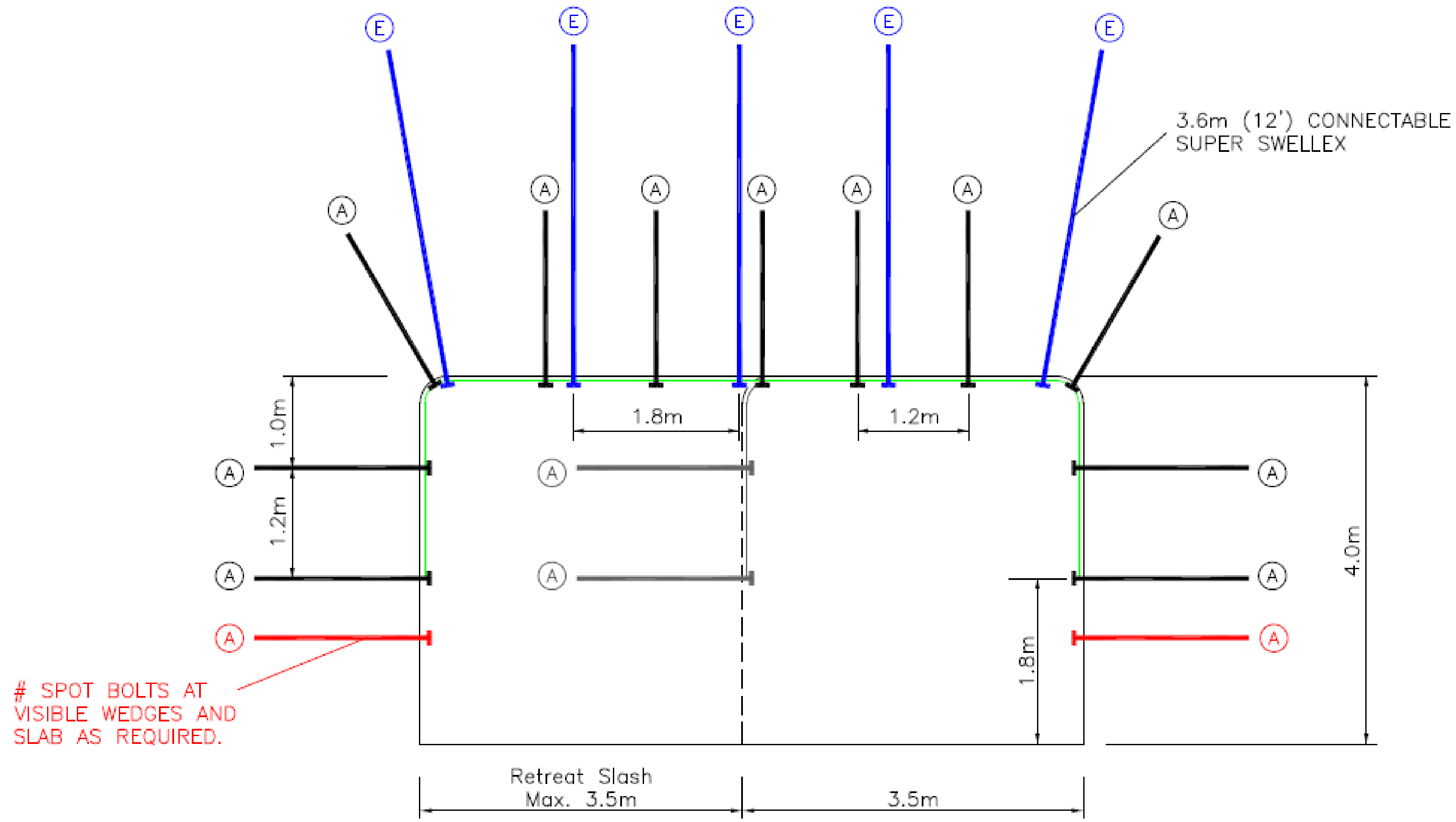
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS	ALEXCO	Designed By: / Checked By: / Drawn By: / SCALE: N.T.S	TYPE IS - II
			FILE NAME: GCMP SUPPORT STANDARDS — REV. 001

WD - I (Max. 7.0mW by 4.0mH)

Fair to Good Ground (45 < GSI < 60)

SECTION



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

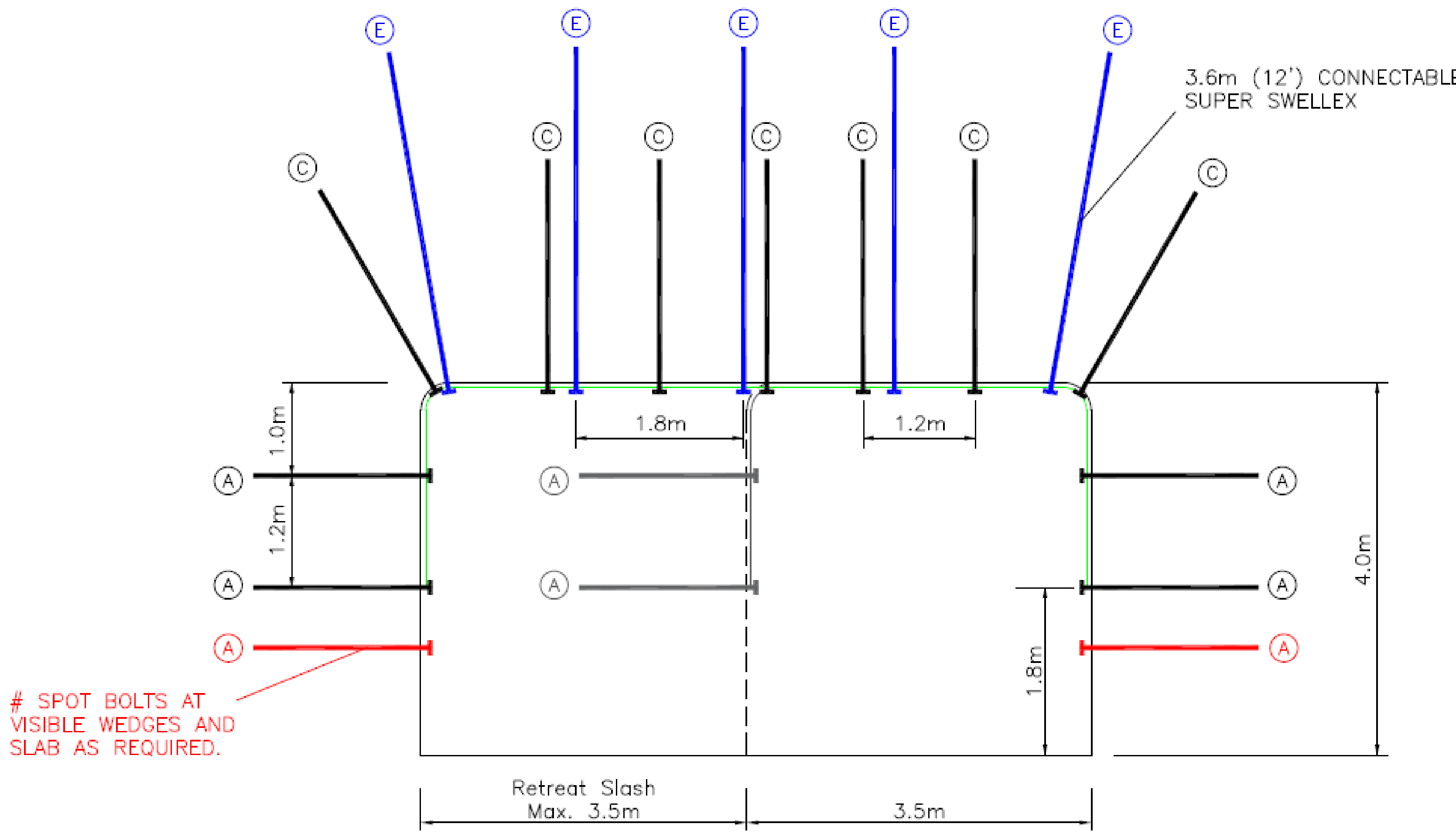
TYPE WD - I

FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

WD - II (Max. 7.0mW by 4.0mH)
Poor to Fair Ground (30 < GSI < 45)

SECTION



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	REG. SWELLEX	2.4m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

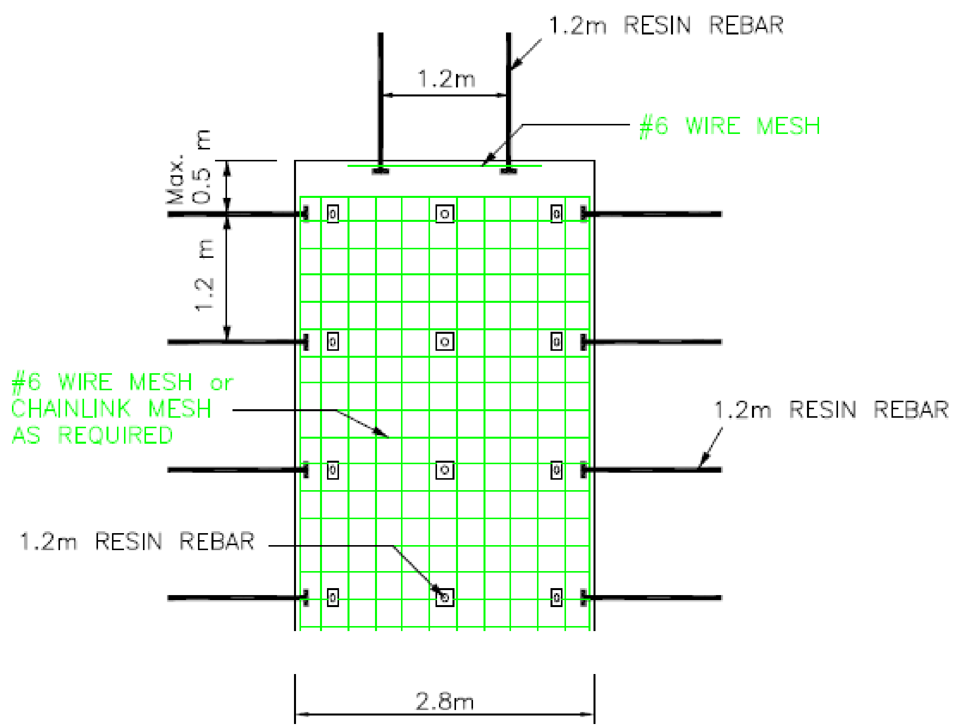
Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

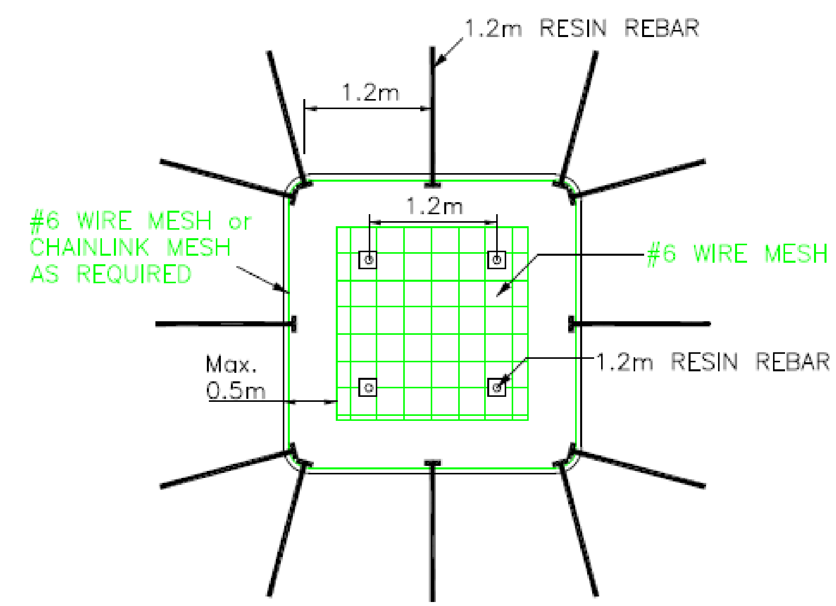
TYPE WD - II

FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

SR - I (2.8mW by 2.8mH)
Fair to Good Ground (45 < GSI < 60)





SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
FACE	RESIN REBAR	1.2m	1.2mX1.2m	–	–	AS NOTED
WALLS	RESIN REBAR	1.2m	1.2mX1.2m	–	–	AS NOTED

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+−10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE SR - I

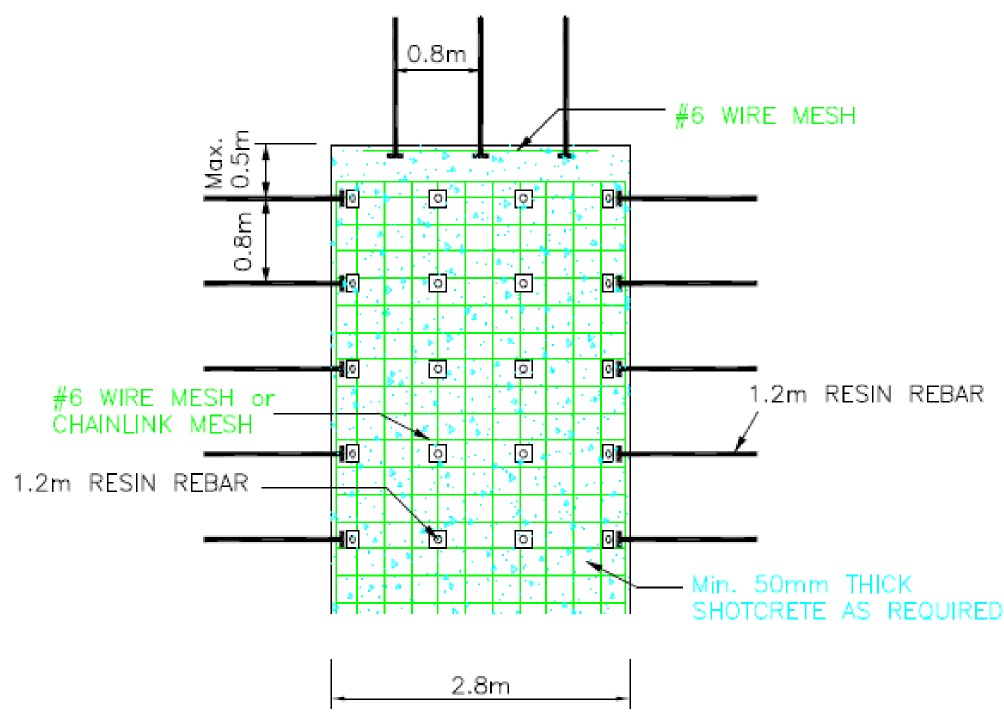
FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

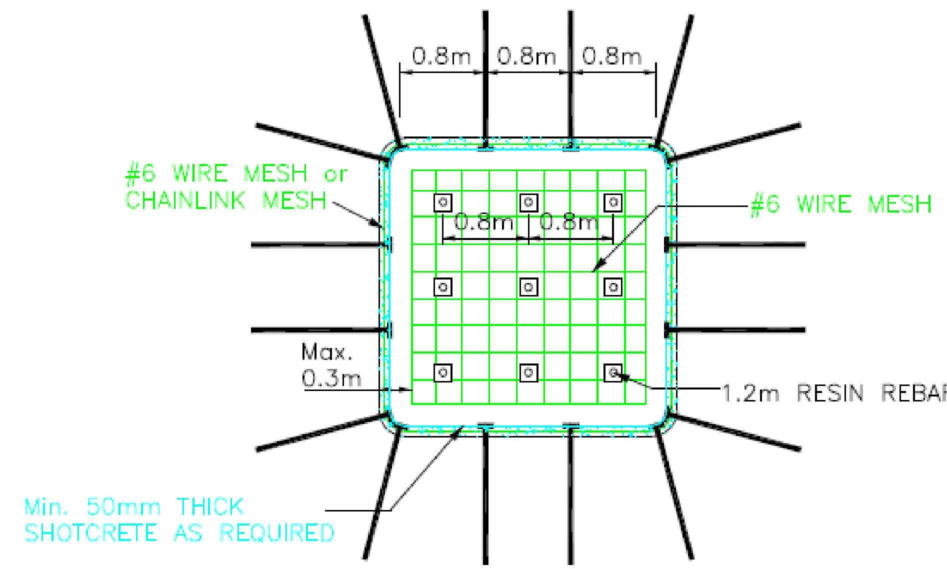
SR - II (2.8mW by 2.8mH)

Ext. Poor to Poor Ground (GSI < 45)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
FACE	RESIN REBAR	1.2m	0.8mX0.8m	REG	50mm AS REQ.	AS NOTED
WALLS	RESIN REBAR	1.2m	0.8mX0.8m	REG	50mm AS REQ.	AS NOTED

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE SR - II

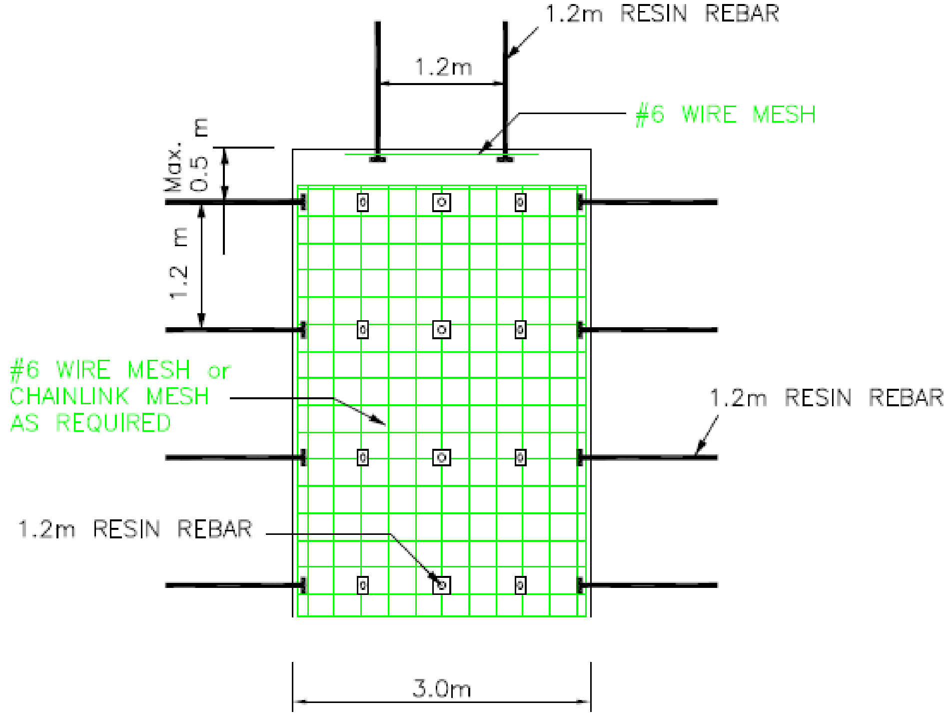
FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

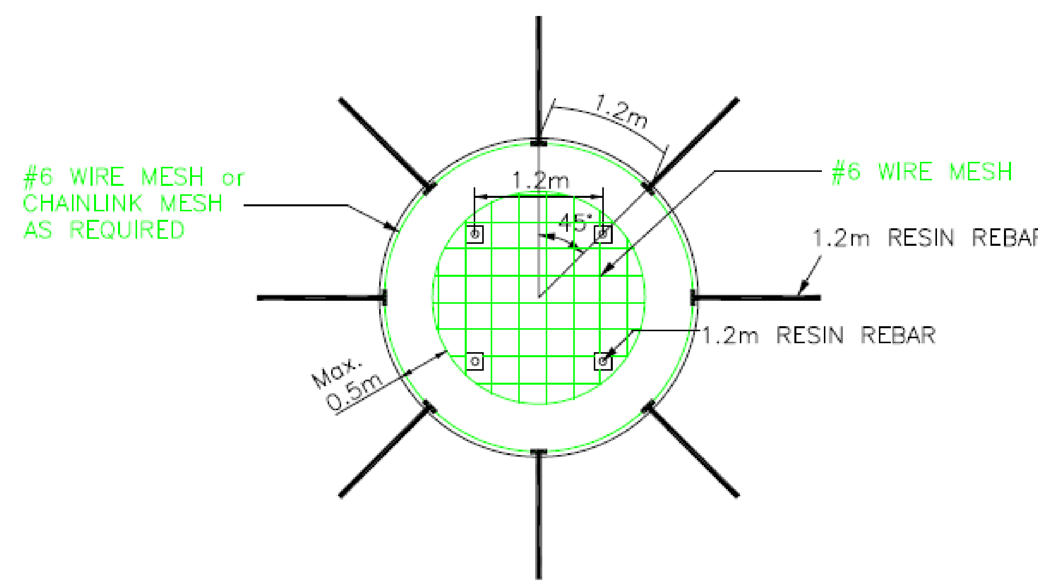
CR - I (D = 3.0m)

Fair to Good Ground (45 < GSI < 60)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
FACE	RESIN REBAR	1.2m	1.2mX1.2m	–	–	AS NOTED
WALLS	RESIN REBAR	1.2m	1.2mX1.2m	–	–	AS NOTED

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+–10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

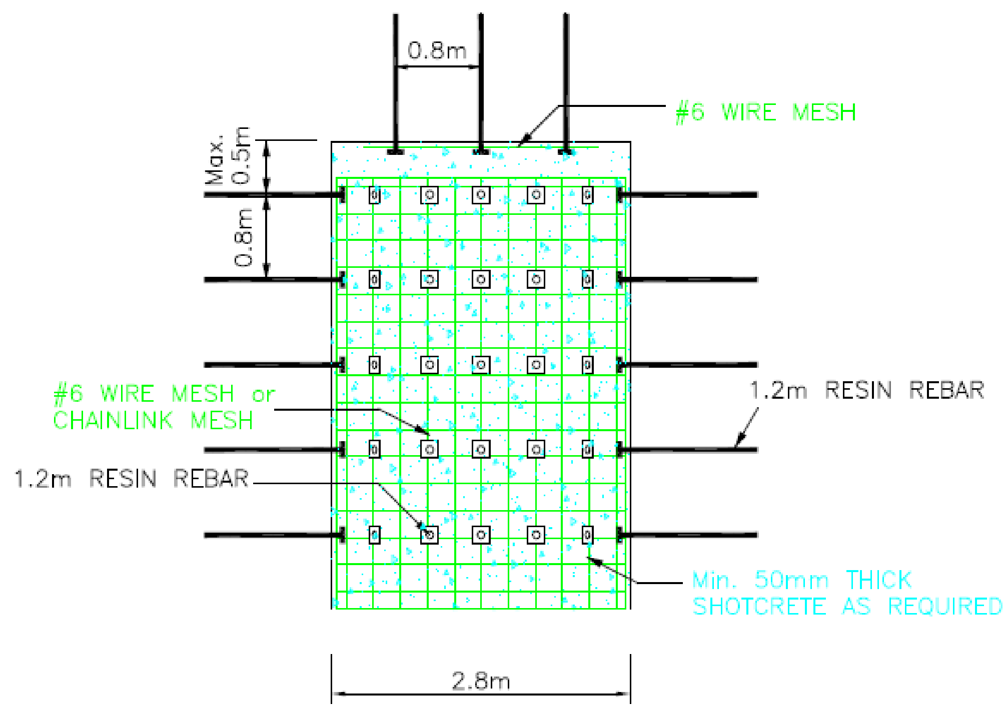
Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE CR - I

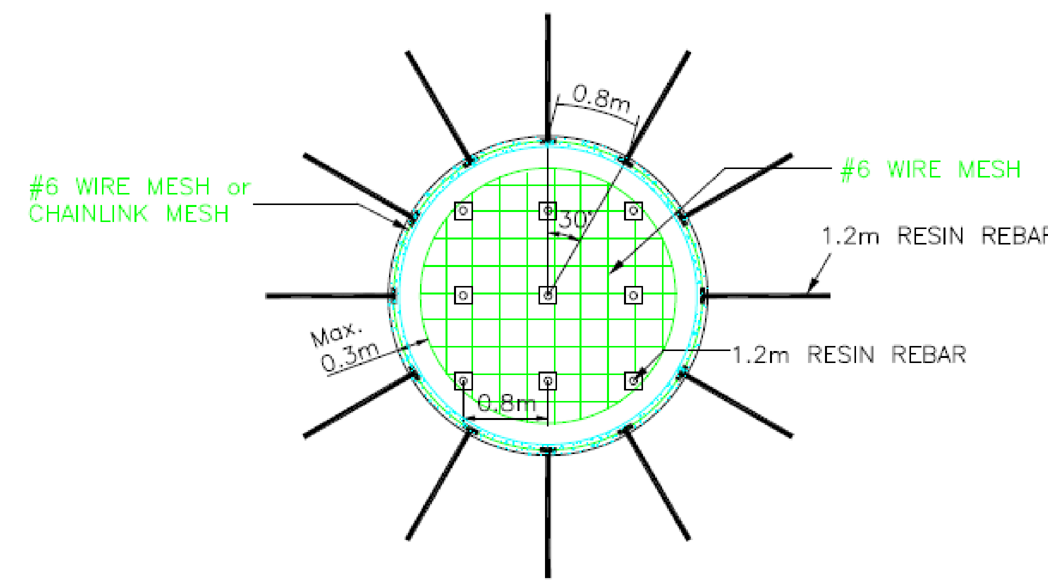
FILE NAME: GCMP SUPPORT STANDARDS | REV. 001

CR - II (D = 3.0m)
Ext. Poor to Poor Ground (GSI < 45)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
FACE	RESIN REBAR	1.2m	0.8mX0.8m	REG	50mm AS REQ.	AS NOTED
WALLS	RESIN REBAR	1.2m	0.8mX0.8m	REG	50mm AS REQ.	AS NOTED

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE N.T.S

TYPE CR - II

FILE NAME GCMP SUPPORT STANDARDS REV. 001

APPENDIX – C.

TRIGGER ACTION RESPONSE PLAN

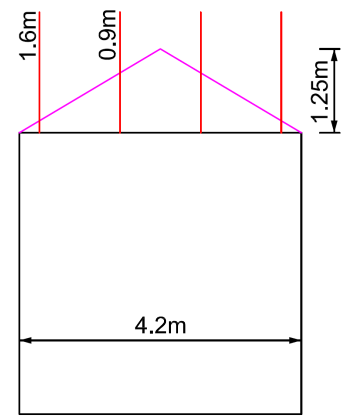
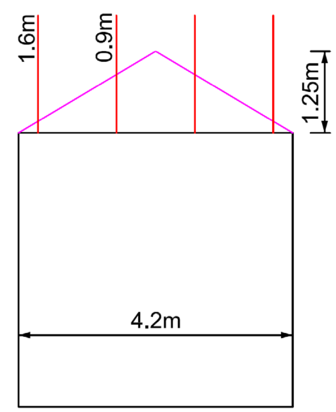
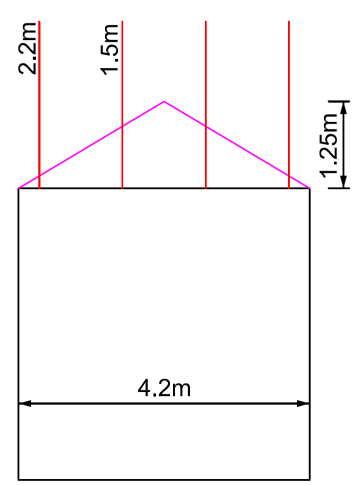
TARP Guideline for Men Entry Openings

Section	Dimension (W x H, m)	Ground Condition		Support Type					Wire Mesh	Additional Support	Comments
		Drilling Condition	Geotechnical Face Mapping Condition		Loc.	Type	Length (m)	Spacing (m x m)			
Main Ramp	4.2 x 4.2	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**Ramp – I**	Back Wall	Rebar Split Set	1.8 1.8	1.2 x 1.2	#6 up to 2.0m from sill		
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**Ramp – II**	Back Wall	Rebar Split Set	2.4 1.8	1.2 x 1.2	#6 up to 1.3m from sill	5mm reg. shotcrete as req.	
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Ramp – III**	Back Wall	Rebar Split Set	2.4 1.8	0.8 x 0.8	#6 up to 1.3m from sill	5mm pre shotcrete as req. 5mm post shotcrete	Spiling as req.
Main Drift	3.5 x 4.0	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**MD – I**	Back Wall	Split Set Split Set	1.8 1.8	1.2 x 1.2	#6 up to 1.8m from sill		
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**MD – II**	Back Wall	R. Swellex Split Set	2.4 1.8	1.2 x 1.2	#6 up to 1.1m from sill	5mm reg. shotcrete as req.	
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**MD – III**	Back Wall	R. Swellex Split Set	2.4 1.8	0.8 x 0.8	#6 up to 1.1m from sill	5mm pre shotcrete as req. 5mm post shotcrete	Spiling as req.
Wide Drift	3.5~7.0 X 4.0	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**WD – I**	Back Wall	Split Set Split Set	1.8 1.8	1.2 x 1.2	#6 up to 1.8m from sill	3.6m connectable 1.8x1.8	3.5m drift with Max. 3.5m slash
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**WD – II**	Back Wall	R. Swellex Split Set	2.4 1.8	1.2 x 1.2	#6 up to 1.1m from sill	3.6m connectable 1.8x1.8 5mm reg. shotcrete as req.	3.5m drift with Max. 3.5m slash
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Development of wide opening is not allowed for this ground condition**							
Remuck	5.0 X 5.5	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**RMK – I**	Back Wall	Rebar Rebar	2.4 2.4	1.2 x 1.2	#6 up to 1.2m from sill		
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**RMK – II**	Back Wall	Rebar Rebar	2.4 2.4	0.8 x 0.8	#6 up to 1.2m from sill		
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Development of wide opening is not allowed for this ground condition**							
Raise	2.8 X 2.8 or D= 3.0	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**SR – I** **CR – I**	Face Wall	Rebar Rebar	1.2 1.2	1.2 x 1.2	#6 Wire mesh or Chainlink		
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**SR – II** **CR – II**	Face Wall	Rebar Rebar	1.2 1.2	0.8 x 0.8	#6 Wire mesh or Chainlink	5mm reg. shotcrete as req.	
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Development of wide opening is not allowed for this ground condition**							
Intersection	R < 6.0	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**IS – I**	Back Wall	Rebar Split Set	2.4 1.8	1.2 x 1.2	#6 up to 1.8m from sill	3.6m connectable 1.8x1.8	3 straps for Pillar support
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**IS - II**	Back Wall	Rebar Split Set	2.4 1.8	1.2 x 1.2	#6 up to 1.1m from sill	3.6m connectable 1.8x1.8 5mm reg. shotcrete as req.	3 straps for Pillar support
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Relocate intersection development to better ground condition**							

APPENDIX – D.

DEAD WEIGHT ANALYSIS

Main Ramp (B = 4.2m) in Fair to Good ground (45 < GSI < 60)

1.8m Split set		***Dead Weight of Wedge Block*** (4.2m × 1.25m) / 2 × 3.2 t/m³ = 8.4 t ***Support Capacity*** (1.6×4.5 + 0.9×4.5x2) / 1.2 = 12.7 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 12.7 / 8.4 = 1.5 (O.K)**
1.8m Rebar		***Dead Weight of Wedge Block*** (4.2m × 1.25m) / 2 × 3.2 t/m³ = 8.4 t ***Support Capacity*** (12 + 0.9×9.0x2) / 1.2 = 23.5 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 23.5 / 8.4 = 2.8 (O.K)**
2.4m Rebar		***Dead Weight of Wedge Block*** (4.2m × 1.25m) / 2 × 3.2 t/m³ = 8.4 t ***Support Capacity*** (12×3) / 1.2 = 30.0 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 30.0 / 8.4 = 3.6 (O.K)**

Main Ramp (B = 4.2m) in Poor to Fair ground (30 < GSI < 45)

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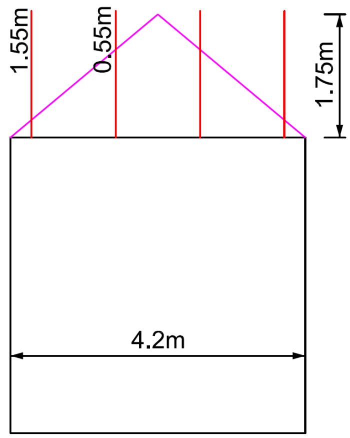
Split set</td>
<td>
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<td><u>Dead Weight of Wedge Block</u>
(4.2m × 1.75m) / 2 × 3.2 t/m^3 = 11.8 t

<u>Support Capacity</u>
(1.55×4.5 + 0.55×4.5x2) / 1.2 = 6.8 t
Bond strength = 4.5 t/m (hard rock)
Breaking strength = 9.0 t
Max. effective length = 9.0/4.5 = 2.0 m

<u>F.S = 11.8 / 6.8 = 0.6 (N.G)</u></td>
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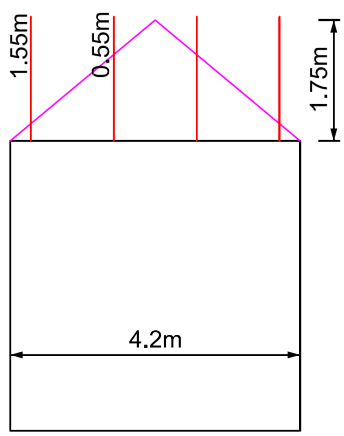
Rebar</td>
<td>
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<td><u>Dead Weight of Wedge Block</u>
(4.2m × 1.75m) / 2 × 3.2 t/m^3 = 11.8 t

<u>Support Capacity</u>
(12 + 0.55×9.0x2) / 1.2 = 18.2 t
Bond strength = 9.0 t/m (hard rock)
Breaking strength = 12.0 t
Max. effective length = 12.0/9.0 = 1.3 m

<u>F.S = 18.2 / 11.8 = 1.5 (O.K)</u></td>
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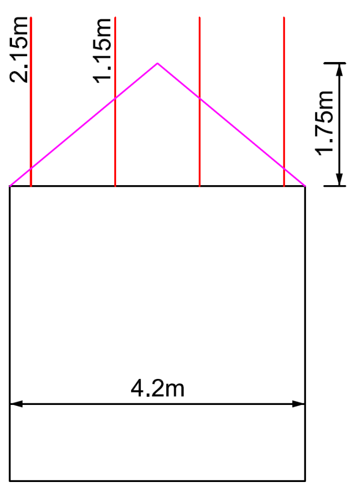
<td>
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<td><u>Dead Weight of Wedge Block</u>
(4.2m × 1.75m) / 2 × 3.2 t/m^3 = 11.8 t

<u>Support Capacity</u>
(12 + 1.15x9x2) / 1.2 = 27.2 t
Bond strength = 9.0 t/m (hard rock)
Breaking strength = 12.0 t
Max. effective length = 12.0/9.0 = 1.3 m

<u>F.S = 27.2 / 11.8 = 2.3 (O.K)</u></td>
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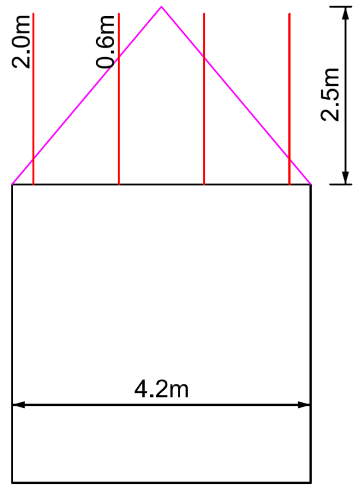
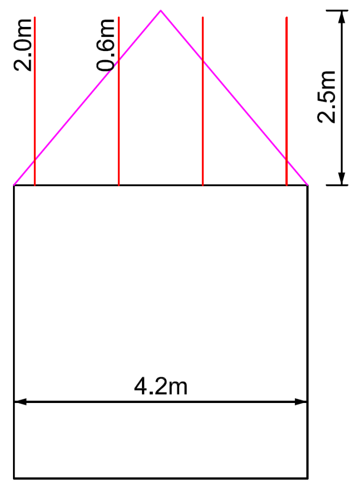
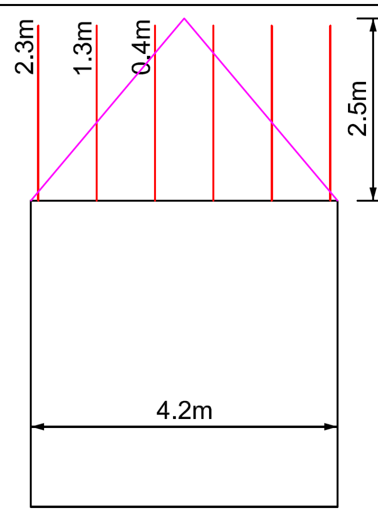
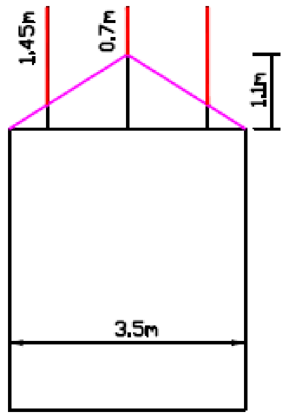
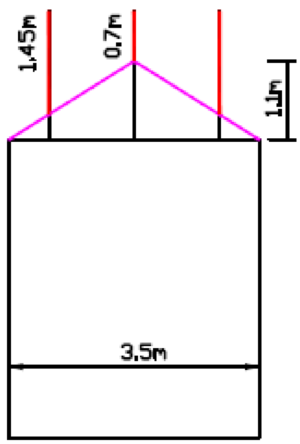
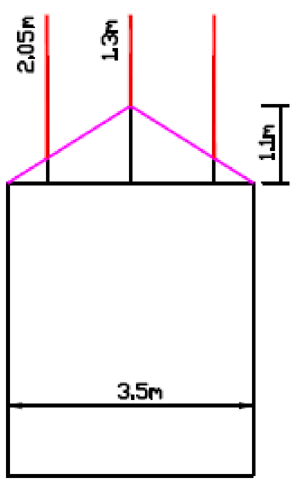
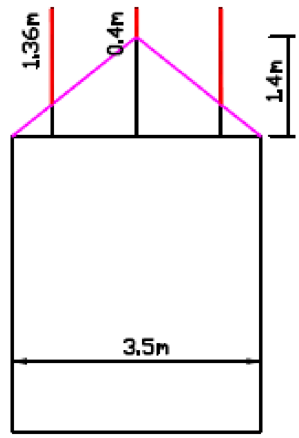
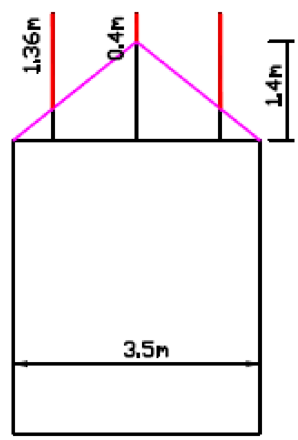
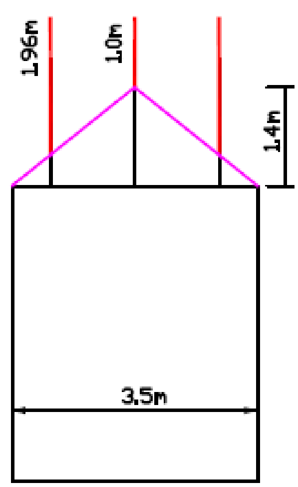
Main Ramp (B = 4.2m) in Extremely Poor ground (20 < GSI < 30)

2.4m Split set		***Dead Weight of Wedge Block*** (4.2m × 2.5m) / 2 × 3.2 t/m^3 = 16.8 t ***Support Capacity*** (9 + 0.6×4.5x2) / 1.2 = 12.0 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 12.0 / 16.8 = 0.7 (N.G)**
2.4m Rebar		***Dead Weight of Wedge Block*** (4.2m × 2.5m) / 2 × 3.2 t/m^3 = 16.8 t ***Support Capacity*** (12 + 0.6×9.0x2) / 1.2 = 19.0 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 19.0 / 16.8 = 1.1 (N.G)**
2.4m Rebar		***Dead Weight of Wedge Block*** (4.2m × 2.5m) / 2 × 3.2 t/m^3 = 16.8 t ***Support Capacity*** (12x3 + 0.4x9x2) / 0.8 = 54.0 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 54.0 / 16.8 = 3.2 (O.K)**

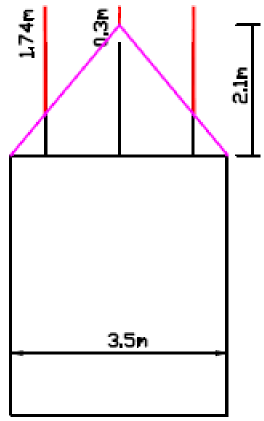
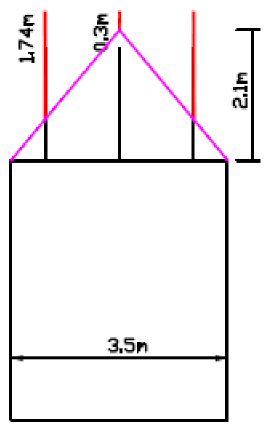
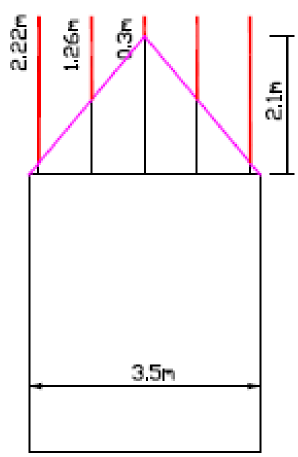
Main Drift (B = 3.5m) in Fair to Good ground (45 < GSI < 60)

1.8m Split set	 	***Dead Weight of Wedge Block*** (3.5m × 1.1m) / 2 × 3.2 t/m³ = 6.2 t ***Support Capacity*** (1.45×4.5×2x0.5 + 0.7×4.5) / 1.2 = 8.1 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 8.1 / 6.2 = 1.3 (O.K)**
1.8m Reg. Swellex	 	***Dead Weight of Wedge Block*** (3.5m × 1.1m) / 2 × 3.2 t/m³ = 6.2 t ***Support Capacity*** (10x2x0.5 + 0.7×7.0) / 1.2 = 12.4 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 12.4 / 6.2 = 2.0 (O.K)**
2.4m Reg. Swellex	 	***Dead Weight of Wedge Block*** (3.5m × 1.1m) / 2 × 3.2 t/m³ = 6.2 t ***Support Capacity*** (10×2×0.5 + 10) / 1.2 = 16.7 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 16.7 / 6.2 = 2.7 (O.K)**

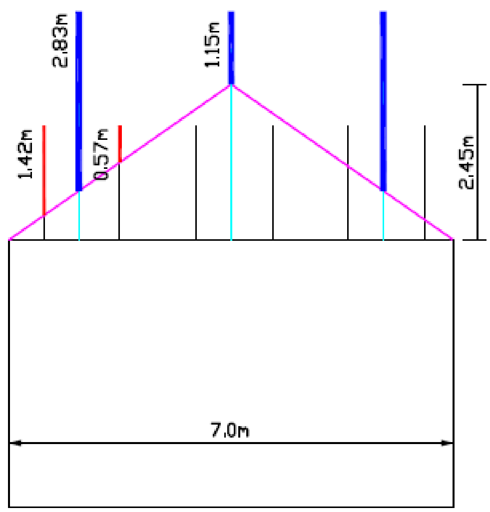
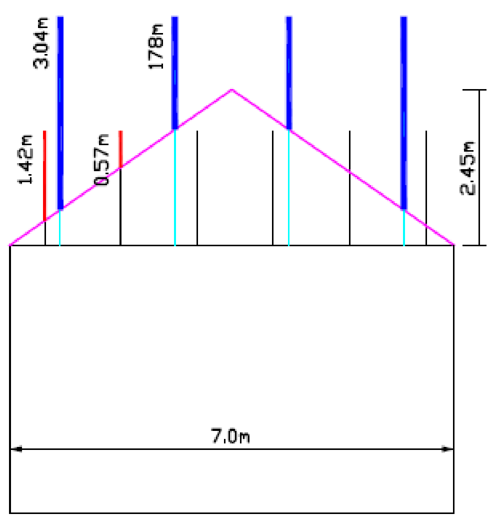
Main Drift (B = 3.5m) in Poor to Fair ground (30 < GSI < 45)

1.8m Split set		***Dead Weight of Wedge Block*** (3.5m × 1.4m) / 2 × 3.2 t/m^3 = 7.9 t ***Support Capacity*** (1.36×4.5×2x0.5 + 0.4×4.5) / 1.2 = 6.6 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 6.6 / 7.9 = 0.8 (N.G)**
1.8m Reg. Swellex		***Dead Weight of Wedge Block*** (3.5m × 1.4m) / 2 × 3.2 t/m^3 = 7.9 t ***Support Capacity*** (1.36x7x2x0.5 + 0.4×7.0) / 1.2 = 10.3 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 10.3 / 7.9 = 1.3 (O.K)**
2.4m Reg. Swellex		***Dead Weight of Wedge Block*** (3.5m × 1.4m) / 2 × 3.2 t/m^3 = 7.9 t ***Support Capacity*** (10×2×0.5 + 1.0x9) / 1.2 = 15.8 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 15.8 / 7.9 = 2.0 (O.K)**

Main Drift (B = 3.5m) in Extremely Poor ground (20 < GSI < 30)

2.4m Split set		***Dead Weight of Wedge Block*** (3.5m × 2.1m) / 2 × 3.2 t/m^3 = 11.8 t ***Support Capacity*** (1.7×4.5×2x0.5 + 0.3×4.5) / 1.2 = 7.5 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 7.5 / 11.8 = 0.6 (N.G)**
2.4m Reg. Swellex		***Dead Weight of Wedge Block*** (3.5m × 2.1m) / 2 × 3.2 t/m^3 = 11.8 t ***Support Capacity*** (10x2x0.5+0.3x7) / 1.2 = 10.1 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 10.1 / 11.8 = 0.9 (O.K)**
2.4m Reg. Swellex		***Dead Weight of Wedge Block*** (3.5m × 2.1m) / 2 × 3.2 t/m^3 = 11.8 t ***Support Capacity*** (10x2x0.5+1.26×7.0x2+0.3x7) / 1.2 = 24.8 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 24.8 / 11.8 = 2.1 (O.K)**

Production Drift (B = 7.0m) in Fair to Good ground (45 < GSI < 60)

1.8m Split set + 3.6m Super Swellex		***Dead Weight of Wedge Block*** (7.0m × 2.45m) / 2 × 3.2 t/m^3 = 27.4 t ***Support Capacity*** (1.42×4.5×2x0.5 + 0.57×4.5x2) / 1.2 + (13x2 + 1.15x10) / 2.4 = 25.2 t Split Set Max. effective length = 9.0/4.5 = 2.0 m Super Swellex Max. effective length = 13.0/10.0 = 1.3 m **F.S = 25.2 / 27.4 = 0.9 (N.G)**
1.8m Split set + 3.6m Super Swellex		***Dead Weight of Wedge Block*** (7.0m × 2.45m) / 2 × 3.2 t/m^3 = 27.4 t ***Support Capacity*** (1.42×4.5×2x0.5 + 0.57×4.5x2) / 1.2 + (13x2 + 13x2) / 1.8 = 38.5 t Split Set Max. effective length = 9.0/4.5 = 2.0 m Super Swellex Max. effective length = 13.0/10.0 = 1.3 m **F.S = 38.5 / 27.4 = 1.4 (O.K)**

Production Drift (B = 7.0m) in Poor to Fair ground (30 < GSI < 45)

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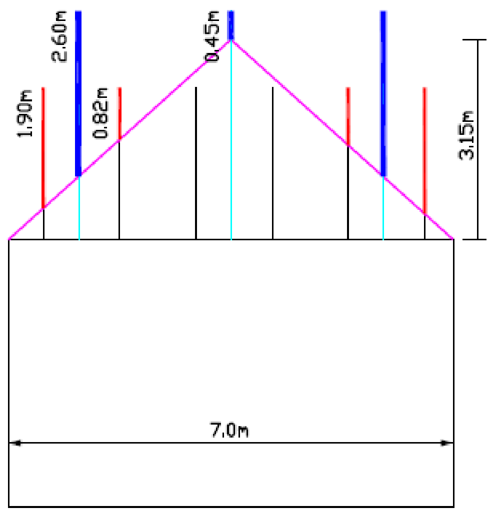
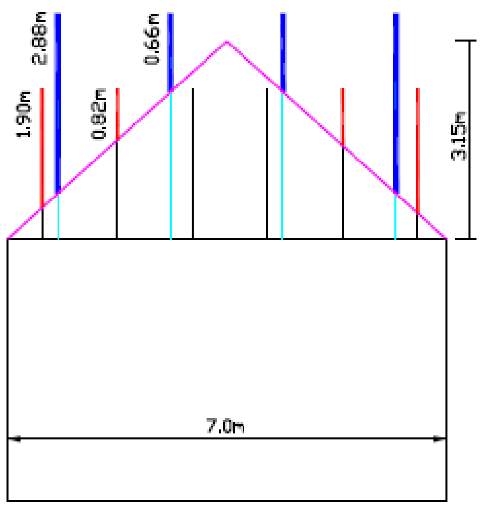
Reg.
Swellex
+
3.6m
Super
Swellex</td>
<td>
</td>
<td><u>Dead Weight of Wedge Block</u>
(7.0m × 3.15m) / 2 × 3.2 t/m^3 = 35.3 t

<u>Support Capacity</u>
(10×2x0.5 + 0.82×7.0x2) / 1.2 +
(13x2 + 0.45x10) / 2.4 = 30.6 t
Reg. Swellex
Max. effective length = 10.0/7.0 = 1.4 m
Super Swellex
Max. effective length = 13.0/10.0 = 1.3 m

<u>F.S = 30.6 / 35.3 = 0.9 (N.G)</u></td>
</tr>
<tr>
<td>2.4m
Reg.
Swellex
+
3.6m
Super
Swellex</td>
<td>
</td>
<td><u>Dead Weight of Wedge Block</u>
(7.0m × 3.15m) / 2 × 3.2 t/m^3 = 35.3 t

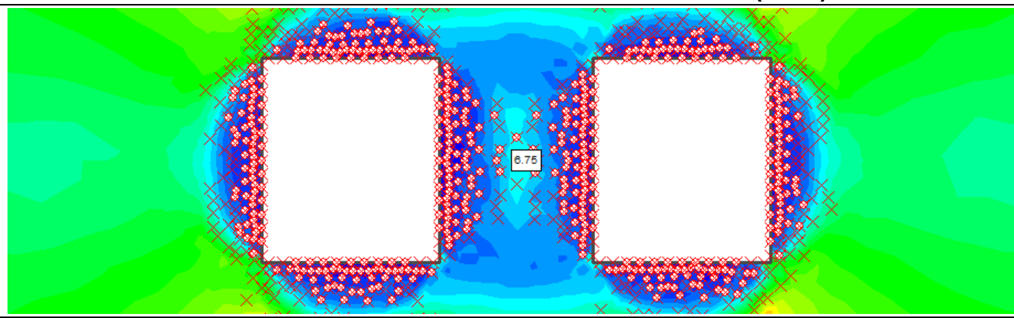
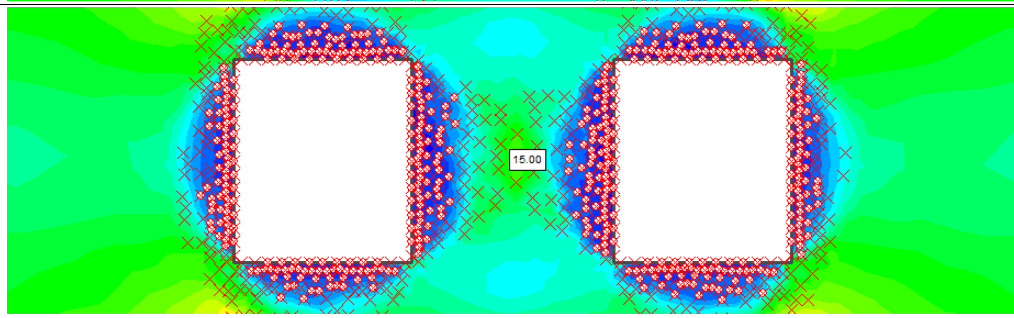
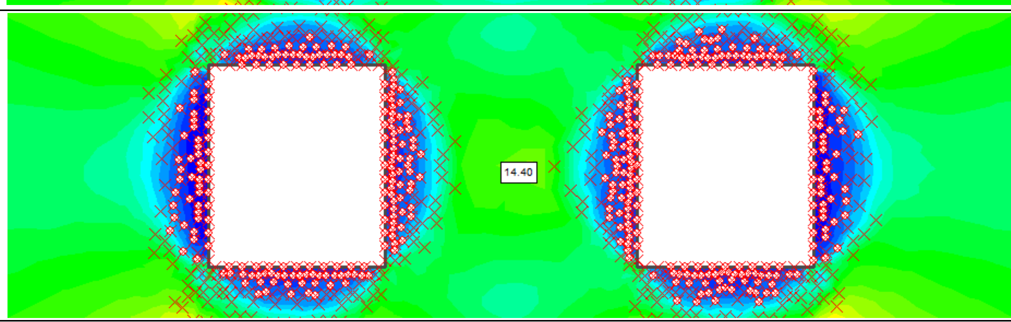
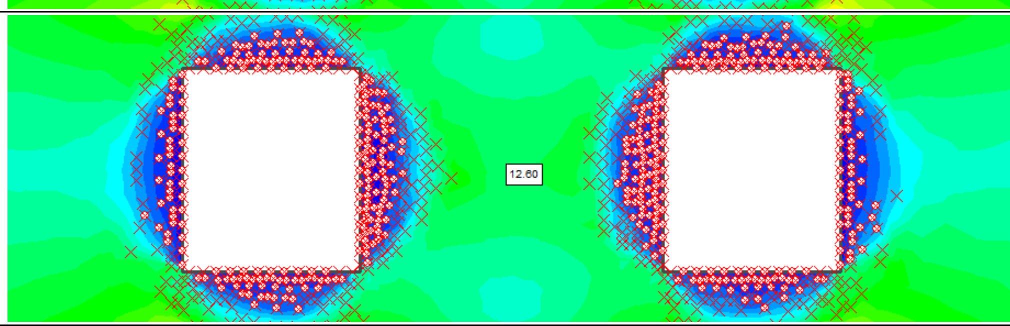
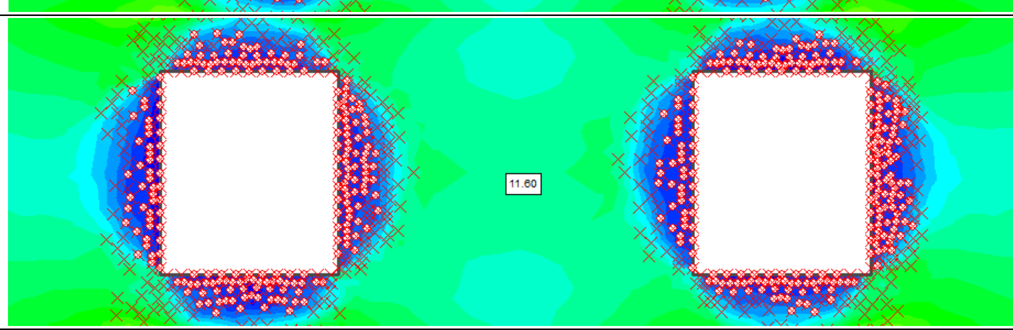
<u>Support Capacity</u>
(10×2x0.5 + 0.82×7.0x2) / 1.2 +
(13x2 + 1.26x10x2) / 1.8 = 39.7 t
Reg. Swellex
Max. effective length = 10.0/7.0 = 1.4 m
Super Swellex
Max. effective length = 13.0/10.0 = 1.3 m

<u>F.S = 46.3 / 35.3 = 1.3 (O.K)</u></td>
</tr>
</table>

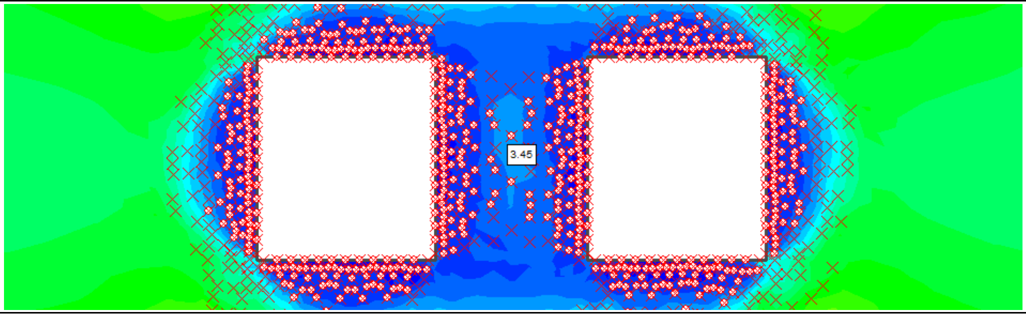
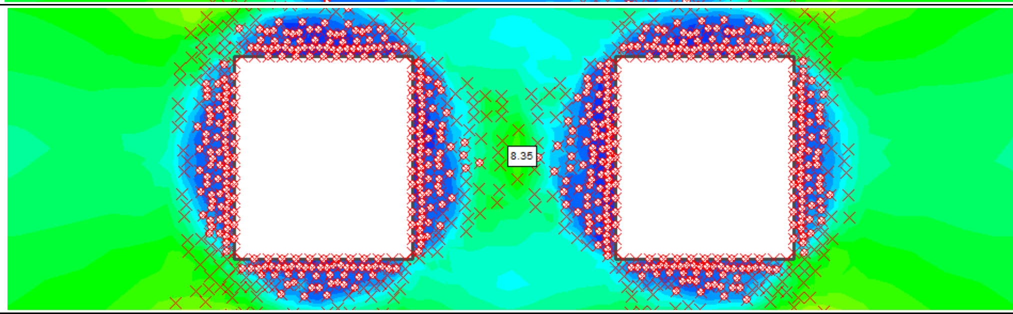
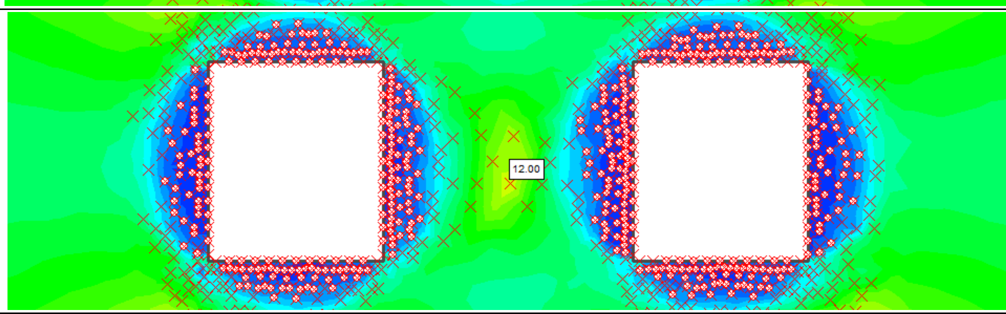
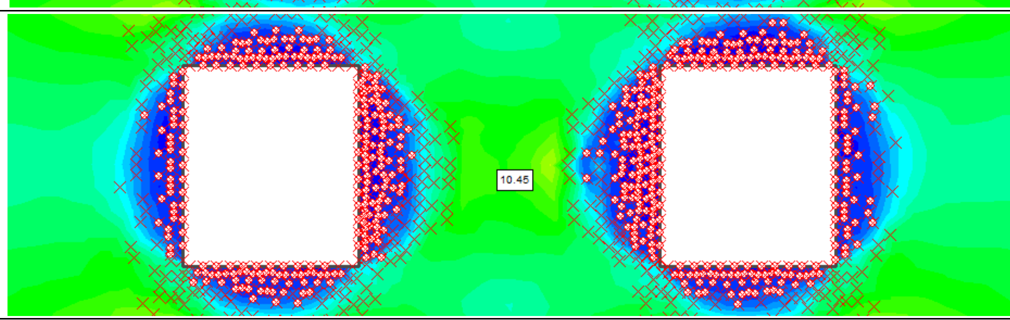
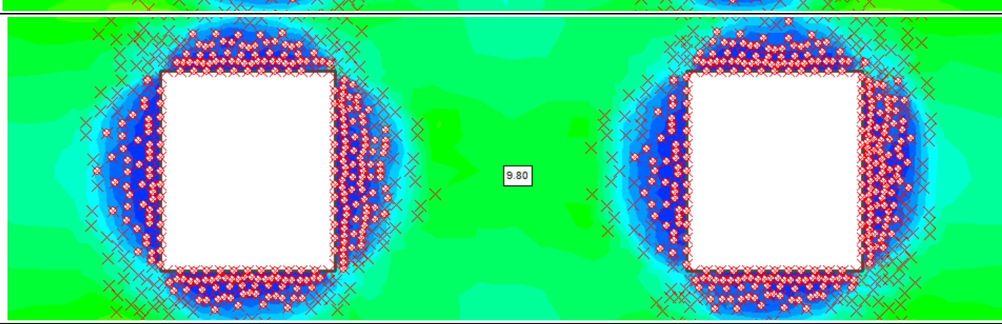
APPENDIX – E.

NUMERICAL CALCULATION (Pillar Stability)

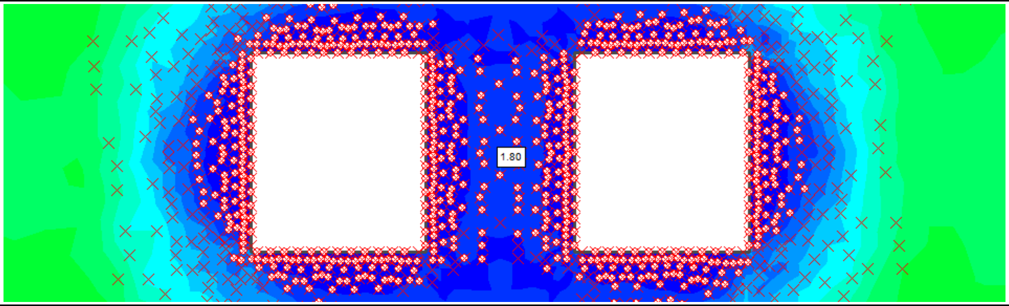
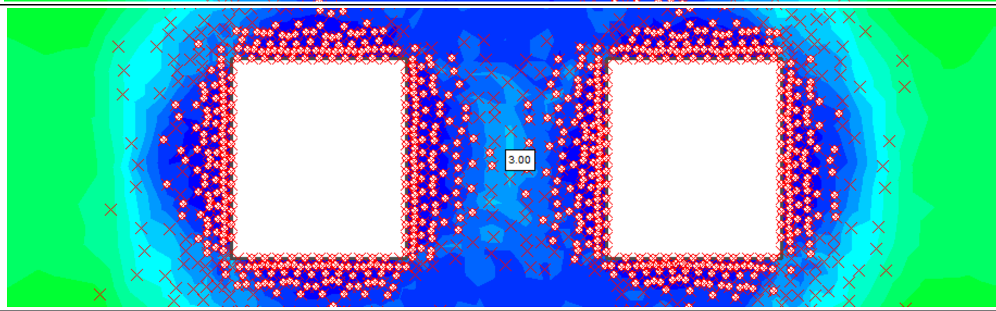
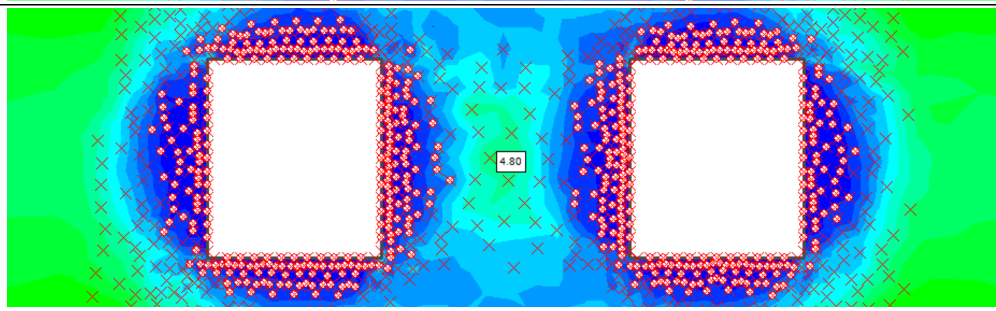
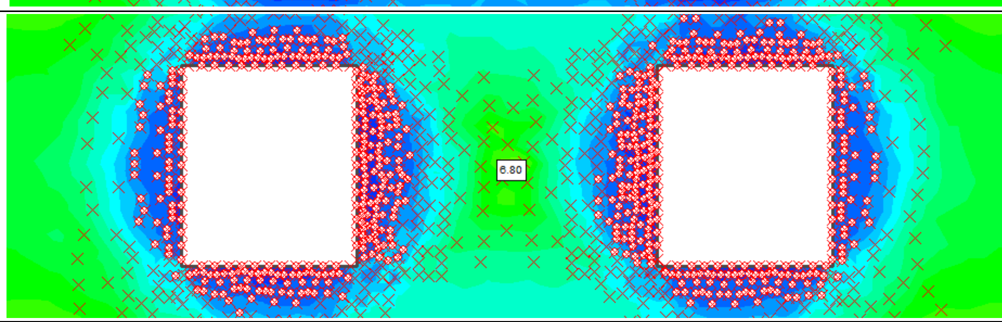
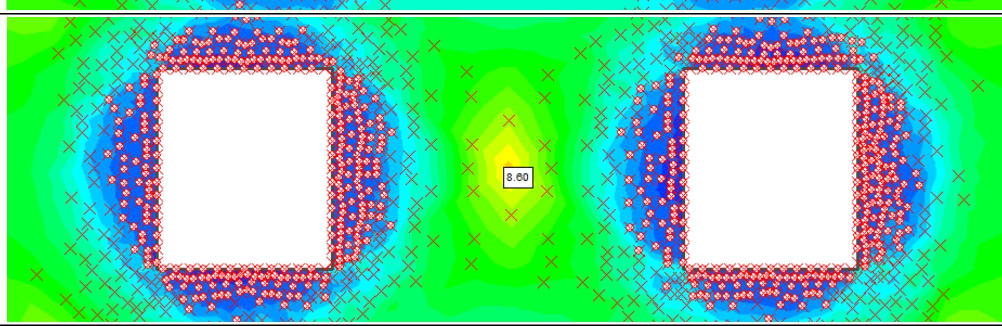
Rib Pillar between 3.5mW x 4.0mH Drift (45 < GSI < 60)

Pilar Width	Yielded Elements and Maximum Stress (σ_{max})
3.0m	
4.0m	
5.0m	
6.0m	
7.0m	

Rib Pillar between 3.5mW x 4.0mH Drift (30 < GSI < 45)

Pilar Width	Yielded Elements and Maximum Stress (σ_{max})
3.0m	
4.0m	
5.0m	
6.0m	
7.0m	

Rib Pillar between 3.5mW x 4.0mH Drift (20 < GSI < 30)

Pilar Width	Yielded Elements and Maximum Stress (σ_{max})
3.0m	
4.0m	
5.0m	
6.0m	
7.0m	

APPENDIX – F.

STABILITY CHART

FOR

STOPE DIMENSION

Stability Graph Analysis - 15m High Stope Wall in Good Ground

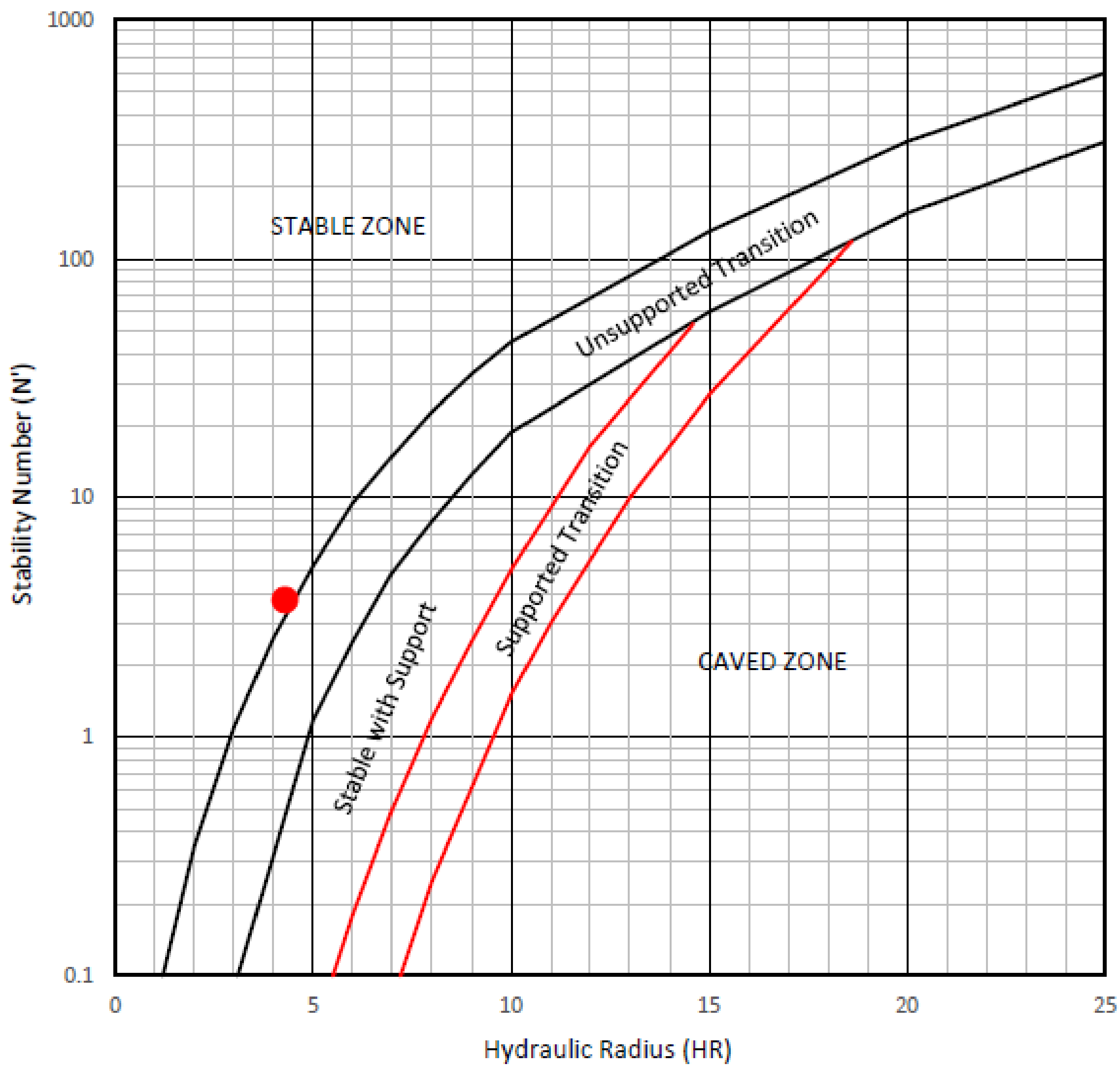
Stope Dimension :	Length =	20	m	Height =	15	m
	Number of Sub-level =	0	levels			

Stability Number, N = Q' × A × B × C

= 5.0 × 0.5 × 0.30 × 5 = **3.75**

Hydraulic Radius = (W × H) / (2W + 2H) = 300 / 70 = **4.29**

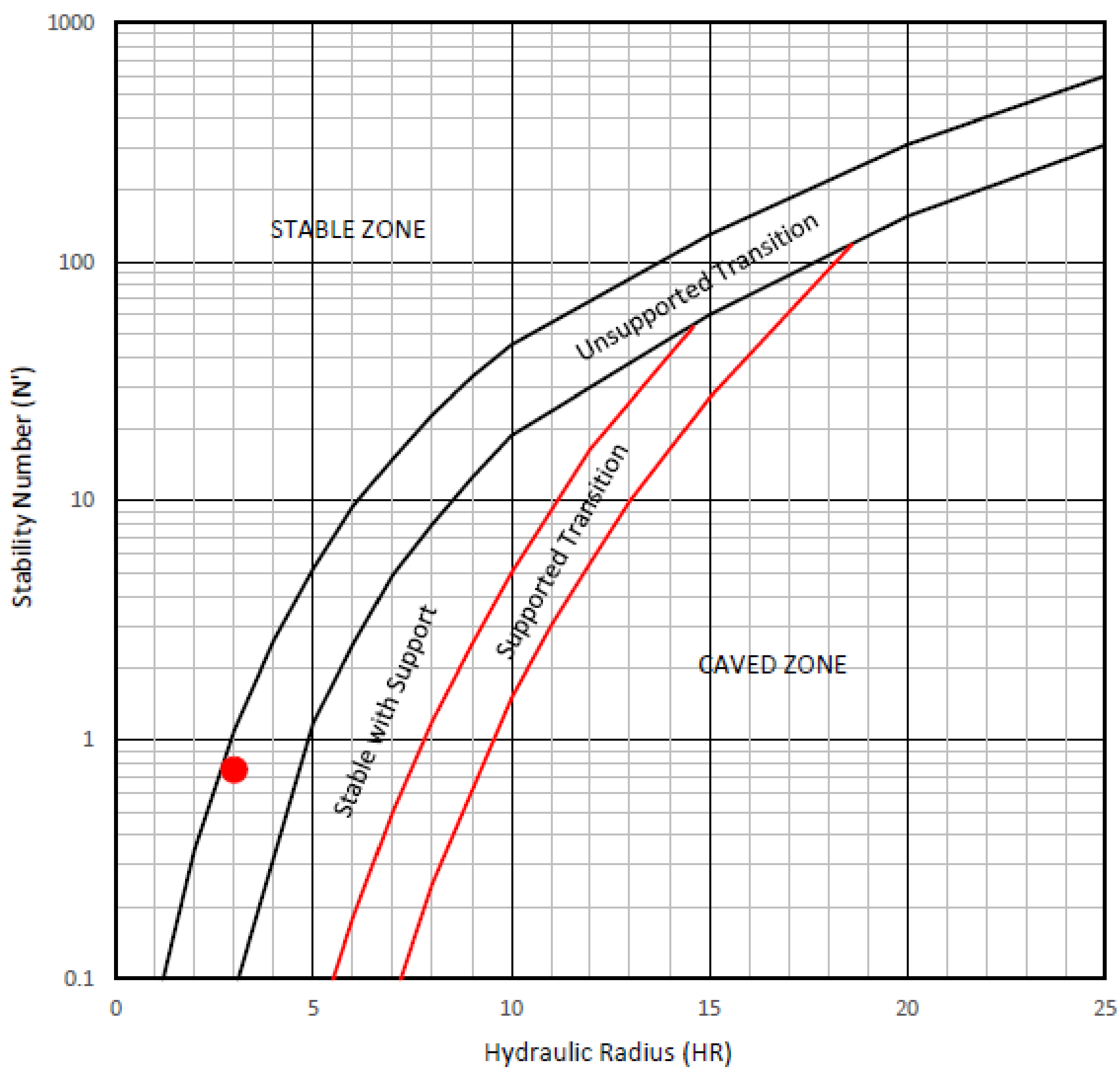
Stability Graph



Stability Graph Analysis - 15m High Stope Wall in Fair Ground

Stope Dimension :		Length =	10	m		Height =	15	m				
		Number of Sub-level =		0	levels							
Stability Number, N	=	Q'	×	A	×	B	×	C				
	=	1.0	×	0.5	×	0.30	×	5	=	**0.75**		
Hydraulic Radius =	(W × H) / (2W + 2H)	=	150	/	50	=	**3.00**					

Stability Graph



Stability Graph Analysis - 15m High Stope Wall in Poor Ground

Stope Dimension : Length = 6 m Height = 15 m

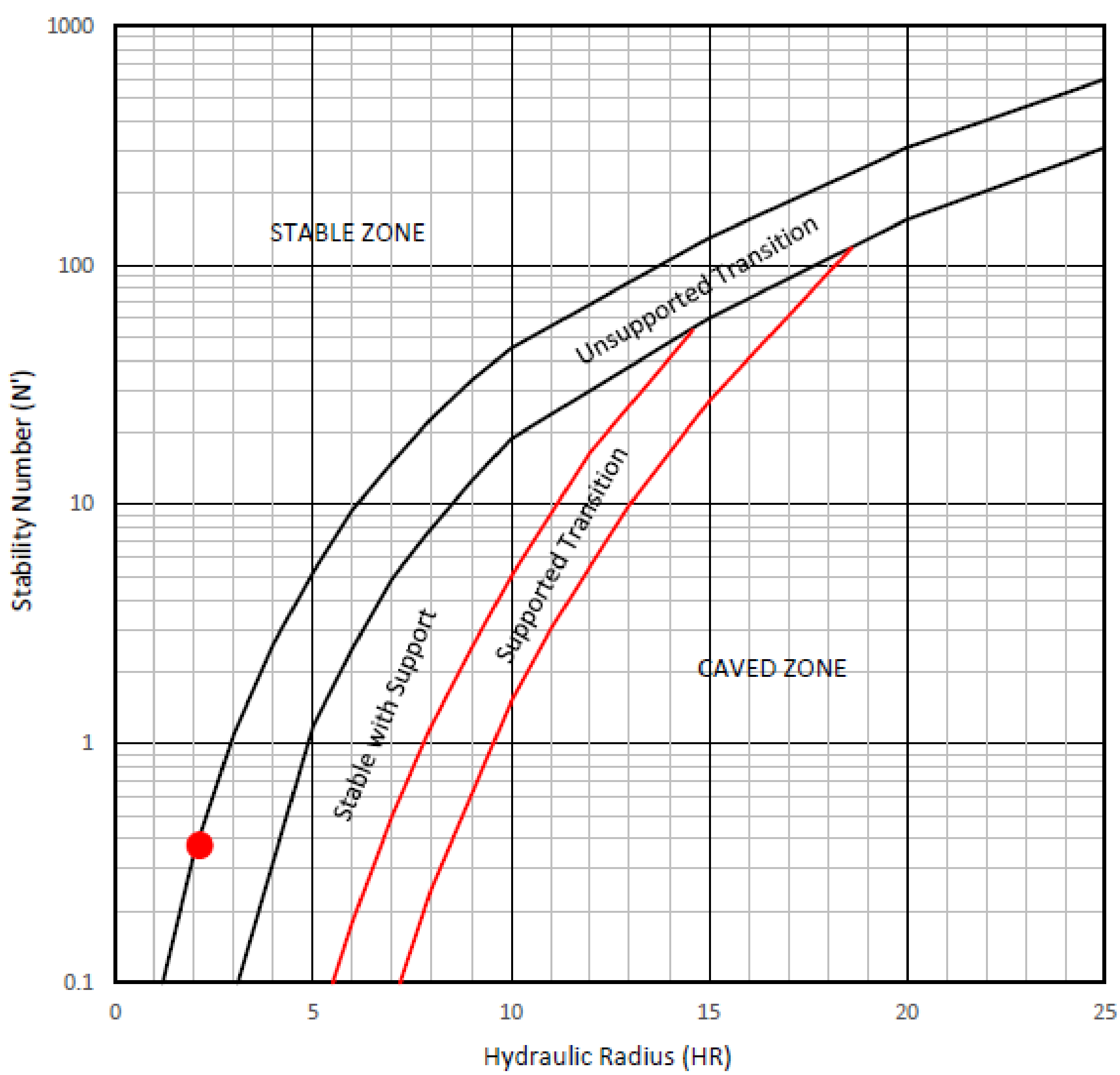
Number of Sub-level = 0 levels

Stability Number, N = Q' × A × B × C

= 0.5 × 0.5 × 0.30 × 5 = **0.38**

Hydraulic Radius = (W × H) / (2W + 2H) = 90 / 42 = **2.14**

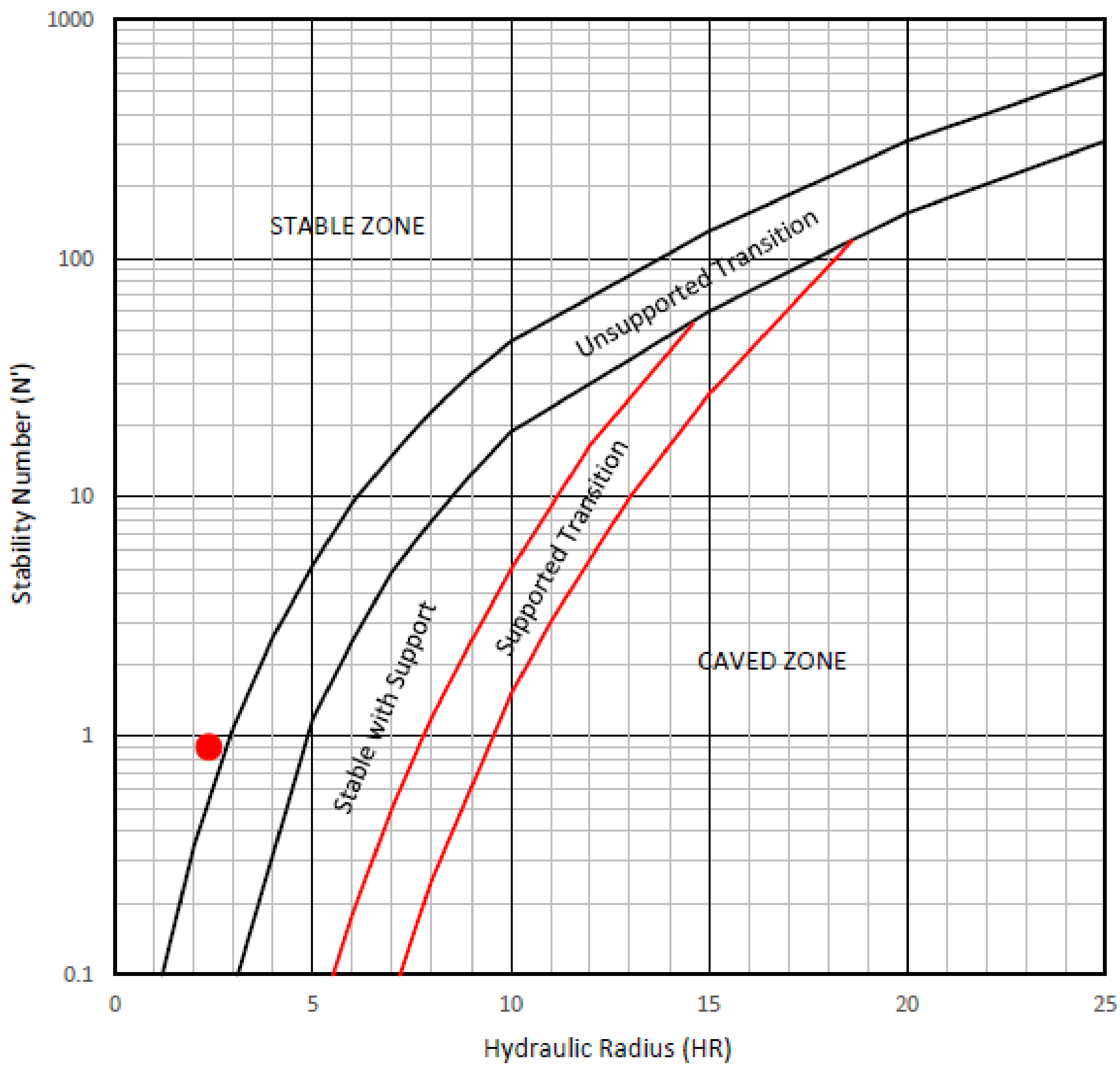
Stability Graph



Stability Graph Analysis - 5m Wide Stope Back in Good Ground

Stope Dimension :	Length =	100	m	Height =	5	m				
	Number of Sub-level =	0	levels							
Stability Number, N =	Q'	×	A	×	B	×	C			
=	5.0	×	0.2	×	0.30	×	3	=	**0.90**	
Hydraulic Radius =	(W × H) / (2W + 2H)	=	500	/	210	=	**2.38**			

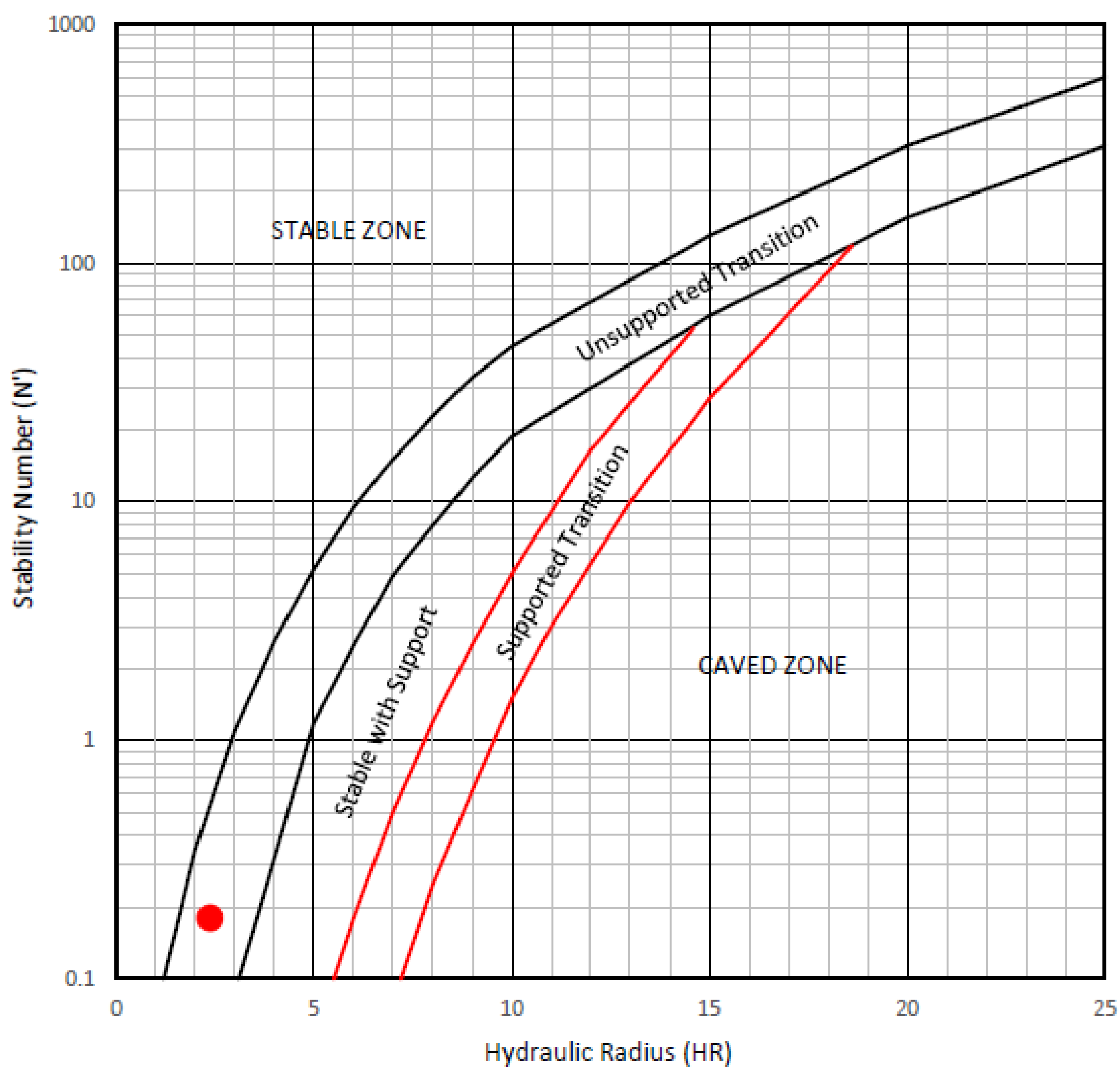
Stability Graph



Stability Graph Analysis - 5m Wide Stope Back in Fair Ground

Stope Dimension :		Length =	100	m		Height =	5	m						
		Number of Sub-level =			0	levels								
Stability Number, N	=		Q'	×	A	×	B	×	C					
		=	1.0	×	0.2	×	0.30	×	3	=	**0.18**			
Hydraulic Radius =	(W × H) / (2W + 2H)	=	500	/	210	=	**2.38**							

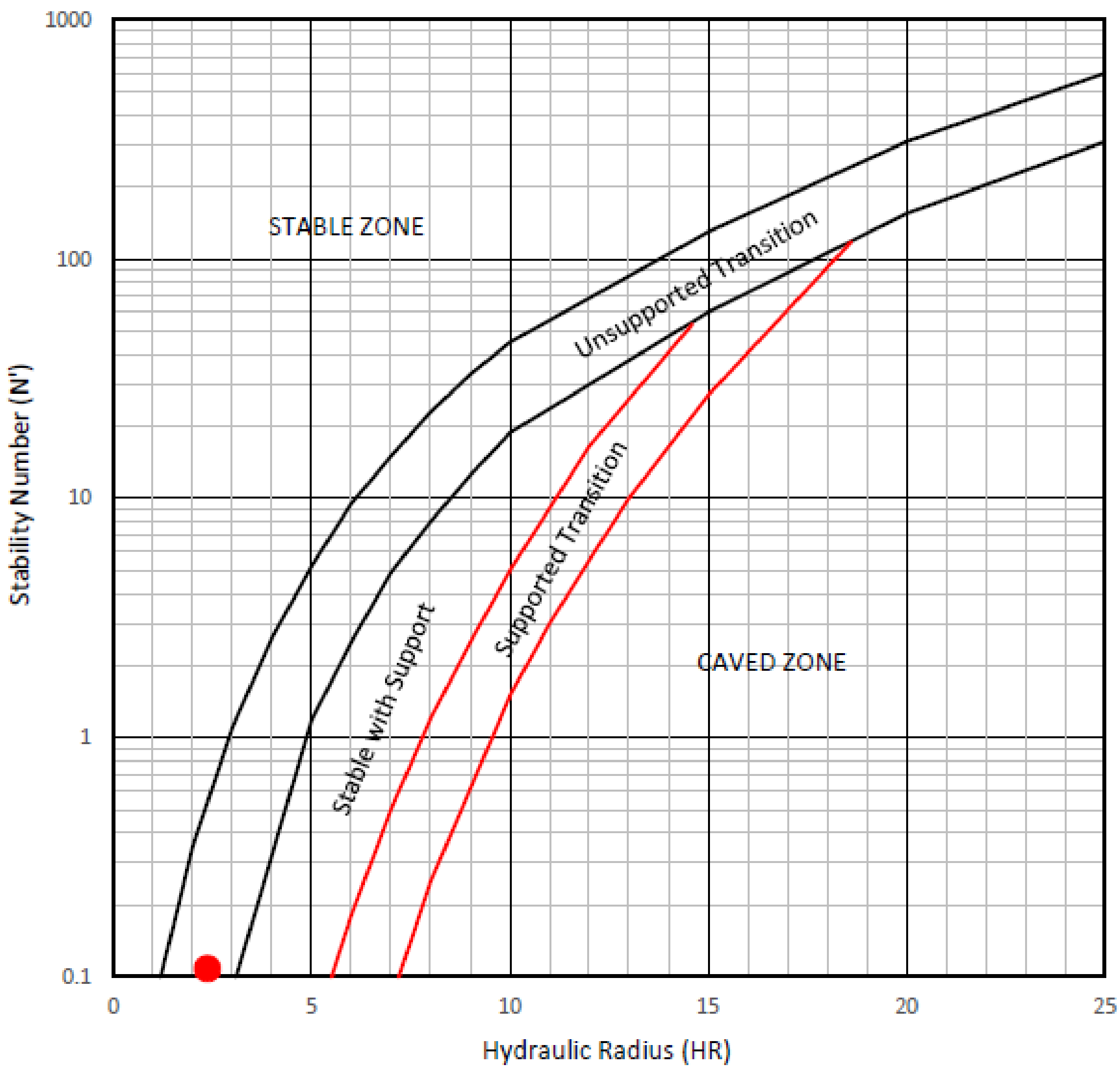
Stability Graph



Stability Graph Analysis - 5m Wide Stope Back in Poor Ground

Stope Dimension :		Length =	100	m		Height =	5	m				
		Number of Sub-level =	0	levels								
Stability Number, N	=	Q'	×	A	×	B	×	C				
	=	0.6	×	0.2	×	0.30	×	3	=	0.11		
Hydraulic Radius =	(W × H) / (2W + 2H)	=	500	/	210	=	2.38					

Stability Graph



APPENDIX – G.

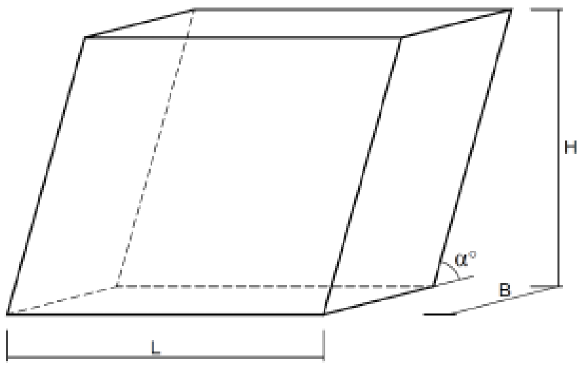
PASTE STRENGTH

For

FILL FACE EXPOSURE

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15	m
L	(Length of Stope Skrike)	:	30	m
B	(Width of Backfill)	:	4	m
α	(angle of stope)	:	70	°

Material Properties of Backfill

γ	(unit weight of fill material)	:	25	kN/m^3
ϕ	(internal friction angle)	:	20	°
K	(Coefficient of fill pressure)	:	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 81.65 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 174.91 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45° + \phi/2)] \cdot \sin(45° + \phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45° + \phi/2)] \cdot \sin(45° + \phi/2)} \right\} = 73.45 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 157.34 \text{ kPa}$$

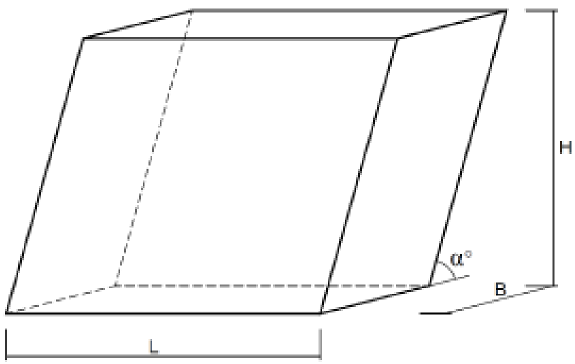
Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin 45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin 45°} \right\} = 73.58 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 157.63 \text{ kPa}$$

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15 m
L	(Length of Stope Skrike)	:	30 m
B	(Width of Backfill)	:	5 m
α	(angle of stope)	:	70 °

Material Properties of Backfill

γ	(unit weight of fill material) :	25	kN/m^3
ϕ	(internal friction angle) :	20	°
K	(Coefficient of fill pressure) :	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 101.71 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \,/\, (1 - \sin\phi) \cdot FS = 217.89 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45° + \phi/2)] \cdot \sin(45° + \phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45° + \phi/2)] \cdot \sin(45° + \phi/2)} \right\} = 86.09 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \,/\, (1 - \sin\phi) \cdot FS = 184.43 \text{ kPa}$$

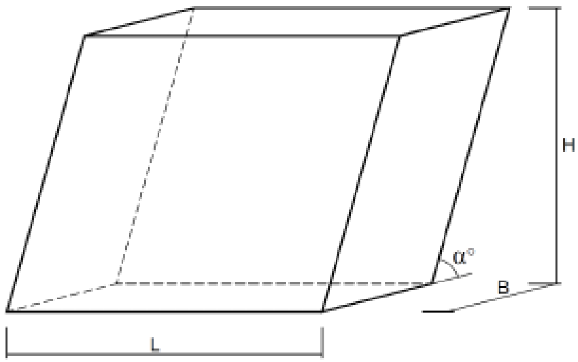
Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin 45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin 45°} \right\} = 86.28 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \,/\, (1 - \sin\phi) \cdot FS = 184.82 \text{ kPa}$$

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15	m
L	(Length of Stope Skrike)	:	30	m
B	(Width of Backfill)	:	6	m
α	(angle of stope)	:	70	°

Material Properties of Backfill

γ	(unit weight of fill material)	:	25	kN/m^3
ϕ	(internal friction angle)	:	20	°
K	(Coefficient of fill pressure)	:	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 121.46 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 260.20 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)} \right\} = 97.26 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 208.34 \text{ kPa}$$

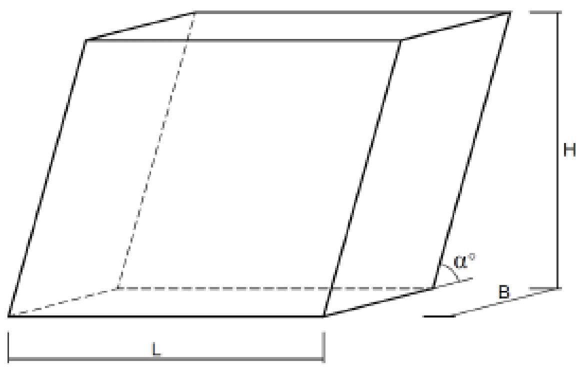
Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin45°} \right\} = 97.49 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 208.85 \text{ kPa}$$

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15	m
L	(Length of Stope Skrike)	:	30	m
B	(Width of Backfill)	:	8	m
α	(angle of stope)	:	70	°

Material Properties of Backfill

γ	(unit weight of fill material)	:	25	kN/m^3
ϕ	(internal friction angle)	:	20	°
K	(Coefficient of fill pressure)	:	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 159.91 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \; / \; (1 - \sin\phi) \cdot FS = 342.56 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45° + \phi/2)] \cdot \sin(45° + \phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45° + \phi/2)] \cdot \sin(45° + \phi/2)} \right\} = 116.07 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \; / \; (1 - \sin\phi) \cdot FS = 248.65 \text{ kPa}$$

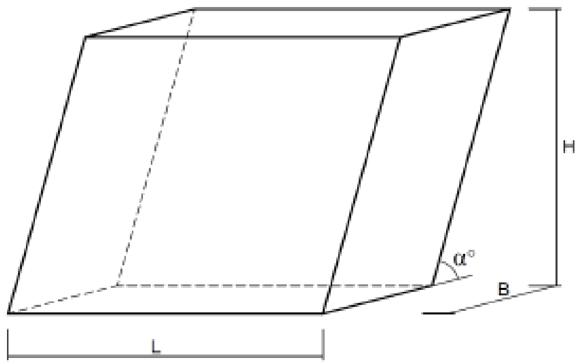
Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin 45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin 45°} \right\} = 116.41 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \; / \; (1 - \sin\phi) \cdot FS = 249.37 \text{ kPa}$$

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15 m
L	(Length of Stope Skrike)	:	30 m
B	(Width of Backfill)	:	10 m
α	(angle of stope)	:	70 °

Material Properties of Backfill

γ	(unit weight of fill material) :	25	kN/m^3
ϕ	(internal friction angle) :	20	°
K	(Coefficient of fill pressure) :	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 196.89 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \,/\, (1 - \sin\phi) \cdot FS = 421.79 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45° + \phi/2)] \cdot \sin(45° + \phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45° + \phi/2)] \cdot \sin(45° + \phi/2)} \right\} = 131.31 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \,/\, (1 - \sin\phi) \cdot FS = 281.30 \text{ kPa}$$

Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin 45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin 45°} \right\} = 131.74 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \,/\, (1 - \sin\phi) \cdot FS = 282.22 \text{ kPa}$$

APPENDIX – H.

NON-CONFORMANCE

RECORD FORM

GROUND CONTROL NON-CONFORMANCE RECORD

To:
From:
CC:
Date:
Re:

Date Non-conformance Recognized: ____________________

Location of Non-conformance: ____________________
(see attached plan)

Type of Non-conformance: ____________________

Required Corrective action: ____________________
(see attached support plan)

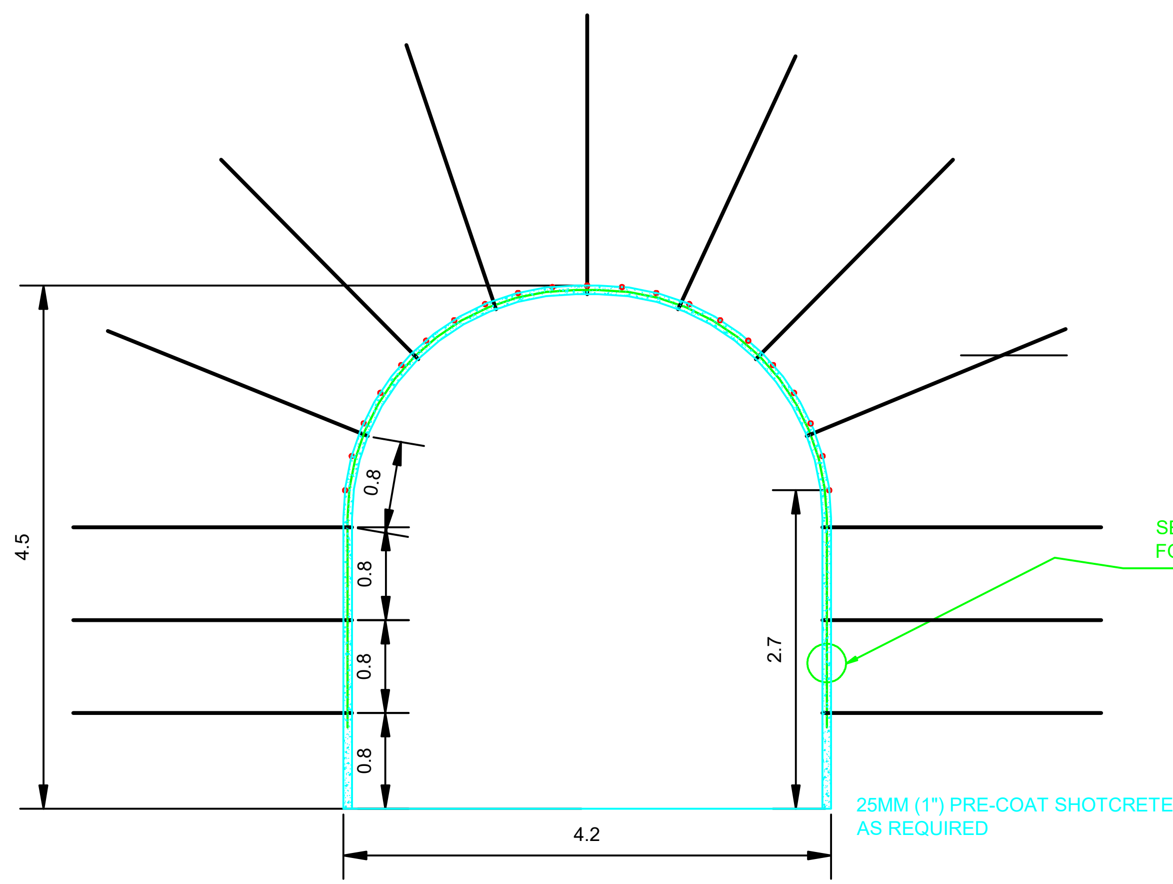
Person(s) responsible for corrective action: ____________________

Date corrective action completed: ____________________

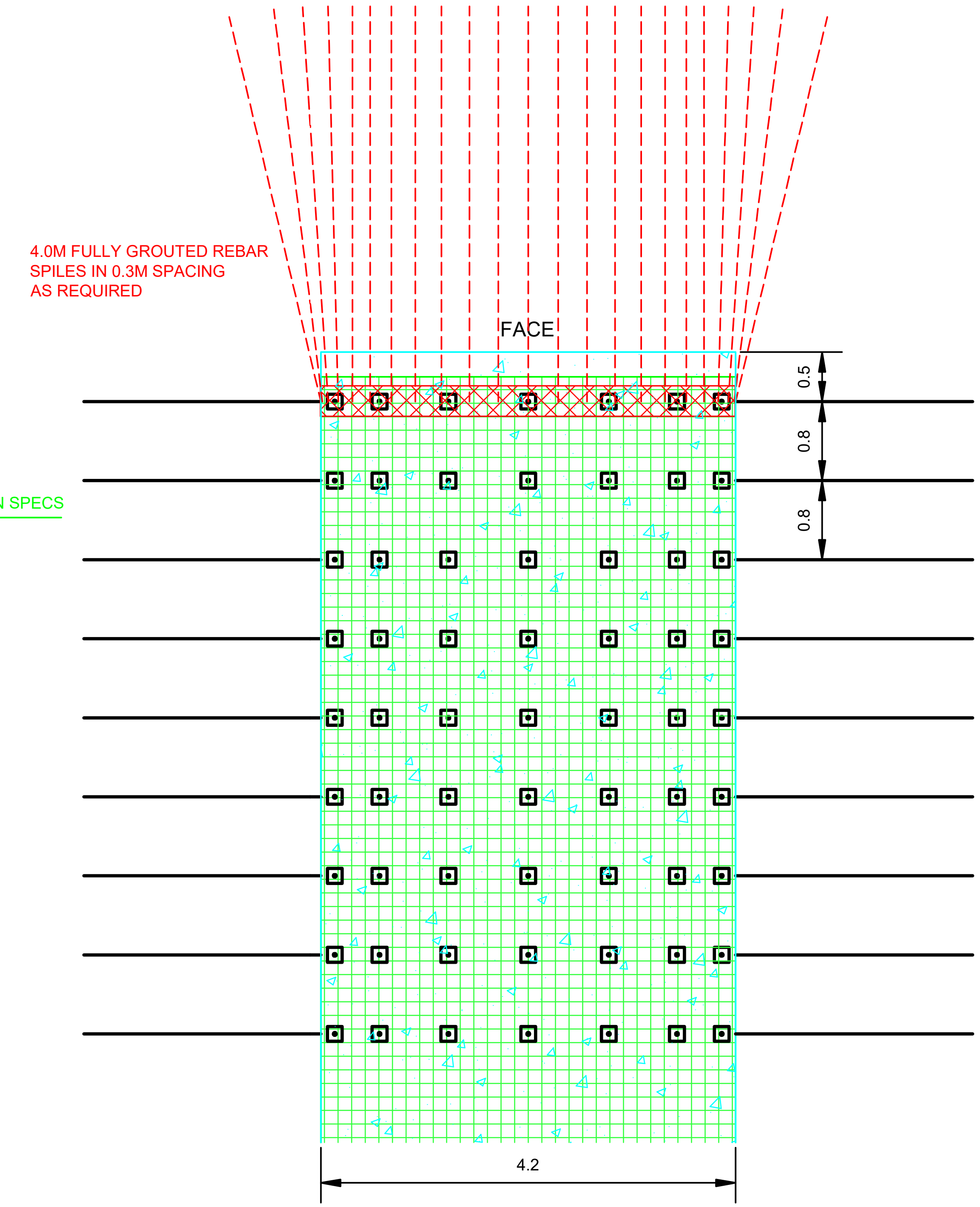
Signoff that corrective action has been completed:

Chief Mine Engineer	
Mine Engineer	
Mine Superintendent	
Geotechnical Engineer	
Shift Boss / Supervisor	

SECTION VIEW



PLAN VIEW



NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5 M FROM THE FACE
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP
8. SPOT BOLT VISIBLE WEDGES AND SLABS AS REQUIRED

SUPPORT ELEMENTS							
LOCATION	ROCK BOLT			SHOTCRETE			MESH
	TYPE	LENGTH	PATTERN	TYPE	PRE-COAT	THICKNESS	
BACK	SWELLEX	2.4M (8')	0.8M X 0.8M	REG	25MM (1")	50MM (2")	4"X4" #6 GAUGE
WALLS	SWELLEX	2.4M (8')	0.8M X 0.8M	REG	25MM (1")	50MM (2")	4"X4" #6 GAUGE

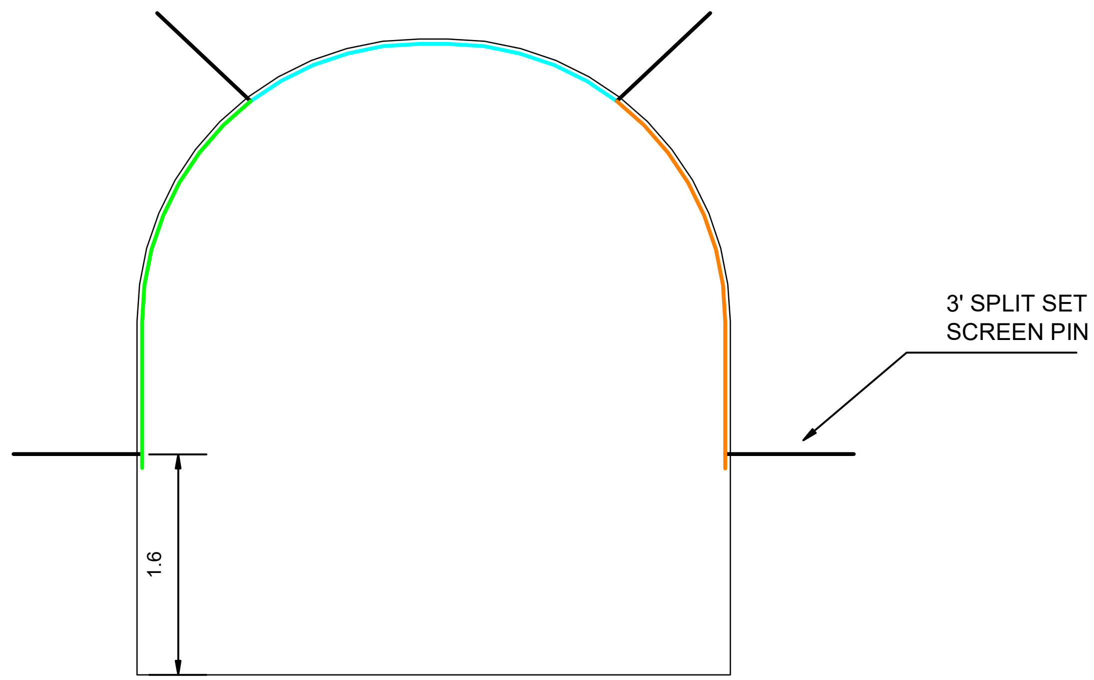
GROUND SUPPORT CONFIGURATIONS

ALEXCO

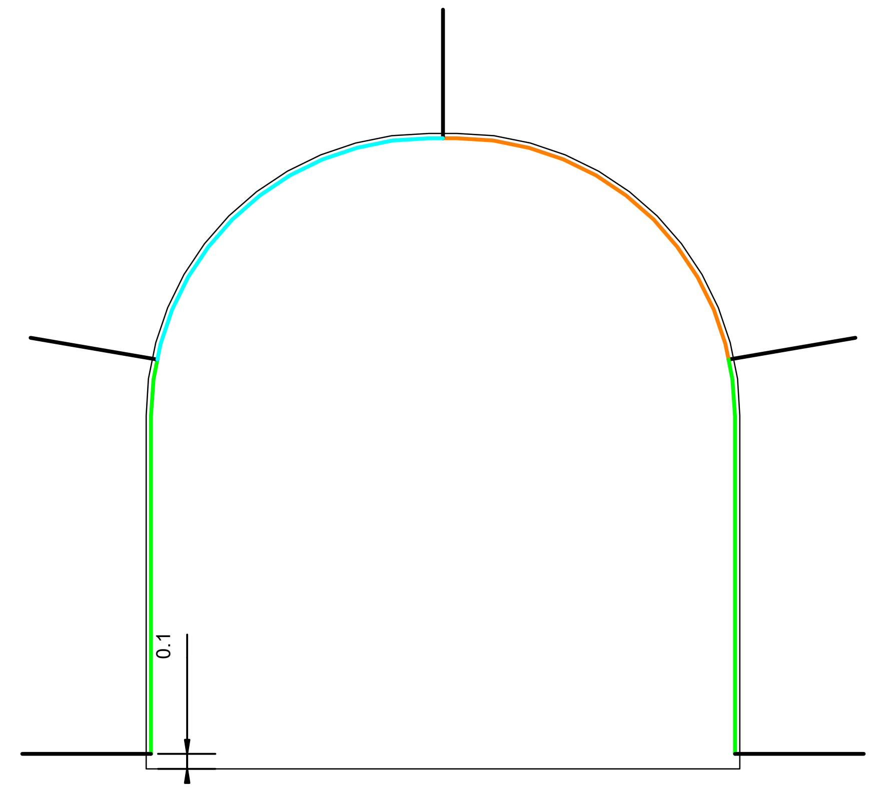
Area	Keno Hill, Yukon Territory, Canada	Rev No.	1
Drawn By	Anna Fazolo		
Date Drawn	23/06/2021		
Scale	N.T.S.		

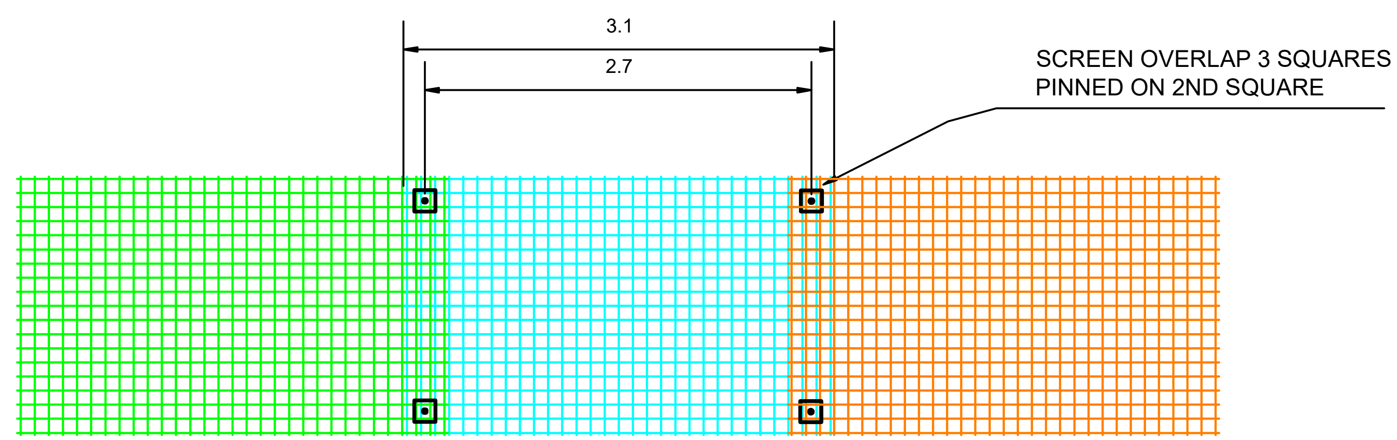
RAMP - CLASS III

SCREEN INSTALLATION IN GOOD GROUND



SCREEN INSTALLATION IN POOR GROUND



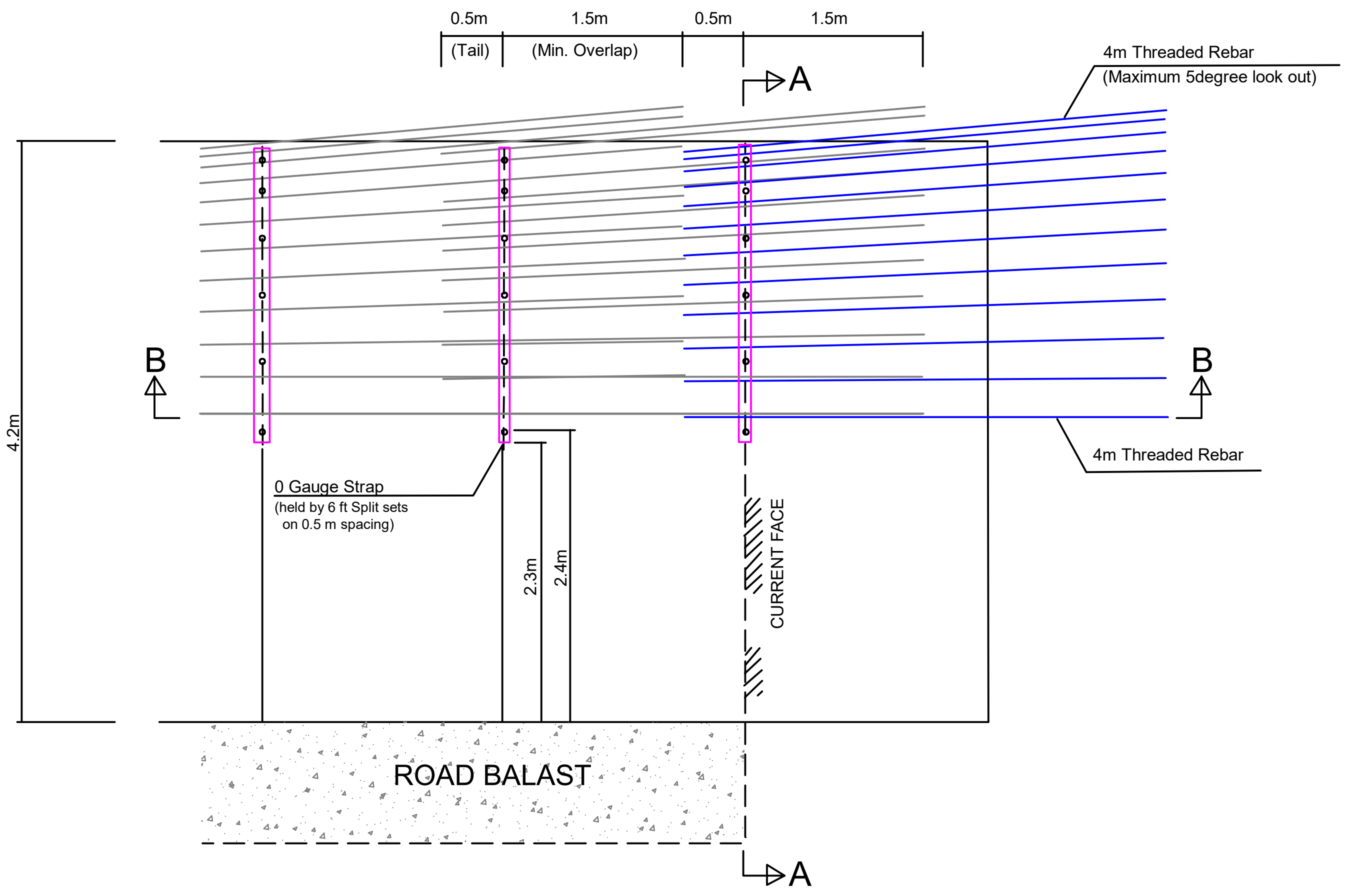


GROUND SUPPORT CONFIGURATIONS

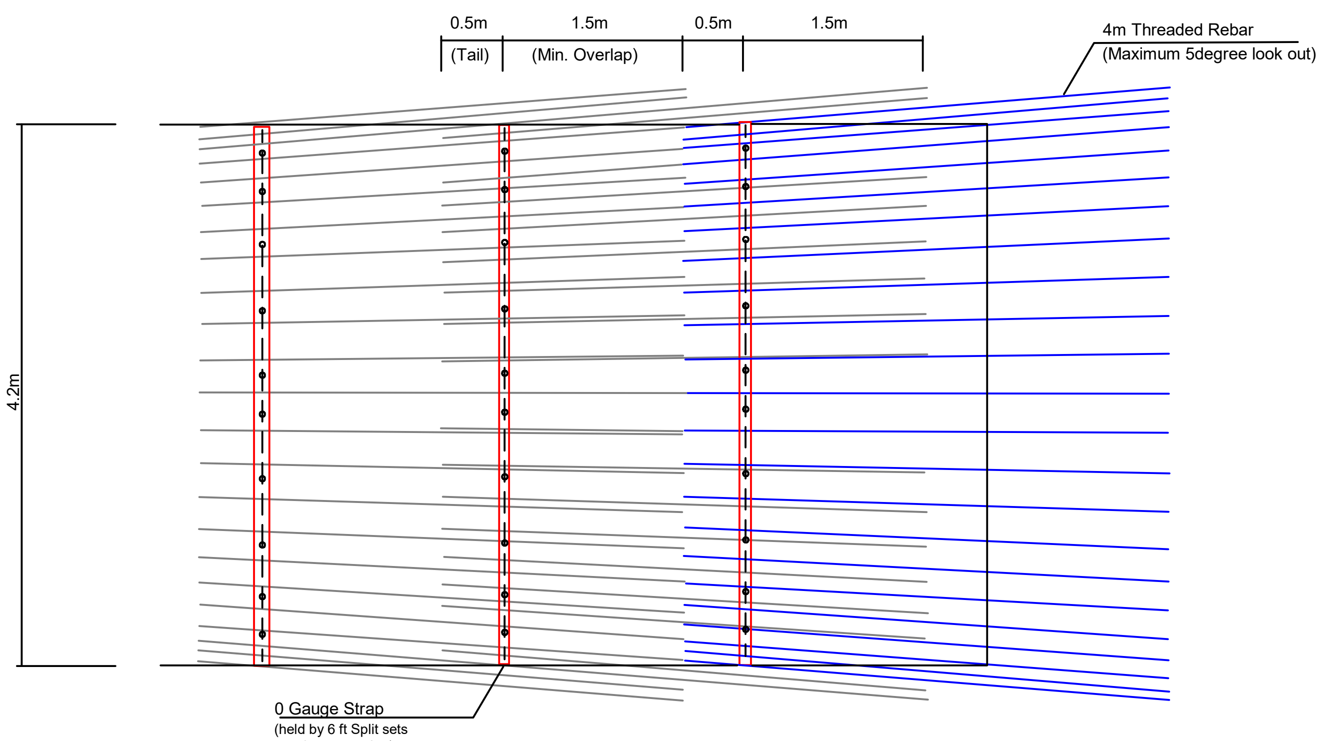
ALEXCO

Area	Keno Hill, Yukon Territory, Canada	Rev No.	1
Drawn By	Anna Fazolo	SCREEN DETAILING	
Date Drawn	23/06/2021		
Scale	N.T.S.		

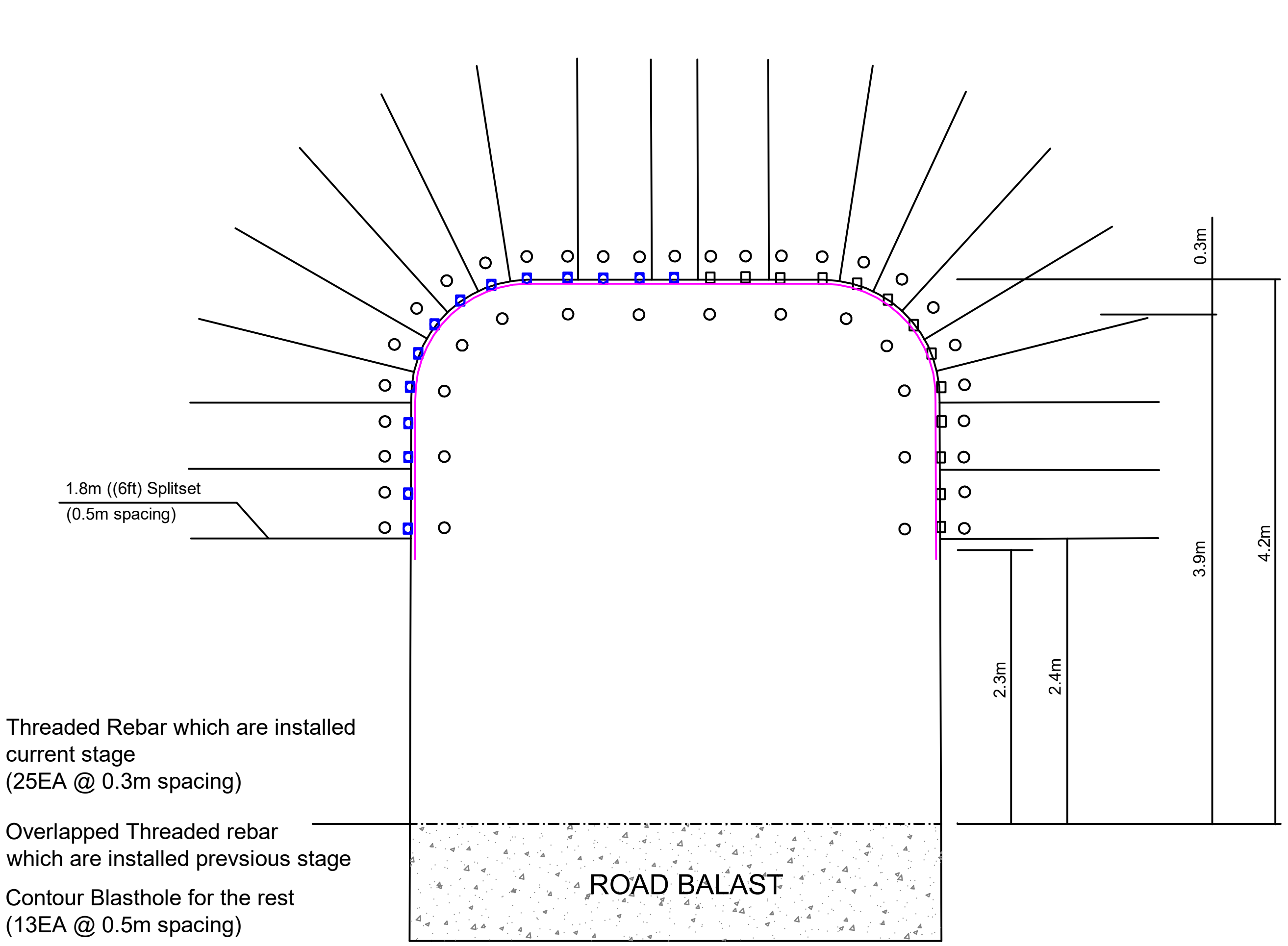
LONG SECTION



PLAN VIEW (SECTION B-B)



CROSS VIEW (SECTION A-A)



- Threaded Rebar which are installed current stage (25EA @ 0.3m spacing)
- Overlapped Threaded rebar which are installed prevsious stage
- Contour Blasthole for the rest (13EA @ 0.5m spacing)

Installation Procedure

Drilling and installing of 4m long threaded rebar

1. Drill holes 0.3m (1ft) along with designed drift periphery to a depth of 3.5m
2. Install a 4m long threaded rebars. A tail of 0.5m (1.5ft) of the rebar should be left out of the hole
3. The spacing between spiling holes should be less than 0.3m (1ft)
4. Contour blast hole should be drilled along with the spilling hole to ensure they remain parallel at all time
5. The rebar should be installed along with drilling, using the other jumbo beam to push them in
6. Holes should be "looked out" slightly from designed grade, with a maximum of 5 degrees.

Pinnng the Tails

1. The tails of the threaded rebarrs are to be *pinned back* using 0 gauge straps and 1.8m (6ft) Splitsets on a 0.5M (1.5ft) spacing

Drilling the remainder of the round

1. When the 4m long threaded rebars have been installed, the remainder of the blast holes can be drilled, Geotechnical engineer will determine the maximum length of the holes (maximum 2.5m)

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Area	Keno Hill, Yukon Territory, Canada
Rev No.	1
Drawn By	kfife
Date Drawn	17/03/2021
Scale	N.T.S.

(SPILING)

Installation Procedure for Spiling

Drilling and Installing of 3m long threaded rebar

1. *Drill holes at a 0.3m (1ft) spacing along the designed drift profile to a depth of 2.5m.*
2. *Install a 3m long threaded rebar. A tail of 0.5m (1.5ft) of the rebar should be left out of the hole.*
3. *Contour blast holes (perimeter holes) should be drilled along with the spiling hole to ensure they always remain parallel.*
4. *The rebar should be installed once the hole is completed, using the other jumbo boom to push them in.*
5. *Holes should be looked out slightly from designed grade, with a maximum of 5 degrees.*

Pinning the Tails

1. *The tails of the threaded rebar are to be pinned back using 0 gauge straps and 1.8m (6ft) split sets on a 0.5m (1.5ft) spacing.*

Drilling the remainder of the round

1. *When the 3m long threaded rebars have been installed, the remainder of the blast holes can be drilled. Maximum round length to not exceed 2.0m when developing through extremely poor ground.*

** Refer to ***Bermingham Operation – Ground Control Management Plan – Report No. 001-2021*** for specification of support standards

APPENDIX C

KENO HILL SILVER DISTRICT, FLAME & MOTH OPERATION

GROUND CONTROL MANAGEMENT PLAN

ALEXCO

Keno Hill Silver District

Flame & Moth Operation

GROUND CONTROL MANAGEMENT PLAN

February 9, 2021

Report No. 001-2021

Alexco Resources Keno Hill Silver District
Flame & Moth Operation
Ground Control Management Plan (GCMP)

First Issue Date: August 21, 2020
Last Modification: February 9, 2021
Final Issue Date:

Version Control

Rev Number	Issue Date	Description & Location of Revision Made	Signatures		
			Originator	Checked	Approved
0	Aug.21 2020	First Draft	W.S	N.C	
1	Sep.7 2020	Add remuck and raise support standard	W.S	N.C	
2	Feb.9 2021	Updated main ramp dimension (4.2mW by 4.2mH)	W.S		
3					
4					
5					

GEOTECHNICAL ENGINEER AUTHORISATION

Authorized

Woo Shin, Ph.D, P.Eng — Date
Rock Mechanics Specialist

MINE MANAGER AUTHORISATION

Authorized

Wayne Zigarlick — Date
Mine General Manager

MINE ENGINEER AUTHORISATION

Authorized

Neil Chambers, P.Eng — Date
Chief Mine Engineer

MINE OPERATION AUTHORISATION

Authorized

Date

Mine Superintendent

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1. GROUND CONTROL MANAGEMENT GUIDELINE

1.1 Strategy

The focus of ground control strategy is the provision of enhanced resources for the collection and utilization of geotechnical information for integration into mine planning and design functions. This will be accomplished by:

- Providing robust geotechnical resources at Alexco Keno Hill Flame & Moth mine site;
- Improving and standardizing the Ground Control Management Plan (GCMP) in use at each of Alexco Keno Hill mine operations;
- Effectively implementing the GCMP at each of Alexco Keno Hill mine operations;
- Developing a structured geotechnical training program, including a ground condition awareness and risk analysis training program; and,

Encouraging a good corporate attitude towards the sponsor and funding of innovative geotechnical research and development.

An effective ground control management strategy for Alexco Keno Hill Silver's operating mine is aimed at qualifying and reducing the geotechnical 'risk' in planning and operation at these mine sites.

1.2 Scope

The scope of this plan is specific to the Alexco Keno Hill Silver Flame & Moth operation and is based on understanding of ground control principals and the geological, geotechnical and mining conditions that apply at the time of the current division.

The GCMP,

- Applied to all underground mine personnel, contractors and visitors who have stated duties under the GCMP;
- Takes effect from the date of issue and is not retrospective;
- Form the basis for training content and specifies requirements for training and competency under the GCMP;
- Outlines the responsibilities and roles of individuals under the GCMP;
- Specifies the Ground Support Rules, requirements for development and production;
- Details the Trigger Action Response Plan (TARP) for both the development and extraction processes; and,
- Does not address controlled or uncontrolled movement of ground resulting in subsidence of ground.

1.3 Detailed Process and Procedures

1.3.1 Mine Design Process

The design of openings, ground support, or pillars should be undertaken in a systemic manner take into general account;

Geological Factors

- Distribution of regional structure
- Distribution of rock types
- Groundwater conditions

Geotechnical Factors

- Back, floor and wall geology and geotechnical parameters
- Known or predicted geological structure and rock defects
- Rock strength parameters (uniaxial compressive strength, cohesion and friction angle)
- In-situ stress
- Expected change in stress accordance with development and extraction sequence
- Ground response from monitoring

Mining Factors

- Excavation dimensions
- Mining methods and sequencing
- Required use of excavation
- Ground support equipment and constraints
- Required life of area or excavation

1.3.2 Ground Control Process

No extraction or development shall take place unless the area has been assessed and an appropriate support system designed, documented and authorized by the Alexco Keno Hill Mine Manager. The GCMP is enacted by following the Ground Control Management Procedures, as listed below,

Table 1.1 Outline of Ground Control Management Procedure

Activities	Summary of Activity
Geotechnical Data Collection	Collection of relevant geological and geotechnical data for characterization of the ground condition.
Modeling, Analysis and Design	Use of the sound geotechnical engineering principles to design excavations (development and production) which are fit for their intended use. Where important, the sequencing of the excavations may be described and any adjustments to the proposed excavation design and/or sequence re-evaluated.

Excavation Performance Monitoring	Ensuring excavations are mined to appropriate dimensions and are properly supported. Where appropriate equipment and/or procedures should be used to monitor conditions.
Remedial Measures	Determination of appropriate, effective techniques for post failure treatment to regain control of excavations as necessary. This includes, but is not limited to, the rehabilitation of failed or old mining areas and ground support, and back analysis of failures, if appropriate.
Producing the GCMP	Incorporating the above into a clear and concise document that can be used as a guide for managing ground conditions. The document should explain the philosophy of the ground control system and list any assumptions used in the design. The plan should be able to be read by third parties to quickly gain an understanding of the principle aspects of ground control at the mine and the procedures and/or processes in place for managing these aspects.

1.4 Objectives

The objectives of the GCMP are as follows;

- Reduce the risk of uncontrolled ground failure
- Contribute to the development and maintenance of a safe working environment
- Contribute to efficient extraction of ore reserves

The objectives are achieved through;

- Identification of hazardous areas and assess associated risks
- Design and implementation of appropriate ground control system
- Communicating known hazards to the workforce in advance of both development and production
- Design and implementation of systems to detect and control change (TARP)
- Design and implementation of procedures associated with ground control including a Standard Operation Procedure (SOP) for installation or ground support
- Providing clear and unambiguous definitions of roles and responsibilities for individuals working under the plan
- Internal and external auditing to assess the effectiveness and degree of compliance with the GCMP and assist in identifying improvement requirements

1.5 Requirements

Human Resources, Equipment, Training, Materials and Systems

Mining operation should provide for sufficient resourcing to implement and maintain a ground control strategy. The key human resource needed to achieve this aim is competent full-time site based geotechnical engineer or a combination of site based personnel and external resources. Other people appointed in the roles listed in Section 3.1, also need adequate training to meet their requirements covered in the GCMP.

Aside from the basic requirement of fulfilling regulatory standards, all equipment used for ground control must be appropriate for its intended use.

Personnel performing ground control tasks must be adequately trained and deemed competent in the correct use of ground control equipment and materials. As such, the mine should provide resources to document the specifications and develop Standard Operating Procedures (SOP's) where necessary or appropriate. It is then the responsibility of the mine to train and access operations in the use of these SOPs.

Data Collection Techniques and Risk Assessment

The collection of suitable, high quality data is the basis for building a solid ground control strategy. Time should be spent determining what type of data can and should be collected for use in the efforts of ground control.

The concept of risk is an integral part of the ground control strategy, such that mitigation of risk to personnel and equipment is routinely considered. System of ground control management should be thought with the practice of assignment the highest practicable level of hazard control whenever possible.

1.6 Ground Control Definitions

Nominal: Refers to an approximate dimension of a drift utilizing the same support requirements.

Primary Support: Ground Support Anchors (GSA) used in conjunction with wire mesh. Accepted bolts of types are 1.8m and 2.4m fully grouted rebar bolts, expandable friction bolts (Swellex) or split tube friction bolts (split set). Accepted wire mesh type are either galvanized chain link mesh or welded wire panels. Welded wire mesh is the preferred type of mesh to use with shotcrete.

Secondary Support: Bolts longer than 2.4m in length. Accepted bolt types are 24 tonnes expandable friction bolts (Super Swellex or Connectable bolt) and cable bolts.

Short Term Drift: Anticipated working life of less than 2 years. In these areas, corrosion protection is not typically necessary. Regular friction bolts are acceptable for the installation. This includes uncoated Swellex bolts.

Long Term Drift: The corrosion protection is required where the working life is anticipated longer than 2 years, depending on the ground water condition. Regular friction bolt and screen may be applicable for these area in dry condition under approval from ground control engineer. Regular friction bolts for long term drift in wet condition should re-bolt under inspection by ground control engineer within 2 years after installation following full-out test results. Installation and quality control program of uncoated bolts for long term drift need to be reviewed and approved by Geotechnical Engineer or designated Engineer under Chief Mine Engineer's supervision.

2. DEVELOPMENT OF GROUND CONTROL MANAGEMENT PLAN

2.1 Development of Formal GCMP

Keno Hill operations, which include Flame & Moth shall conduct site based risk assessments to support the development of GCMP and related activities. Risk assessments generally includes but are not limited to the following key consideration:

- Geotechnical assessment and monitoring
- Ground stability, surface subsidence and potential in-rush (air, mud, bodies of water, etc.)
- Material and equipment selection criteria
- Identification of required standard operation or work procedures (based on consequences)
- Workforce training and competency levels
- Mining methods and operations planning criteria (including excavation size and sequence)
- Significant changes in opening plans or ground conditions
- Ground condition monitoring methods (focused on earliest possible detection)
- Emergency response planning.

2.2 Content of the GCMP

Generally, the following information may be included within the GCMP:

- A process of technical mine planning
- Technical competency requirements of personnel and resources involved in the management of ground control (including inspection) and analysis of technical data
- The technical data utilized in modeling, design, excavation and support methods
- Procedures to allow person to work in conditions where the hazards have been identified, formally assessed and controlled, standard operating procedures for work in such areas have been produced

- Methods, materials certification, and quality criteria for stability enhancement such as rock fixtures, plates, backfill, barricades, shotcrete, cribbing, wire mesh, etc.
- Corrective action for removal of loose or unconsolidated materials
- The ongoing inspection processes for ground control conditions which specify corrective action and emergency procedures. Rock mass conditions should be monitored for all departures from normal
- Specifications of monitoring equipment for type, location and frequency of data collection and review.

2.3 Consistent of Systemic Approach

The GCMP presents a systematic approach to allow the reader to understand the important aspects of ground control for the mine. Factual information should be clearly separable from any inferred or analytical judgements proposed in the document. These should be a logical flow from data collection, analysis and design to monitoring and back analysis work.

Given that rock is a dynamic material and mining is a dynamic process, the geotechnical engineer must usually make a general assumption about the property of a given rock mass for design purpose at a given time in the mining process. Reliance is then placed on an "observational approach" to monitor the effectiveness of the design and the appropriateness of the design assumptions. The concept of the observational approach was first described by Terzaghi and Peck in 1967 and can be outlined as follows:

- Decide on some sort of initial mine layout
- Begin mining
- Monitor the rock mass response – normally visual or by monitoring equipment
- Redesign based on the observed field conditions – model calibration.

Using this iterative process, the geotechnical engineer builds a case for the reliability of their assumptions over time and in doing so becomes more confident in predicting better safety and accuracy of technical design.

Detailed ground control management system and underground mine and mine planning system based on the concept suggested Terzaghi and Peck (1967) are shown in Figure 2.1 and 2.2

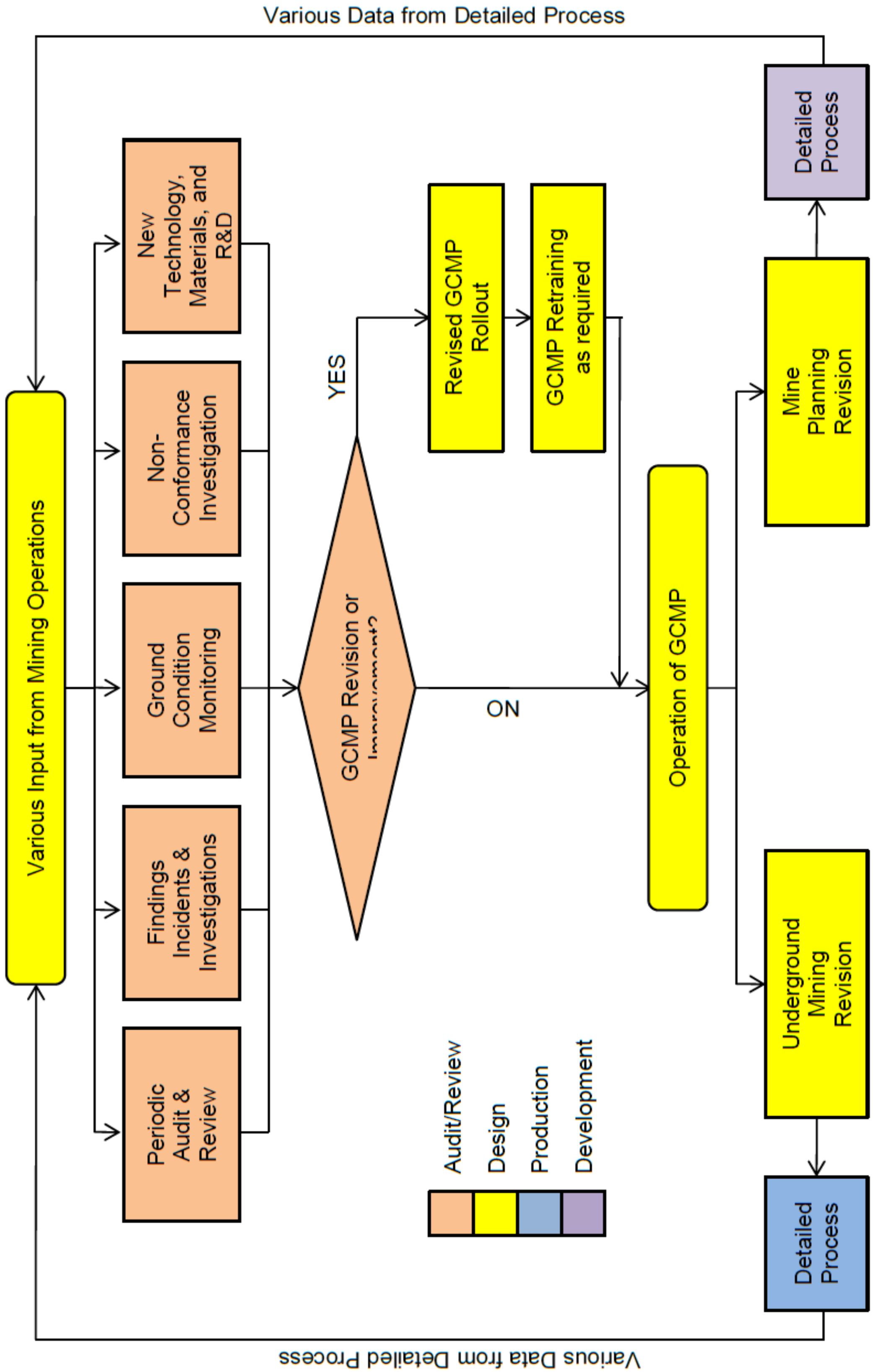


Figure 2.1. Ground Control Management System

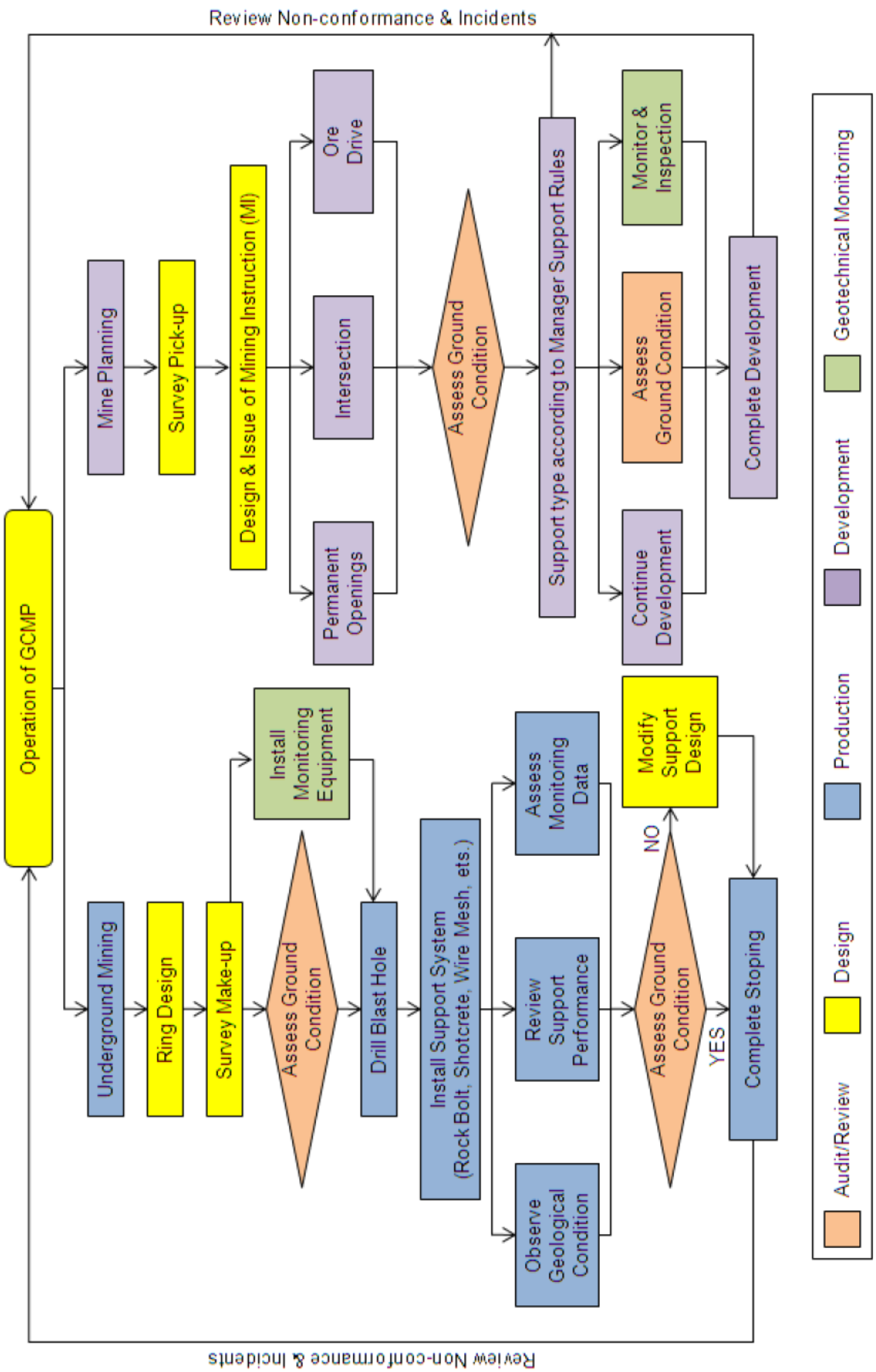


Figure 2.2. Detailed Underground Mining and Mine Planning system

2.4 Risk Assessment

The main focus of the Keno Hill Flame & Moth Mine Ground Control Management Plan is to facilitate early recognition and timely control of ground control hazards by the underground workforce. It is recognized that not all hazards are predictable and accurately defined in advance of mining by such methods as exploration, geological evaluation and therefore the GCMP must remain responsive to ground conditions and mining variations to reduce the risks to an acceptable level.

2.4.1 Hazard Identification

The key hazard associated with underground development in regard to ground control is rock fall due to;

- Geological structure
- Over-excavation
- Groundwater
- Ground movement
- Stress change
- Drill and blast techniques

Geological Structure

Geological structures include normal faults, strike slip faults and folds. These can have an adverse impact on conditions primary through weakening the rock mass conditions and creating unstable wedges in the back and walls.

Over-ecavation

Increasing the span or heights over the specified dimensions can have an adverse impact because;

- The capacity of the ground to support itself may be exceeded
- By increasing the size of the potential wedge over the capacity of the ground support elements.

Groundwater

Ground water in the general back or walls can have an adverse impact on ground control. Water can weaken the immediate ground or reduce the integrity of ground support, particularly cement based support element such as shotcrete and grout. It can have a lubricating effect on slip and joints.

Water can be from;

- Natural source along with discontinuity
- Exploration drill holes

Ground Movement

Ground movement is a result of post mining relaxation or change in local conditions. Ground movement can be monitored for underground mine with various instruments, from relatively simple disto-meter and Ground Movement Monitor (GMM) to multi-point extensometers. Change in rate of movement may mean that the primary or secondary support design may need to be supplemented or access to that area restricted.

Stress Change

Changes in ground stress can lead to loading ground support and possible failure. At Keno Hill underground this is not likely to occur around all underground openings including main ramp, production drift and long hole stope access drift areas but may become apparent in development at depth. Indicators of stress may include flattering or buckling or rock bolt plates, straining of cable plates, bird caging of secondary support tendons, spalling of shotcrete and unusual popping sound caused by rock burst.

Unusual roof noise: audible cracking, squeaking or “banging” observed in the backs or walls generally indicate that the ground is “working”. This is a sign of ground instability which can lead to loss of control and ground failure. To date this has not been reported at Keno Hill underground. Because this noises associate with major faults, immediate notice by miners and special remedial action were required for this case.

Drill and Blast Techniques

Drill and blast is the one major variable that can be controlled. Ground control can be enhanced by ensuring that drilling is to design and the appropriate explosives and numbers are used when firing development headings. Drill and blasting techniques should limit collateral damage to host rock surrounding the excavation.

2.4.2 Likelihood and Consequence of Occurrence of the Risk

The likelihood of occurrence can be based on both past experiences and judgements; it must be clearly stated which,

In some circumstances the likelihood of a potential failure may be quantified from failure record in Keno Hill Ground Control Risk Assessment Report (Appendix-A). The report should be used to record all back and/or wall failures that occurred in any supported ground. A failure that requires an Incident Report shall be recorded in the Keno Hill Mine Incident Investigation Report.

2.4.3 Risk Assessment Report

The risk associated with ground related and other identified hazards are estimated by considering the “Consequence, Exposure and Probability of the Hazard”. During the daily and weekly meetings risk shall be reviewed and if required highlighted so that appropriate action can be taken.

2.4.4 Trigger Action Response Plan

The aim of a Triger Action Response Plan (TARP) is to ensure a response to changed ground conditions at an early stage. The TARP for use in Keno Hill Flame & Moth mine is shown in Appendix-B. From the empirical guideline and numerical study, 11 different types of ground supporting regimes are recommended for the Keno Hill FM UG mine depending on ground condition, life time of openings and development geometry conditions. The TARP provides a list of indicators, observable at operator level that can be used to guide the selection of the appropriate Support Type as defined by the Ground Support Standards (see Section 5.4)

The key indicators are;

- Rock qualification (GSI)
- Contact orientations between FW/HW of Fault zone
- Presence and condition of the geotechnical structures

In addition of the Geotechnical Engineer may dictate extra support based on geotechnical monitoring or visual inspections.

The Geotechnical Engineer or Supervisor will conduct an inspection of the area in the event the ground Support Type is changed.

2.5 Ground Support Installation Guidelines

The designed support shall be installed to established standard Flame & Moth mine operating procedures and as outlined in Keno Hill Ground Support Rules and TARP's.

Operators shall observe the ground conditions and monitor effectiveness of ground support installation (e.g. drilling rates, water loss/gain, bolting problems, voids etc) and report any unusual conditions and action the TARP's. The operators shall only use approved (UG Mine Manager or Supervisor) installation equipment and support hardware.

The requirements for ground support installation are listed below;

2.5.1 Split Tube Friction Bolt

- Bit gauge is critical for this type of bolts, and the hole size should be monitored to maintain the inside-diameter (ID) between 35 to 38 mm. Holes should be drilled 150 mm longer than the bolt to ensure pressure on the plate when installed.
- If split sets are the primary ground support in an area, secondary support will also may be required if the excavation is a long-term excavation under inspection by geotechnical engineer.
- The use of drive-time tests is also useful.

- Pull tests should be conducted routinely and can be done in the current headings. It should be ensured that a large portion of the testing be undertaken in the excavation back, as this area has the highest risk of poor support performance related failure.
- Report all split set defects to the technical department so that they can follow up with manufacturer on quality control.

2.5.2 Expendable Friction Bolt – Standard and Super

- Bit gauge is less important for optimum performance. Use a 32 mm to 43 mm bit with standard bolt (12 ton) installation and a 43 mm to 52 mm bit for super bolt (24 ton). Pull testing has demonstrated that 38 mm diameter bits give optimum anchorage for standard bolt. Undersized or oversized bits will reduce the anchorage capacity of friction bolts.
- It is important that the bolt is pressurized to the recommended 300 bar. Using a pressure less than the recommended value will reduce the anchorage capacity.
- It is important that the bolt is held at the 300 bar of pressure for a full 6 seconds as per the manufacturer's recommendation. Failure to hold the pressure for this length of time could reduce the anchorage of the bolt. This is a function of the bolt and not the pump and so the guideline should be followed regardless of the pump power.
- Re-pressurizing a friction bolt can give an indication if the bolt has been damaged in the cases where the damage would cause it to leak and not hold pressure (for example the bolt was sheared off).
- Pull tests should be conducted routinely.
- Report all friction bolt defects to the technical department so that they can follow up with manufacturer on quality control.

2.5.3 Cable Bolt

- Cable bolt should have an interrupted lay at approximately 0.6 m to 0.9 m centers to provide anchorage along the length of the bolt. Garford pattern cables are normally used.
- Cable bolts should be fully grouted with cement grout using a grout tube and bleeder tube. The grout tube (20 mm) always terminates at the lowest point on the bolt (the collar in an up hole and the toe of the hole for a hole drilled angled down). Bleeder tubes (10 -12 mm) always terminate at the highest point on a bolt (the toe on an up hole or the collar for a hole drilled angled down). Tubes should never extend past the fish hook anchor of the cable to prevent them from being bent back and kinked.
- The quality of the grout is important to the effectiveness of cable bolt support. The water/cement ratio should be approximately 0.35 to 0.4. Grout that is too thin will reduce bolt strength. Grout that is too thick will make proper grouting difficult.

- The use of grout additives, such as BASF's "Flow Cable", should be considered in order to optimize the grout' characteristics.
- Bolts are grouted until return of grout is achieved from the bleeder tube. Both tubes are pinched off to keep them grout filled. Empty bleeder tubes create voids along the cable and reduce its anchorage.
- If plates are used they should be installed no sooner than 48 hours after grouting, to ensure that the grout has had time to cure.

2.5.4 Rebar Bolt

- Two types of resin capsules have been recommended: the first is of a quick set type that will set within 30 seconds after mixing. The second type is a slower set that activates after about two to five minutes. The fast set capsules are to be installed at the end of the hole followed by the slower set cartridges. This practice allows the bolts to be pre-tensioned prior to the setting of the full column resin (with approximately 1 ton of pre-tension for each 20 to 25 foot-pounds of torque). This process clamps the rock mass together and then secures the bolt.
- Mixing instructions should be adhered to, paying special attention to the number of revolutions (of the jackleg or stopper) while the bolt is being spun for mixing, and the hole-gauge.
- Resin needs to be within its shelf life (the expiration date is marked on the end of the box) and in good condition. Damaged or stale resin must be disposed of, unused, in an environmentally appropriate site-specific manner.
- Pull testing should be done on a regular basis. If testing indicates that the bolts have not been installed properly, then they should be individually checked and new bolts installed in the immediate vicinity to replace them. If subsequent testing shows that the bolt installation remains sub-standard, the issue needs to be escalated, and the miner's supervisor needs to become involved with corrective action. All testing needs to be documented.
- No more than 100 mm of thread (sometimes referred to as the tail) should be allowed to stick out of the hole beyond the collar. If the tail is longer than this, then new bolt should be installed immediately adjacent to the bolt in question.

2.5.5 Wire Mesh

- Mesh must be 100mm x 100mm welded mesh.
- Mesh may be pinned with friction bolts, but all other bolts must be the prescribed type and at correct bolt spacing and ring spacing.

- Adjacent sheets of mesh must overlap by 3 squares with the bolt pinning them together in the middle (second) row of overlap.
- Always advance your wire from supported ground. When working with jackleg do not drill holes beyond the next row of bolts to be installed – the “one hole, one bolt, policy”.
- As far as practicable once installed mesh must be pushed to fit shape of the excavation to guard against voids forming behind the shotcrete once it is applied.

2.5.6 Shotcrete

- All Headings are to be hydro scaled prior to shotcrete application to ensure any loose material is washed away and to remove excess dust, both of which contribute to shotcrete fallouts.
- All shotcrete applied to headings will be as per the prescribed mix design.
- Shotcrete thicknesses must be comply with the relevant Ground Support Type currently applicable to that specific heading;
- All headings are considered non-entry for a period of 1 hour after shotcreting to allow the shotcrete to achieve 1MPa, which is the industry standard for shotcrete re-entry strengths;
- Where mesh is not applied fiber reinforced shotcrete as per the prescribed mix design will be used;
- Where shotcrete is unavailable for any reason all development shall use mesh for the relevant Ground Support Type.
- Where ground conditions dictate fiber reinforced shotcrete will be applied before installing mesh with shotcrete then being sprayed over the mesh.

3. ROLES AND RESPONSIBILITY

The Mine General Manager or designated personnel has the overall responsibility for implementation, review and revision of the GCMP and is the only official who may authorize the GCMP, its review and revisions.

The Ken Hill Underground Mine technical team, in conjunction with operation staff, will determine the appropriate levels of development support, monitoring and hazard response for all headings and stopes.

3.1 Ground Control Management Responsibilities

Relevant personnel (employees, staff, contractors and visitors) entering Keno Hill Silver District Operation should be made aware of and take note of their responsibilities under the Keno Hill Underground GCMP, relevant regulations and implied duty of care.

The Keno Hill mine GCMP defines the specific responsibilities of key personnel in terms of the Flame & Moth underground mining process.

Mine Manager / Chief Engineer

- Ensure the requirements of the GCMP are compiled with
- Shall approve and sign all Managers Support Rules
- Shall oversee and drive the GCMP and ensure the GCMP and TARP are audited annually
- Appoint and ensure that the necessary resources are provided to manage the GCMP
- Ensure budgets are sufficient to provide for adequate geological/geotechnical understanding of the mining environment
- Provide guidance and input as required

Mine Superintendent

- Ensure the requirements of the GCMP are compiled with
- Ensure sufficient materials are on site to implement the Ground Support Rules
- Ensure clear communication of the GCMP to all Cementation contracting personnel
- Shall communicate operational deficiencies and improvements in the GCMP to relevant technical support personnel
- Ensure channels of communication are open for the operators to make suggestions regarding the GCMP
- Provide guidance and input ground support as required

Mine/Geotechnical Engineer

- Ensure that GCMP is taken into account in mine design
- Arrange the annual internal and external auditing of the GCMP
- Provide guidance and input to ground control as required
- Responsible for ground support in the mine
- Provide geotechnical input into the ground control management process at Flame & Moth Mine

- Undertake regular inspections of their work areas, specifically back and wall support, making reports of any non-conformance or deterioration
- Facilitate the design of the various Support Types, in terms of Ground Support Rules
- Ensure that required testing of support performance is carried out
- Manage the installation, reading and interpretation of monitoring equipment and ensure findings are communicated to management in a timely manner
- Ensure ongoing monitoring occurs of the ground control and geotechnical/geological environmental
- Determine and communicate trigger levels and TARP

Geologist

- Shall gather data and information, in so far as it relate to geological and geotechnical parameters and record that information in face mapping, line mapping and database
- Report areas of concern to the Geotechnical Engineer, Supervisor or other relevant staff
- Provide advice on any geological issues as they relate to ground support
- Shall ensure that the geological model is updated and ensure that the geology and structure indicated on the plans is correct

Shift Boss/Supervisor

- Ensure that those people under their charge who have responsibilities under the GCMP understand and perform those duties
- Contribute to the design and implementation of the various Support Types
- Communicate minutes and outcomes of all meetings to all mining crews
- Undertake inspections of the backs and walls or the mine and ground support
- Ensure crews are reporting all unusual visual observations, ground noise or ground (control) related events on their plods or end of shift reports
- Ensure that the appropriate changes in support hardware are made in accordance with the Underground Inspection Memo, TARP’s and other instructions
- Quality control: ensure Shift Supervisors and Operators are aware of and conduct necessary QC checks on installed ground support.

Operators

- Develop headings and install support in accordance with the Ground Support rules

- Verbally report any changes or anomalies in ground conditions or support behavior to the Shift Boss / Supervisors
- Install monitoring tools as instructed
- Quality Control: ensure the necessary QC checks on installed ground support are conducted in a timely manner

Geotechnical Consultant

- Provide advice on any geotechnical issues raised by the Mine Manager, Chief Engineer, Mine/Geotechnical Engineer or other technical support team
- Periodically review and manage change / update of the GCMP

3.2 Other Key Personnel

Mine Surveyor

- Shall report to the Mine Engineer, Shift Bass/Supervisor and Mine/Geotechnical Engineer any development or intersection that exceeds design dimensions
- Survey the locations of all types of monitoring instruments and boreholes drilled through the mine and record

Safety and Training Officer

- Assist with the development of training modules that address the GCMP in conjunction with the Geotechnical Engineer
- Develop and maintain a comprehensive training and assessment plan and maintain records of any training and assessment conducted in compliance with the GCMP

3.3 Temporary Delegation of Responsibilities

The Keno Hill mine system of mining on a 24 hours per day, 7 days a week basis (with personnel requiring rostered time off), requires particular attention when considering available personnel. Where staffs are absent or unavailable, it is the responsibility of individuals to provide clear and unambiguous delegation of their authority to appropriate proxy. Such delegation should be made in writing (including e-mail) and will include details of;

- Contact details for the proxy
- Duration of delegation
- Any potential limitations of duty with respect to the proxy
- Resource authorization of the proxy
- Any specific instructions to the proxy

4. GEOTECHNICAL DATA ASSESSMENT

The original Preliminary Economic Assessment level geotechnical study conducted by SRK (SRK 2013a) and geotechnical assessment updated in 2016 based on the field work completed by Alexco during 2013 and 2014 (SRK 2016). Petram Mechanica also conducted simple geotechnical assessment using seven unoriented HQ size double barrel diamond borehole logging data provided by Alexco geology team for study of Decline Ramp Ground Support (Petram 2018a) and Vent-shaft Raisebore Stability Assessment (Petram 2018b). More recently, Jacobs reviewed the diamond drill hole data collected by Alexco geology team and rock property testing laboratory data completed by Golder (2018). This information, along with previous technical reports as mentioned above and site visit completed 2018 has been used to assess ground condition and generate mine design parameters for Keno Hill FM mine.

4.1 Geological Domains and Structural Feature Sets

The Flame & Moth deposit is a narrow vein divided into Lightning and Christal mining areas, which are separated by the cross-passing Mill fault. The fault offsets Christal in a south-east direction by approximately 120 m relative to Lightning as shown in plan view of Flame & Moth deposit (Figure 4.1). The deposit is hosted within district scale sedimentary rock units know locally as the Lower Schist, Central Quartzite, and Upper Schist. Base on drill hole database provided by Alexco, the quartzite is considered to be of fair rock mass quality, while larger schist packages and graphitic schist in the immediate vein hanging wall and foot wall zones are considered to be of extremely poor to poor rock mass quality.

To understand the ground conditions at the Keno Hill FM mine, geological domains were identified for each deposit. Preliminary geotechnical parameters were assessed using major lithology units as identified by Alexco geology team. Geotechnical domains are outlined below on which geotechnical designs have been based:

- Quartzite domain – waste development
- Schist domain – waste development
- Faults domain – waste and production development
- Ore Vein domain – production development

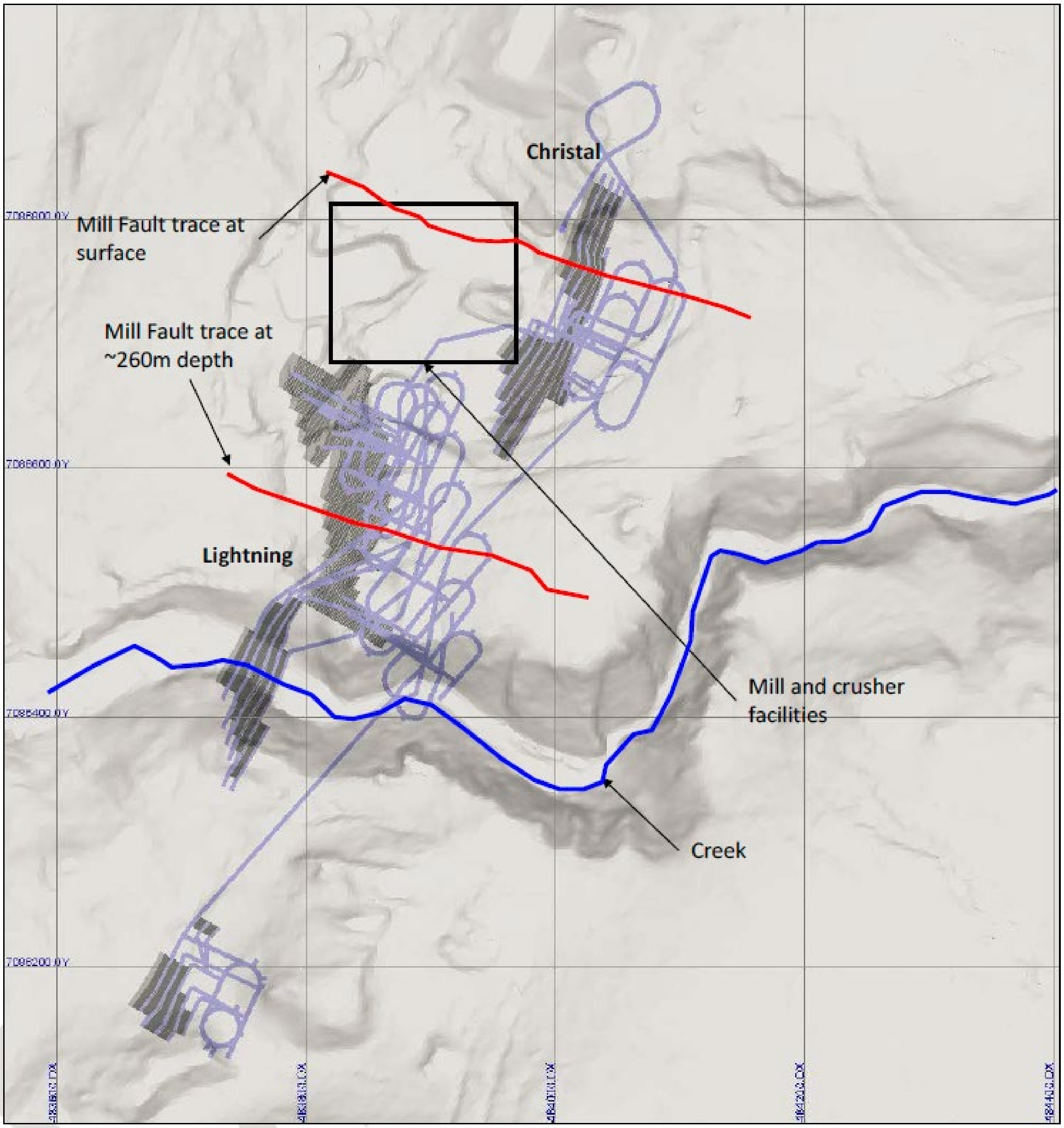


Figure 4.1 Plan view of Flame & Moth

Alexco provided wireframe models of the preliminary mine design shapes (Figure 4.2). Planned mining methods are small Long Hole Open Stopes (LHOS) in fair ground and Cut and Fill (C&F) mine in poor to fair ground, both utilizing cemented and uncemented rock backfill. In LHOS mine areas, all available geotechnical drill hole data proximal to the planned stope hanging wall was applied for stability analysis, including

- Christal zone (lower)
- Lightning zone (West)

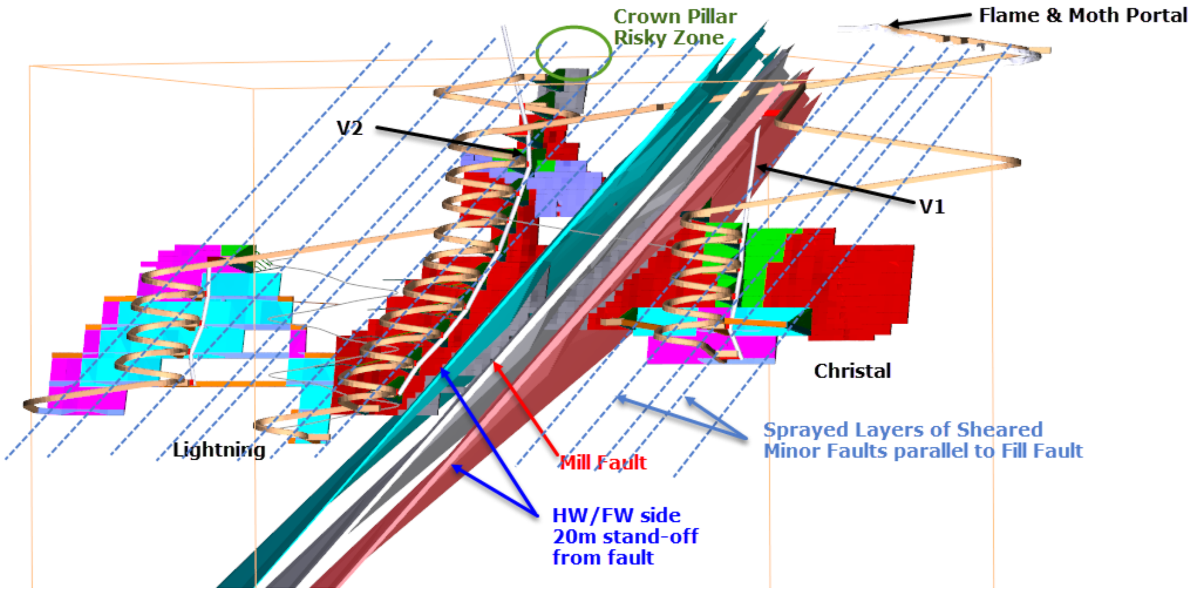


Figure 4.2 Longitudinal section view (looking north-west) of Flame & Moth

4.2 Rock Mass Classification

Rock mass classification was conducted using the Norwegian Geotechnical Institute's tunneling quality index (the NGI Q-system), as proposed by Barton et al. (1974), where Q value is determined from the following relationship,

$$Q = \frac{RQD}{J_n} \cdot \frac{J_r}{J_a} \cdot \frac{J_w}{SRF}$$

Where,

RQD: Rock Quality Designation

Jn: Joint set number

Jr: Joint roughness number

Ja: Joint alteration number

Jw: Joint water reduction factor

SRF: Stress Reduction Factor

Table 4.1 Rock Quality Categories by Q-System (Barton et al, 1974)

Q	< 0.1	0.1 - 1	1 - 4	4 - 10	10 - 40	40 - 100	100 <
Description	Extremely Poor	Very Poor	Poor	Fair	Good	Very Good	Extremely Good

Table 4.2 Flame & Moth rock mass classification by SRK

Domain	RQD (%)	IRS (MPa)	RMR	Q
Quartzite	70 - 90	90 – 150	55 – 65	**3.4 – 10.3**
Schist	50 – 90	20 – 50	40 – 55	**0.6 – 3.4**
Fault	30 - 60	20 - 40	30 - 45	**0.2 – 1.1**

SRK (2016) determined Q value using data collected from drill core at Flame & Moth, correlated to the condition of drill core and underground observations from previous Bellekeno mine. The final rock mass classifications have been engineered based on the anticipated ground conditions and SRK recommended the geotechnical parameters based on rock classification for each domain should be reviewed and adjusted during initial mining to optimize design, and to reflect the actual ground conditions encountered. Table 4.2 presents the estimated rock mass classification by SRK.

Jacobs also developed basic descriptive statistics and histograms for each geotechnical domain to better understand the statistical variability and character within each data set. This information was used to identify representative values for each Q input value. In the case of Jw and SRF, site experience, assumed far-field stress conditions, and typical depth of mining were applied. The Q value estimations for domains in Flame & Moth are summarized in Table 4.3

Underground tour for currently developing Flame & Moth decline ramp and first remuck drift was conducted by Woo Shin accompanied with Alexco Chief Engineer and Mine Manager during site visit in July 2018. Multi layers of highly sheared geotechnical structures which formed parallel to Mill Fault in blocky Quartzite main domain were observed from portal to first remuck drift as shown in Figure 4.3.

Table 4.3 Summary of NGI Q value for Flame & Moth deposit

Input	Quartzite		Schist		Faults		Ore Vein	
RQD	Mean (drill core)	40	Mean (drill core)	25	Mean (drill core)	20	Mean (drill core)	50
Jn	2 Joint sets	4	2 Joint sets	4	2 Joint sets	4	2 Joint sets	4
Jr	Undulating, smooth	2	Undulating, smooth	2	Undulating, smooth	2	Undulating, smooth	2
Ja	Non-softening, fine	3	Non-softening, fine	3	Non-softening, medium	3	Non-softening, medium	3
Jw	Dry (minor inflow)	1	Wet (drips/rain)	0.7	Wet (drips/rain)	0.7	Dry (minor inflow)	1
SRF	Low stress	2.5	Low stress	2.5	Low stress	2.5	Low stress	2.5
Q		**2.7**		**1.3**		**0.9**		**3.3**

Figure 4.3 Ground condition of first remuck drift in main decline ramp

Alexco geology team also provided various Mill Faults drill intersections during the site visit. Mill Faults zone is primarily composed of 'Extremely Poor' to 'Poor' rock quality with fairly thick clay/silty gauge and highly fractured characteristics. Ground condition in near fault HW and FW is also can be expected to be of 'Poor' quality which is similar to conditions within fault zone, and the thickness near the proposed development locations is approximately 2 to 4 m wide. Example of Mill Faults intersections are shown in Figure 4.4.

Figure 4.4 Example of Mill Faults drill intersections

Rock mass classification using Geological Strength Index (GSI) chart based on previous core logging summaries and underground observation of decline ramp development was conducted in this section to simplify mapping face and decision-making process of ground support in the field. The GSI, introduced by Hoek and Brown (1998), provides a system for estimating the reduction in rock mass strength for different geological conditions as identified by field observation. The rock mass characterization using GSI is straightforward and it is based on the visual impression of the rock structure, in terms of blocky and surface condition of discontinuities such as joint roughness. Range of GSI values for each geotechnical domain for Flame & Moth were summarized in Figure 4.5 and table 4.4.

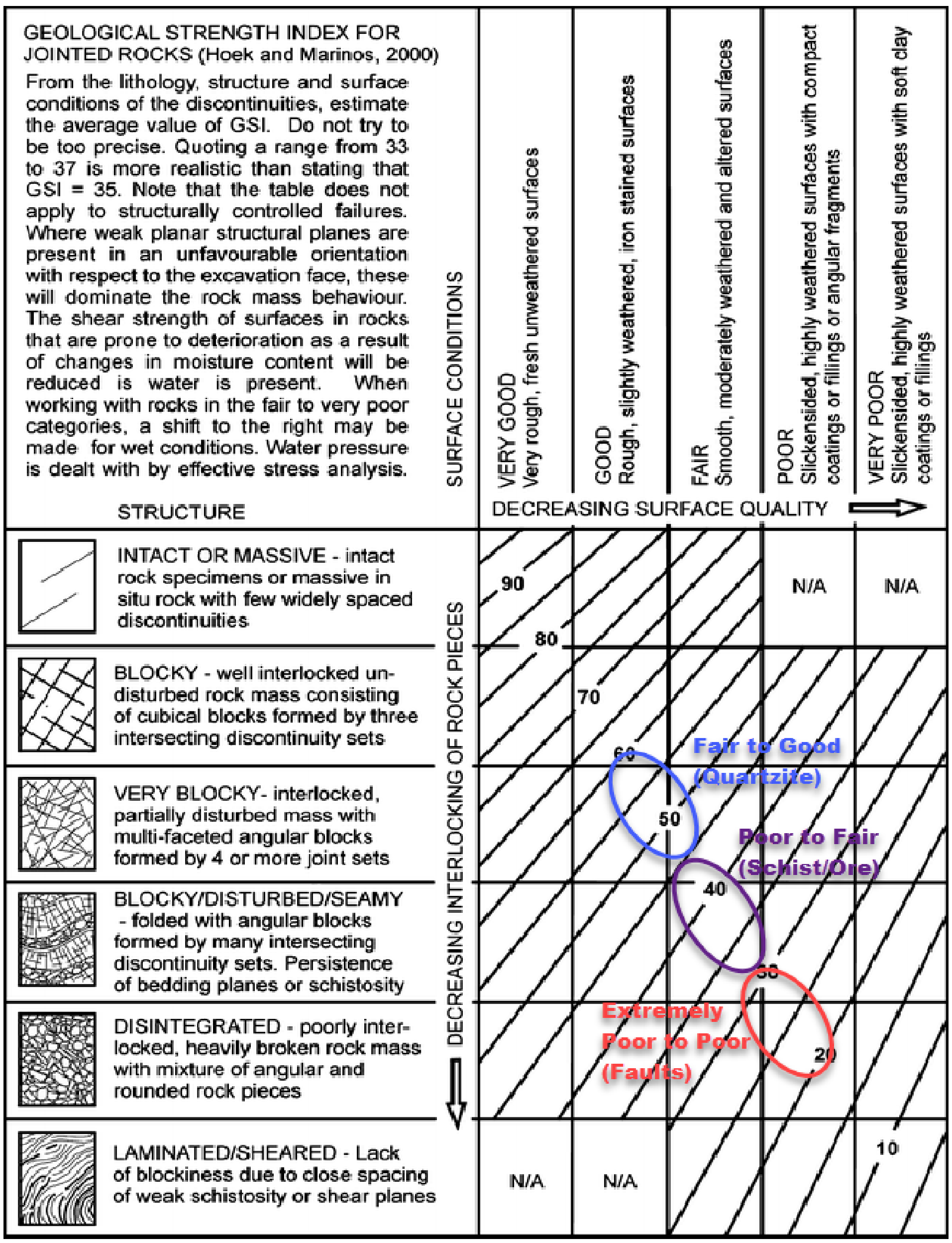


Figure 4.5 Estimated range of representative rock mass for geotechnical domains

Table 4.4 Summary of applied rock mass classification for Flame & Moth mine

Rock Mass Quality	Domain	GSI			Q
		Structure	Surface	Value	
Fair to Good	Quartzite	Blocky Very Blocky	Rough Smooth	**45 -60**	**2.0 – 6.0**
Poor to Fair	Schist Ore Vein	Very Blocky Seamy	Smooth Weathered	**30 – 45**	**0.3 – 2.0**
Ext. Poor to Poor	Faults	Disintegrated Foliated	Slickensided	**20 - 30**	**0.05 – 0.3**

4.3 Rock Mass Properties

Rock mass strength criteria and material properties were estimated for each domain using geotechnical data to conduct numerical and empirical assessment. The Hoek-Brown failure criterion was applied, which requires the GSI rock mass classification scheme to be initially assessed.

Hoek and Brown (1980a, 1980b) proposed a method for obtaining estimates of the strength of jointed rock masses, based upon an assessment of the interlocking of rock blocks and the condition of the surface between these blocks. The Hoek-Brown criterion for all geotechnical domains was estimated using the approach outlined by Hoek et al (2003).

Table 4.5 Applied rock mass properties for the ground support analyses

Rock Mass Properties		Quartzite	Schist/Vein	Faults/Vein
Intact Rock Strength, UCS (MPa)		50	45	25
Geological Strength Index, GSI		50	40	25
Young's Modulus, E_i (GPa)		75	50	7.5
Disturbance Factor, D		0.3	0.3	0.2
Hoek-Brown Constant	m_b	1.5	1.0	0.8
	a	0.5	0.5	0.5
	s	0.002	0.0006	0.0001
Rock Mass Modulus, Em (GPa)		50	40	5
Poisson's Ratio, v		0.3	0.3	0.3

5. GROUND SUPPORT DESIGN – MEN ENTRY OPENINGS

5.1 Opening Dimensions

Men entry design span for main ramp and production drifts have been reviewed based on the critical span curve presented by Ouchi et al. (2004) as shown in Figure 5.1. From this work, the back span for openings in fair to good ground which is mostly in quartzite, some schist and ore vein domains was ranged from 5 m to 9 m. However, maximum critical span in poor ground, faults and some ore vein domain, is limited less than 4 m, which means immediate ground support such as pre-spraying of shotcrete before installing primary ground support by pattern bolting with screen. Possible span of heading in extremely poor ground is less than 2.5m which lies on the boundary between unstable and potentially unstable back condition and, if wider than the critical span heading is required in this low rock mass quality ground, pre-ground support method, spilling and/or grouted pore-poling, may will be required.

To mine the full mineralized width using C&F mine method in central Lightening and upper Christal deposits, wide drift with retreat slashed (up to 7 m wide ore body) or backfill with side drift (ore body width ranging from 7 m to 10 m) will be required depend on ground condition. For the production drifts with wide span near the surface, the use of shotcrete girder structure and/or artificial pillar support can be further evaluated to increase opening span, stability and recovery.

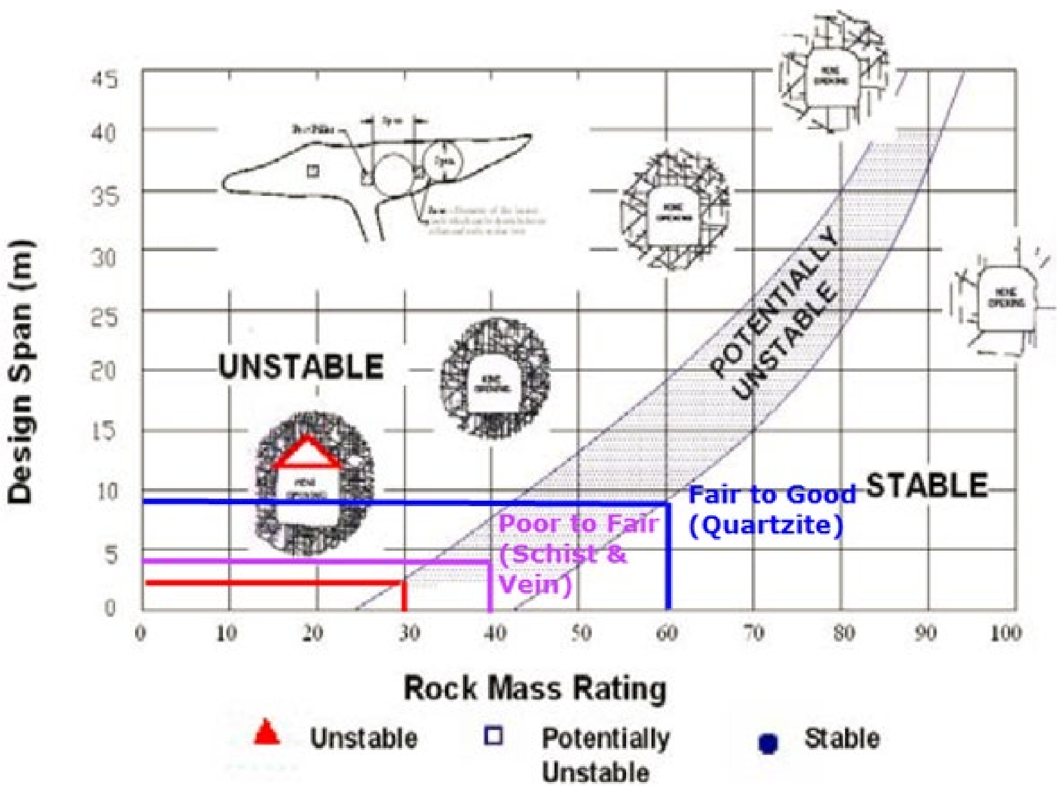


Figure 5.1 Recommended design span from Ouchi critical span curve

The planned opening dimensions that are to be used to access and mine ore bodies, for which support will be required, are given in Table 5.1

Table 5.1 Planned dimensions of men entry openings

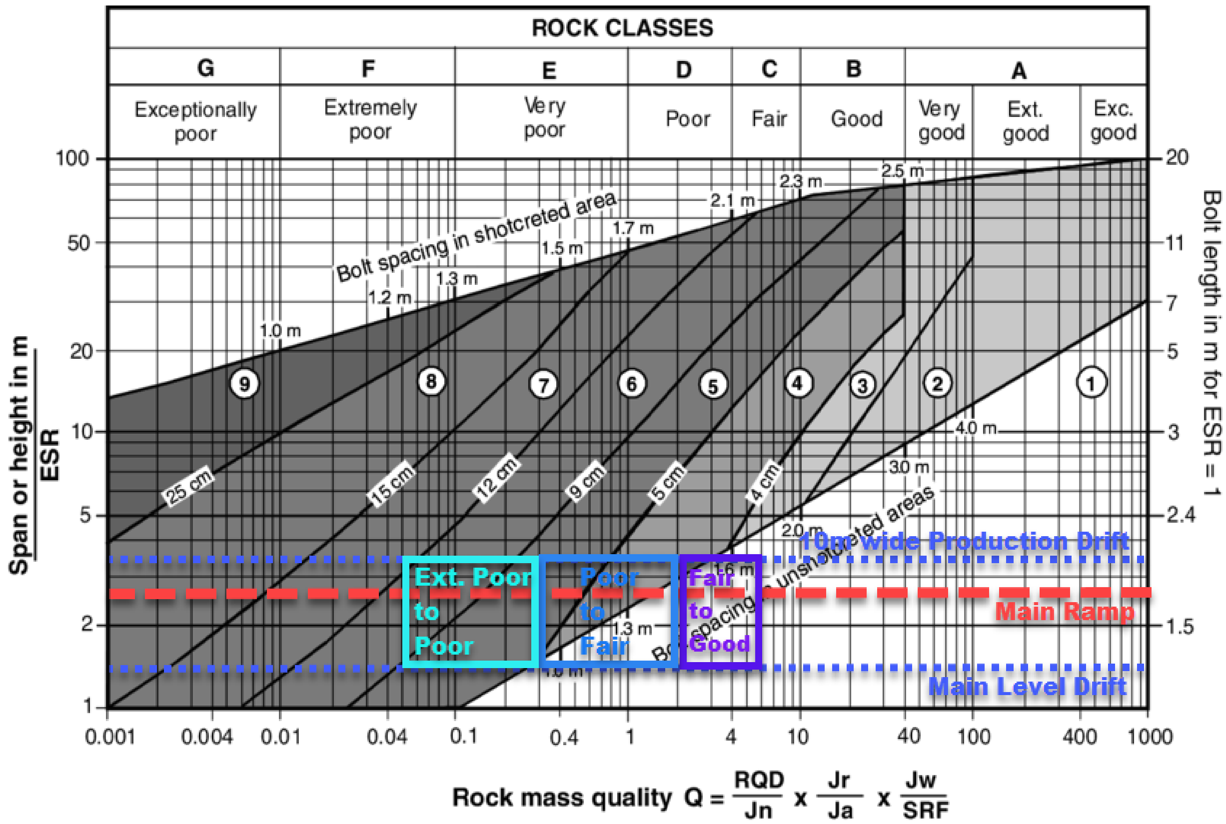
Opening Development	Dimension (W x H, m)
Main Ramp	4.2 x 4.2
Level Access Drift	3.5 x 4.0
Production Drift	3.5 ~ 10.0 x 4.0
Take Down Back (TDB) retreat Drift underneath Backfill	5.5 x 6.0
Raise	2.8 x 2.8 or D = 3.0

5.2 Support Requirements from Empirical Q Support Guideline

The ground support guidelines for main drifts (blue) and main ramp (red) are plotted in Figure 5.2. The value of Excavation Support Ratio (ESR) in the chart is relate to the intended use of the excavation and to the degree of security which is demanded of the support system installed to maintain the stability of the excavation for the planned stand-up time. For Flame & Moth mine, two broad categories of excavation are supported: a) Long term infrastructure, main ramp, for which the ESR values is 1.6 and b) short term mining excavations for which the suggested ESR value is 3 (Table 5.2).

Table 5.2. The value of ESR related to the intended use of the excavation and to the degree of security which is demanded of the support system installed to maintain the stability of the excavation. (Barton et al, 1974)

	Excavation Category	ESR
A	**Temporary mine openings**	**3 - 5**
B	**Permanent mine openings, water tunnels for hydro power (excluding high pressure penstocks), pilot tunnels, drifts and heading for excavations**	**1.6**
C	Storage rooms, water treatment plants, minor road and railway tunnels, civil defense chambers, portal intersections.	1.3
D	Power stations, major road and railway tunnels, civil defense chambers, portal intersections.	1.0
E	Underground nuclear power stations, railway stations, sports and public facilities, factories	0.8



REINFORCEMENT CATEGORIES;

1. Unsupported.
2. Spot bolting (SB).
3. Systematic bolting (B).
4. Systematic bolting with 40-100 mm unreinforced shotcrete
5. Fiber reinforced shotcrete, 50-90 mm, and bolting.
6. Fiber reinforced shotcrete, 90-120 mm, and bolting.
7. Fiber reinforced shotcrete, 120-150 mm, and bolting.
8. Fiber reinforced shotcrete, >150 mm, with reinforced ribs of shotcrete and bolting.
9. Cast concrete lining.

Figure 5.2. Estimated ground support requirements for temporary mine drifts and permanent infrastructure openings based on the empirical Q-support guideline.

According to Barton chart, ground support category for most openings in fair to good ground and some short-term openings in poor to fair fall into category 1 which means openings can stand-up without supports. However, long-term openings such as main ramp in poor ground or all openings in faults zone area need to apply proper ground support in timely manner.

Barton et al (1974) also provide additional information on rock bolt length, maximum span of rock bolt. According to Barton et al, the length, L, of rock bolts can be estimated from the excavation width (B) and ESR value, and rock bolt span can be calculated using Q-value and ESR. Both empirical correlations and ground support patterns for different ground conditions using empirical methods are summarized in Table 5.3.

Table 5.3 Ground support estimation for men entry openings using empirical correlation suggested by Barton et al. (1974)

			Support Category	Rock bolt length (m) $L = 2 + 0.15B / ESR$	Bolt spacing (m) $S = 2 \times ESR \times Q^{0.4}$
Main Ramp (ESR = 1.6)	B = 4.2m	Q = 4	(1)	2.4	5.6
		Q = 1	(4), (5)		3.2
		Q = 0.1	(6)		1.3
Level Access (ESR = 3.0)	B = 3.5m	Q = 4	(1)	2.2	10.4
		Q = 1	(1)		6.0
		Q = 0.1	(5)		2.4
Wide Ore Drift	B = 7.0m	Q = 4	(1), (4)	2.4	10.4

5.3 Stand-up Time Analysis

The stand-up time of unsupported spans is one of the fundamental issues in mine development. The Bieniawski diagram (Figure 5.3) shows the relationship between the unsupported span and stand-up time of an excavation with reference to its rock mass quality. The basic relationship that governs stand-up time is:

- For a given rock mass quality, a stand-up time decrease as the unsupported roof span become wider, and
- For a given roof span, a stand-up time decrease as the rock mass quality becomes poorer.

Using data collected from Flame & Moth mine, stand-up time for two different roof span in three different ground conditions were estimated based on the Bieniawski diagram as shown in Figure 5.3.

The stand-up time of Openings with 3.7 m span and 7.0 m span in fair rock mass (GSI =50) can be assumed 20 days and 4 days respectively. The other cases, face of main ramp or regular level drift in extremely poor fault zone (GSI =25) would be stand-up less than an hour according to Bieniawski. This chart can be apply for the delay time of ground support for current developing faces and this time does not means stand-up time for whole mine drift.

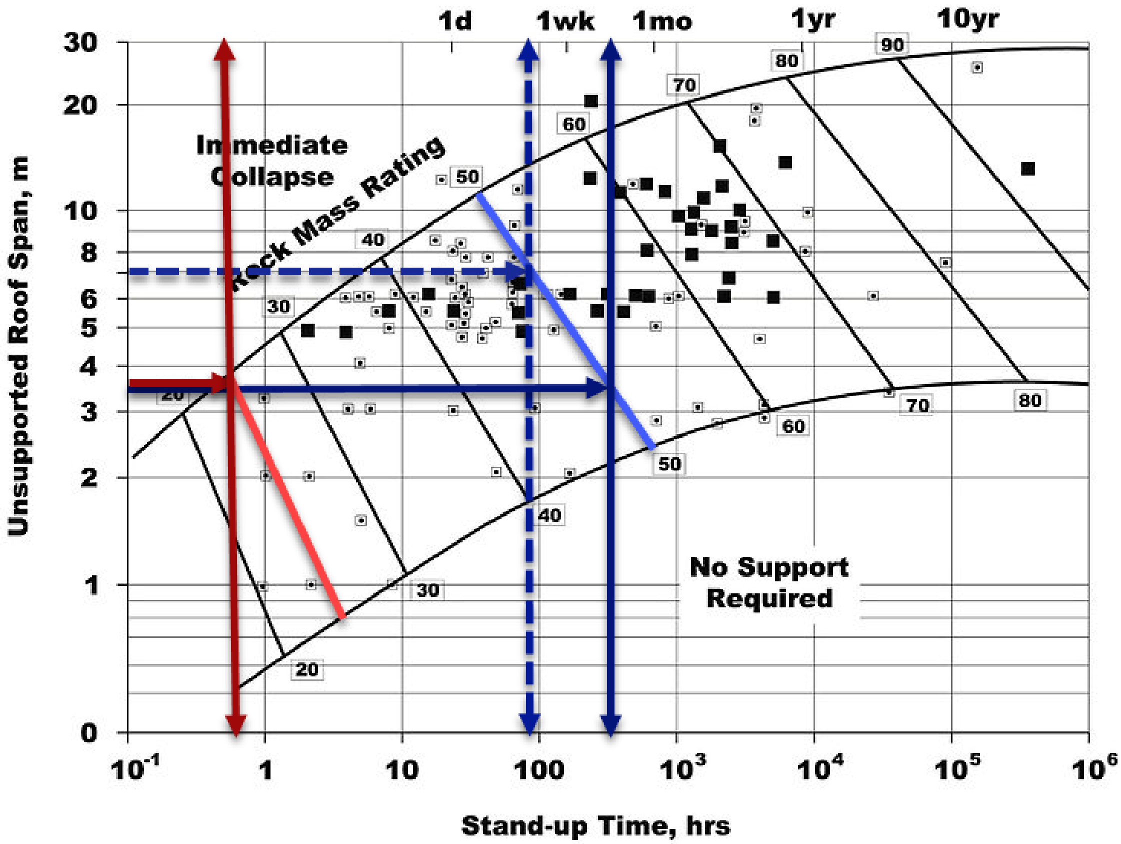


Figure 5.3 Relationship between stand-up time, roof span and RMR after Bieniawski (1989)

5.4 Ground Support Standards for Men Entry Openings

Ground support standards for men entry openings, such as main decline ramp, level access drift, vertical ventilation/escape raise and possible wide production drift in different ground conditions are recommended in this section. Although some decline ramp and main access drift in fair to good ground are able to stand-up relatively long period without ground support based on Barton's empirical Q support chart and Bieniawski's stand-up chart, minimum ground support using 1.8m to 2.4m long rock bolts for the back and walls are required to prevent possible wedge failure or unconsolidated back and wall sloughing caused by blasting damage.

Split sets or Swellex (expandable friction bolt) are preferred as a primary support element for production drift with relatively short term of opening period to reduce cycle-time of ground support installation. As decline ramp need to keep open longer period, fully grouted rebar bolts can be recommended for the ramp support. Minimum ground support standards for Flame & Moth mine underground men entry openings are summarized in Table 5.4 and detailed ground support regimes for each opening are shown in Appendix – B.

Table 5.4 Ground support standards for Flame & Moth men entry openings

Type	Ground Condition	Ground Support Standards (Bolt space)	
Decline Ramp (permanent openings)			
Ramp – I	Fair to Good (45 < GSI < 60)	Back	1.8m Rebar (1.2m x 1.2m)
		Wall	1.8m Split set (1.2m x 1.2m), 1.8m from sill
Ramp – II	Poor to Fair (30 < GSI < 45)	Back	2.4m Rebar (1.2m x 1.2m), SC as req.
		Wall	1.8m Split set (1.2m x 1.2m), 1.2m from sill
Ramp – III	Ext. Poor (GSI < 30)	Back	2.4m Rebar (0.8m x 0.8m), 2” SC, Spilling as req.
		Wall	2.4m Rebar (0.8m x 0.8m), 2” SC, Spilling as req.
Main Access Drift (opening less than 3 years)			
MD – I	Fair to Good (45 < GSI < 60)	Back	1.8m Swellex (1.2m x 1.2m)
		Wall	1.8m Split set (1.2m x 1.2m), 1.8m from sill
MD – II	Poor to Fair (30 < GSI < 45)	Back	2.4m Swellex (1.2m x 1.2m), SC as req.
		Wall	1.8m Split set (1.2m x 1.2m), 1.2m from sill
MD – III	Ext. Poor (GSI < 30)	Back	2.4m Swellex (0.8m x 0.8m), 2” SC, Spilling as req.
		Wall	2.4m Swellex (0.8m x 0.8m), 2” SC, Spilling as req.
Wide Production Retreat Drift (3.5 m ~ 7.0 m)			
WD – I	Fair to Good (45 < GSI < 60)	Back	MD-I + 3.6m Connectable (2.4m x 2.4m)
		Wall	MD- I
WD – II	Poor to Fair (30 < GSI < 45)	Back	MD-II + 3.6m Connectable (1.8m x 1.8m), 2” SC as req.
		Wall	MD-II
Remuck			
RMK – I	Fair to Good (45 < GSI < 60)	Back	2.4m Rebar (1.2m x 1.2m)
		Wall	2.4m Rebar (1.2m x 1.2m), 1.2m from sill
RMK - II	Poor to Fair (30 < GSI < 45)	Back	2.4m Rebar (0.8m x 0.8m)
		Wall	2.4m Rebar (0.8m x 0.8m), 1.2m from sill
Raise			
SR – I CR – I	Fair to Good (45 < GSI < 60)	Face	1.2m Rebar (1.2m x 1.2m)
		Wall	1.2m Rebar (1.2m x 1.2m)
SR – II CR – II	Poor to Fair (30 < GSI < 45)	Face	1.2m Rebar (0.8m x 0.8m), 2” SC as Req.
		Wall	1.2m Rebar (0.8m x 0.8m), 2” SC as Req.
Intersection			
IS – I	Fair to Good (45 < GSI < 60)	Back	Ramp/MD-I + 3.6m Connectable (2.4m x 2.4m)
		Pillar	3 rows of strap with 1.8m Split set
IS - II	Poor to Fair (30 < GSI < 45)	Back	Ramp/MD-II+ 3.6m Connectable (1.8mx1.8m), 2” SC
		Pillar	Screen + 3 rows of strap with 1.8m Split set, 2” SC

No intersection and Wide ore drift in extremely poor ground

The Ground Support Standards form the basis for all ground support and are to be installed according to specification. It is the responsibility of the operator to report and deviation to the standard and the reason for it.

In advance geotechnical ground conditions (e.g. fair to good ground, presence of structures, expected corrosion), the Ground Support Standards shall be review and additional support recommendations will be made by Geotechnical Engineer or designated personnel. The Ground Support Standards cannot be reduced without recommendation by Geotechnical Engineer and approved by Mine Manager.

The Ground Support Standard will be revised as experience is gained upon excavation of the mine. The support regimes employed at Flame & Moth mine are composed of main ramp, main access drift, ore extraction drift and intersection (Appendix – B). Intersections pose a higher risk for ground instability than normal development due to the large spans and repeated blasting damage. Specific regimes for intersection area also have been formulated to support the increased span both horizontally as well as vertically.

- 4-way intersections are to be avoid as much as possible
- Intersection in extremely poor aground must be relocated
- Over-excavation should be minimized

The Ground Support Standards specify the ground support required in all development

- There are 9 basic support types and 2 intersection support types depending on ground conditions and development geometry.
- No Ground Support Standard was recommended for wide ore extraction drift, remuck and intersection in extremely poor ground. Special mine and support plans need to develop for these activities in such ground condition.

The Trigger Action Response Plan (TARP) as summarized in Appendix - C specifies the circumstances under which a change in support type is to occur.

- The TARP provides a description of ground condition indicators which, where observed separately or individually may indicate a change in Support Standard for individual headings

Copies of the GCMP shall be kept in the Shift Supervisor’s office and Mine Superintendent’s office, Engineering Main office and the crew lineup meeting room. The Ground Support Standards and TARP should be prominently displayed. The Supervisor shall ensure that all Shift Supervisors responsible for ground support during development are familiar with the GCMP, Ground Support Standards and TARP.

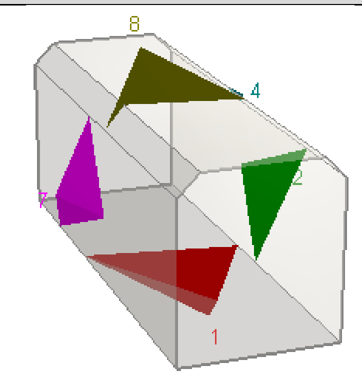
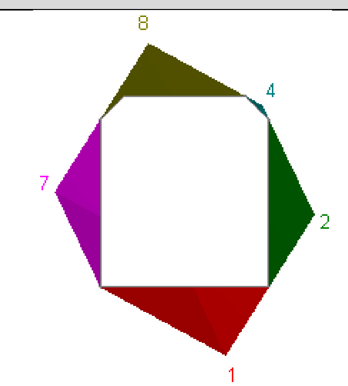
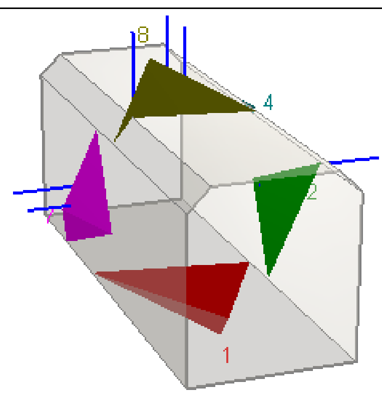
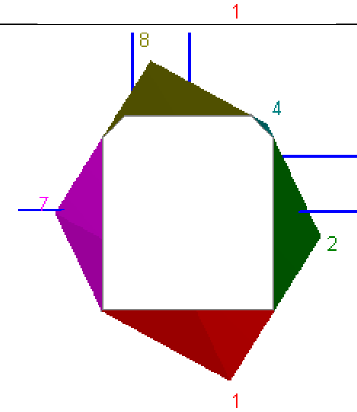
5.5 Kinematic Wedge Stability Analysis

Three most prominent sets of joint including major joint set parallel to Mill Faults were identified during site visit and these joint sets were used in the kinematic analysis to identify the potential for wedge failure around the stopes and development in the proposed underground excavations. (The RocScience program UNWEDGE was used for the analysis). For the analysis, it was assumed that the joint sets are ubiquitous, continuous and planar, and as such does not take into consideration joint spacing and persistence. This usually results in a lower factor of safety and a more conservative assessment of the excavation geometry.

Value for cohesion and tensile strength were set to zero for both the foliation and joints in the analysis, and the field stress was set to 1 MPa lithostatic, to prevent the formation of unrealistic high aspect wedge in the analysis.

A wedge analysis for the man entry excavations was conducted using opening size of 3.7m wide by 4.2 m high. The model was conducted without inclusion of support; and in instances where unstable blocks were identified, ground support was added to the model and re-evaluate the factor of safety for the wedge failure. The ground support recommended in Section 5.4 provides enough support pressure to prevent the wedge generated from falling out of the back (Table 5.5).

Table 5.5 Kinematic wedge analysis results for the main drift with 3.7 mW × 4.2 mH dimension

Support	Perspective	Front	FoS
Before Support			Lower Right wedge [2] FS: 15.584 Weight: 0.043 MN Upper Right wedge [4] FS: 28.085 Weight: 0.000 MN Lower Left wedge [7] FS: 9.615 Weight: 0.043 MN Roof wedge [8] FS: 0.000 Weight: 0.038 MN
After Support			Lower Right wedge [2] FS: 21.307 Weight: 0.043 MN Upper Right wedge [4] FS: 28.085 Weight: 0.000 MN Lower Left wedge [7] FS: 14.657 Weight: 0.043 MN Roof wedge [8] FS: 7.955 Weight: 0.038 MN

5.6 Verification of Applied Support Standard

5.6.1 Assessment of Damaged/Disturbed Zone around Openings

Damaged/Disturbed Zone(DZ) around two different dimensions openings, B = 3.7m and 7.0m, in different ground condition were estimated by numerical parametric study. Using elasto-plastic model in RS2 two-dimensional numerical analysis package, the depth of DZ around openings can be estimated from Strength Factor (SF) because If the Strength Factor is less than 1, this indicates that the stress in the material exceeds the material strength (i.e. the material would fail, if a plasticity analysis were carried out). From the work it is indicated that the ratio between wedge height (H_w) and opening width (B) changes relate to opening width (B) and ground condition as shown in Table 5.6 and 5.7. For the main ramp and drift 0.3B, 0.4B and 0.6B can be assumed as a possible failure depth for the opening in different ground condition respectively (Table 5.6). 0.35B and 0.45B can be considered as a DZ for the wide ore extraction drift with 7 m of width in fair to good and poor to fair ground (Table 5.7).

Table 5.6 Damaged/Disturbed zone at the back of main ramp openings

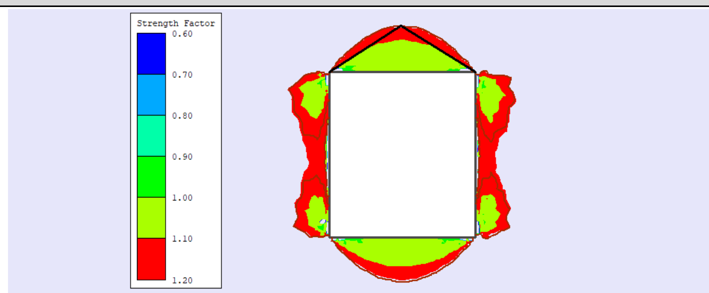
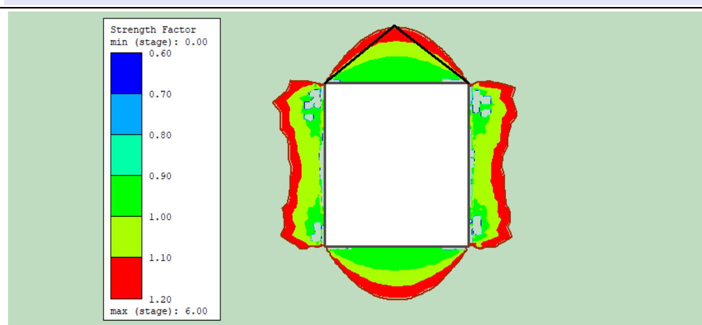
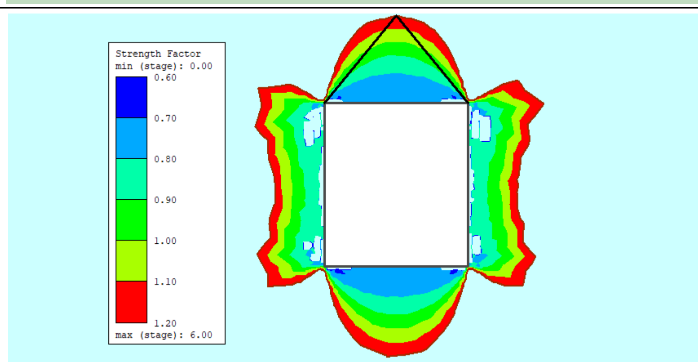
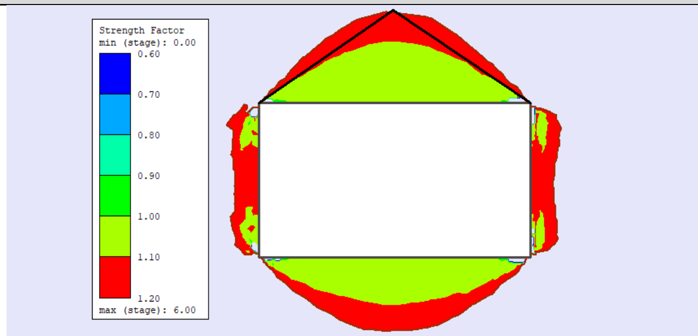
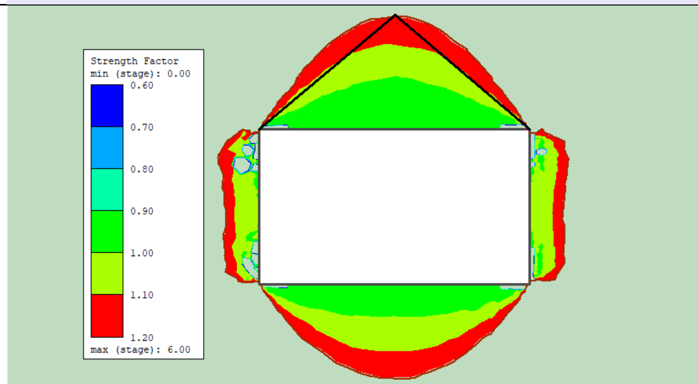
Ground	Strength factor and damaged	DZ/B
Fair to Good		B = 3.7m DZ = 1.1m **DZ/B = 0.3**
Poor to Fair		B = 3.7m DZ = 1.5m **DZ/B = 0.4**
Ext. Poor to Poor		B = 3.7m DZ = 2.2m **DZ/B = 0.6**

Table 5.7 Damaged/Disturbed zone at the back of 7m wide ore extraction drift

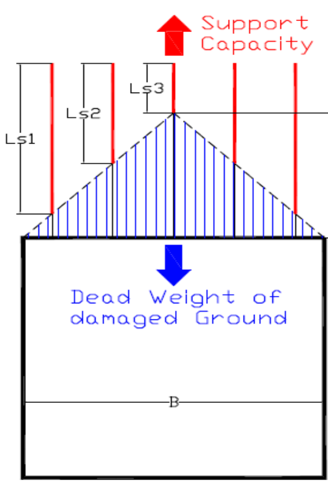
<table>
<tr><th>Ground</th><th>Strength factor and damaged</th><th>DZ/B</th></tr>
<tr><td>Fair
to
Good</td><td>
</td><td>B = 7.0m
DZ = 2.45m

DZ/B = 0.35</td></tr>
<tr><td>Poor
to
Fair</td><td>
</td><td>B = 3.7m
DZ = 3.1m

DZ/B = 0.45</td></tr>
</table>

5.6.2 Dead Weight Analysis

Safety factors for all support patterns associate with ground conditions and opening dimensions were estimated by Dead Weight analysis. Outline of Dead Weight analysis is illustrated in Figure 5.4. Safety factor is the capacity of rock bolts installed at the back against weight of failed wedge block. The weight of wedge can be calculated by opening width and failure depth, capacity of rock bolts should be estimated using the installed length beyond the wedge.



Dead Weigh
$= 1/2 \times \Upsilon_{\text{rock mass}}\ (t/m^3) \times B \times DZ \times \text{Bolt Space}$
Support Capacity
$= \text{Bond strength} \times (2L_{s1}/2 + 2L_{s2} + L_{s3})$

FS = Support Capacity / Dead Weight

FS > 1.5 : permanent opening (Ramp)
FS > 1.2 : temporary opening (Main Drift)

Figure 5.4 Factor of Safety from Dead Weight analysis

Factor of Safety (FoS) for three different support patterns with 3.7 m and 3.5 m wide heading in different conditions ground were estimated using Dead Weight analysis (Appendix – D) and results of the analysis were summarized in Table 5.8. Minimum 1.5 of FoS is required for decline ramp support as a permanent opening and FoS of 1.2 is considered for minimum FoS for main drift as a temporary production opening. According to long life of opening, 1.8 m and 2.4 m long fully grouted rebar were recommended for the back support of decline ramp. As a primary support for temporary opening, 1.8 m split set and 2.4m regular swellex can be recommended depend upon ground condition. 3.6 m long connectable super swellex and pre/post shotcrete also need to apply for openings in extremely poor ground. Pre-support with spills may requires as an additional ground support in extremely poor ground because of less than an hour of stand-up time (Figure 5.3).

Table 5.8 Factor of Safety (FoS) from Dead Weight analysis (Appendix – D)

Opening Type	Ground Condition	Ground Support (Spacing) **Factor of Safety**		
Decline Ramp B = 4.2m	Fair to Good (45 < GSI < 60)	1.8m Split Set (1.2m x 1.2m) **FoS = 1.5**	1.8m Rebar (1.2m x 1.2m) **FoS = 2.8**	2.4m Rebar (1.2m x 1.2m) **FoS = 3.6**
	Poor to Fair (30 < GSI < 45)	1.8m Split Set (1.2m x 1.2m) **FoS = 0.6**	1.8m Rebar (1.2m x 1.2m) **FoS = 1.5**	2.4m Rebar (1.2m x 1.2m) **FoS = 2.3**
	Extremely Poor (GSI < 30)	2.4m Split Set (1.2m x 1.2m) **FoS = 0.7**	2.4m Rebar (1.2m x 1.2m) **FoS = 1.1**	2.4m Rebar (0.8m x 0.8m) **FoS = 3.2**
Main Drift B = 3.5m	Fair to Good (45 < GSI < 60)	1.8m Split Set (1.2m x 1.2m) **FoS = 1.3**	1.8m Swellex (1.2m x 1.2m) **FoS = 2.0**	2.4m Swellex (1.2m x 1.2m) **FoS = 2.7**
	Poor to Fair (30 < GSI < 45)	1.8m Split Set (1.2m x 1.2m) **FoS = 0.8**	1.8m Swellex (1.2m x 1.2m) **FoS = 1.3**	2.4m Swellex (1.2m x 1.2m) **FoS = 2.0**
	Extremely Poor (GSI < 30)	2.4m Split Set (1.2m x 1.2m) **FoS = 0.6**	2.4m Swellex (1.2m x 1.2m) **FoS = 0.9**	2.4m Swellex (0.8m x 0.8m) **FoS = 2.1**
Production Drift B = 7.0m	Fair to Good (45 < GSI < 60)	3.6m Connect. (2.4m x 2.4m) **FoS = 0.9**	3.6m Connect. (1.8m x 1.8m) **FoS = 1.4**	
	Poor to Fair (30 < GSI < 45)	3.6m Connect. (2.4m x 2.4m) **FoS = 0.9**	3.6m Connect. (1.8m x 1.8m) **FoS = 1.3**	

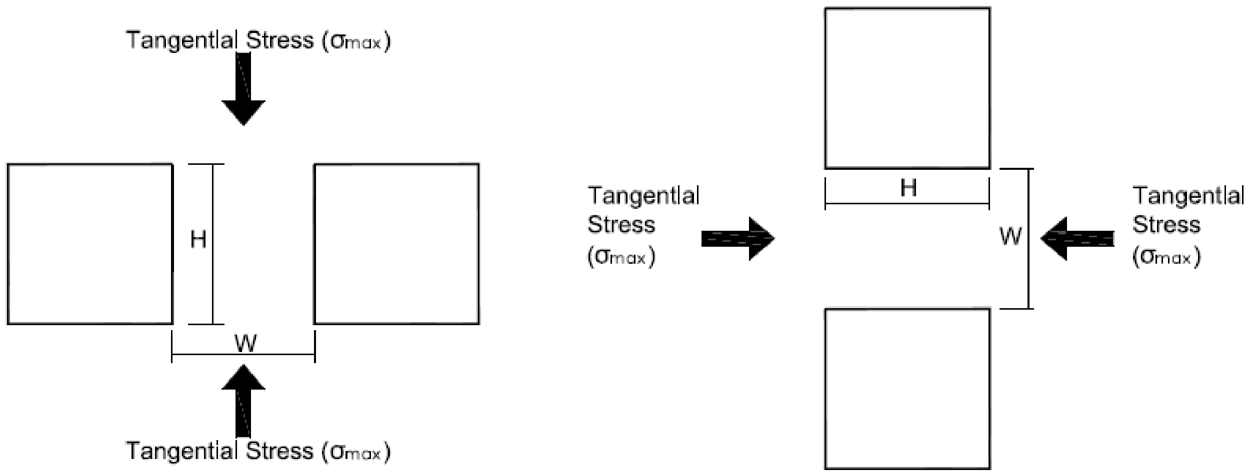


Figure 5.5 Dimension of rib and sill pillar

5.7 Pillar Design for Men Entry Openings

5.7.1 Pillar Geometry

Pillars are usually designed to be rectangular or square shapes in both plan and section. The design of pillars relates the strength to pillar shape. Figure 5.5 illustrates the pillar dimension. It is important to note that the pillar height is defined relative to the direction of the maximum stress. For example, for sill pillar, the pillar height is actually in the horizontal direction as the maximum pillar stress will be in the horizontal (Figure 5.5).

5.7.2 Pillar Failure Modes

There are three modes of pillar failure which are commonly observed underground: (1) structurally controlled failure; (2) stress induced progressive failure; and (3) pillar burst,

Structurally controlled failure

Most rock masses contain pre-existing failure plane (discontinuities) known as joints, faults, etc. Structurally controlled failure occurs when the pillars are oriented unfavorably with respect to the discontinuities present within the rock mass. Failure of these planes is usually in the form of shear movement along the plane. This type of failure is often observed as corners of pillars coming off along wall defined planes.

Progressive failure

The second mode of failure is termed stress-induced progressive failure. This is observed as slabs spalling off the walls of the pillars. The progressive spalling mode of failure, otherwise known as “hour-glassing”, is generally observed in squat pillars where the skin of the pillar which has little confinement and high tangential stresses causes cracking and slab formation parallel to the direction of the major principal stress in the pillars.

Kaiser et al (1996) suggested that the first stage of stress-induced failure was the 'hour-glass' effect commonly observed in hard rock pillar failure (Figure 5.6). They suggested that the failed material should be turned 'baggage' because if unsupported it simply forms detached slabs. The extent of this spalling failure could be predicted by,

$$\Delta = \frac{h\ (or\ W)}{2 \cdot sin\alpha} \cdot \left(1 - \frac{sin\alpha}{tan\alpha}\right)$$

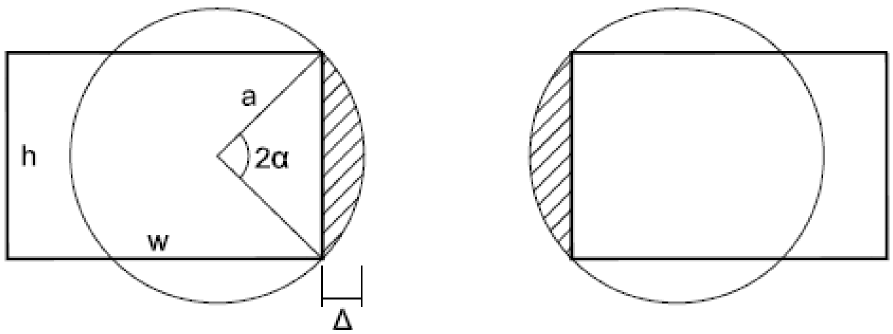


Figure 5.6 Definition of baggage (after Kaiser, McCreath and Tannant, 1996)

Initially the core of the pillar remains intact after spalling failure, because it is still confined and, hence, the pillar still remains most of its load carrying capacity. As spalling occurs, the stresses flowing through the pillar are redistributed to the intact pillar rock. The loss of the slabs relaxes the confinement on the adjacent intact core rock in the pillar and further damage then occurs to the newly exposed pillar wall surfaces (Figure 5.7). If this type of progressive failure is allowed to propagate too far, then the intact core of the pillar can reach a critical cross-sectional area and fail.

If the loads around an opening were sufficient to cause additional stress-induced failure (Figure 5.7), the depth of the failure could be approximated by the linear relationship given by (Martin, 1990),

$$\frac{df}{a} = 1.34 \cdot \frac{\sigma_{max}}{\sigma_c} - 0.57$$

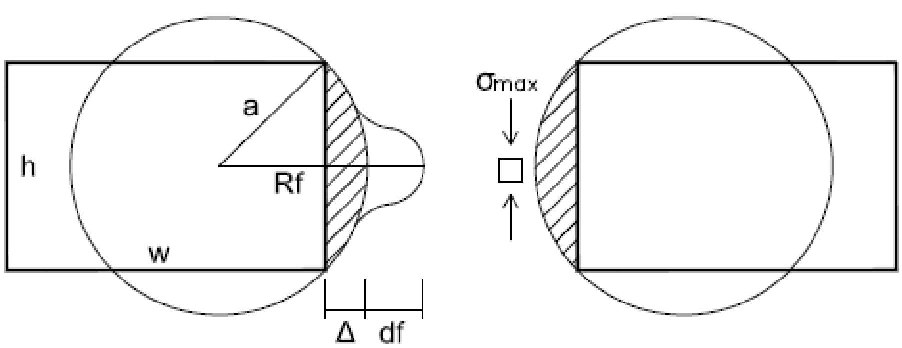


Figure 5.7 Depth of stress-induced failure (after Kaiser, McCreath and Tannant, 1996)

It is important to note that the above equation the stress-induced failure propagates when the maximum tangential stress exceeds approximately one third of uniaxial compressive strength.

Pillar Bursting

The third mode of failure encountered in pillar is pillar burst. This mode of failure is usually encountered when the following two constraints are satisfied: (1) The stress in the pilaar must exceed the strength; and (2) the local mine stiffness must be less than that of the pillar.

Based on the work by Martin (1990) and Kaiser et al (1996) when the pillar stress exceeds 1/3 of the uniaxial compressive strength of the rock the first constraint is generally satisfied. Once the strength of the pillar exceeded, the violence of the failure is governed by stiffness of the surrounding mine environment. If the local mine stiffness is high compared to the post-peak stiffness of the pillar, then the failure will be nonviolent (stress-induced progressive failure mode). However, if the local mine stiffness is low, less than that of the pillar, then the failure will be violent as more energy is put into the failing pillar.

5.7.3 Pillar Design

Pillar stability analyses against stress-induced progressive failure were conducted for Flame & Moth mine pillar design because structurally controlled failure can be controlled by additional spot bolting during regular basis geotechnical inspection and possibility of pillar bursting in this mine is low according to mine stiffness and given low in situ stress condition.

Table 5.9 Maximum stresses, extent of damaged depth in pillars

Opening Dimension (3.5mW x 4.0mH)		Pillar Width, Wp (m)				
		3.0	4.0	5.0	6.0	7.0
σ_{max}/σ_c	Fair to Poor (45<GSI<60)	Failed	1.0	0.96	0.84	0.77
	Poor to Fair (30<GSI<45)	Failed	Failed	1.0	1.0	0.85
	Extremly Poor (GSI<30)	Failed	Failed	Failed	Failed	1.0
DZ Δ + df (m)	Fair to Poor (45<GSI<60)	-	2.84	2.79	2.62	2.53
	Poor to Fair (30<GSI<45)	-	-	2.84	2.84	2.64
	Extremly Poor (GSI<30)	-	-	-	-	2.84
DZ/Wp (%)	Fair to Poor (45<GSI<60)	-	**71**	**56**	44	36
	Poor to Fair (30<GSI<45)	-	-	**57**	**47**	38
	Extremly Poor (GSI<30)	-	-	-	-	**40**

Maximum tangential stresses in rib pillar area with 5 different pillar width from 3 m to 7 m were estimated using 2-dimensional numerical analyses (Phase2) for different conditions of ground and the results were summarized in Appendix – E. Damaged/disturbed depth (Δ + df) and percentage of failure area in rib pillars were assumed in accordance with maximum pillar stresses and pillar dimensions. Maximum stress, progressive stress-induced Excavation Damaged ⁄ Disturbed Depth (EDZ), and the percentage of damaged area in pillar (EDZ⁄Pillar width) from the analysis were summarized in Table 7.11 and Figure 7.9.

The results from numerical analysis (Appendix – E) on 3 m wide pillars shows stress induced failure propagate whole pillar area regardless of ground conditions. Pillar in fair to good condition ground with 4 m width shows less than 75% of damaged depth ratio to pillar width which means pillar will be stable with additional support and the ratio shows lower than 60% if the pillar width is wider than 5 m which noted that the pillar should be wider than 5.0m without additional support in fair to good ground. If ground condition of pillar location is poor to fair, pillar width must be wider than 5 m with additional support plan. However, according to this stability analysis, wider than 7 m of pillar width is required for the opening with 3.5 m wide by 4 m high dimension in extremely poor to poor ground and shotcrete to the pillar walls and displacement monitoring are strongly recommended.

6. OPTIMIZATION OF LONGHOLE STOPE DIMENSION

6.1 Stress Change surrounding Longhole Stope

To determine proper dimension of longhole stopes and mining sequence it is requested that understand stress path change caused by development of longhole mine. Failure is a result of rock mass relaxation and that is defined as a reduction in stress static parallel to wall excavation. Wedge failure occurs when the minor principal stress is below or equal to zero as shown in Figure 6.1 stress path A. The severity of sloughing (stress path B) also possible failure mode for the longhole stope and the failure is related directly to the rock tensile strength. However, rock mass has a self-supporting capacity depending on the material properties and geological structures.

6.2 Maximum Stope Strike Length

The widely used empirical tool for a maximum stope strike length is the stability graph method. The method is developed by Mathews et al (1981) and defined by Potvin (1988). The stability graph method associates the stability number to the hydraulic radius of a stope. The graph helps to access the stability of an opening according to the stope hydraulic radius. The stability number (N) can be calculated by the following equation,

$$N' = Q' \times A \times B \times C$$

Where, N: Stability number

Q': Modified NGI Q value with stress reduction factor

A: Stress factor – ratio of intact rock strength to applied stress

B: Joint orientation factor – relative orientation of dominant structure with respect to the excavation surface

C: Gravity factor – influence of gravity on the stability of the face being considered.

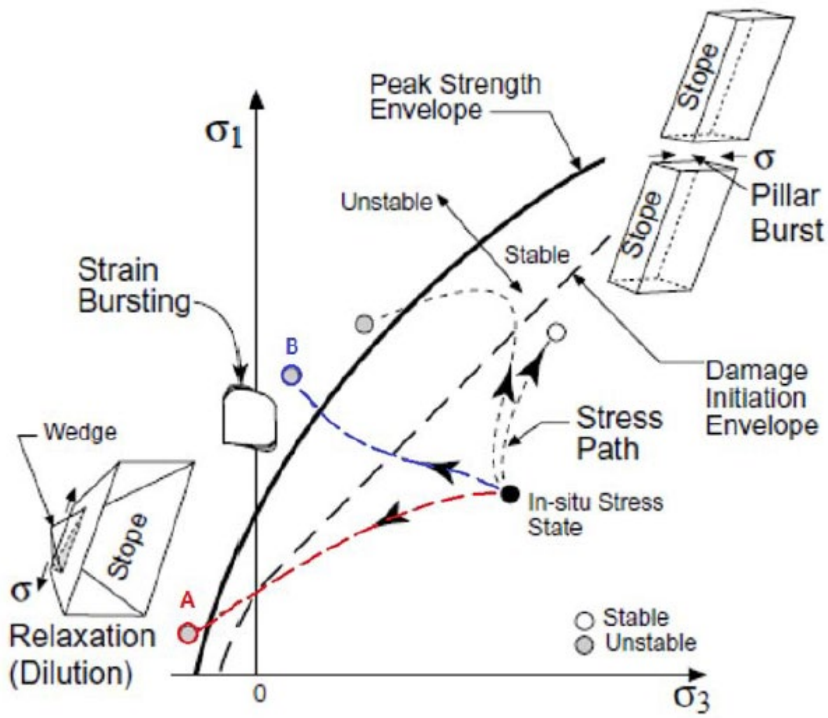


Figure 6.1. Possible stress path for a longhole stope (Martin et al, 1999)

Table 6.1. Stability Graph assumption for stope designs

Parameter	Value	Design Assumption
Q'	5 1 0.5	Fair to good ground (45 < GSI < 60) Poor to Fair ground (30 < GSI < 45) Extremely Poor to Poor ground (20 < GSI < 30)
A	0.5 (wall) 0.2 (back)	Assume induced stresses concentrate above and adjacent to back. Walls are generally destressed
B	0.3 (wall) 0.3 (back)	Conservative assumption based on structural variability in all domain
C	5.0 (wall) 3.0 (back)	Defined based on critical discontinuity set assuming horizontal structure in back and structure parallel to wall

Q' values of 5, 1, and 0.5 were used in combination with A, B, and C inputs to calculate permissible stope strike length for 15 m high and 5 m wide stope in three different category of ground conditions (fair to good, poor to fair, poor). Input parameters for Stability Graph are summarized in Table 6.1 and recommendation of maximum stope strike lengths for each ground condition are shown in Table 6.2 and Figure 6.2.

Table 6.2. Maximum Stope Strike Length Recommendation

Ground Condition	Basic Stope Height / Width: 15 m / 5 m		
	N'	HR	Max. Strike Length (m)
Fair to Good (45 < GSI < 60)	Wall: 3.8 Back: 0.9	Wall: 4.5 Back: 5.2	**W: 20m with 15m high (unsupported)** B: over 100m with 5m wide (supported)
Poor to Fair (30 < GSI <45)	Wall: 0.8 Back: 0.2	Wall: 2.9 Back: 3.5	**W: 10m with 15m high (unsupported)** B: over 100m with 5m wide (supported)
Ext. Poor to Poor (20 < GSI < 30)	Wall: 0.4 Back: 0.12	Wall: 2.1 Back: 3.0	**W: 6m with 15m high (unsupported)** B: over 100m with 5m wide (supported)

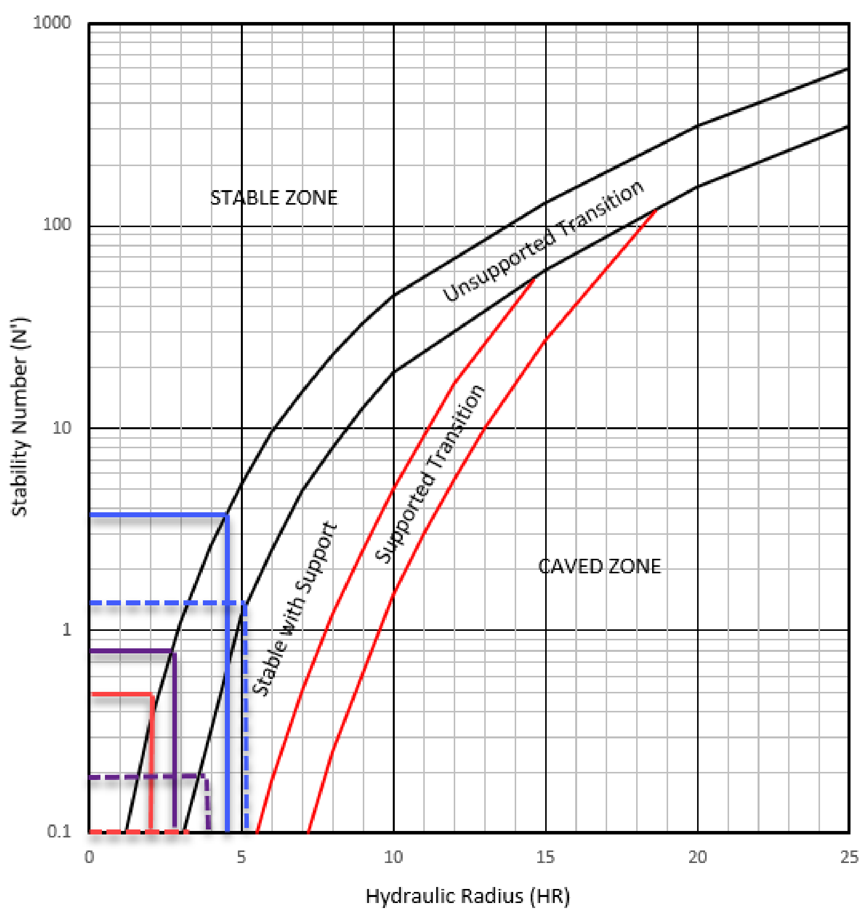


Figure 6.2. Stope Stability Plot for unsupported stopes (Potvin, 1988)

6.3 Ground Support for Longhole Stope

According to stope stability analysis, 20 m to 30 m of maximum stope strike length for 15m high stope can be recommendable depending on ground condition in poor to good ground. However, these stopes assume the use of proper rib and sill pillars that clamp the edges of the various conditions of stope surface. If an Avoca method, continuous stope development following backfill, is used as a extraction method, these stope strike lengths and stope heights will need to be reduced, as the Avoca fill does not provide the same level of support/stiffness as pillars. Stope heights would need to be limited 15 m. Previously recommended stope lengths have to include 5 to 10 m length of the previous stope because of the unconsolidated rock fill.

In this case of open stopping, the use of cable bolting to increase spans and lengths of stope has been used with varying degrees of success. Cable bolting to increase the potential dimension of stopes generally falls into two categories: pattern bolting across the full span or supporting the stope from cable bolting drifts located adjacent to the stopes, or targeted cable bolting to locally improve the rock mass.

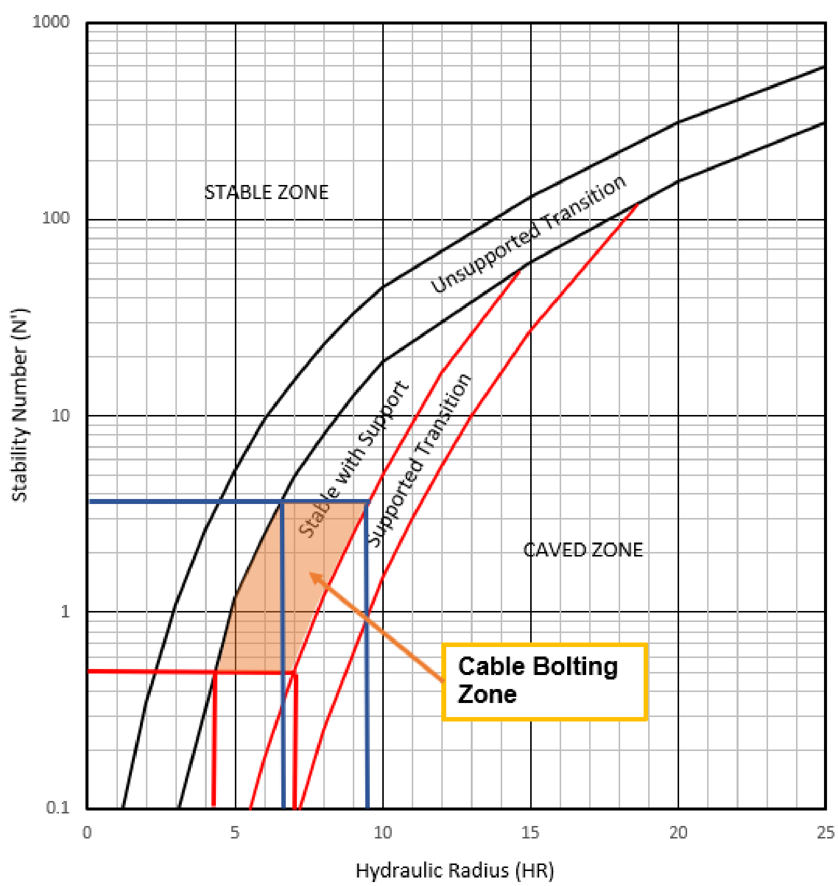


Figure 6.3 Cale bolt design zone for open stopes using modified stability graph method

Hutchinson and Diederichs (1996) discussed the work of Nickson (1992) where the use of localised high density cable bolting has allowed an increased total stope span (or height) by reducing the unsupported span. The theory behind this approach is that a block of reinforced rock on the stope perimeter has an effect similar to that of a pillar, and provides localised support. Smaller sub spans are then formed between these reinforced blocks allowing greater total spans to be opened up.

The blocks marked on Figure 6.3 represent the zones where cable bolting is considered appropriate for potential improvement to stope dimension. This shows that based on the range of N' values and calculated hydraulic radius values, some improvement to the stope dimension may be possible.

6.4 Stope Rib Pillar

If longhole stopes filled with uncemented fill, rib pillars between longhole stopes are required. The function of rib pillars is to ensure the stability of the longhole stopes and in particular the HW during mining and to keep the uncemented fill in the adjacent stope. There are two factors to consider when estimating stope pillar dimensions; the load applied to the pillar must be determined; and, the strength of the pillar to which a suitable safety factor is applied.

Pillar strength was estimated using the formula below from Potvin *et al* (1989). The strength of the pillar is a function of the pillar aspect ratio (the width to height ratio, W/H), the Unconfined Compressive Strength (UCS) of the intact rock, and a calibration factor to account for specific regional stress and rock conditions.

$$\text{Pillar Strength (in MPa)} = 0.42 * \text{UCS} * \text{W/H}$$

Upper and lower bound estimates of pillar strength and stress were calculated for the largest stope size of 15 m height and a 30 m strike span. The rib-size recommendations presented in Table 8.3 were calculated using tributary area theory, which incorporates; the calculated pillar strength, the in situ stresses, and an appropriate factor of safety (of around 1.2 in this case).

Table 6.3 Rib Pillar Recommendations

Back Width (m)		3.0	5.0	7.0
Rib Pillar Width (m)	45 < GSI < 60	5.5	7.0	9.5
	30 < GSI < 45	6.5	8.5	11.0

7. STOPE BACKFILL

7.1 Introduction

The use of cemented backfill is an increasingly important of underground mine operations and is becoming a standard practice for use in many cut & fill and longhole mine around the world. Cemented Rock Fill (CRF) can be considered as a primary backfill method for Flame & Moth mine to allow maximize pillar recovery of the narrow vein ore in longhole mine and optimize ground support for conventional underhand/overhand cut & fill mine area. The use of CRF not only provides ground support to the pillar and wall, but also helps prevent caving and roof falls, and minimize dilution of ore, which enhances productivity.

7.2 Backfill as a Ground Support and Ground Control Element

7.2.1 Backfill Target Design Criteria

Cemented backfill design criteria will be based on target backfill properties, which will be dependent on the backfill function, the mining system conditions, and other site-specific factors. Key target design criteria include;

- Geotechnical properties
- Distribution and placement criteria
- Environmental performance
- Socio-Economic performance

Where backfill is required for ground support or to provide a working floor, backfill strength is the primary geotechnical property. Backfill strength can be increased with the cement or other binding agents. A related geotechnical property pf backfill is liquidation potential, which is dependent on physical and mechanical properties of the tailing material but not major factor for CRF in Flame & Moth mine.

Distribution and placement criteria including system capacities and scheduling are based on the requirements of the mining system as well as the rheological properties of the material.

Environmental considerations have played a growing role in the determination of backfill target properties in recent years. Mining operations face increasing pressure to reduce and limit surface waste disposal of tailings. Target backfill functions and design criteria are critical to improving underground environmental health and safety working conditions also affects target backfill functions and design criteria.

Inevitably socio-economic performance of the backfill influences backfill design. As a primary resource, mining has a significant effect on the local, regional economy in terms of employment and income. The advancement of technology that contributed to the sustainability of environment will serve to enhance the continued economic viability of the mining industry. However, even though the technology may be available to meet the required geotechnical and environmental criteria and the logistical parameters for transportation and placement, the backfill system design will go no further if the cost is too high. The sophistication of backfill systems contributes to relatively high capital costs, but it is important that some of the less tangible cost benefits such as those relate to environmental factors and potential increased mining recoveries be accurately factored into trade-off studies comparing backfill to alternative backfill methods.

7.2.2 Strength of Backfill

As a target backfill property, the required strength of fill will depend on its intended function and site specific factors pertaining to rock mass quality. If the function of the fill is to provide a working floor, as in cyclical mining methods, the curing time must be short and the fill must provide early strength to support personnel and mechanized equipment. For delayed type backfilling, the backfill must achieve and maintain longer term stability and be capable of providing a free standing wall to enable pillar recovery and the mining of secondary stopes with minimal dilution.

Backfill strength can be greatly increased by the addition of binding agents. The most common biding agent used in backfills is Portland cement. Portland cement, containing lime, iron, silica and alumina components, sets and hardens in hydration reactions.

7.3 Design of Required Backfill Strength

7.3.1 Strength Design for Backfill Face Exposure

In order to maximize ore recovery, it is very common to return for mine pillar after primary ore recovery. While this is being done, large vertical heights of massive backfill may be exposed. For delayed backfill, as used in open stopping operations, the fill must be stable when free standing wall faces are exposed during pillar recovery. It is necessary that the fill has sufficient strength to remain free-standing during and after the process of pillar extraction by resisting the blast effect.

In the difficulty of numerical modeling, many mine engineers still rely on 2-dimensional limit equilibrium analyses along with calculated Factor of Safety (FoS) to determine fill expose stability. These analyses typically result in an over conservative estimate of the limiting strength which increase the cost of backfill operations. However, 2-dimensional and pseudo 3-dimensional empirical models have been developed to account for arching effects, cohesion and friction along sidewalls (Mitchell et al, 1982; Smith et al, 1982; Arioglu, 1984; Mitchell & Roettger, 1989; Chen & Jiao, 1991; Yu, 1992).

Narrow Exposed Fill Face

This design method accounts for arching effects on confined fill by adjacent side walls (Figure 7.1) using Terzaghi's vertical pressure model. Based on 2-dimensional finite element modeling, Askew et al (1978) proposed the following formula to determine the design fill compressive strength;

$$UCS_{design} = \frac{1.25 \cdot B}{2 \cdot K \cdot tan\varphi}\left(\gamma - \frac{2 \cdot c}{B}\right)\left[1 - exp\left(-\frac{2 \cdot H \cdot K \cdot tan\varphi}{B}\right)\right] \cdot FoS$$

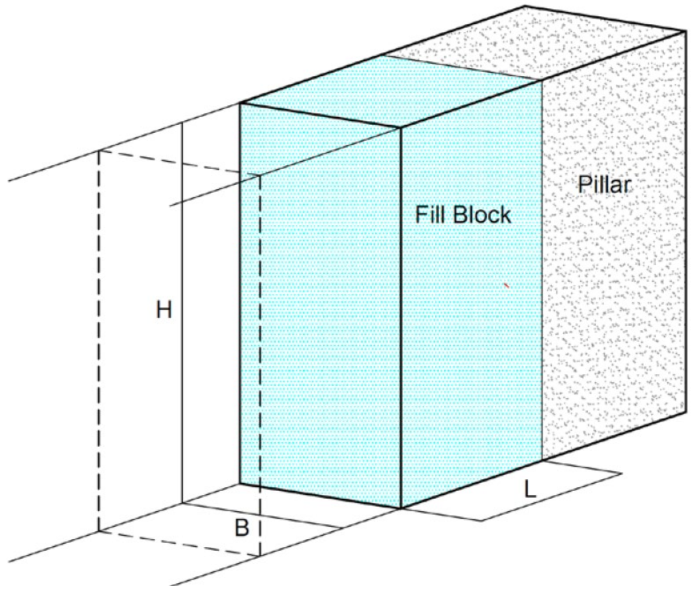


Figure 7.1 Narrowly exposed fill face mechanism

Exposed Friction Fill Face

This design refers to an exposed fill where both opposite sides of the fill are against stope walls (Figure 7.2). By assuming that there is shear resistance between the fill and stope walls due to the fill cohesion, the design UCS can be determined by the following relationship (Mitchell, 1982);

$$UCS_{design} = \frac{(\gamma \cdot B - 2 \cdot c)\left[H - \frac{1}{2} \cdot tan\left(45^{\circ} + \frac{\varphi}{2}\right)\right] \cdot sin\left(45^{\circ} + \frac{\varphi}{2}\right)}{B} \cdot FoS$$

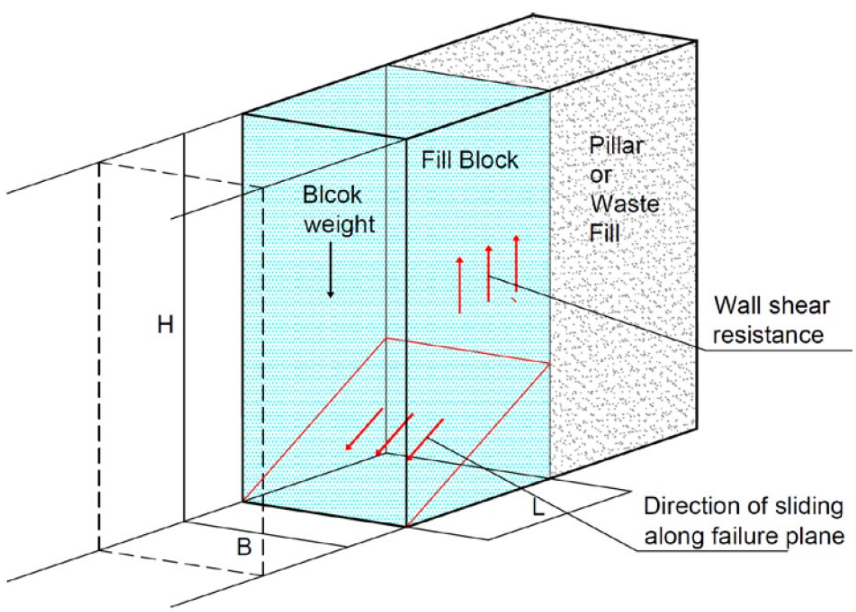


Figure 7.2 Confined block with shear resistance mechanism

Exposed Frictionless Fill Face

The compressive strength of backfill is mainly due to binding agents and any strength contributed from friction can be considered negligible for the long term (i.e. φ = 0). For a frictionless material (Figure 7.3), cohesion is assumed to be half of the UCS (c = UCS/2). Thus, the design UCS can be evaluated by the following relationship proposed by Mitchell et al (1982);

$$UCS_{design} = \frac{(\gamma \cdot B - 2 \cdot c)\left[H - \frac{L}{2}\right] \cdot \sin(45^{\circ})}{B} \cdot FoS$$

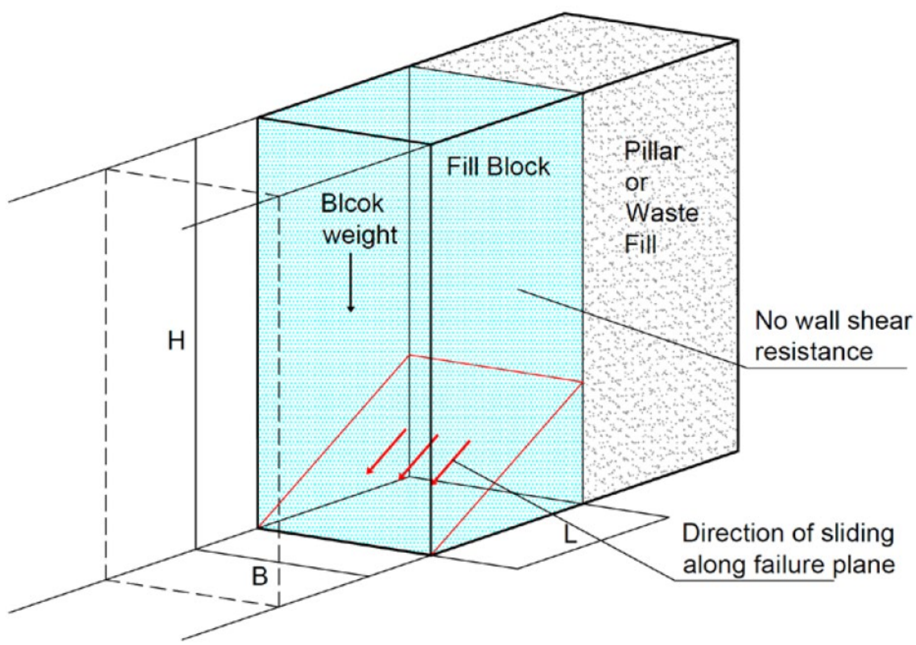


Figure 7.3 Confined block without shear resistance mechanism

Where,

B = width of stope

H = total height of filled stope

K = coefficient of fill pressure (K = 1/[1+2tan2(φ)])

C = cohesive strength of fill (kPa)

Φ = angle of internal friction of fill (°)

γ = bulk unit weight of the fill (kN/m^3)

FoS = Factor of Safety

Required Strength (UCS) for Fill Face Exposure

Required backfill strength for different width of fill face exposures were evaluated using three different relationship with different shear failure mechanism. The detailed evaluation results are shown in Appendix – G and required backfill strength for exposed fill face ranging from 4 m to 10 m are summarized in Table 7.1. The continuous longhole extraction for 4 m thick ore deposit need backfill with minimum 175 MPa of UCS but more than 400 MPa of backfill will required to place for 10 m wide longhole stope backfill.

Table 7.1 Required backfill strength for fill face exposure

Exposed Face Width (m)		4 m	5 m	6 m	8 m	10 m
Required Strength (MPa)	No Friction Shear	**175**	**218**	**260**	**343**	**422**
	Shear for Wall and Plane	157	184	208	249	281
	No Shear for Wall	158	185	209	250	282

7.3.2 Strength Design for Underhand Cut Stability

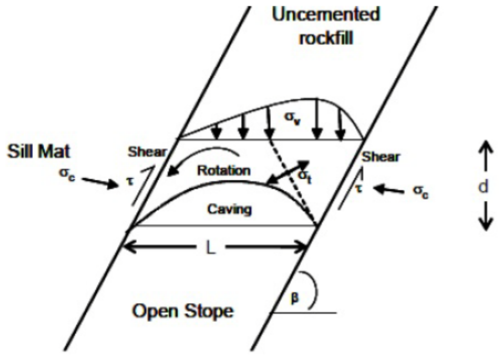
Failure modes of backfill for Underhand Developing

The methodology of span design under paste fill is complex because many different factors affect the overall stability, as shown in Figure 7.4 (a). The failure modes and combination thereof should be analyzed with respect to the cement paste properties, stope geometry, and other factors relate to filling practice, such as cold joints and gaps above not tightly filled.

For the underhand cut design, Factor of Safety (FoS) against four different types of failure mode can be estimated from limit equilibrium analysis summarized by Mitchell (1991) and illustrated in Figure 7.4 (b).

Caving failure would occur when the unsupported weight of backfilled sill material exceeds the tensile strength of the material. The caving is assumed to extend to a semi-circular arch shape defined by L/2 where L is the undercut span. This failure is assumed to be related only to the self-weight of the material, independent of external loadings. Other than the sill drive geometry, the assumed tensile strength of the material is the critical factor to consider in this analysis.

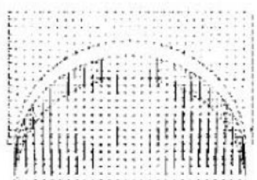
Flexural failure would occur when the moments due to bending of the sill mat under its self-weight plus the vertical stresses applied to the sill exceed the moment capacity of the sill material. Following this analysis, the tensile strength of the material and thickness of the sill would provide the main resistance to flexural instability.



L : Span of the underhand-cut stope
γ : Unit weight of paste fill
σ_t : Tensile strength of the cement fill
d : Thickness of paste sill
σ_c : Horizontal confinement (assumed zero – conservative)
σ_v : Vertical stress above paste sill (uncemented rockfill)
τ : Shear strength along fill and wall contact
β : Stope wall dip angle

(a) Schematic showing typical failure mode after Mitchell (1991)

Caving Failure



$$FS_{caving} = [\,(8 \times \sigma_t)/\pi\,]/[\,L \cdot \gamma\,]$$

Flexural Failure

$$FS_{flexural} = [2\cdot(\sigma_t+\sigma_c)/(\sigma_v+d\cdot\gamma)]/[(L/d)^2]$$

Sliding Failure

$$FS_{sliding} = [2\cdot(\tau/\sin^2\beta)\cdot(d/L)]/[\sigma_v+d\cdot\gamma]$$

Rotational Failure

$$FS_{rotation} = [(d^2\cdot\sigma_t)/(L\cdot(L-d\cdot(\cot\beta\cdot\sin^2\beta)))]/[\sigma_v+d\cdot\gamma]$$

(b) Limit equilibrium analysis of typical failure modes

Figure 7.4 Limit equilibrium criteria developed by Mitchell (figure from Pakalnis et al. 2005)

Sliding or shear failure along the sill mat abutments would occur when the weight of the backfill material, in combination with the vertical loads emplaced on the sill mat, exceed the shear strength of the paste material. For the assessment of UCS against sliding, shear strength (τ) is defined by initial failure strength of UCS test.

Rotational failure strongly depends on backfill thickness (d) as shown in Figure 7.4.

Minimum required backfill strength for underhand cut from 3 m to 10 m wide were estimated against four different failure modes and summarized in Table 7.2. 4.0 m of sill thickness and 1.5 of FoS were applied for the analysis to evaluate design backfill strength for underhand cut developments with Cut & Fill mine method. From the analysis results it is noted that minimum 540 KPa of backfill strength will be required for 3.5 m span underhand cut and the strength must achieve more than 1700 KPa for the production drift underneath 10 m wide backfill span (Table 7.2).

Table 7.2 Required backfill strength for underhand cut drives (FoS = 1.5)

Backfill Span (m)		3.5	4	5	6	8	10
Design Strength against Failure Modes (KPa)	Caving	540	615	765	920	1225	1530
	Flexural	35	100	150	210	375	590
	Sliding	310	410	510	615	820	1020
	Rotational	115	210	430	620	1100	1700

8. GROUND CONTROL PROGRAM - IMPLEMENTATION

8.1 Risk Assessment and Hazard Identification

Risk assessment and hazard identification involves the systematic examination of any activity, location or operational system. The risks and hazards are identified and the likelihood and potential consequences of an event are reviewed so that planned approaches to manage the risk exist. This GCMP should be re-assessed and updated by an authorized person or group on an annual basis, or before any major change is made to the mine design, method, or equipment used. It should be made available for examination, in conjunction with the mine design, on request by any relevant parties.

8.2 The Mines Act and Other References

This GCMP should be read and implemented within the context of the prevailing legislative framework (as defined by "The Mines Act"), industry-accepted best practice, and Health and Safety policies, guidelines, and targets, as amended from time-to-time.

Some of the most important sections of the Mines Act which deal specifically with ground control should be reiterated. And they deal with the *"examination of workings"* and the *"daily examination and report book"*:

Examination of Workings

- All active workings shall be examined by the certified shift boss or supervisor with assigned responsibility to ascertain that they are in a safe working condition, as often as the nature of the work necessitates.
- All persons working underground shall have their work areas inspected by a shift boss or supervisor at least twice per shift.

Daily Examination and Report Book

- The person making the examination shall record all unusual and hazardous conditions and corrective actions taken or proposed in a daily examination and report book and sign the report as a record of the conditions found. For underground mines the record shall include a report on each working place examined.
- The report shall be read and countersigned by the corresponding supervisor on the oncoming shift and the unusual and/or hazardous conditions discussed with the workers before they are permitted to resume operations in the areas indicated in the record.
- In addition, all mining personnel are responsible for recognizing poor ground conditions in active headings and notifying supervision so appropriate action can be taken.

The miner(s) assigned to specific work areas are responsible for examining and testing for loose ground. The miner(s) assigned to a specific work area shall examine and, where applicable, test ground conditions in areas where work is to be performed, prior to work commencing, after blasting, and as ground conditions warrant during the work shift.

8.3 Communication

A communication process that ensures a two-way flow of information between operations and mine management shall be fostered.

8.3.1 Communication Process

The process shall ensure that;

- Operators are provided with an understanding of expected conditions, anticipated support, mining procedures and any relevant changes in support design prior to implementation.
- Personnel are aware of typical warning signs which suggest that installed support may be inadequate and need review.
- Close communication exists between all members working under the GCMP.
- Management has an early opportunity to respond to unexpected mining conditions and/or support system behavior.

Communication channels may include;

- Geotechnical Daily Logging Book
- Start of roster meeting
- Underground inspections
- Daily/weekly planning meeting
- Support rules and drawings
- Plans and sections
- Shift reports
- Toolbox meeting

- Safety meetings
- TARP's and work procedures
- Tell tale and other monitoring forms
- Incident reports
- Inspection checklists

8.3.2 Non-conformance and Corrective Action

Treatment of non-conformances and corrective actions under the GCMP will be in accordance with the framework defined bellow;

- Identification and notification of non-conformances
- Documentation of non-conformances using the relevant Keno Hill Flame & Moth mine forms (Appendix-H)
- Identification of potential corrective actions that may be applied
- Determination of required corrective actions (taking into account impacts of change including potential additional hazards and effects on other operations)
- Allocation and recording of responsibilities and target dates for completion of corrective actions
- Monitoring and review of non-conformances and progress of completion of corrective actions (generally conducted at Monthly Planning Meetings, additionally as required or warranted)
- Record of completion and closure of corrective actions by responsible person
- Storage of records

The Geotechnical Engineer shall maintain a Ground Control Non-conformance register.

8.3.3 Identification of Non-conformances

Non-conformances may be identified through means including;

- Observations and inspections by Alexco Underground personnel, contractors, consultants and visitors
- Monitoring of ground control performances
- TARP
- Incidents and incident investigations
- Internal audits (including systematic and non-systematic audits by Alexco technical staffs, materials and equipment suppliers and routine inspection)
- External audit typically done by 3rd party consultants (including by the Mining Inspectorate and systematic periodic audit)

Non-conformances will be reviewed at the Monthly Planning Meetings.

8.3.4 Corrective Action

The adequacy and effectiveness of corrective actions, allocation of responsibility, target completion date and progress towards completion will be reviewed and adjusted as appropriate/required at the Weekly Planning Meetings.

8.4 Monitoring

8.4.1 Ground Inspections

Routine ground inspection needs to be conducted by miners, supervision and technical staff. Additionally, quality testing of ground support will be conducted to determine the effectiveness of installations in supporting the ground. Internal reviews of standards need to be conducted to ensure applicability of ground support standards to evolving conditions as the mine matures.

The routine ground inspections, which should be conducted on a daily basis, are part of the "workplace inspection" each miner should conduct prior to the commencement of work. The supervisors need to verify that the workplace inspection has been done by the workers, miners, and would also need to inspect headings themselves.

On a weekly basis, the main travel-ways and haulages need to be inspected by both supervisors and technical staff.

8.4.2 Ground Control Logbook

A single book and set of plans that provides a record of ground control related issues, falls-of-ground (FOG), incidents/accidents, remedial measures, etcetera, needs to be kept. It enables easy review during meetings and at times when a single repository of information and data is required – but mostly, it ensures that ground control is adequately addressed at all levels of the organization. The regularly updated plans can be posted in the start-of-shift meeting areas for reference and discussion.

8.4.3 Overbreak Measurement Program

Overbreak tolerances of 15% by volume are considered good in most operations. Within the vein it will be critical to limit the overbreak as much as possible to avoid increasing the excavation span. With best practices, drilling and loading overbreak and loosening of the ground surrounding the excavation can be minimized.

Mining faces under geological control should be clearly delineated by the mine geologists prior to the face mark-up and drilling. The face should be photographed to record the geologists" decisions before the rock-face is worked on for the advance.

8.4.4 Design Effectiveness

The overbreak evaluation program will provide effective feedback on the drilling and blasting practices. Other measurements should be considered to assess the GCMP. These measures should be ones that can be readily collected and are meaningful, for example; rehabilitation requirements, excavation deformation measurements, shotcrete cracking, accidents/incidents, FOG, fill dilution, and so on.

8.4.5 Ground Support Quality Assurance and Quality Control (QA/QC)

QC is a critical part of ground control – for which a stand-alone guideline will be developed and used within the context of pre-existing SOPs.

Major points covered in the QA/QC process, apart from the individual support members‟ quality and installation procedures, include tangible means to achieve solid ground control – and they are:

- Installed rebar rockbolts, Swellex and split-set friction anchors should be randomly tested to ensure consistent. Effective installation methods are practiced at all times in mining operations. Drill-bit sizes should be reviewed daily by the Shift Boss or equipment operator, to ensure that the required drill-hole size is achieved.
- Testing of support elements should be performed monthly. On these occasions, 1 % of total installed rebar and each of the various FSA‟s (Friction Support Anchors) should be pulled from random sites in the mine. Over the first three months of mining, the quantity of support installation, support unit performance and excavation performance, should be evaluated on a weekly basis. This will enable the short-term assessment of the suitability of the proposed (and implemented) support units.
- If a new type of bolt is planned for use and/or ground conditions have changed, additional pull tests are required. These should be undertaken both in the back and in the sidewalls, in the range of ground conditions in which the bolts are being proposed to be used.
- Records of all tests should be documented and maintained. These reports should be distributed to appropriate personnel for review and submitted for remedial and/or corrective measures where required – in a way that reflects the urgency of the case in-hand.
- A documented bi-annual inspection should be instituted in which the corrosion of splits-sets (and other steel elements) are monitored during the life of the excavations.
- Rehabilitation should be completed in areas in which the support capacity (of the original support units) does not meet, or is unable to adequately support, the required life-of-opening expectation. Rehabilitation with rebar support elements should be done in these instances.

8.5 Review

8.5.1 Conforming to Regulatory Requirements

Regulatory requirements should be adhered to on all fronts, on a daily basis. Ground support materials employed at the mine should conform to the Canadian Standards Association specification, as detailed in “CAN/CSA-M430-90 (R2007) Roof and Rock Bolts, and Accessories”. Daily workplace inspections should be carried out, and the main haulages and travel-ways should be inspected weekly (or more regularly if weaker ground conditions or excavation performance warrants it).

8.5.2 Examination of Ground Conditions

All underground workers should be trained in the examination, and testing, for loose or unsafe ground conditions. This should occur prior to work commencing, after blasting, and at any time during the work shift if ground conditions change.

Underground haulage and travel ways, surface area high walls, and banks adjoining travel ways need to be inspected weekly or more often if ground conditions change.

8.5.3 Re-evaluate Failure Modes and Update Risk Management Studies

As experience is gained in the mining of the access-excavations and the extraction of the ore deposit, the potential modes of failure, the ground control practices, and the mining approach should be re-evaluated. They should be adjusted to reflect the increased understanding of the rock mass and its behaviour. The re-evaluation may precipitate an amendment to the base assumptions upon which the ground control design was built which may, iteratively, affect the minimum ground control standards for those conditions. The re-evaluation should be conducted annually.

8.5.4 Peer Review of Standard Work Practices

Ground control implementation guidelines should be made available for discussion, review, and comment, by any person at any time. This dialog will ensure the applicability of the various work standards as they apply to the installation and performance of ground support at the mine. A formal peer review of the standard work practices should be conducted on an annual basis.

8.5.5 External Review of the GCMP

The mine should provide for an external audit of the GCMP to be conducted annually. The overall plan should be revised and-or amended on an as needed basis and as conditions change. Any of these revisions should be vetted and „signed-off‟ by a qualified geotechnical professional.

9. ONGOING DATA COLLECTION

Once mining commences, a formal geotechnical data acquisition program needs to be invoked that includes excavation mapping, geotechnical logging and excavation and support performance monitoring. From these processes, support designs, excavation and stope dimensions are modified to reflect any changes in ground conditions as the extraction of the deposit progresses.

9.1 Diamond Drill Core Logging

Geotechnical information collected from core logging forms the basis for recommendation of appropriate mining methods, ground support designs, and stable mining geometries prior to (and during ongoing) mining excavation. Without these data, accurate rock mass quality estimates are difficult which leads to either a high risk of failure (economic and-or mining excavation) or a very conservative design methodology. For the purposes of mine development and sustainment, core logging for engineering parameters should be done for at least the amount of core suggested in Table 9.1.

Table 9.1 Suggested percentage of cored bore holes geotechnical logging

Stage of Mine Development	Suggested Percentage Logged
Feasibility Study	100 %
Operating Mine	35 % - 75 %

Geotechnical Logging of Boreholes

Geotechnical logging of boreholes´ rock-core (for engineering parameters) should, where possible, be a representative sample of the FW, ore, and HW conditions (and country or host rocks). This enables a balanced view to be formed on the inherent variability in the ground conditions and allow recognition and delineation of discrete geotechnical domains.

Geotechnical Logging Code

The diamond-drilled rock-core should be logged for the following main parameters used in the calculation of RQD, Q", RMR, and other measures of rock mass quality or condition.

Basic Geotechnical Parameters

- Total Core Recovery (TCR)
- Magnetic Susceptibility
- Orientation Comment
- Rock Quality Designation (RQD)
- Orientations Offset
- Notes: Other Geotechnical Observations

Detailed Geotechnical Parameters

- Intact Rock Strength (IRS) Strong
- Intact Rock Strength (IRS) Weak
- Percent Weak IRS Material
- Total Discontinuity Features : All
- Total Foliations Total
- Open Joints Foliation
- Angle: Alpha Foliation
- Angle: Beta number of Joint sets
- Joint Set Definition: Core Axis Angle, Roughness, Alteration, Fill Comments, geotechnical observations

Structural Geology - General

- Location
- Description/Quality
- Total Joints
- Alpha, Beta, Gamma Angles

Core Orientation

- Depth
- Feature Type
- Roughness
- Alteration
- Fill
- Oriented Structural Features
- Confidence
- Notes: General structural observations

Point Load Test

- Location
- Core Size
- Test Diameter
- Foliation Orientation
- Guage Roughness
- Failure Mode
- Test Quality
- Comments: General Observations

9.2 Geotechnical Mapping

During the mine development cycle, recognized ground control concerns should be addressed immediately. This is achieved mostly by the operator knowledge base, engineering design work applied to obtain a required profile, and the inherent (and post-excavation) stability of the rock mass.

Before any cycle begins, however, the design process involved for development headings should consider a range of aspects, for example:

- Geotechnical mapping requirements (for the building of an accurate geotechnical model of the rock mass)
- Geological and geotechnical domains of the area and local rock mass
- Engineering design process used for the profile, drilling, explosive selection, charge up and sequencing of rounds
- Geotechnical methodology used, and assumptions used, for determining the ground reinforcement and support of the heading

- Type, method, and timing of the support and reinforcement installation
- Engineering practice as applied on the site‟s excavations (blasting, ground support)
- Steps undertaken if the ground conditions vary from expected conditions, and remedial measures available
- Operator observations of installed systems, and effectiveness
- Operator training, and commitment to following and improving procedures
- Established and well-used communication channels which effectively relate ground conditions and-or work quality to those in a position to quickly implement remedial action.

Geotechnical Mapping Requirements

Geotechnical mapping records features of the rock mass which may influence the stability of an excavation, in both the short and long-term. These factors include:

- Representative face and sidewall photography and sketch-maps of significant features Intact strength of the rock, both estimated and measured
- Orientation, spacing, persistence, roughness, aperture, infill-type and shear strength of the mapped discontinuities
- The visible effects of water on the discontinuities and intact rock.

The amount of detail (or „resolution‟ of survey and mapping) required, depends on a number of factors, which are all related to the ultimate use of these data:

- Rock mass structure and fabric
- Analysis method,, and the resolution of engineering application, for example; local (heading), regional (mine), and so on…
- Level of refinement, or the number of iterations, used in the analysis.

Types of Geotechnical Mapping

The various techniques used for structural mapping of a rock mass can be divided into three main categories:

- Spot and-or face-mapping;
- Lineal mapping, which is an effective „fast-sampling‟ method
- Window mapping at pre-determined or random exposures

The objective and ultimate use for the data, as well as the mapping method employed, dictate the required amount or sample density of the data to be collected. In situations where fault-structures are not obvious or easily discernible from the available rock exposure and/or rock cores, the sample data sets should aim to record sufficient data to readily discern the fault from within the background random or ordered discontinuity suites.

The choice of mapping method to use depends on the extent of the exposed rock face and the ultimate use to which the data will be applied. The advantage of window over lineal mapping is that it reduces the sampling bias due to discontinuity orientation, as well as requiring less rock-exposure for a statistically significant result. It also provides a better representation of the trace-length distribution. A disadvantage is that a measure of the spacing distance between two very widely spaced discontinuities may be under-represented in the data set. Notwithstanding these limitations, this type of mapping should be conducted for each stope in all mining zones.

9.3 Deformation Monitoring

If higher risk local situations or areas are noted, they should be monitored for signs of deformation, and the results used to assess the potential for instability. Measurement of sidewall and back displacement in stopes and drifts should be undertaken using industry standard geotechnical instrumentation technology wherever possible. Tunnel displacement monitoring stations should be installed in higher risk areas. This monitoring will facilitate the development of ground reaction curves, and in so doing, will enable the suitability analysis of the existing support systems.

10. INCIDENT INVESTIGATION

Following the occurrence of an incident related to uncontrolled ground movement, general priorities will be;

- Removal of personnel from positions of potential harm
- To eliminate hazards sufficiently to enable safe recovery or treatment of injured personnel
- Investigation, data collection and reporting
- Securing the back and walls
- Recovering equipment and resumption of development/production

Alexco Keno Hill Health and Safety guidelines provide guidance as to responsibilities, communications, reporting and other requirements for incident investigation (Appendix-A).

Incidents will be reviewed at Special Meetings, Safety Meetings and Monthly Planning Meetings.

10.1 Guideline for Incident Investigation

Appropriately experienced personnel will be used in incident investigation. Consideration should be given to whether external opinion or other particular skills are also required.

Records of all investigations, including associated analysis, conclusions, recommended actions and action completion will be maintained by the Geotechnical Engineer.

As relevant, ground control incident investigation may include;

- Inspection of the incident site
- Photography and sketches of the incident site
- Soliciting of verbal and written statements from personnel involved in the incident

- Soliciting of verbal and written statements from personnel associated with the incident (e.g. Supervisor, Shift Supervisors, Leading Hands, Operators)
- Compilation of a chronology of events
- Review of equipment and materials in use
- Assessment of compliance with the GCMP
- Review of data
- Review of design
- Back analysis
- Review of ground support design or operating practice
- Review the GCMP

10.2 Incident Statutory Reporting Requirements

Ground related incidents will be reported to the relevant authorities by the Mine Manager or designated personnel as required by the appropriate regulations.

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APPENDIX – A.

RISK EVALUTION TABLE

AND

ASSESSMENT MATRIX

Team-Based Risk Assessment – Consequence, Exposure & Probability Risk Evaluation Tables

	CONSEQUENCE						SEVERITY FACTOR
	Financial	**Compliance**	**Reputation**	**Communities Impact**	**Health and Safety**	**Environment**	
C1	>$100M one off or NPV, or >$40M annually	Potential jail terms for executives. Very high company fines. Operations suspended or severely reduced by authorities. Loss of water licence and/or forfeiture of land lease.	Extended and widespread international condemnation.	Total social breakdown, significant damage to highly valued cultural objects or structures Irreparable and prolonged impact	Multiple fatalities; multiple cases of fatal chronic disease	Massive widespread, irreversible environmental damage. Could close mine permanently	**100**
C2	$20M - $100M NPV, or $8M - $40M annually	Major regulatory breach; potential for severe fines and prosecutions; Multiple, serious litigation.	Serious public or media attention with international coverage Alexco CEO exposure	Very serious social impacts ; Irreparable and widespread	Single fatality, Quadriplegia, paraplegia; fatal chronic disease	Significant, local, irreversible impact; likely short-term mine closure	**50**
C3	$5M - $20M NPV, or $2M - $8M annually	Potential for significant prosecution and fines. Very serious litigation, including class action.	Serious national media, NGO attention and public concern. Product Group CEO exposure	Significant social impacts and/or damage to culturally significant objects.	Serious permanent disabling injury or disease eg. blindness	Potential prosecution/ conviction. Negative perception. Significant but reversible	**25**
C4	$1M - $5M NPV, or $400K - $2M annually	Major breach of regulation; Potential for major fines; Major litigation or major legal issue.	Significant adverse national media, public and NGO attention. Alexco Managing Director exposure	Ongoing social impacts and damage to culturally significant objects. Major non-compliance with PA's or SEMA. Mostly reparable	Serious disabling injury. (Rehabilitation required) Loss of an arm or leg. Noise induced hearing loss	Non or compromised compliance with environmental obligations; generally reversible impact	**10**
C5	$100K - $1M NPV, or $40K - $400K annually	Serious internal non-compliance; serious regulatory breach; prosecution with moderate fines; Potential for investigation or report to authority.	Attention from media and/or heightened concern by local community. Criticism by NGOs; DDMI General Manager exposure	Medium term social impacts on local community. Serious non-compliance with PA's. Mostly reparable	Loss of a finger, broken leg or arm, asthma (e.g. LTI >2 wks)	Serious degradation or harm to environment but reversible.	**5**
C6	$20K - $100 NPV, or $5K - $40K annually	Minor legal issue, minor infraction of regulation; no fines (warning), no litigation.	Minor adverse local public or media attention and complaints. Alexco Manager exposure	Minor impact to social structures. Minor non-compliance with PA's. Fully reparable	Medical treatment injuries or illness (e.g. MTI or LTI <2 wks)	Minor impact requiring regulatory reporting	**1**
C7	$5K - $20K NPV, or $2K - $5K annually	Minor non-compliance with internal policy.	Public concern restricted to local complaints. Alexco manager issue	Very minor impact. Fully reparable.	Minor medical/first aid treatment eg. Dust in eye (no MTI/LTI)	Nuisance only; minimal impact	**0.5**

2. EXPOSURE TO THE RISK		
LEVEL	**EXPOSURE DESCRIPTION**	**S.F**
E1	Continuous or several times per day or several employees once per day	**10**
E2	Approximately once per day	**6**
E3	Once per week to once per month	**3**
E4	Once per month to once per year	**2**
E5	Once a year to once every ten years	**1**
E6	Rarely, but it has been know to occur	**0.5**
E7	No exposure identified	**0.1**

3. PROBABILITY OF OCCURRENCE OF UNWANTED EVENT		
LEVEL	**PROBABILITY DESCRIPTION**	**S.F**
P1	Always 90% to 100%	**10**
P2	Frequent 51% to 90%	**9**
P3	Common: heard of it happening a number of times 30% to 50%	**5**
P4	Probable – Have heard of it happening 11% to 30%	**3**
P5	Possible – Could happen 6%to 10%	**1**
P6	Unlikely 1% to 5%	**0.5**
P7	Extremely Unlikely (less than 1%)	**0.1**

Risk Evaluation

Risk Rating = 1 **Consequence** x 2 **Exposure** x 3 **Probability**

DDMI Risk Rating	Risk Level	YZC Risk Determination	Action	Minimum Notification and Accountability
>3000	**Extreme**	Class V	Risks that significantly exceed the risk acceptance threshold and need urgent and immediate attention	President / COO
1501 - 3000	**Very High**	Class IV	Risks that exceed the risk acceptance threshold and require proactive management	General Manager / VP Responsible
501 - 1500	**High**	Class III	Risks that exceed the risk acceptance threshold and require proactive management	General Manager
101 - 500	**Moderate**	Class II	Risks that exceed the risk acceptance threshold and require review of controls and required mitigations.	Department Manager
0 -100	**Low**	Class I	Risks that are below the risk acceptance threshold and do not require active management	

Flame & Moth Underground Mine Project Risk Ranking Matrix for Job Hazard Analysis

		PROBABILITY					RISK ASSESSMENT CATEGORY
		A	B	C	D	E	
CONSEQUENCE	1	**1**	**2**	**4**	**7**	**11**	CRITICAL
	2	**3**	**5**	**8**	**12**	**16**	HIGH
	3	**6**	**9**	**13**	**17**	**20**	MODERATE
	4	**10**	**14**	**18**	**21**	**23**	LOW
	5	**15**	**19**	**22**	**24**	**25**	

Potential sequence and probability details

Potential **CONSEQUENCE** of the incident	
1	Could kill, permanently disable or cause very serious damage
2	Could cause serious injury (major LTI) or major damage
3	Could cause typical MTC / LTI or moderate damage
4	Could cause First Aid injury or minor damage
5	Could not cause injury or damage

PROBABILITY of this occurring again	
A	ALMOST CERTAIN to happen
B	LIKELY to happen at some point
C	MODERATE, POSSIBLE, it might happen
D	UNLIKELY, not likely to happen
E	RARE, practically impossible

Flame & Moth Project Risk Assessment

Minimum impact – Work your plan

Some disruption – Re-evaluate the control measures in order to reduce the overall risk

Unacceptable major disruption likely – Re-evaluate the control measures with the Supervisor. Determine lower risk options

Unacceptable major disruption likely – Job sut down for re-evaluation with Superintendent and the job team

Keno Hill Flame and Moth Project Priority of Risk Controls

1. **Elimination** – Controlling the hazard at source
2. **Substitution** – Replacing one substance or activity with a less hazardous
3. **Engineering** – Installing guards on machinery
4. **Administration** – Policies and procedures for safe work practices
5. **Personal Protective Equipment** – Respirators, earplugs, etc.

Flame & Moth Ground Control Risk Assessment Form

Area/Location/Activity		
Unwanted Events/Potential Loss		
Cause/s		
Impacts		
Inherent Risk	Type of Loss	
	Consequence	
	Exposure	
	Probability	
	Risk Ranking	
	Risk Level	
Controls		
Contingency		
Residual Risk	Type of Loss/Benefit	
	Consequence	
	Exposure	
	Probability	
	Risk Ranking	
	Risk Level	
Recommendations/Actions		
Who		
When		

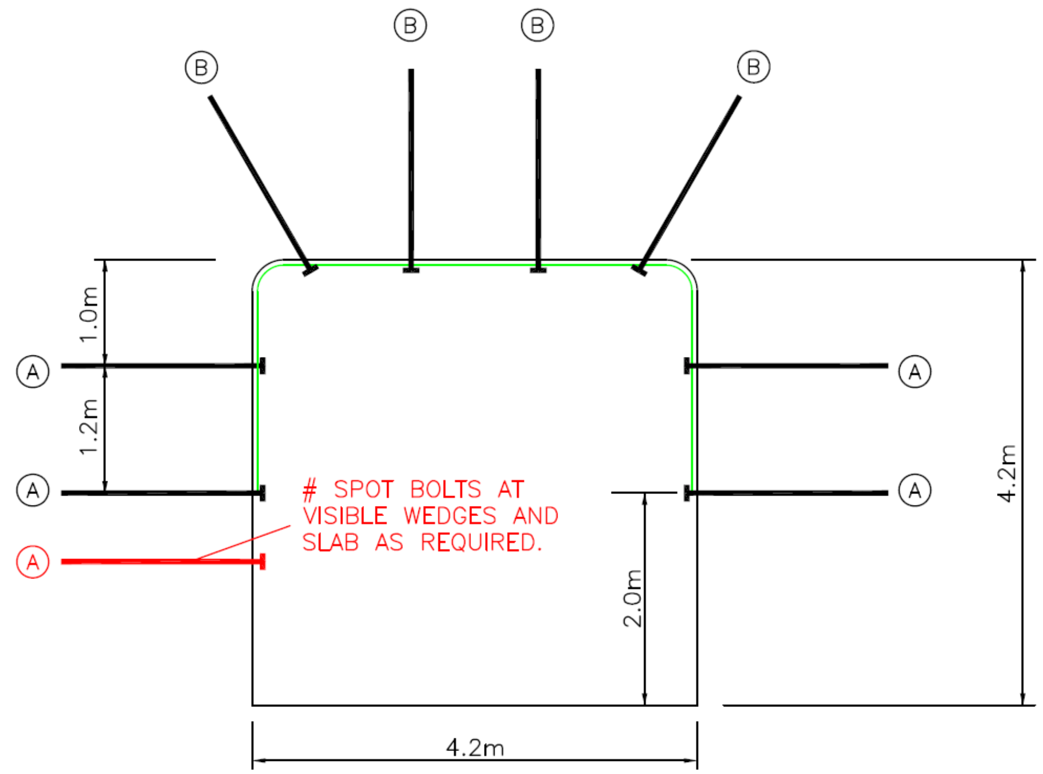
APPENDIX – B.

SPECIFICATION OF SUPPORT STANDARDS

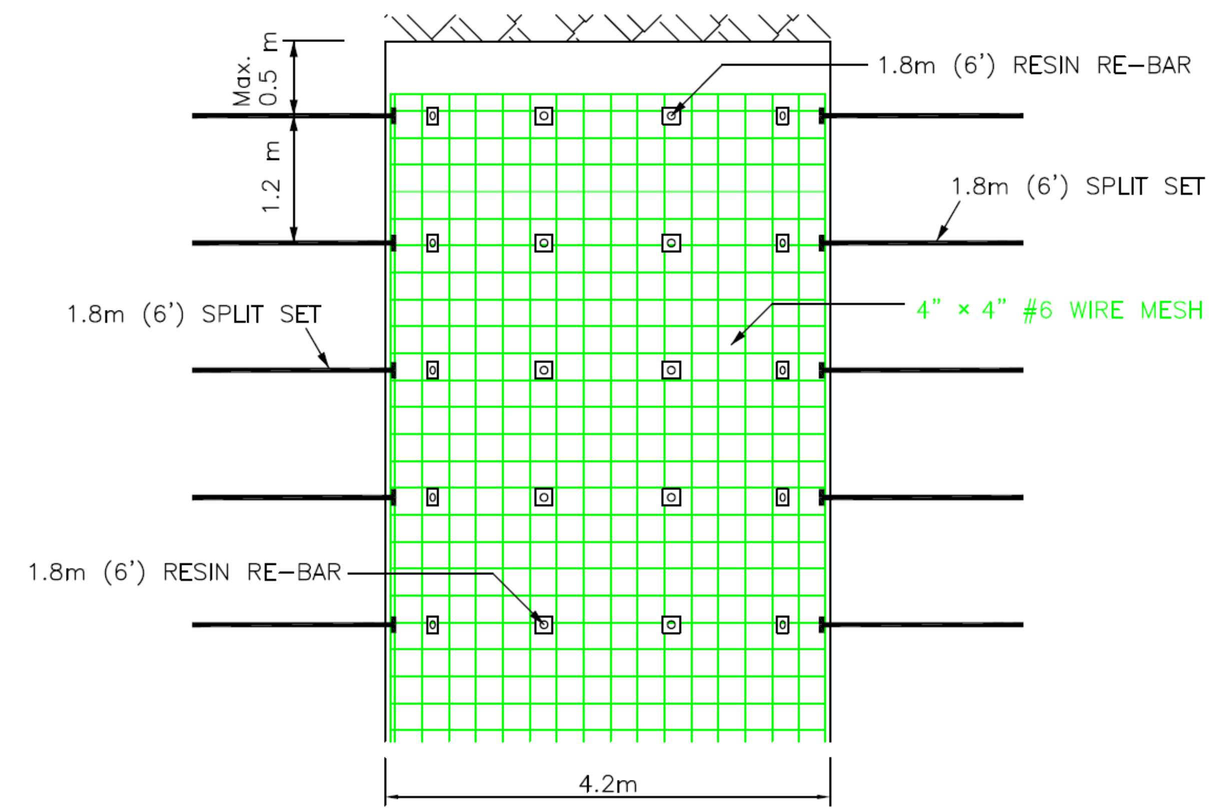
RAMP - I (4.2mW by 4.2mH)

Fair to Good Ground (60 > GSI > 45)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN RE–BAR	1.8m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN RE–BAR	1.8
C	12T SWELLEX	2.4
D	RESIN RE–BAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+–10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE N.T.S

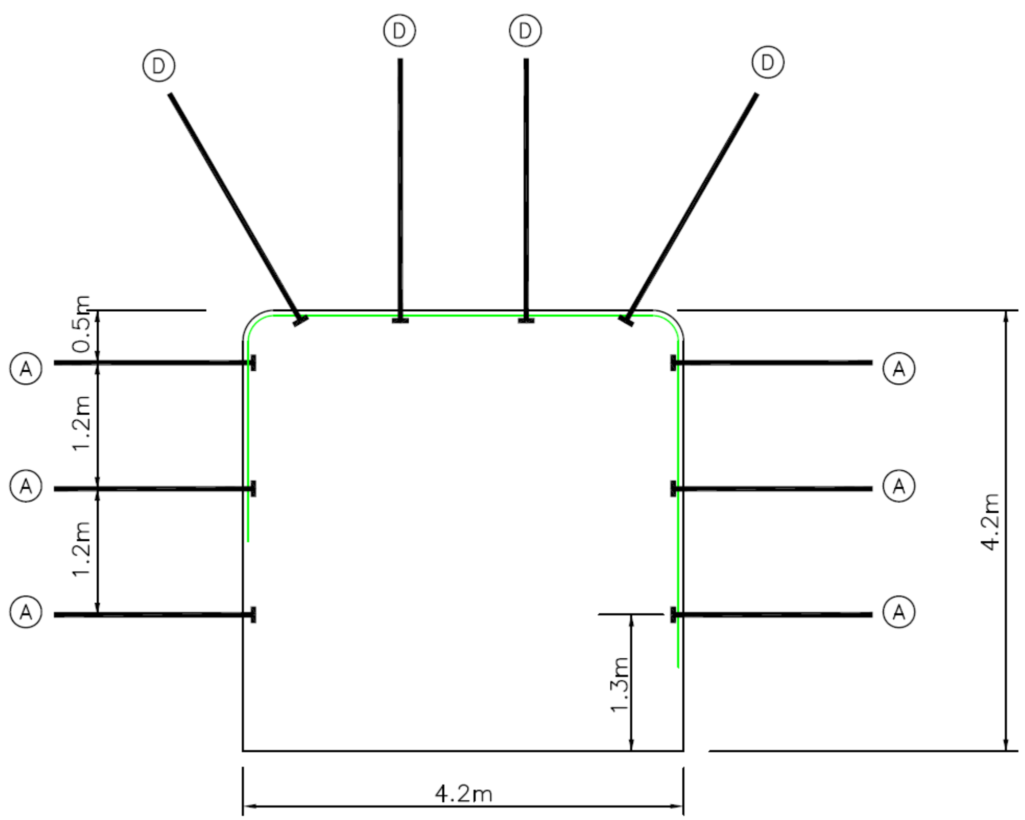
TYPE RAMP - I

FILE NAME GCMP SUPPORT STANDARDS REV. 001

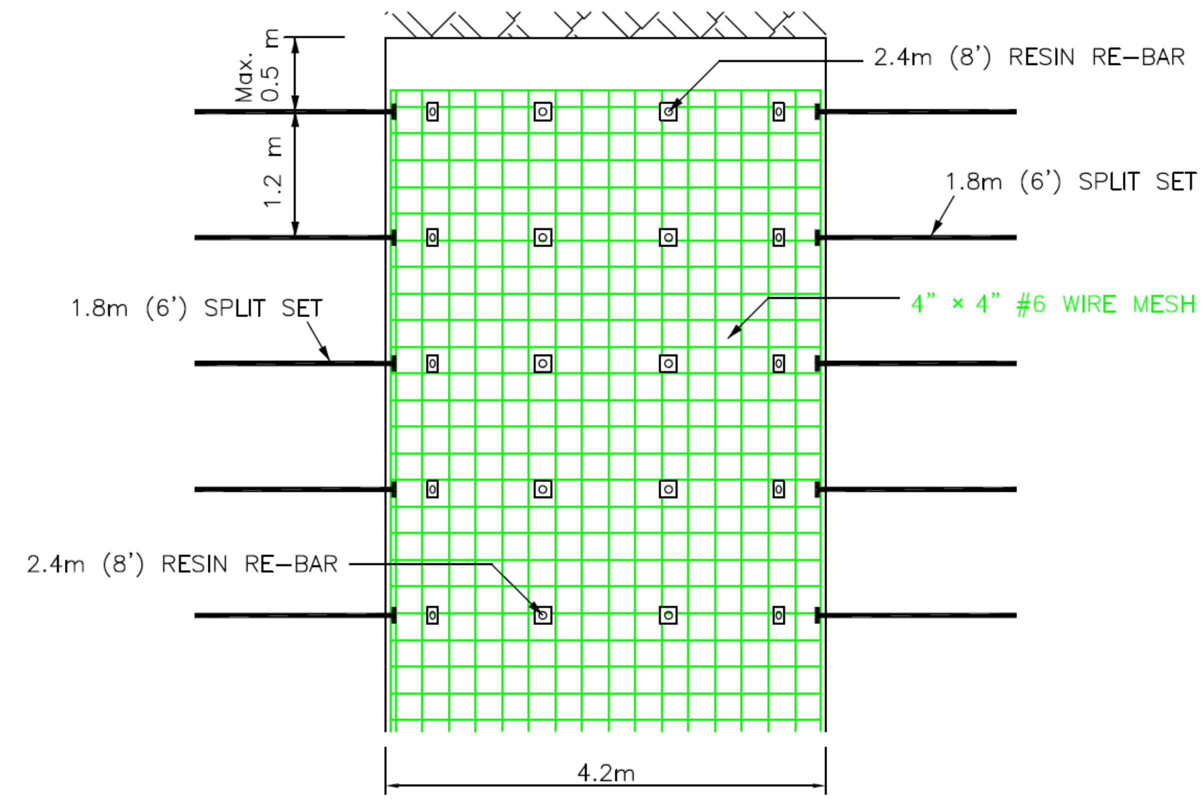
RAMP - II (4.2mW by 4.2mH)

Poor to Fair Ground (45 > GSI > 30)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN RE–BAR	2.4m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN RE–BAR	1.8
C	12T SWELLEX	2.4
D	RESIN RE–BAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

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6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE RAMP - II

FILE NAME: GCMP SUPPORT STANDARDS

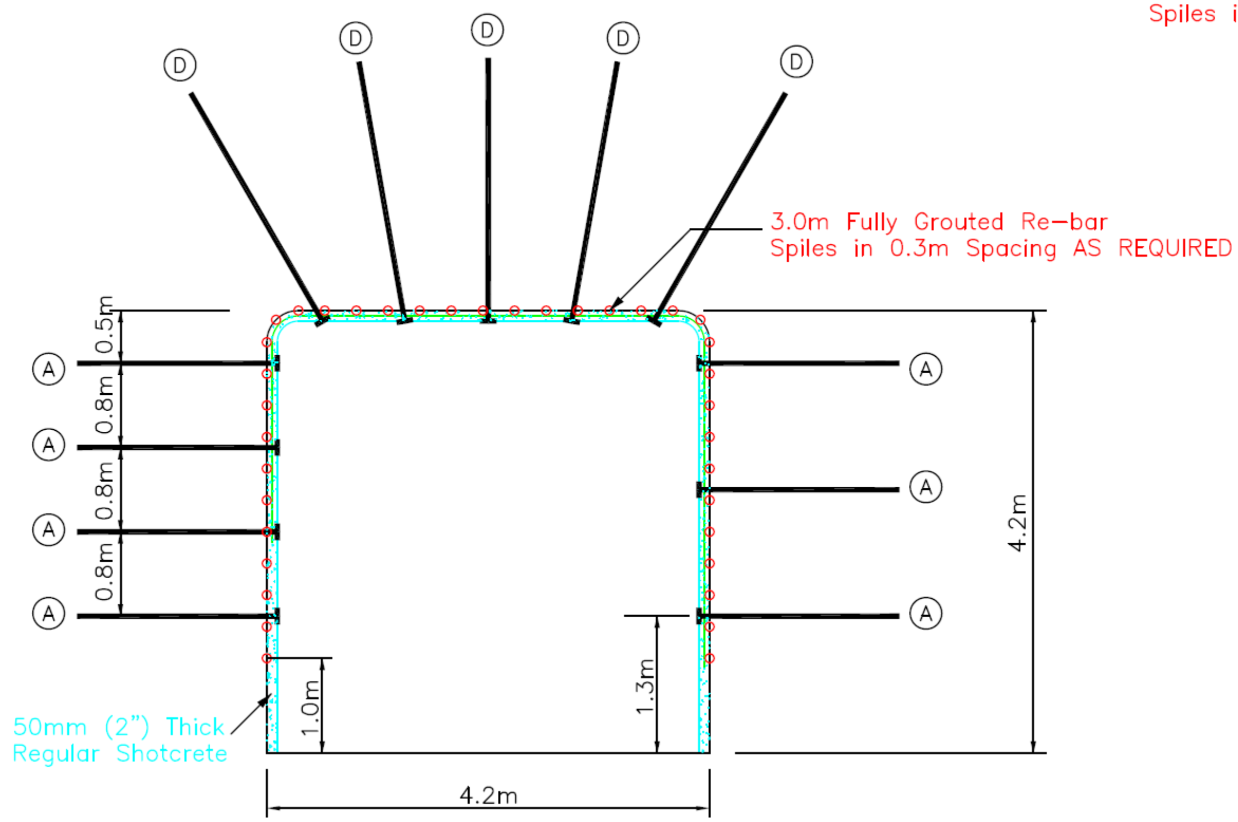
REV. 001

RAMP - III (4.2mW by 4.2mH)

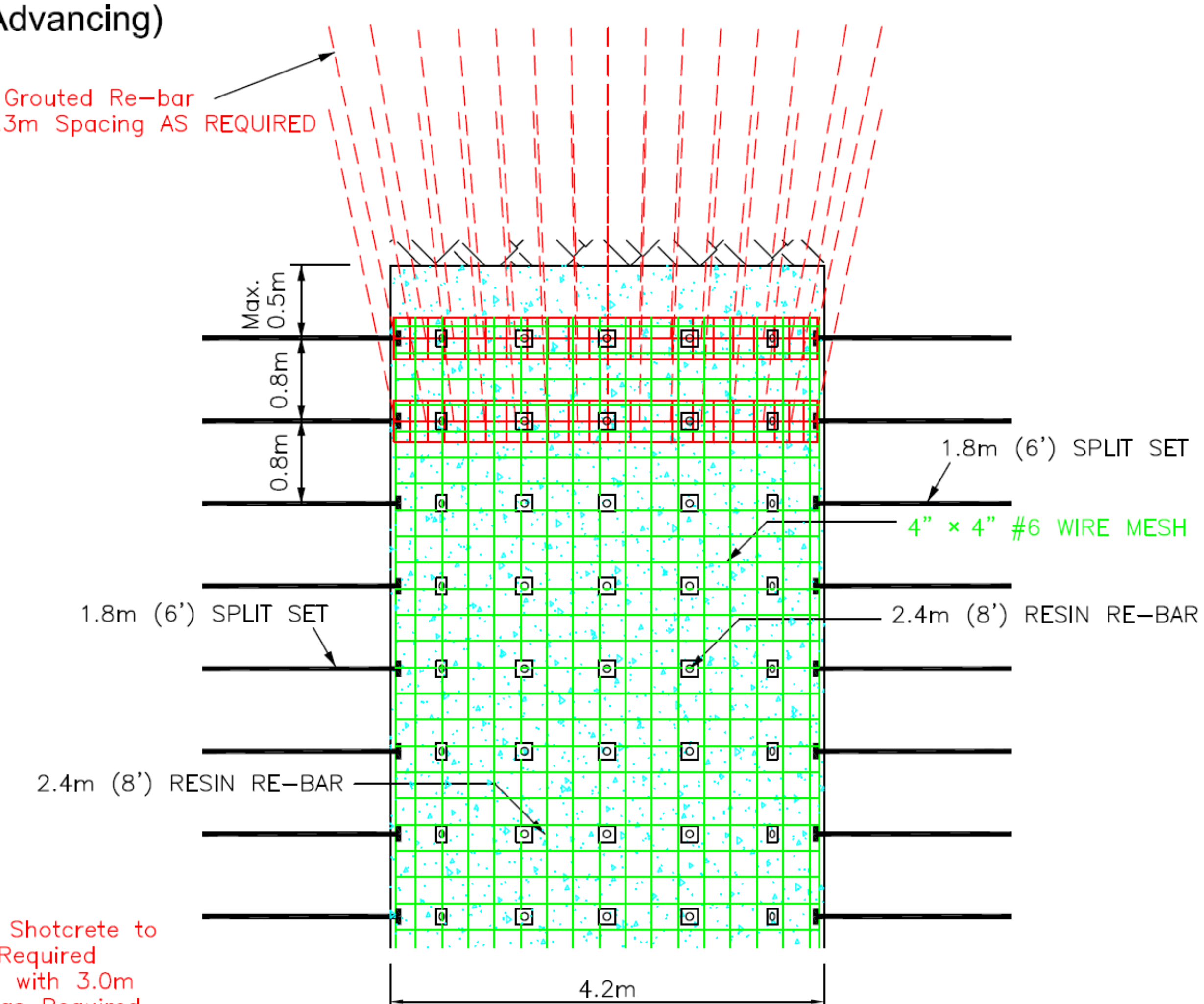
Extremely Poor Ground (GSI < 30)

(Short Round - Max. 2m Advancing)

SECTION



PLAN



− 25mm (1") Flash−Coat Shotcrete to Back and Shoulder as Required
− Spiling at 0.3m centers with 3.0m Grouted Re−bar Spiles as Required

SUPPORT ELEMENTS

LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN RE−BAR	2.4m	0.8mX0.8m	REG	25mm (PRE) 50mm (POST)	AS NOTED
WALLS	SPLIT SET	1.8m	0.8mX0.8m	REG	25mm (PRE) 50mm (POST)	AS NOTED

SUPPORT BOLT TABLE

No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN RE−BAR	1.8
C	12T SWELLEX	2.4
D	RESIN RE−BAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
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8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:	
Checked By:	
Drawn By:	
SCALE:	N.T.S

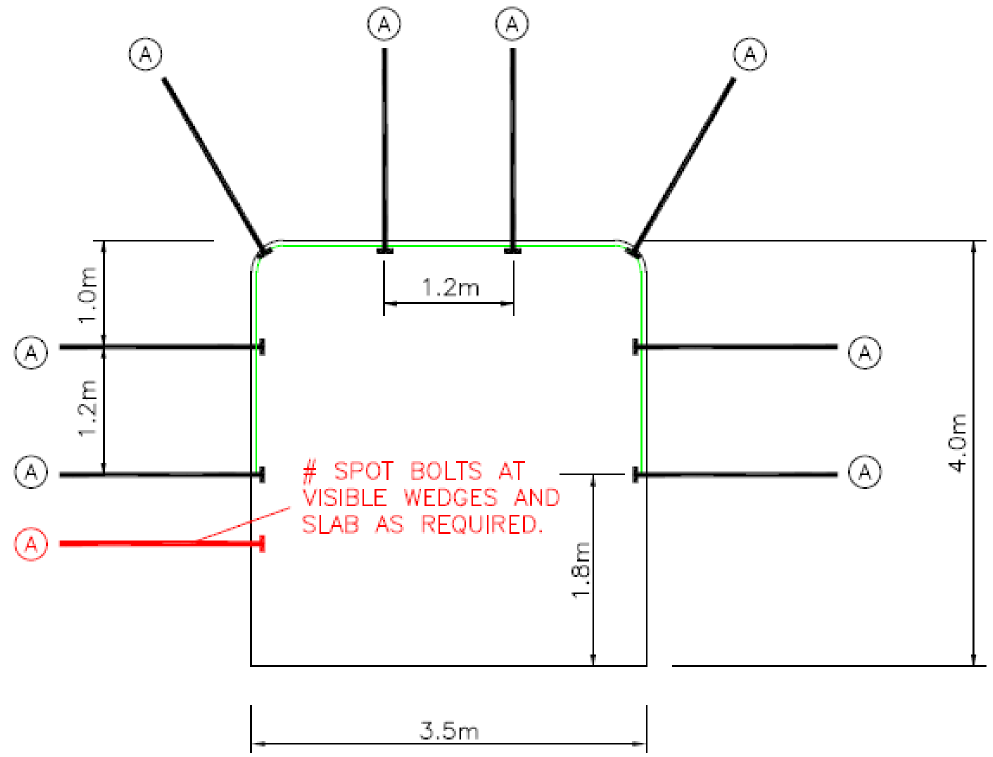
TYPE RAMP - III

FILE NAME: GCMP SUPPORT STANDARDS

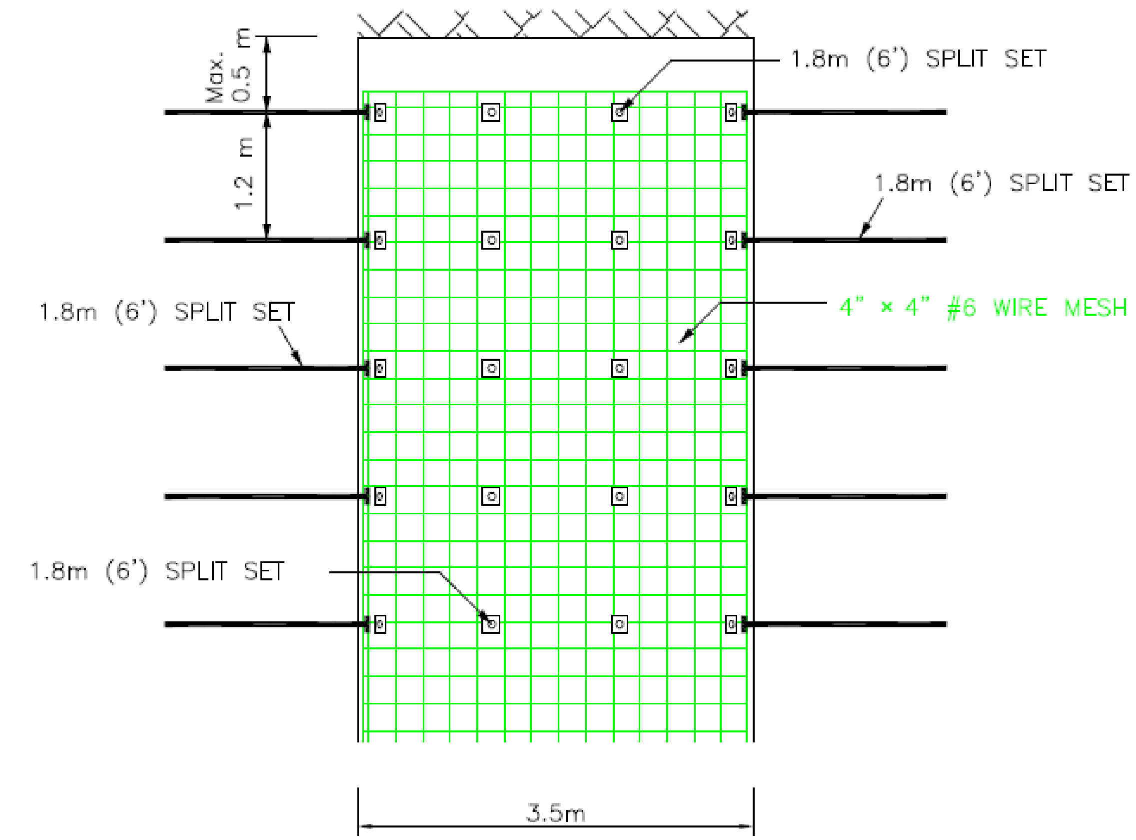
REV. 001

MD - I (3.5mW by 4.0mH)
Fair to Good Ground (45 < GSI < 60)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
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7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

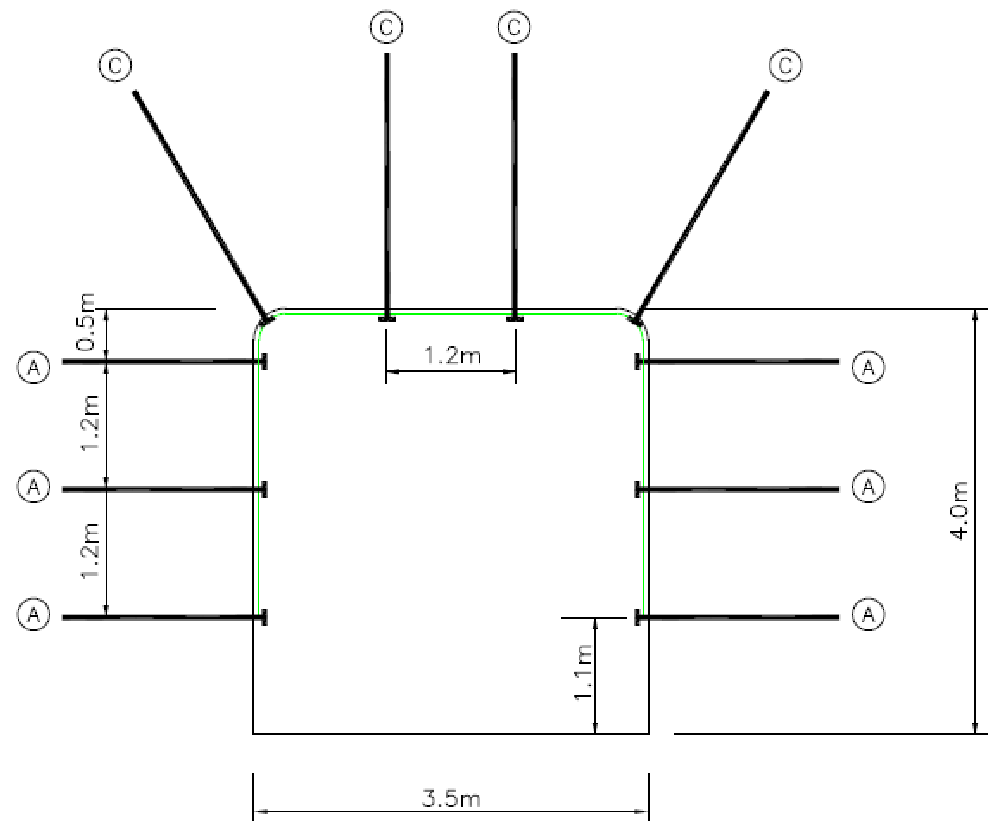
TYPE MD - I

FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

MD - II (3.5mW by 4.0mH)
Poor to Fair Ground (30 < GSI < 45)

SECTION



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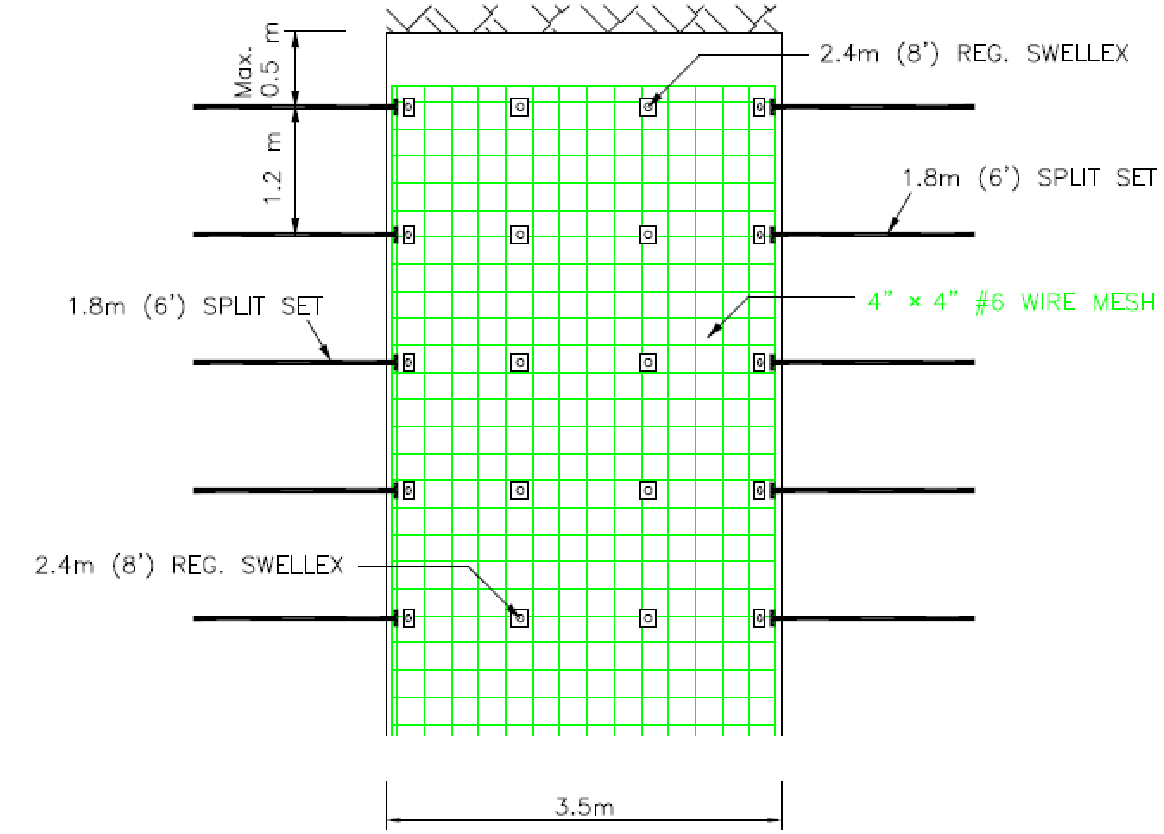

SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	REG. SWELLEX	2.4m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES
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8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE MD - II

FILE NAME: GCMP SUPPORT STANDARDS

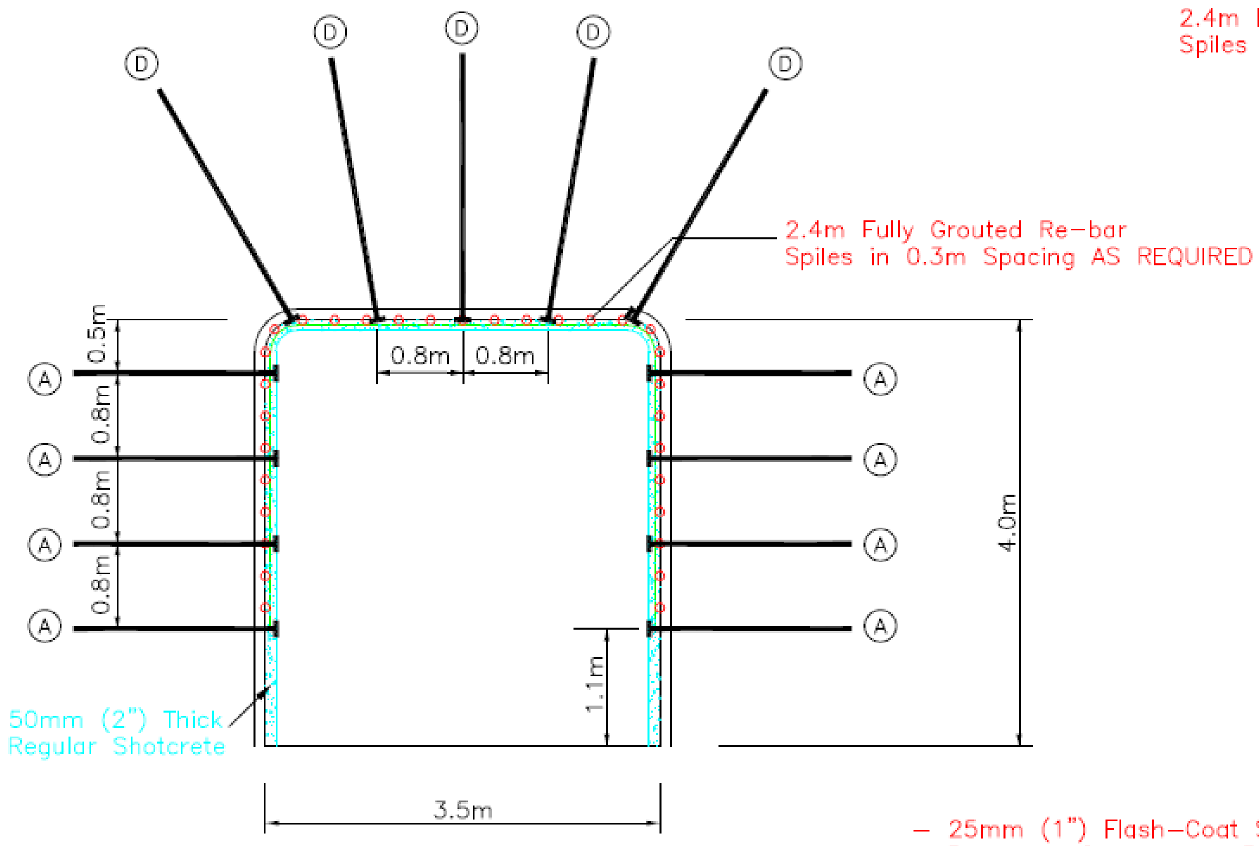
REV. 001

MD - III (3.5mW by 4.0mH)

Extremely Poor Ground (GSI < 30)

(Short Round - Max. 2m Advancing)

SECTION



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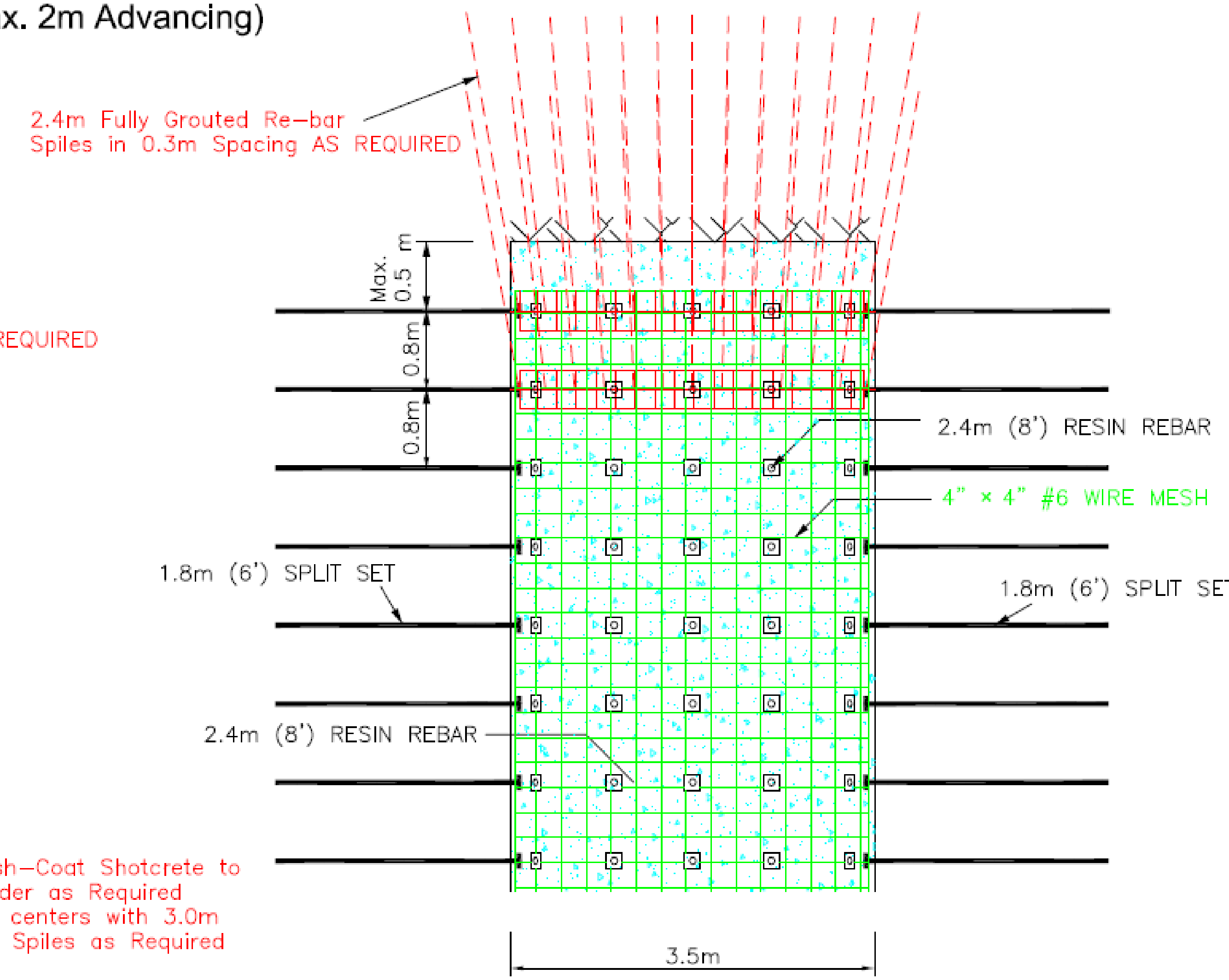

– 25mm (1") Flash-Coat Shotcrete to Back and Shoulder as Required
– Spiling at 0.3m centers with 3.0m Grouted Re-bar Spiles as Required

SUPPORT ELEMENTS

LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	REG. SWELLEX	2.4m	0.8mX0.8m	REG	25mm (PRE) 50mm (POST)	AS NOTED
WALLS	SPLIT SET	1.8m	0.8mX0.8m	REG	25mm (PRE) 50mm (POST)	AS NOTED

SUPPORT BOLT TABLE

No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

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8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

TYPE MD - III

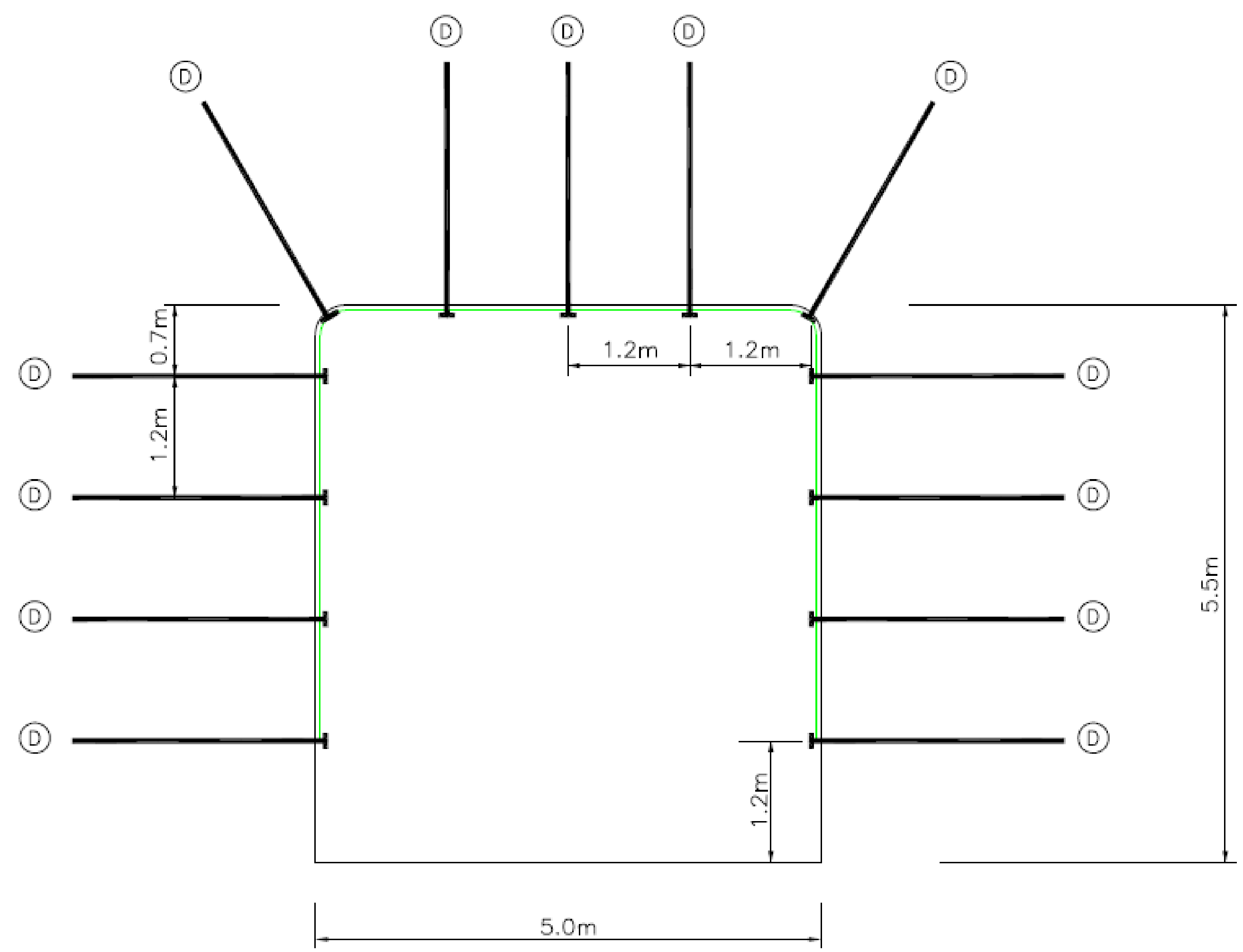
FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

RMK - I (5.0mW by 5.5mH)

Fair to Good Ground (45 < GSI < 60)

SECTION



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN REBAR	2.4m	1.2mX1.2m	–	–	AS NOTED
WALLS	RESIN REBAR	2.4m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
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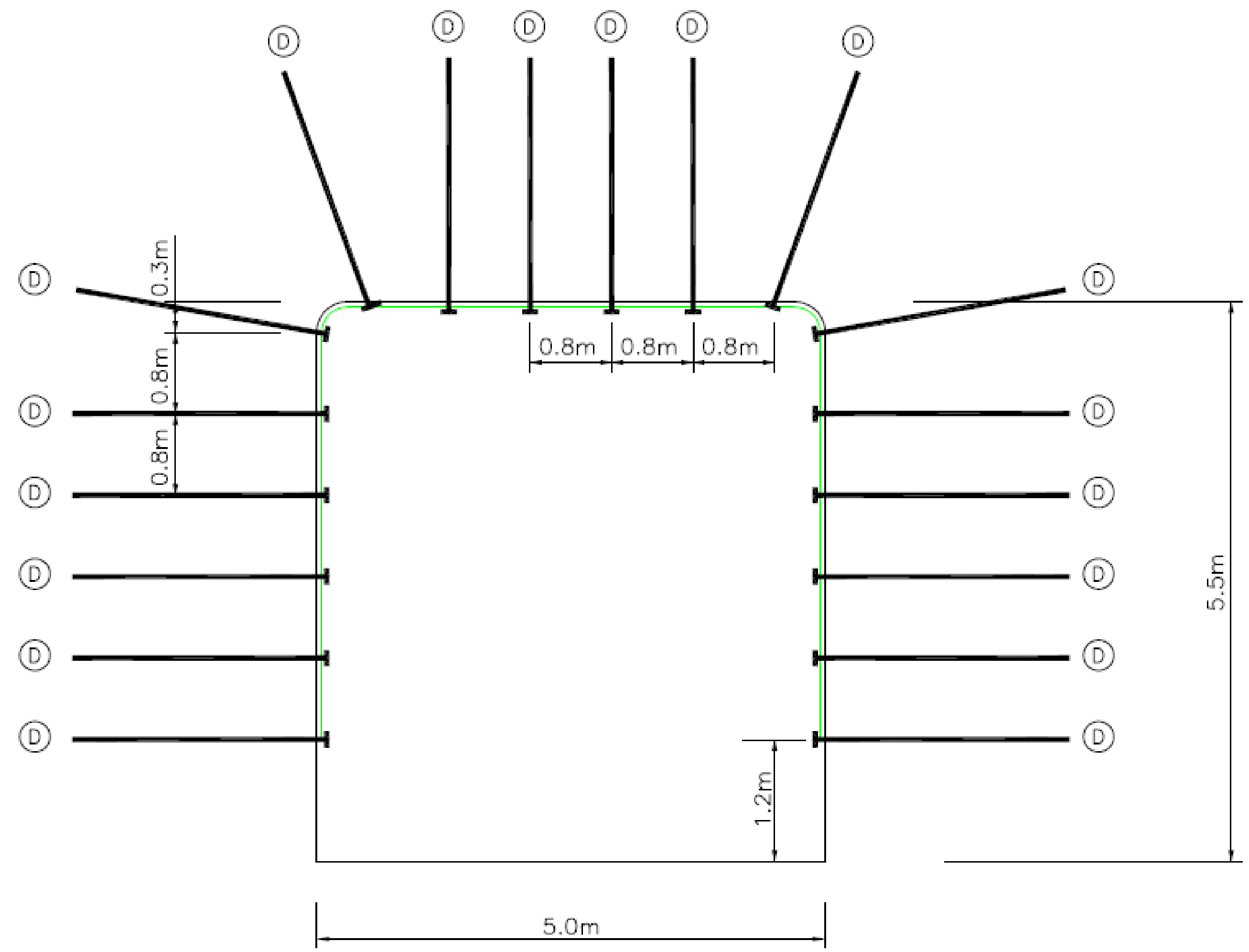
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS	ALEXCO	Designed By: Checked By: Drawn By: SCALE: N.T.S	TYPE RMK - I FILE NAME: GCMP SUPPORT STANDARDS	REV. 001

RMK - I (5.0mW by 5.5mH)

Poor to Fair Ground (30 < GSI < 45)

SECTION



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	RESIN REBAR	2.4m	0.8mX0.8m	–	–	AS NOTED
WALLS	RESIN REBAR	2.4m	0.8mX0.8m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
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GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE N.T.S

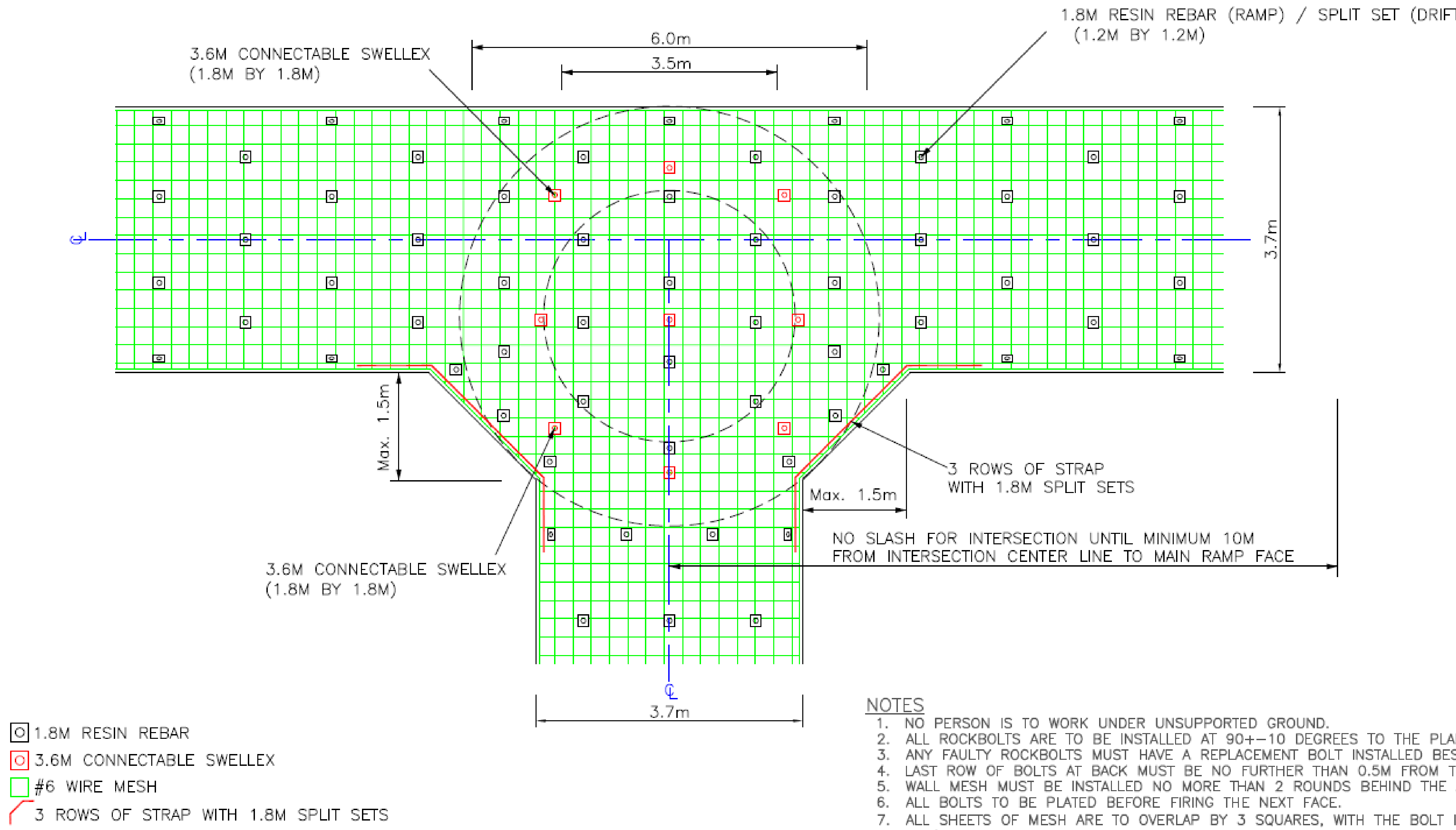
TYPE RMK - II

FILE NAME GCMP SUPPORT STANDARDS

REV. 001

IS - I

Fair to Good Ground (60 > GSI > 45)



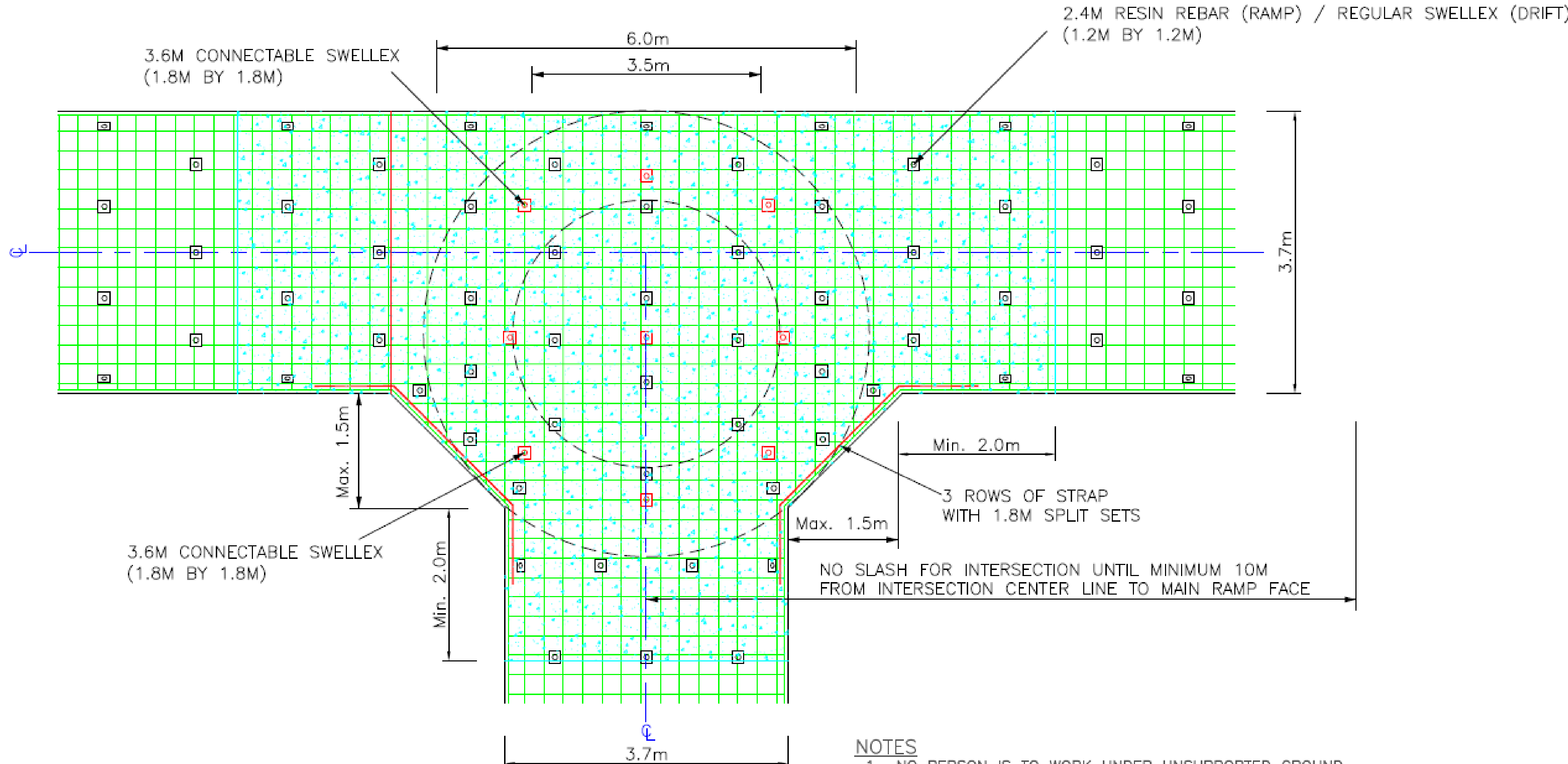
NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS	ALEXCO	Designed By: Checked By: Drawn By: SCALE: N.T.S	TYPE IS - I FILE NAME: GCMP SUPPORT STANDARDS REV. 001

IS - II

Poor to Fair Ground (45 > GSI > 30)



- 2.4M RESIN REBAR (RAMP) / 2.4M REGULAR SWELLEX (DRIFT)
- 3.6M CONNECTABLE SWELLEX
- #6 WIRE MESH
- 3 ROWS OF STRAP WITH 1.8M SPLIT SETS
- 50mm REGULAR POST SHOTCRETE OR 50mm FIBER PRE SHOTCRETE AS REQUIRED

NOTES

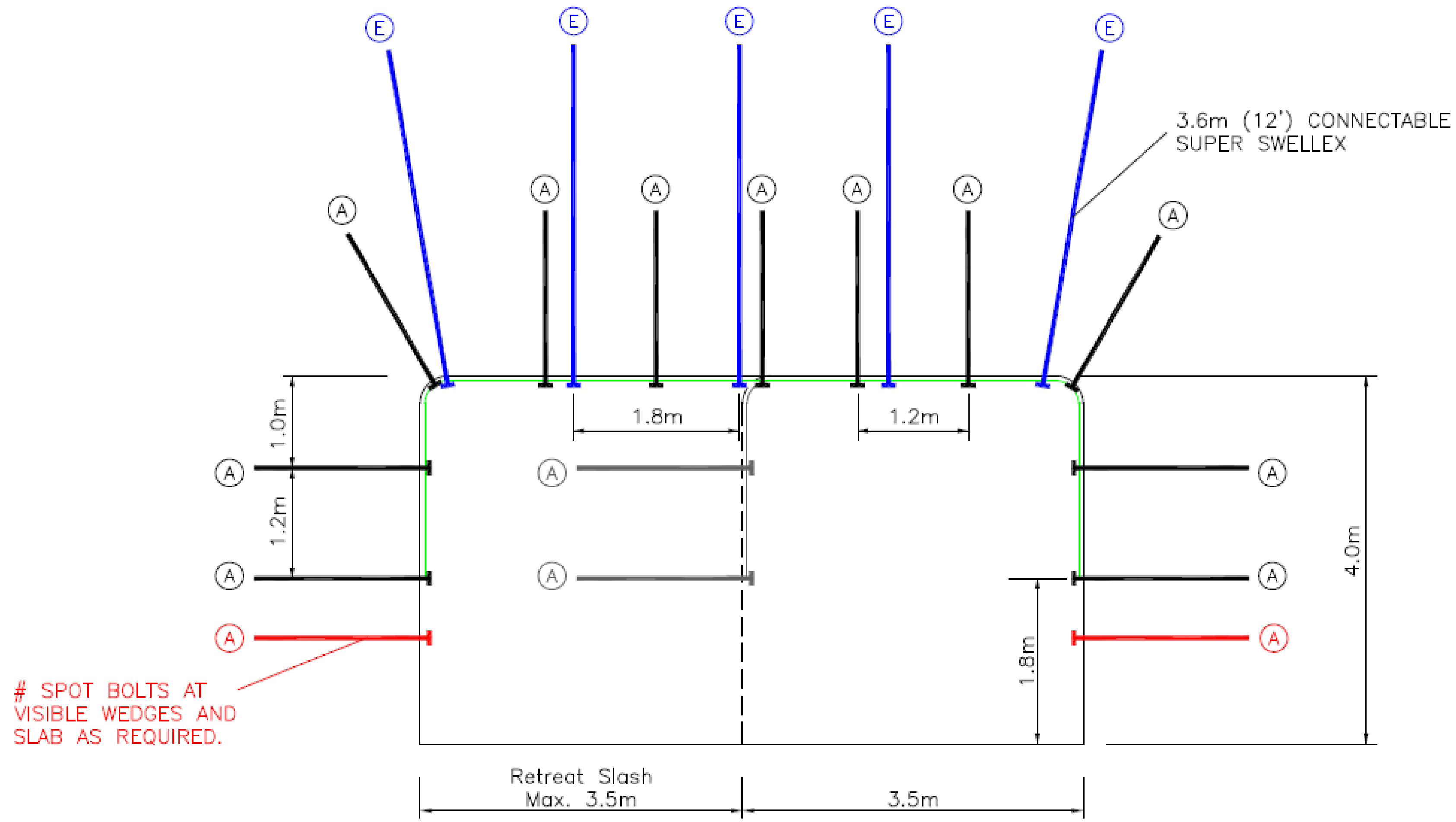
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS	ALEXCO	Designed By: Checked By: Drawn By: SCALE: N.T.S	TYPE IS - II FILE NAME: GCMP SUPPORT STANDARDS	REV. 001

WD - I (Max. 7.0mW by 4.0mH)

Fair to Good Ground (45 < GSI < 60)

SECTION



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

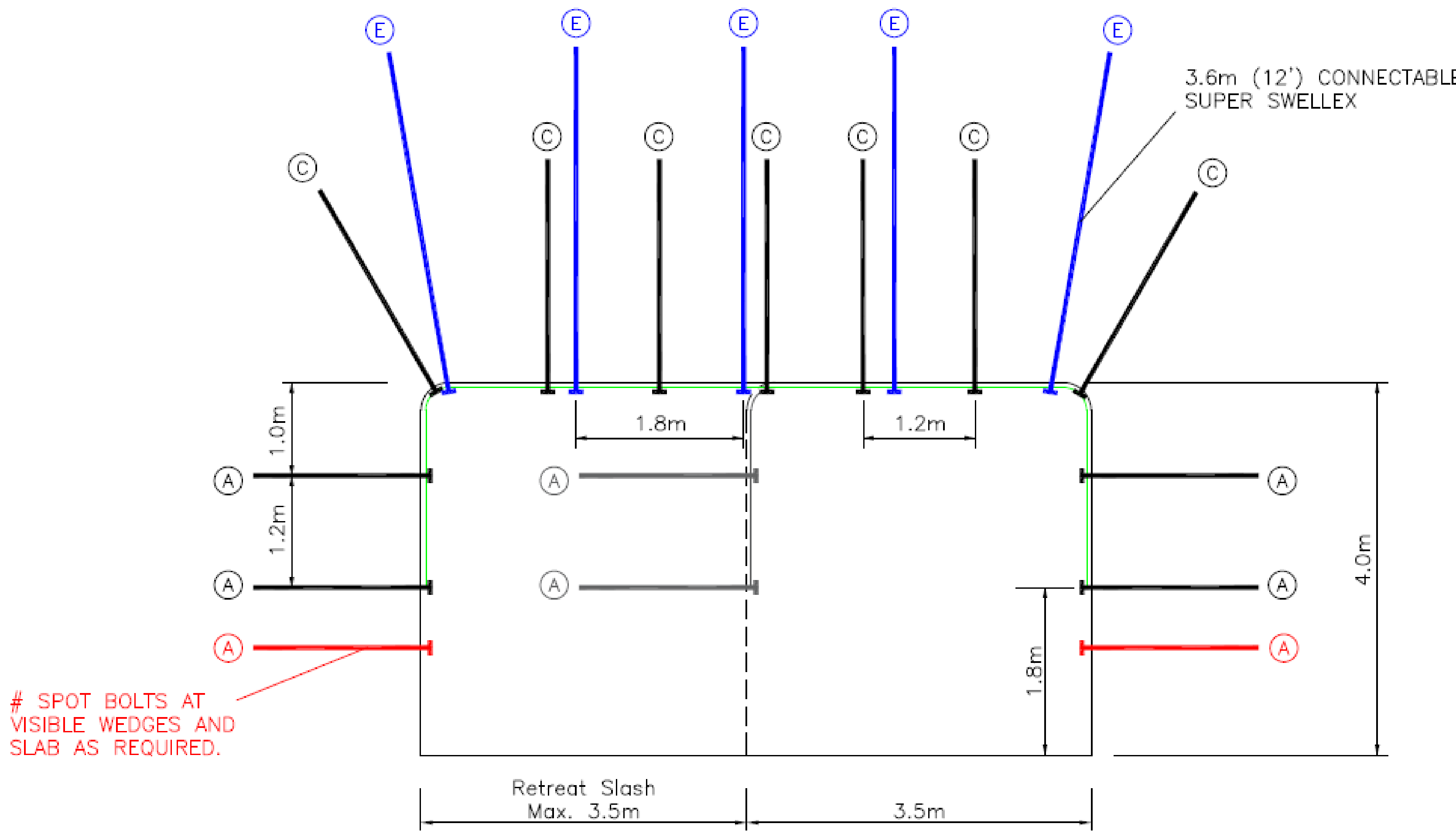
TYPE WD - I

FILE NAME: GCMP SUPPORT STANDARDS

REV. 001

WD - II (Max. 7.0mW by 4.0mH)
Poor to Fair Ground (30 < GSI < 45)

SECTION



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
BACK	REG. SWELLEX	2.4m	1.2mX1.2m	–	–	AS NOTED
WALLS	SPLIT SET	1.8m	1.2mX1.2m	–	–	AS NOTED

SUPPORT BOLT TABLE		
No.	TYPE	LENGTH (m)
A	SPLIT SET	1.8
B	RESIN REBAR	1.8
C	12T SWELLEX	2.4
D	RESIN REBAR	2.4
E	24T CONNECTABLE	3.6
F	CABLE BOLT	5.0

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE: N.T.S

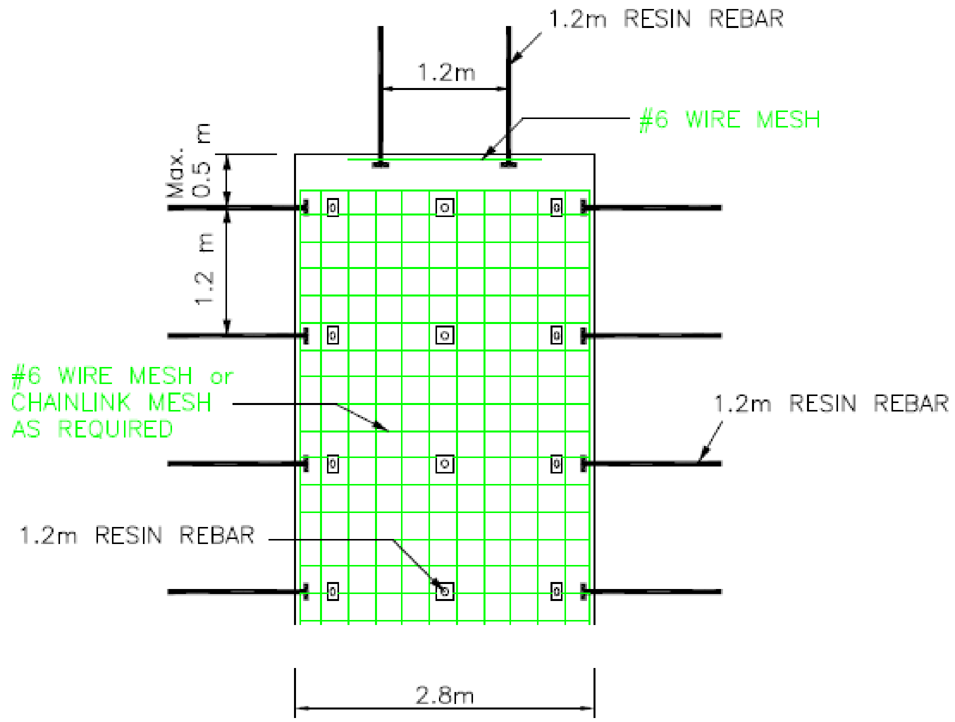
TYPE WD - II

FILE NAME: GCMP SUPPORT STANDARDS

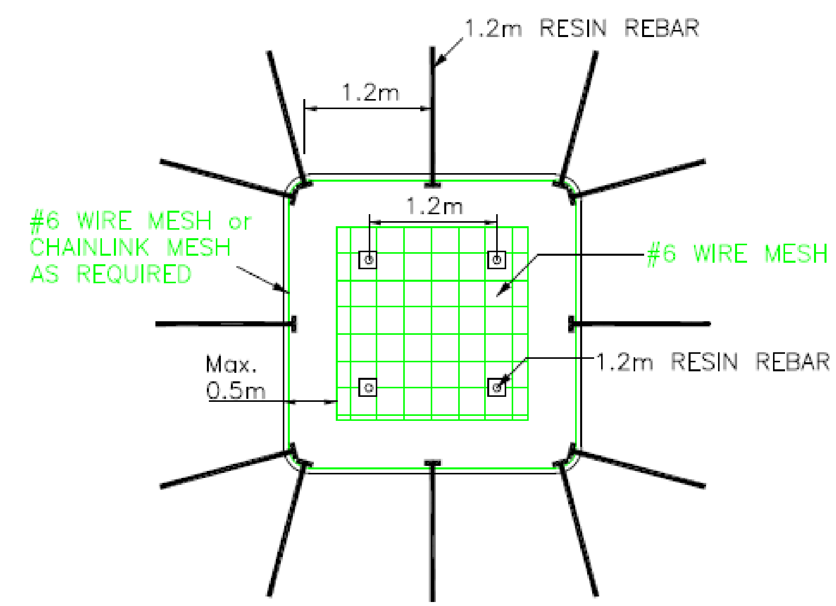
REV. 001

SR - I (2.8mW by 2.8mH)
Fair to Good Ground (45 < GSI < 60)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
FACE	RESIN REBAR	1.2m	1.2mX1.2m	–	–	AS NOTED
WALLS	RESIN REBAR	1.2m	1.2mX1.2m	–	–	AS NOTED

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

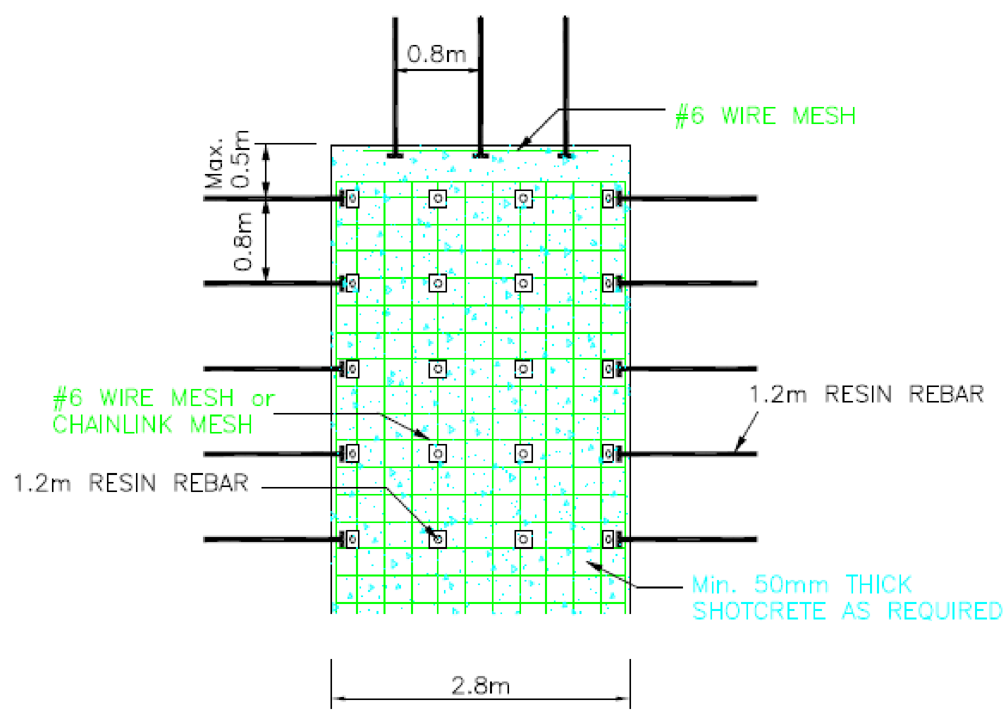
Designed By:
Checked By:
Drawn By:
SCALE N.T.S

TYPE SR - I

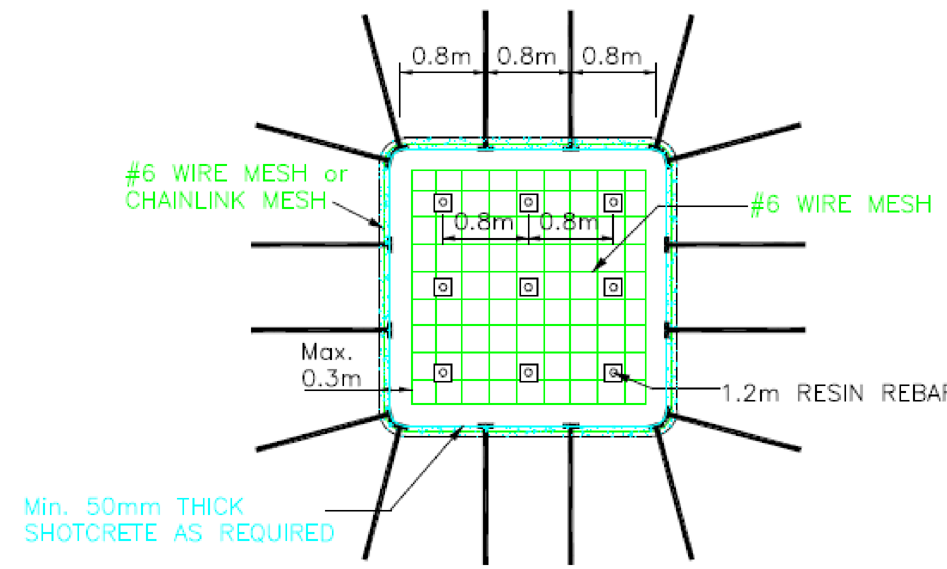
FILE NAME GCMP SUPPORT STANDARDS | REV. 001

SR - II (2.8mW by 2.8mH)
Ext. Poor to Poor Ground (GSI < 45)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
FACE	RESIN REBAR	1.2m	0.8mX0.8m	REG	50mm AS REQ.	AS NOTED
WALLS	RESIN REBAR	1.2m	0.8mX0.8m	REG	50mm AS REQ.	AS NOTED

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
4. LAST ROW OF BOLTS AT BACK MUST BE NO FURTHER THAN 0.5M FROM THE FACE.
5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

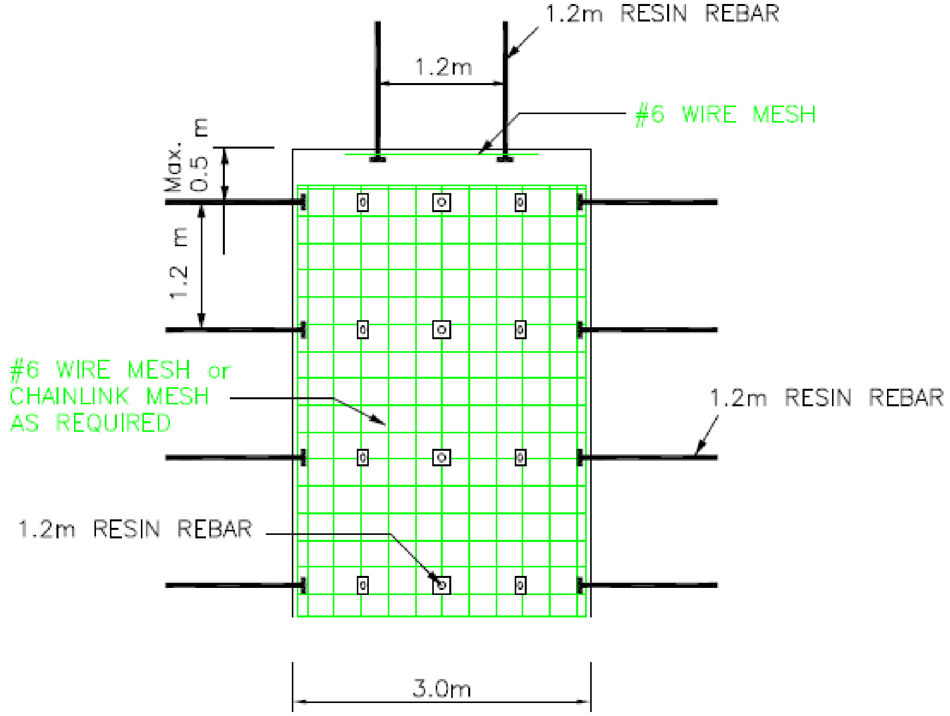
Designed By:	
Checked By:	
Drawn By:	
SCALE:	N.T.S

TYPE SR - II

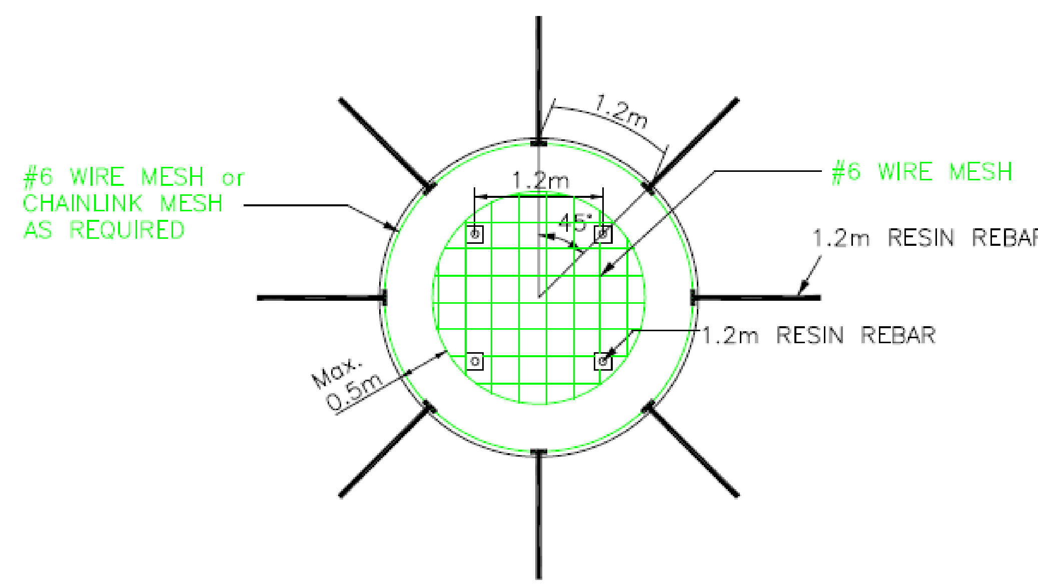
FILE NAME: GCMP SUPPORT STANDARDS | REV. 001

CR - I (D = 3.0m)
Fair to Good Ground (45 < GSI < 60)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
FACE	RESIN REBAR	1.2m	1.2mX1.2m	–	–	AS NOTED
WALLS	RESIN REBAR	1.2m	1.2mX1.2m	–	–	AS NOTED

NOTES
1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+–10 DEGREES TO THE PLANNED PROFILE.
3. ANY FAULTY ROCKBOLTS MUST HAVE A REPLACEMENT BOLT INSTALLED BESIDE IT.
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5. WALL MESH MUST BE INSTALLED NO MORE THAN 2 ROUNDS BEHIND THE ACTIVE FACE.
6. ALL BOLTS TO BE PLATED BEFORE FIRING THE NEXT FACE.
7. ALL SHEETS OF MESH ARE TO OVERLAP BY 3 SQUARES, WITH THE BOLT IN THE MIDDLE (2ND ROW) OF THE OVERLAP.
8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:	
Checked By:	
Drawn By:	
SCALE:	N.T.S

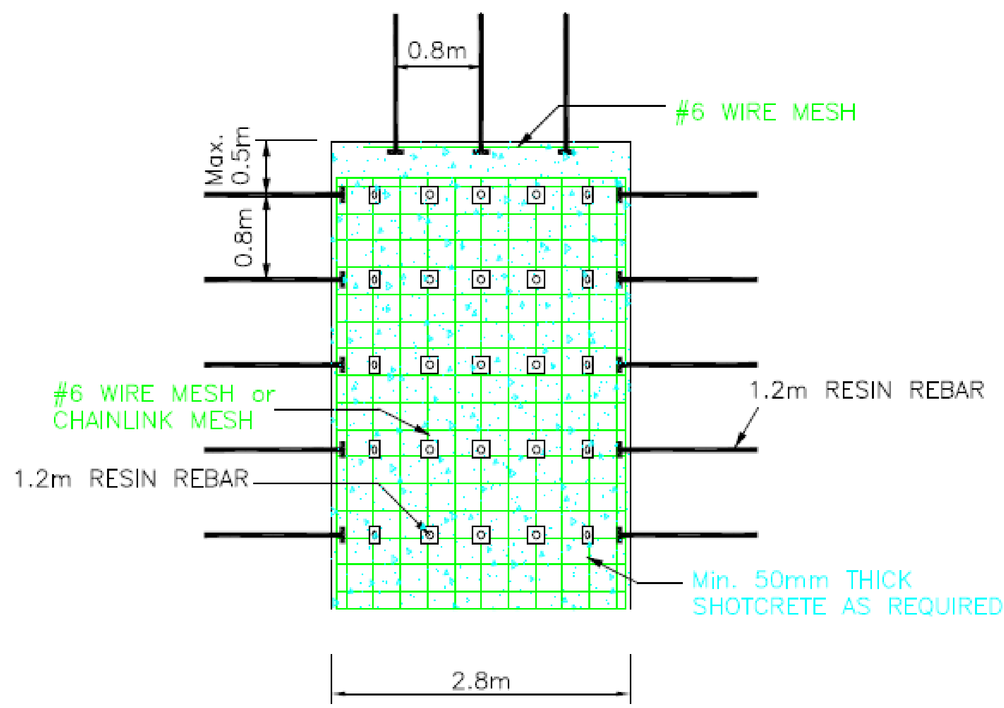
TYPE CR - I

FILE NAME: GCMP SUPPORT STANDARDS | REV. 001

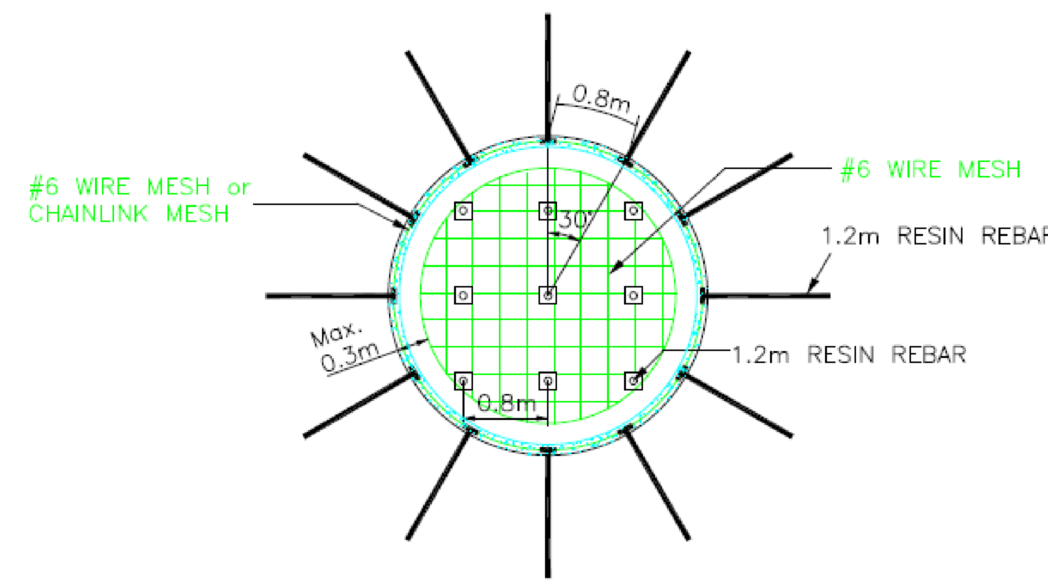
CR - II (D = 3.0m)

Ext. Poor to Poor Ground (GSI < 45)

SECTION



PLAN



SUPPORT ELEMENTS						
LOCATION	ROCK BOLT			SHOTCRETE		#6 MESH
	TYPE	LENGTH	PATTERN	TYPE	THICKNESS	
FACE	RESIN REBAR	1.2m	0.8mX0.8m	REG	50mm AS REQ.	AS NOTED
WALLS	RESIN REBAR	1.2m	0.8mX0.8m	REG	50mm AS REQ.	AS NOTED

NOTES

1. NO PERSON IS TO WORK UNDER UNSUPPORTED GROUND.
2. ALL ROCKBOLTS ARE TO BE INSTALLED AT 90+-10 DEGREES TO THE PLANNED PROFILE.
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8. SPOT BOLTS VISIBLE WEDGES AND SLABS AS REQUIRED

GROUND SUPPORT CONFIGURATIONS

ALEXCO

Designed By:
Checked By:
Drawn By:
SCALE N.T.S

TYPE CR - II

FILE NAME GCMP SUPPORT STANDARDS
REV. 001

APPENDIX – C.

TRIGGER ACTION RESPONSE PLAN

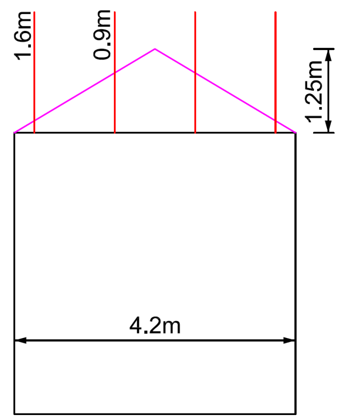
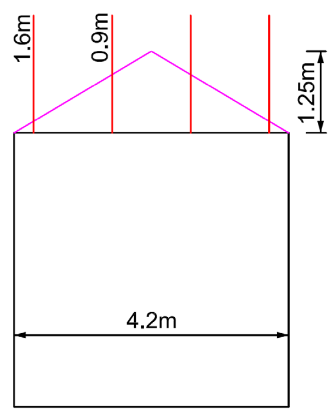
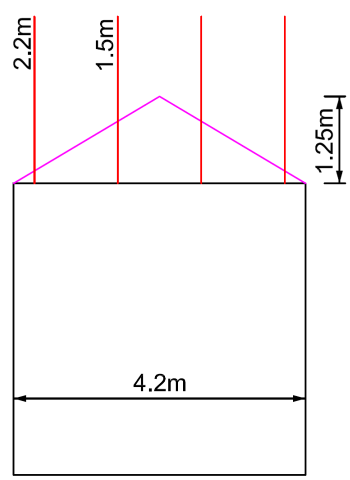
TARP Guideline for Men Entry Openings

Section	Dimension (W x H, m)	Ground Condition		Support Type					Wire Mesh	Additional Support	Comments
		Drilling Condition	Geotechnical Face Mapping Condition		Loc.	Type	Length (m)	Spacing (m x m)			
Main Ramp	4.2 x 4.2	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**Ramp – I**	Back	Rebar	1.8	1.2 x 1.2	#6 up to 2.0m from sill		
					Wall	Split Set	1.8				
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**Ramp – II**	Back	Rebar	2.4	1.2 x 1.2	#6 up to 1.3m from sill	5mm reg. shotcrete as req.	
					Wall	Split Set	1.8				
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Ramp – III**	Back	Rebar	2.4	0.8 x 0.8	#6 up to 1.3m from sill	5mm pre shotcrete as req. 5mm post shotcrete	Spiling as req.
					Wall	Split Set	1.8				
Main Drift	3.5 x 4.0	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**MD – I**	Back	Split Set	1.8	1.2 x 1.2	#6 up to 1.8m from sill		
					Wall	Split Set	1.8				
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**MD – II**	Back	R. Swellex	2.4	1.2 x 1.2	#6 up to 1.1m from sill	5mm reg. shotcrete as req.	
					Wall	Split Set	1.8				
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**MD – III**	Back	R. Swellex	2.4	0.8 x 0.8	#6 up to 1.1m from sill	5mm pre shotcrete as req. 5mm post shotcrete	Spiling as req.
					Wall	Split Set	1.8				
Wide Drift	3.5~7.0 X 4.0	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**WD – I**	Back	Split Set	1.8	1.2 x 1.2	#6 up to 1.8m from sill	3.6m connectable 1.8x1.8	3.5m drift with Max. 3.5m slash
					Wall	Split Set	1.8				
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**WD – II**	Back	R. Swellex	2.4	1.2 x 1.2	#6 up to 1.1m from sill	3.6m connectable 1.8x1.8 5mm reg. shotcrete as req.	3.5m drift with Max. 3.5m slash
					Wall	Split Set	1.8				
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Development of wide opening is not allowed for this ground condition**							
Remuck	5.0 X 5.5	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**RMK – I**	Back	Rebar	2.4	1.2 x 1.2	#6 up to 1.2m from sill		
					Wall	Rebar	2.4				
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**RMK – II**	Back	Rebar	2.4	0.8 x 0.8	#6 up to 1.2m from sill		
					Wall	Rebar	2.4				
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Development of wide opening is not allowed for this ground condition**							
Raise	2.8 X 2.8 or D= 3.0	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**SR – I** **CR – I**	Face	Rebar	1.2	1.2 x 1.2	#6 Wire mesh or Chainlink		
					Wall	Rebar	1.2				
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**SR – II** **CR – II**	Face	Rebar	1.2	0.8 x 0.8	#6 Wire mesh or Chainlink	5mm reg. shotcrete as req.	
					Wall	Rebar	1.2				
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Development of wide opening is not allowed for this ground condition**							
Intersection	R < 6.0	Fair – Good	Partially disturbed very blocky ground. More than four discontinuities. Smooth surface (45<GSI<60)	**IS – I**	Back	Rebar	2.4	1.2 x 1.2	#6 up to 1.8m from sill	3.6m connectable 1.8x1.8	3 straps for Pillar support
					Wall	Split Set	1.8				
		Poor – Fair	Folded with angular blocks. Intersecting discontinuity sets. Bedding planes or schistosity (30<GSI<45)	**IS - II**	Back	Rebar	2.4	1.2 x 1.2	#6 up to 1.1m from sill	3.6m connectable 1.8x1.8 5mm reg. shotcrete as req.	3 straps for Pillar support
					Wall	Split Set	1.8				
		Ext. Poor - Poor	Poorly interlocked. Heavily broken rock mass with mix of angular and rounded rock pieces (GSI < 30)	**Relocate intersection development to better ground condition**							

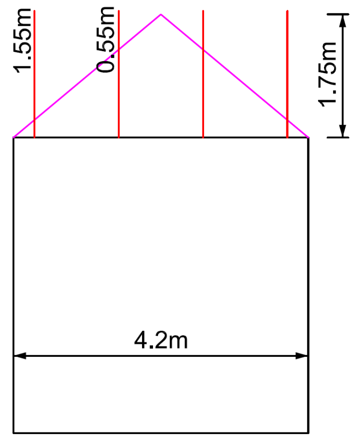
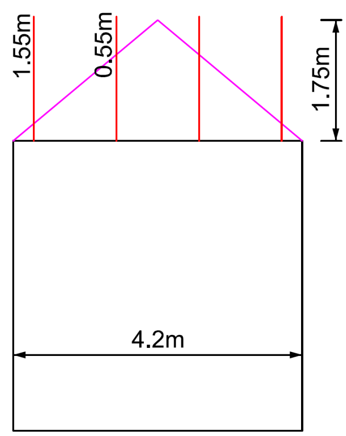
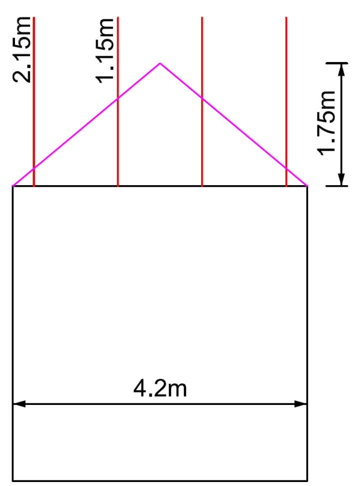
APPENDIX – D.

DEAD WEIGHT ANALYSIS

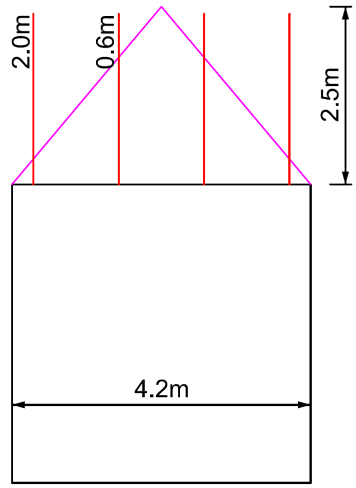
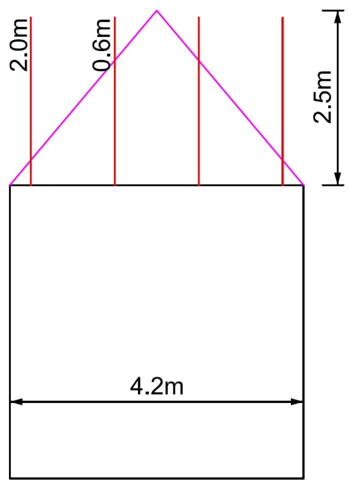
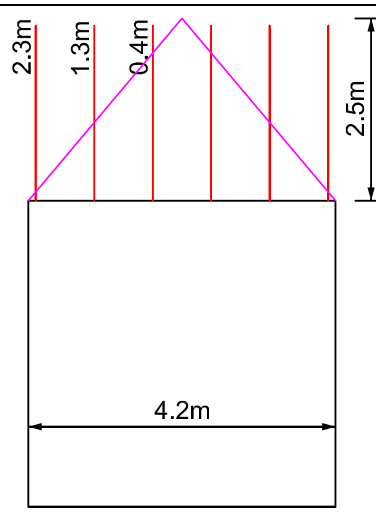
Main Ramp (B = 4.2m) in Fair to Good ground (45 < GSI < 60)

1.8m Split set		***Dead Weight of Wedge Block*** (4.2m × 1.25m) / 2 × 3.2 t/m³ = 8.4 t ***Support Capacity*** (1.6×4.5 + 0.9×4.5x2) / 1.2 = 12.7 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 12.7 / 8.4 = 1.5 (O.K)**
1.8m Rebar		***Dead Weight of Wedge Block*** (4.2m × 1.25m) / 2 × 3.2 t/m³ = 8.4 t ***Support Capacity*** (12 + 0.9×9.0x2) / 1.2 = 23.5 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 23.5 / 8.4 = 2.8 (O.K)**
2.4m Rebar		***Dead Weight of Wedge Block*** (4.2m × 1.25m) / 2 × 3.2 t/m³ = 8.4 t ***Support Capacity*** (12×3) / 1.2 = 30.0 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 30.0 / 8.4 = 3.6 (O.K)**

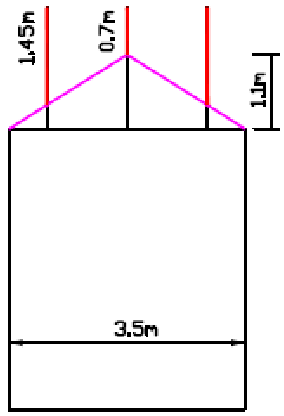
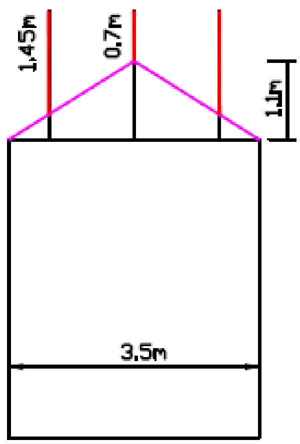
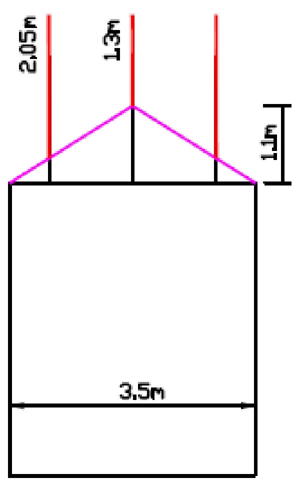
Main Ramp (B = 4.2m) in Poor to Fair ground (30 < GSI < 45)

1.8m Split set		***Dead Weight of Wedge Block*** (4.2m × 1.75m) / 2 × 3.2 t/m^3 = 11.8 t ***Support Capacity*** (1.55×4.5 + 0.55×4.5x2) / 1.2 = 6.8 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 11.8 / 6.8 = 0.6 (N.G)**
1.8m Rebar		***Dead Weight of Wedge Block*** (4.2m × 1.75m) / 2 × 3.2 t/m^3 = 11.8 t ***Support Capacity*** (12 + 0.55×9.0x2) / 1.2 = 18.2 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 18.2 / 11.8 = 1.5 (O.K)**
2.4m Rebar		***Dead Weight of Wedge Block*** (4.2m × 1.75m) / 2 × 3.2 t/m^3 = 11.8 t ***Support Capacity*** (12 + 1.15x9x2) / 1.2 = 27.2 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 27.2 / 11.8 = 2.3 (O.K)**

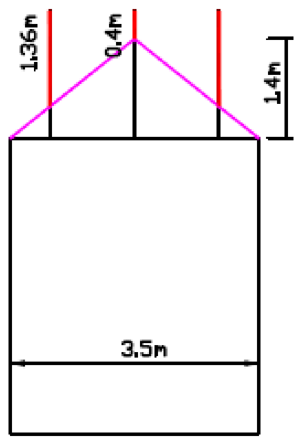
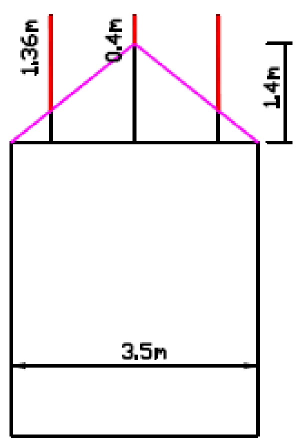
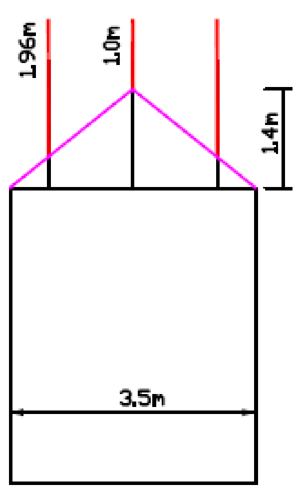
Main Ramp (B = 4.2m) in Extremely Poor ground (20 < GSI < 30)

2.4m Split set		***Dead Weight of Wedge Block*** (4.2m × 2.5m) / 2 × 3.2 t/m³ = 16.8 t ***Support Capacity*** (9 + 0.6×4.5x2) / 1.2 = 12.0 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 12.0 / 16.8 = 0.7 (N.G)**
2.4m Rebar		***Dead Weight of Wedge Block*** (4.2m × 2.5m) / 2 × 3.2 t/m³ = 16.8 t ***Support Capacity*** (12 + 0.6×9.0x2) / 1.2 = 19.0 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 19.0 / 16.8 = 1.1 (N.G)**
2.4m Rebar		***Dead Weight of Wedge Block*** (4.2m × 2.5m) / 2 × 3.2 t/m³ = 16.8 t ***Support Capacity*** (12x3 + 0.4x9x2) / 0.8 = 54.0 t Bond strength = 9.0 t/m (hard rock) Breaking strength = 12.0 t Max. effective length = 12.0/9.0 = 1.3 m **F.S = 54.0 / 16.8 = 3.2 (O.K)**

Main Drift (B = 3.5m) in Fair to Good ground (45 < GSI < 60)

1.8m Split set		***Dead Weight of Wedge Block*** (3.5m × 1.1m) / 2 × 3.2 t/m³ = 6.2 t ***Support Capacity*** (1.45×4.5×2x0.5 + 0.7×4.5) / 1.2 = 8.1 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 8.1 / 6.2 = 1.3 (O.K)**
1.8m Reg. Swellex		***Dead Weight of Wedge Block*** (3.5m × 1.1m) / 2 × 3.2 t/m³ = 6.2 t ***Support Capacity*** (10x2x0.5 + 0.7×7.0) / 1.2 = 12.4 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 12.4 / 6.2 = 2.0 (O.K)**
2.4m Reg. Swellex		***Dead Weight of Wedge Block*** (3.5m × 1.1m) / 2 × 3.2 t/m³ = 6.2 t ***Support Capacity*** (10×2×0.5 + 10) / 1.2 = 16.7 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 16.7 / 6.2 = 2.7 (O.K)**

Main Drift (B = 3.5m) in Poor to Fair ground (30 < GSI < 45)

1.8m Split set		***Dead Weight of Wedge Block*** (3.5m × 1.4m) / 2 × 3.2 t/m^3 = 7.9 t ***Support Capacity*** (1.36×4.5×2x0.5 + 0.4×4.5) / 1.2 = 6.6 t Bond strength = 4.5 t/m (hard rock) Breaking strength = 9.0 t Max. effective length = 9.0/4.5 = 2.0 m **F.S = 6.6 / 7.9 = 0.8 (N.G)**
1.8m Reg. Swellex		***Dead Weight of Wedge Block*** (3.5m × 1.4m) / 2 × 3.2 t/m^3 = 7.9 t ***Support Capacity*** (1.36x7x2x0.5 + 0.4×7.0) / 1.2 = 10.3 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 10.3 / 7.9 = 1.3 (O.K)**
2.4m Reg. Swellex		***Dead Weight of Wedge Block*** (3.5m × 1.4m) / 2 × 3.2 t/m^3 = 7.9 t ***Support Capacity*** (10×2×0.5 + 1.0x9) / 1.2 = 15.8 t Bond strength = 7.0 t/m (hard rock) Breaking strength = 10.0 t Max. effective length = 10.0/7.0 = 1.4 m **F.S = 15.8 / 7.9 = 2.0 (O.K)**

Main Drift (B = 3.5m) in Extremely Poor ground (20 < GSI < 30)

<table>
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Split
set</td>
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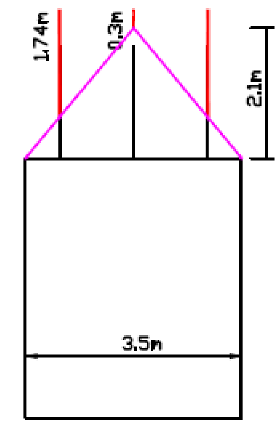


</td>
<td>

Dead Weight of Wedge Block

(3.5m × 2.1m) / 2 × 3.2 t/m^3 = 11.8 t

Support Capacity

(1.7×4.5×2x0.5 + 0.3×4.5) / 1.2 = 7.5 t

Bond strength = 4.5 t/m (hard rock)

Breaking strength = 9.0 t

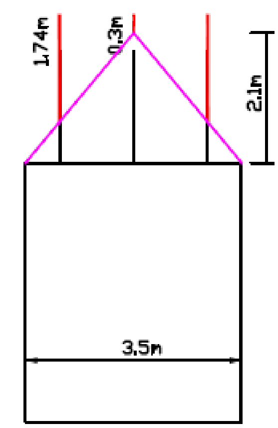
Max. effective length = 9.0/4.5 = 2.0 m

F.S = 7.5 / 11.8 = 0.6 (N.G)

</td>
</tr>
<tr>
<td>2.4m

Reg.

Swellex</td>
<td>



</td>
<td>

Dead Weight of Wedge Block

(3.5m × 2.1m) / 2 × 3.2 t/m^3 = 11.8 t

Support Capacity

(10x2x0.5+0.3x7) / 1.2

= 10.1 t

Bond strength = 7.0 t/m (hard rock)

Breaking strength = 10.0 t

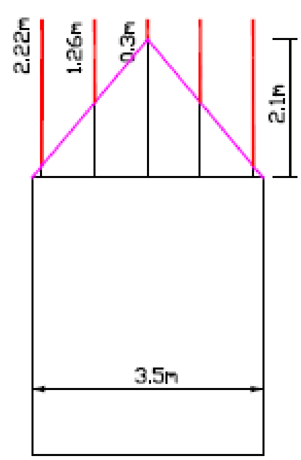
Max. effective length = 10.0/7.0 = 1.4 m

F.S = 10.1 / 11.8 = 0.9 (O.K)

</td>
</tr>
<tr>
<td>2.4m

Reg.

Swellex</td>
<td>



</td>
<td>

Dead Weight of Wedge Block

(3.5m × 2.1m) / 2 × 3.2 t/m^3 = 11.8 t

Support Capacity

(10x2x0.5+1.26×7.0x2+0.3x7) / 1.2

= 24.8 t

Bond strength = 7.0 t/m (hard rock)

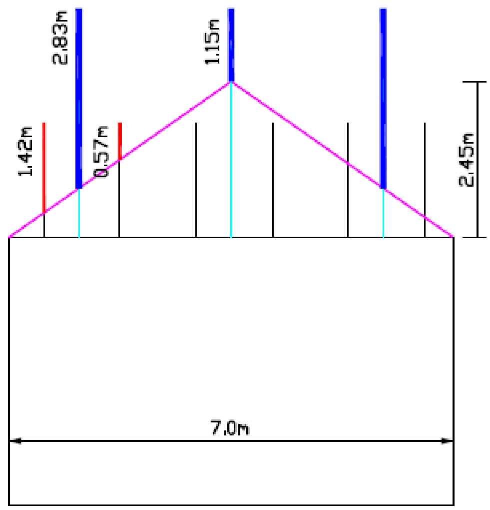
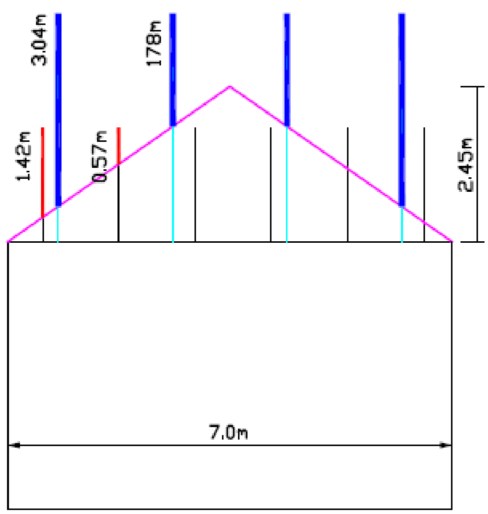
Breaking strength = 10.0 t

Max. effective length = 10.0/7.0 = 1.4 m

F.S = 24.8 / 11.8 = 2.1 (O.K)

</td>
</tr>
</table>

Production Drift (B = 7.0m) in Fair to Good ground (45 < GSI < 60)

1.8m Split set + 3.6m Super Swellex		***Dead Weight of Wedge Block*** (7.0m × 2.45m) / 2 × 3.2 t/m^3 = 27.4 t ***Support Capacity*** (1.42×4.5×2x0.5 + 0.57×4.5x2) / 1.2 + (13x2 + 1.15x10) / 2.4 = 25.2 t Split Set Max. effective length = 9.0/4.5 = 2.0 m Super Swellex Max. effective length = 13.0/10.0 = 1.3 m **F.S = 25.2 / 27.4 = 0.9 (N.G)**
1.8m Split set + 3.6m Super Swellex		***Dead Weight of Wedge Block*** (7.0m × 2.45m) / 2 × 3.2 t/m^3 = 27.4 t ***Support Capacity*** (1.42×4.5×2x0.5 + 0.57×4.5x2) / 1.2 + (13x2 + 13x2) / 1.8 = 38.5 t Split Set Max. effective length = 9.0/4.5 = 2.0 m Super Swellex Max. effective length = 13.0/10.0 = 1.3 m **F.S = 38.5 / 27.4 = 1.4 (O.K)**

Production Drift (B = 7.0m) in Poor to Fair ground (30 < GSI < 45)

<table>
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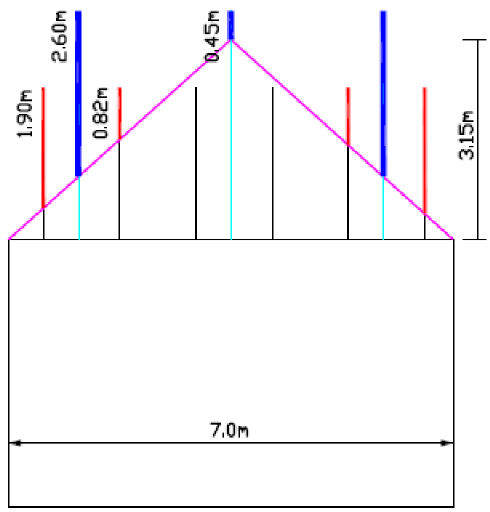
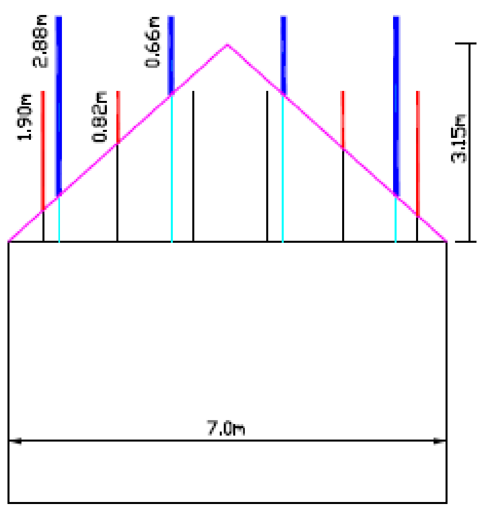
<td>2.4m
Reg.
Swellex
+
3.6m
Super
Swellex</td>
<td>
</td>
<td><u>Dead Weight of Wedge Block</u>
(7.0m × 3.15m) / 2 × 3.2 t/m^3 = 35.3 t

<u>Support Capacity</u>
(10×2x0.5 + 0.82×7.0x2) / 1.2 +
(13x2 + 0.45x10) / 2.4 = 30.6 t
Reg. Swellex
Max. effective length = 10.0/7.0 = 1.4 m
Super Swellex
Max. effective length = 13.0/10.0 = 1.3 m

<u>F.S = 30.6 / 35.3 = 0.9 (N.G)</u></td>
</tr>
<tr>
<td>2.4m
Reg.
Swellex
+
3.6m
Super
Swellex</td>
<td>
</td>
<td><u>Dead Weight of Wedge Block</u>
(7.0m × 3.15m) / 2 × 3.2 t/m^3 = 35.3 t

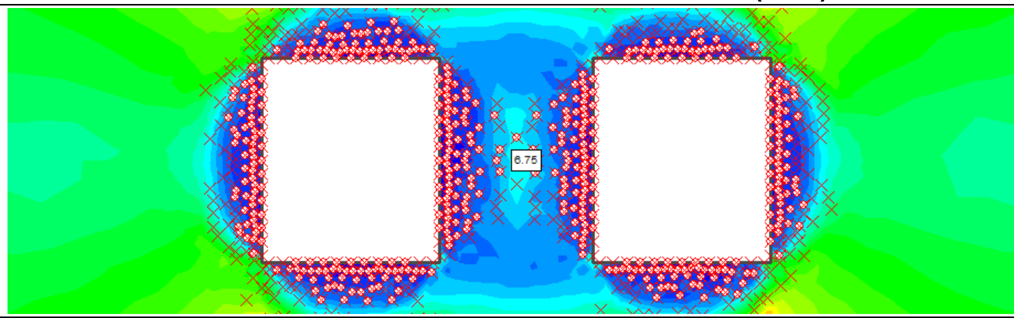
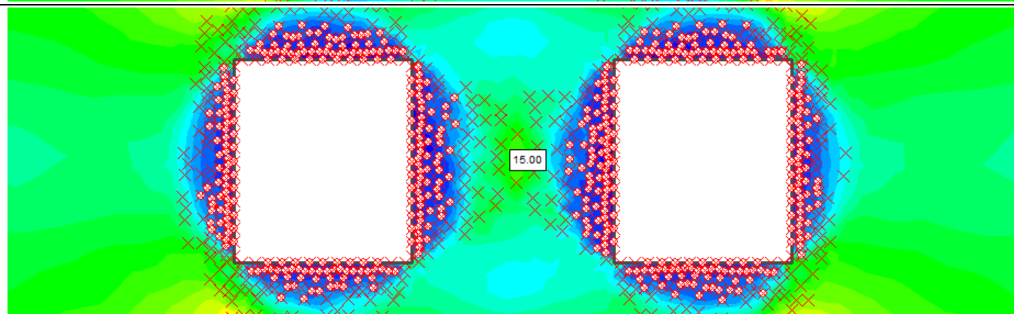
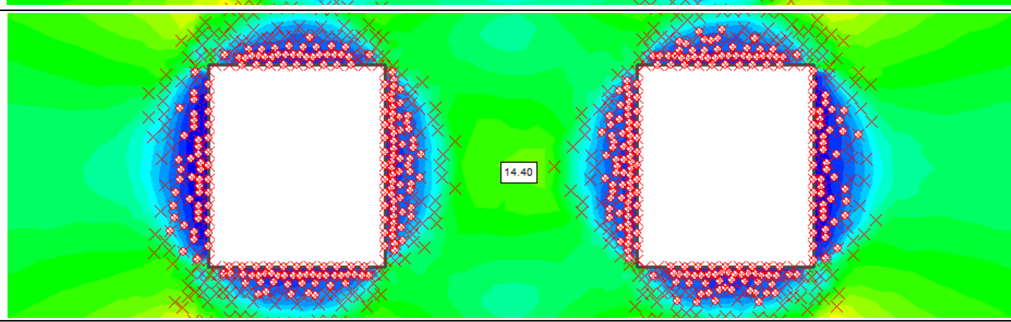
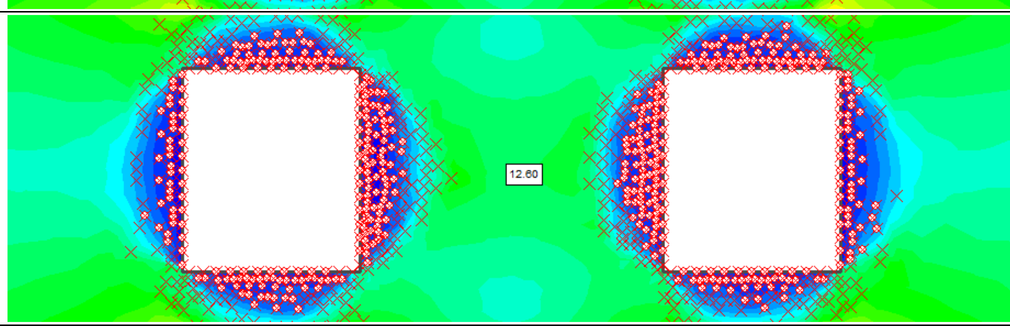
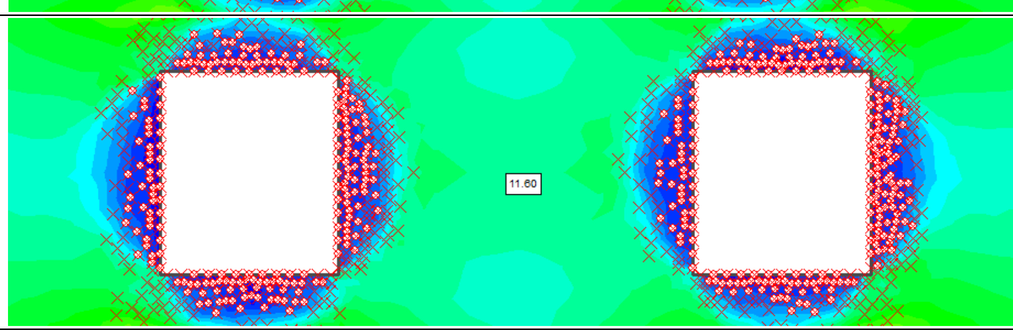
<u>Support Capacity</u>
(10×2x0.5 + 0.82×7.0x2) / 1.2 +
(13x2 + 1.26x10x2) / 1.8 = 39.7 t
Reg. Swellex
Max. effective length = 10.0/7.0 = 1.4 m
Super Swellex
Max. effective length = 13.0/10.0 = 1.3 m

<u>F.S = 46.3 / 35.3 = 1.3 (O.K)</u></td>
</tr>
</table>

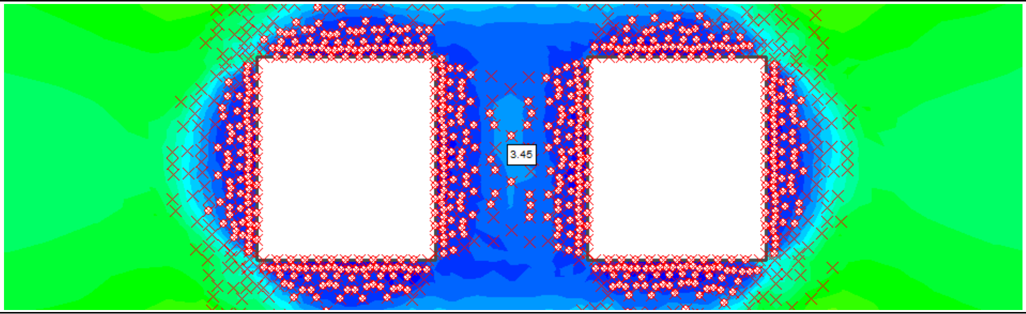
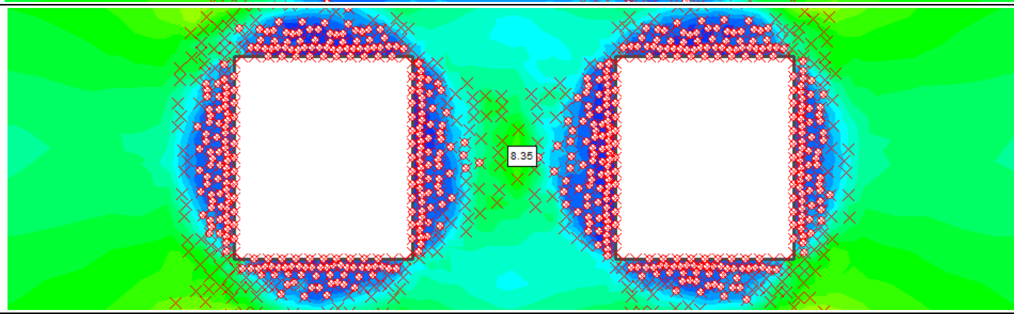
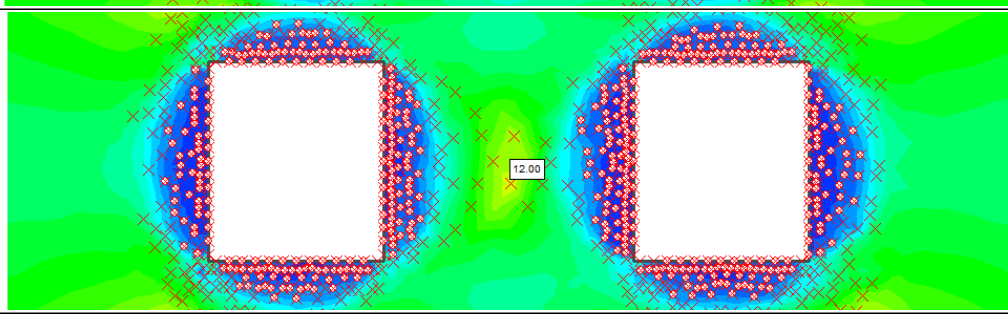
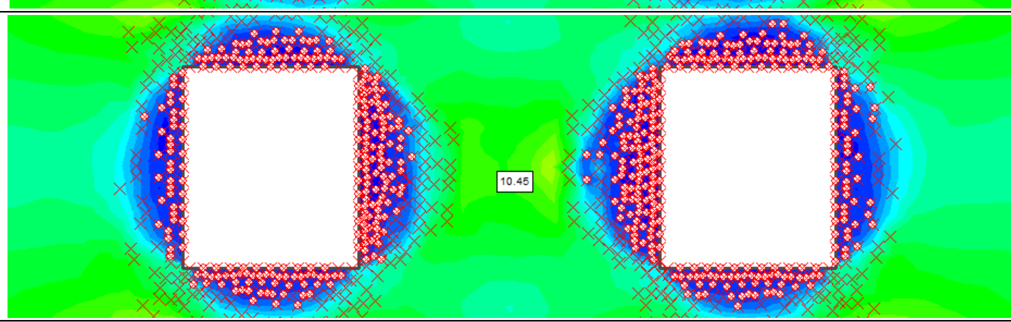
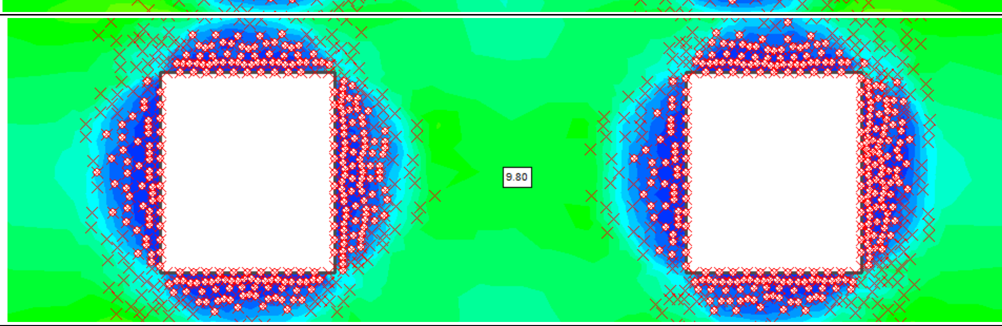
APPENDIX – E.

NUMERICAL CALCULATION (Pillar Stability)

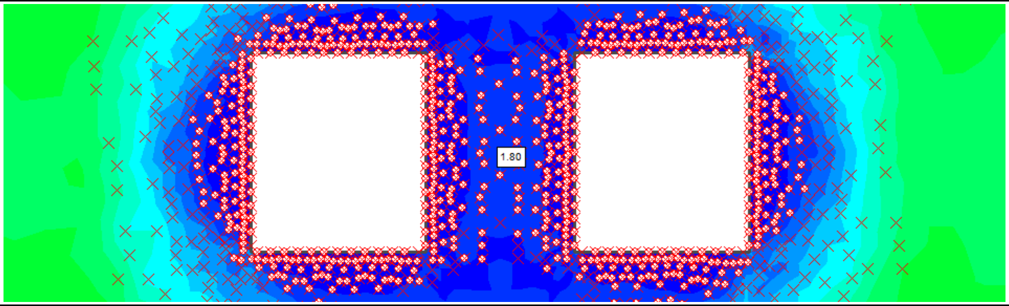
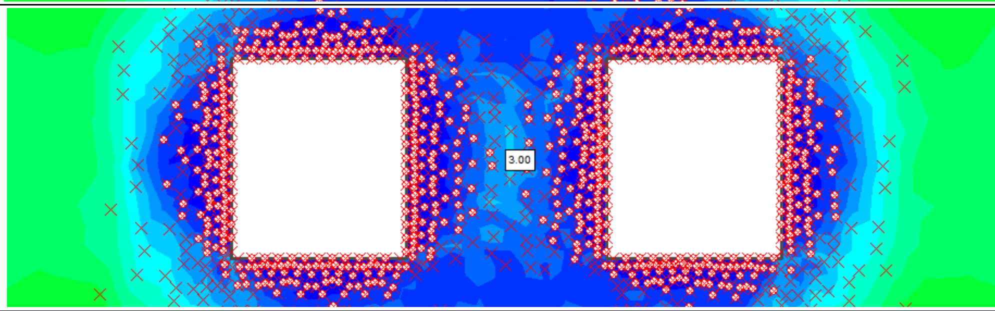
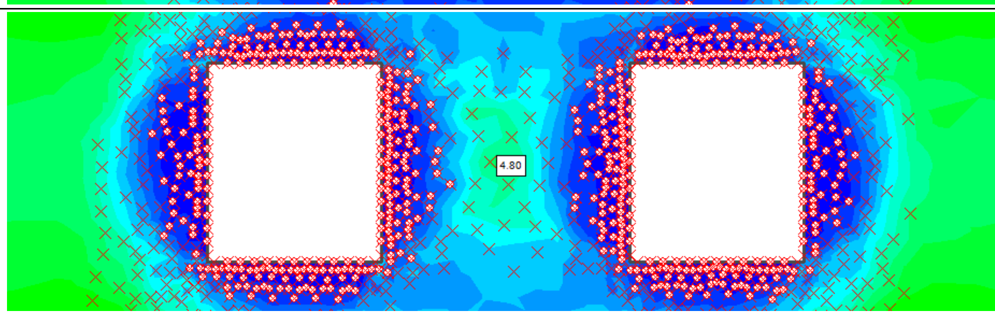
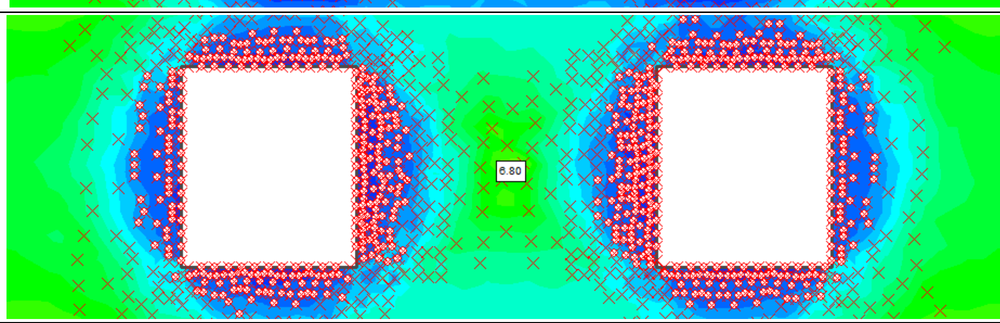
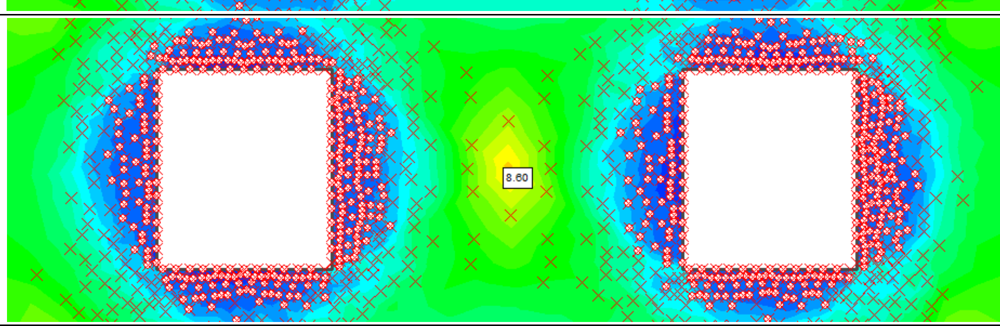
Rib Pillar between 3.5mW x 4.0mH Drift (45 < GSI < 60)

Pilar Width	Yielded Elements and Maximum Stress (σ_{max})
3.0m	
4.0m	
5.0m	
6.0m	
7.0m	

Rib Pillar between 3.5mW x 4.0mH Drift (30 < GSI < 45)

Pilar Width	**Yielded Elements and Maximum Stress (σ_{max})**
3.0m	
4.0m	
5.0m	
6.0m	
7.0m	

Rib Pillar between 3.5mW x 4.0mH Drift (20 < GSI < 30)

Pilar Width	Yielded Elements and Maximum Stress (σ_{max})
3.0m	
4.0m	
5.0m	
6.0m	
7.0m	

APPENDIX – F.

STABILITY CHART

FOR

STOPE DIMENSION

Stability Graph Analysis - 15m High Stope Wall in Good Ground

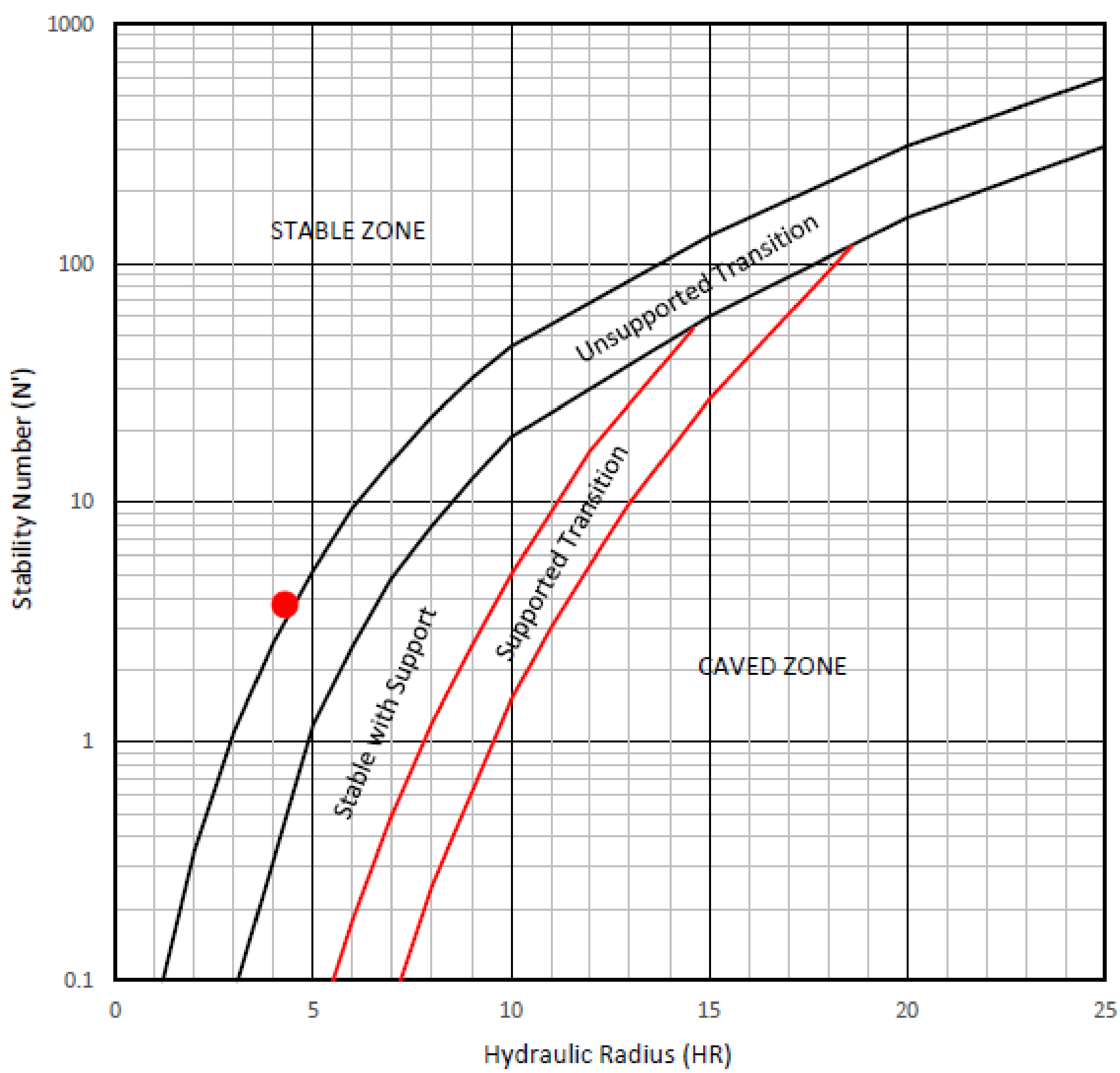
Stope Dimension : Length = 20 m Height = 15 m

Number of Sub-level = 0 levels

Stability Number, N	=	Q'	×	A	×	B	×	C		
	=	5.0	×	0.5	×	0.30	×	5	=	**3.75**

Hydraulic Radius = (W × H) / (2W + 2H) = 300 / 70 = **4.29**

Stability Graph



Stability Graph Analysis - 15m High Stope Wall in Fair Ground

Stope Dimension : Length = 10 m Height = 15 m

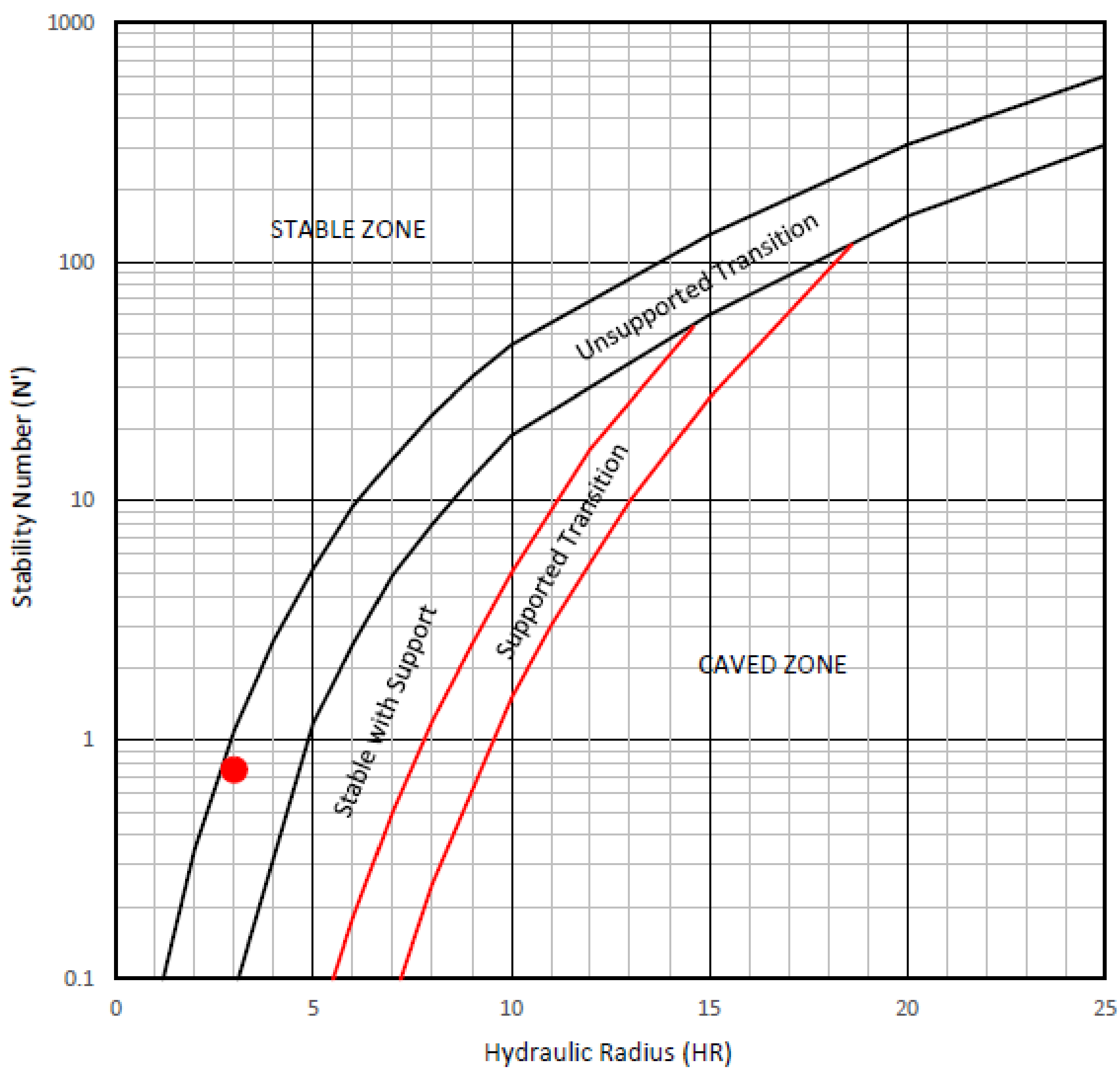
Number of Sub-level = 0 levels

Stability Number, N = Q' × A × B × C

= 1.0 × 0.5 × 0.30 × 5 = **0.75**

Hydraulic Radius = (W × H) / (2W + 2H) = 150 / 50 = **3.00**

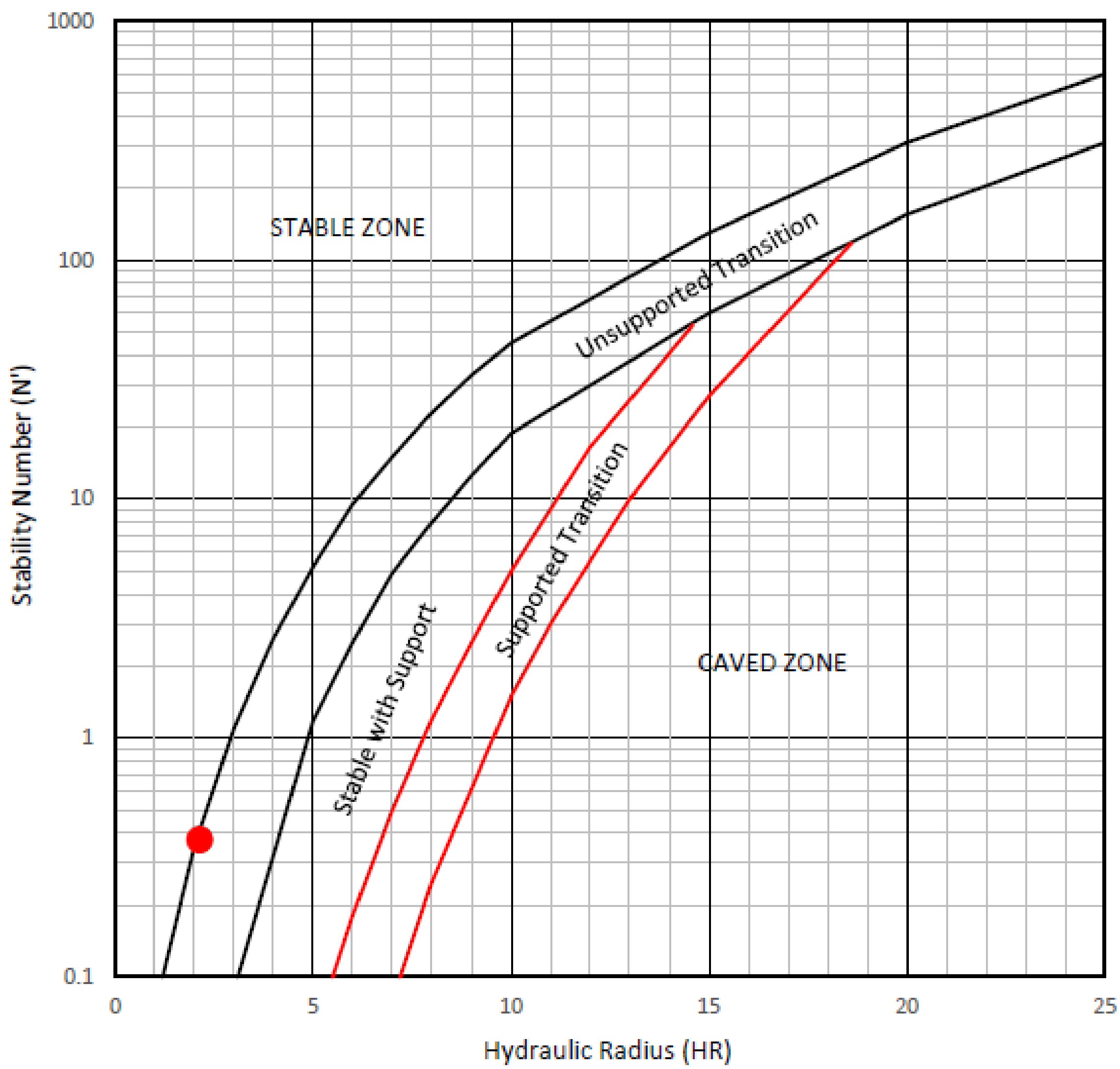
Stability Graph



Stability Graph Analysis - 15m High Stope Wall in Poor Ground

Stope Dimension :		Length =	6	m		Height =	15	m					
		Number of Sub-level =			0	levels							
Stability Number, N	=		Q'	×	A	×	B	×	C				
		=	0.5	×	0.5	×	0.30	×	5	=	0.38		
Hydraulic Radius =	(W × H) / (2W + 2H)	=	90	/	42	=	2.14						

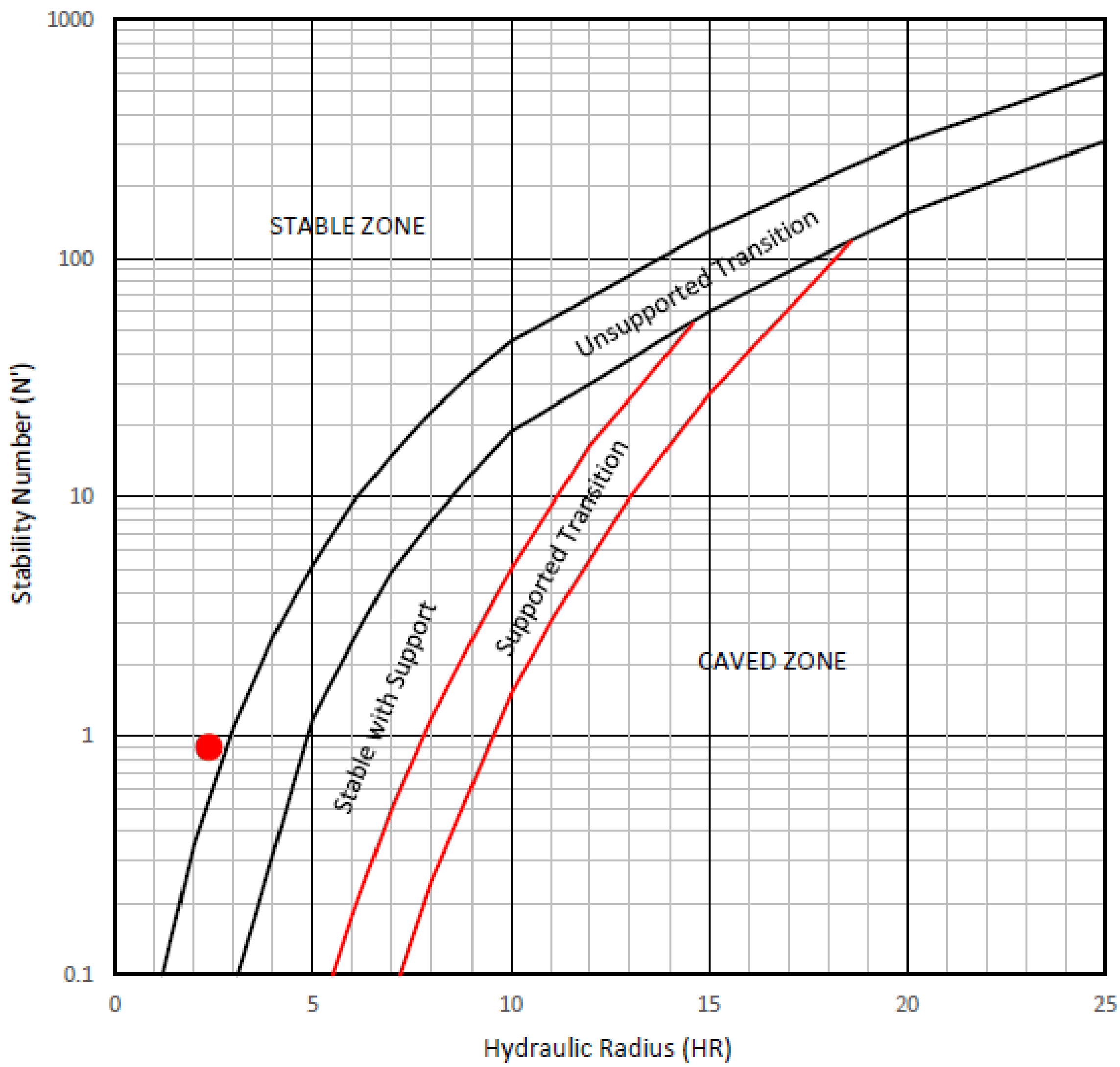
Stability Graph



Stability Graph Analysis - 5m Wide Stope Back in Good Ground

Stope Dimension :		Length =	100	m		Height =	5	m			
		Number of Sub-level =		0	levels						
Stability Number, N	=	Q'	×	A	×	B	×	C			
	=	5.0	×	0.2	×	0.30	×	3	=	**0.90**	
Hydraulic Radius =	(W × H) / (2W + 2H)	=	500	/	210	=	**2.38**				

Stability Graph



Stability Graph Analysis - 5m Wide Stope Back in Fair Ground

Stope Dimension : Length = 100 m Height = 5 m

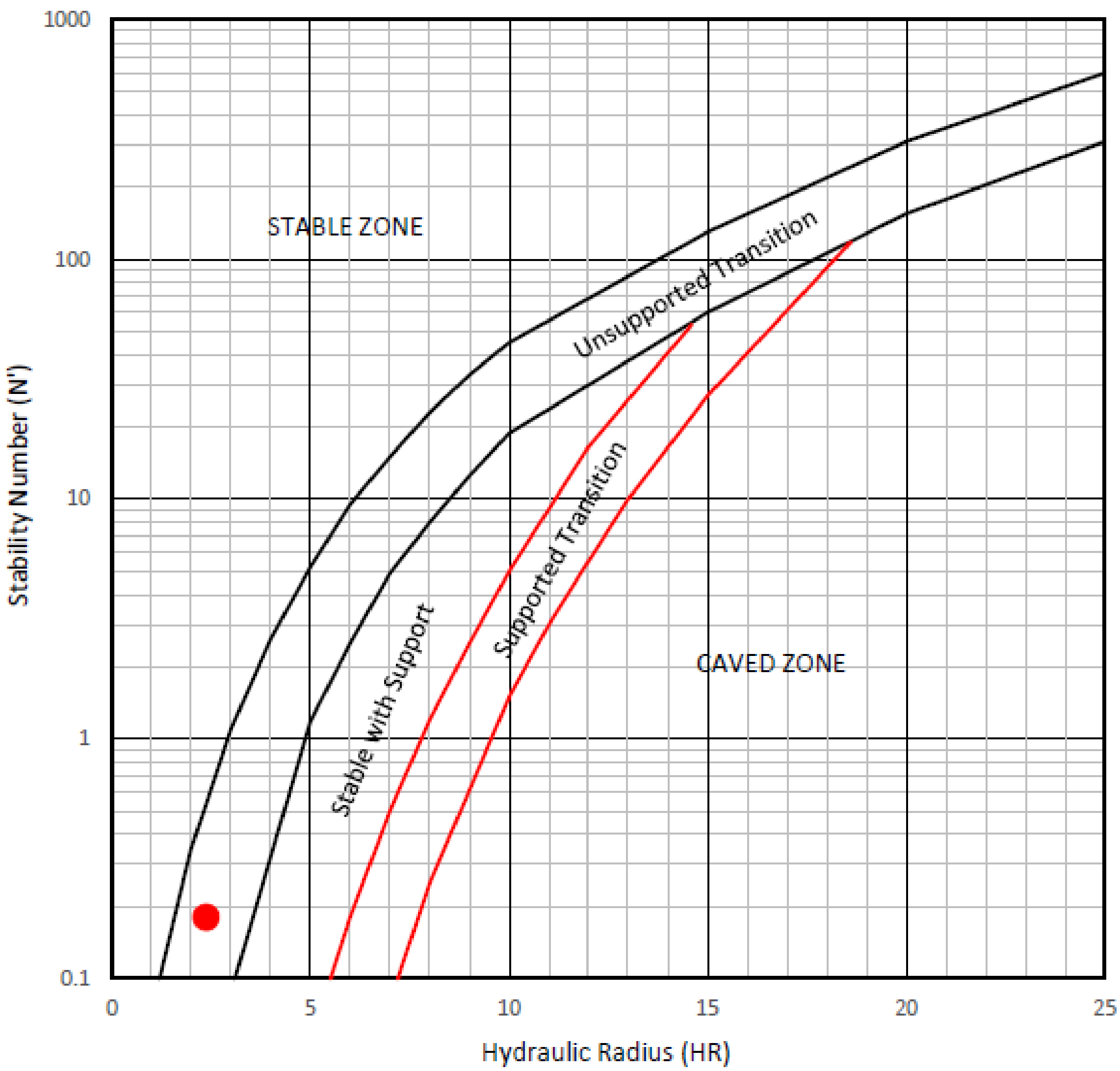
Number of Sub-level = 0 levels

Stability Number, N = Q' × A × B × C

= 1.0 × 0.2 × 0.30 × 3 = **0.18**

Hydraulic Radius = (W × H) / (2W + 2H) = 500 / 210 = **2.38**

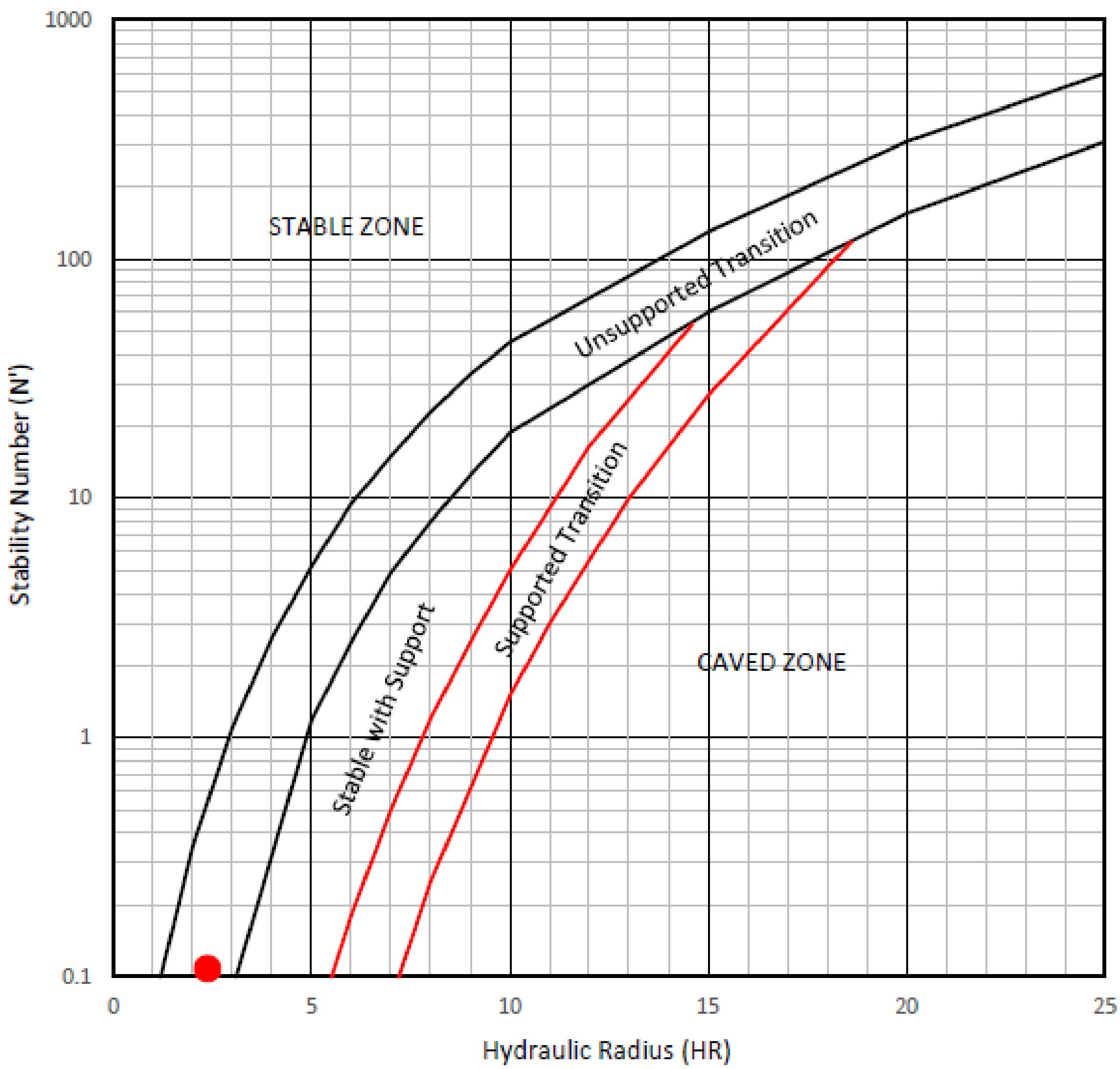
Stability Graph



Stability Graph Analysis - 5m Wide Stope Back in Poor Ground

Stope Dimension :		Length =	100	m		Height =	5	m			
		Number of Sub-level =		0	levels						
Stability Number, N	=	Q'	×	A	×	B	×	C			
	=	0.6	×	0.2	×	0.30	×	3	=	**0.11**	
Hydraulic Radius =	(W × H) / (2W + 2H)	=	500	/	210	=	**2.38**				

Stability Graph



APPENDIX – G.

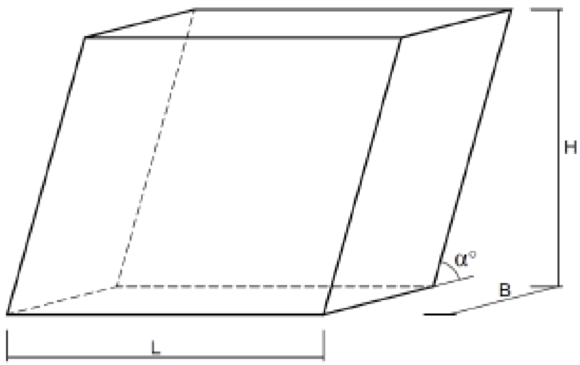
PASTE STRENGTH

For

FILL FACE EXPOSURE

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15	m
L	(Length of Stope Skrike)	:	30	m
B	(Width of Backfill)	:	4	m
α	(angle of stope)	:	70	°

Material Properties of Backfill

γ	(unit weight of fill material)	:	25	kN/m^3
ϕ	(internal friction angle)	:	20	°
K	(Coefficient of fill pressure)	:	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 81.65 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 174.91 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)} \right\} = 73.45 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 157.34 \text{ kPa}$$

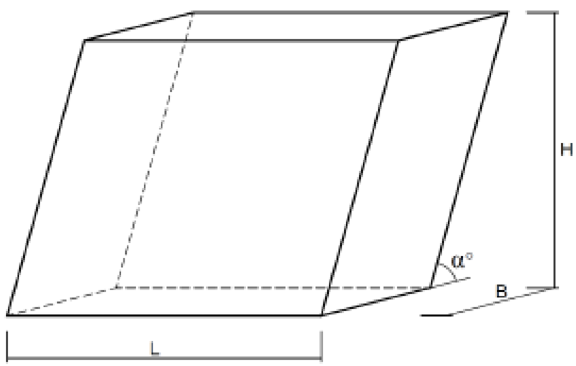
Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin45°} \right\} = 73.58 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 157.63 \text{ kPa}$$

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15 m
L	(Length of Stope Skrike)	:	30 m
B	(Width of Backfill)	:	5 m
α	(angle of stope)	:	70 °

Material Properties of Backfill

γ	(unit weight of fill material) :	25	kN/m^3
φ	(internal friction angle) :	20	°
K	(Coefficient of fill pressure) :	0.79	$(K = 1 / (1 + 2\tan^2 \phi))$

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 101.71 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 217.89 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)} \right\} = 86.09 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 184.43 \text{ kPa}$$

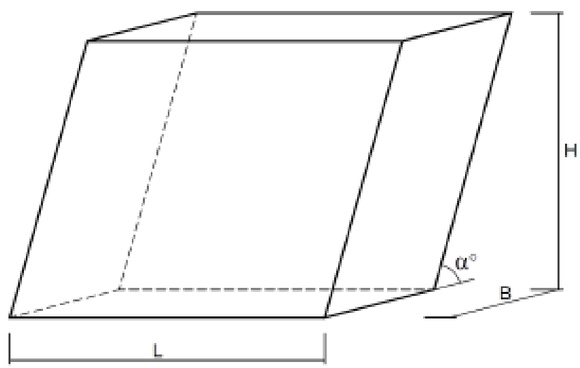
Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin 45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin 45°} \right\} = 86.28 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 184.82 \text{ kPa}$$

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15	m
L	(Length of Stope Skrike)	:	30	m
B	(Width of Backfill)	:	6	m
α	(angle of stope)	:	70	°

Material Properties of Backfill

γ	(unit weight of fill material)	:	25	kN/m^3
ϕ	(internal friction angle)	:	20	°
K	(Coefficient of fill pressure)	:	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 121.46 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 260.20 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)} \right\} = 97.26 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 208.34 \text{ kPa}$$

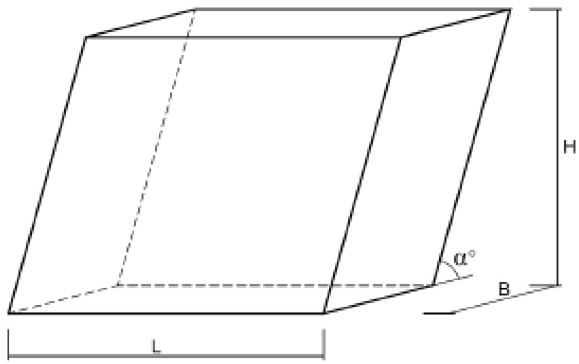
Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin 45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin 45°} \right\} = 97.49 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 208.85 \text{ kPa}$$

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15	m
L	(Length of Stope Skrike)	:	30	m
B	(Width of Backfill)	:	8	m
α	(angle of stope)	:	70	°

Material Properties of Backfill

γ	(unit weight of fill material)	:	25	kN/m^3
ϕ	(internal friction angle)	:	20	°
K	(Coefficient of fill pressure)	:	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS$_{design}$ for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 159.91 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \;/\; (1 - \sin\phi) \cdot FS = 342.56 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45^\circ+\phi/2)] \cdot \sin(45^\circ+\phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45^\circ+\phi/2)] \cdot \sin(45^\circ+\phi/2)} \right\} = 116.07 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \;/\; (1 - \sin\phi) \cdot FS = 248.65 \text{ kPa}$$

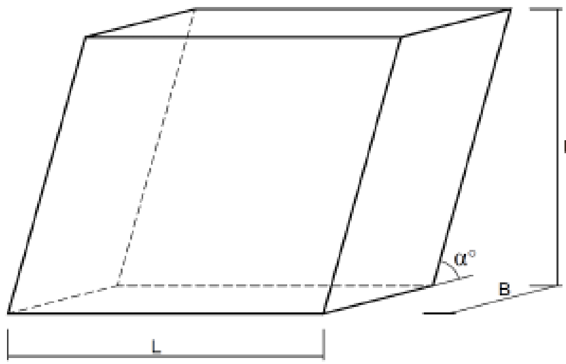
Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin 45^\circ}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin 45^\circ} \right\} = 116.41 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi \;/\; (1 - \sin\phi) \cdot FS = 249.37 \text{ kPa}$$

Major Input Parameters

Dimension of Backfill Stope



H	(Height of Backfill)	:	15	m
L	(Length of Stope Skrike)	:	30	m
B	(Width of Backfill)	:	10	m
α	(angle of stope)	:	70	°

Material Properties of Backfill

γ	(unit weight of fill material)	:	25	kN/m^3
ϕ	(internal friction angle)	:	20	°
K	(Coefficient of fill pressure)	:	0.79	($K = 1 / (1 + 2\tan^2 \phi)$)

Factor of Safety

FS : 1.5

Required CRF UCS_{design} for Self Standing with Free Face

Narrow exposed backfill face

$$2C_{design} = \left\{ \frac{[(1.25 \cdot B/(2 \cdot K\tan\phi)) \cdot \gamma \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]}{[\cos\phi/(1-\sin\phi)+1.25/(2 \cdot K \cdot \tan\phi) \cdot [1-\exp(-2 \cdot H \cdot K \cdot \tan\phi/B)]]} \right\} = 196.89 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 421.79 \text{ kPa}$$

Exposed frictional fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2) \cdot \tan(45°+\phi/2)] \cdot \sin(45°+\phi/2)} \right\} = 131.31 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 281.30 \text{ kPa}$$

Exposed frictionless fill face

$$2C_{design} = \left\{ \frac{\gamma \cdot [H-(L/2)] \cdot \sin 45°}{\cos\phi/(1-\sin\phi)+(1/B) \cdot [H-(L/2)] \cdot \sin 45°} \right\} = 131.74 \text{ kPa}$$

$$\sigma_{design} = 2 \cdot C_{design} \cdot \cos\phi / (1 - \sin\phi) \cdot FS = 282.22 \text{ kPa}$$

APPENDIX – H.

NON-CONFORMANCE

RECORD FORM

GROUND CONTROL NON-CONFORMANCE RECORD

To:
From:
CC:
Date:
Re:

Date Non-conformance Recognized: ______________________

Location of Non-conformance: ______________________
(see attached plan)

Type of Non-conformance: ______________________

Required Corrective action: ______________________
(see attached support plan)

Person(s) responsible for corrective action: ______________________

Date corrective action completed: ______________________

Signoff that corrective action has been completed:

Chief Mine Engineer	
Mine Engineer	
Mine Superintendent	
Geotechnical Engineer	
Shift Boss / Supervisor	