Ground Support 2023 – J Wesseloo (ed.)

doi:10.36487/ACG_repo/2325_38

Evolution of ground support designs for blasthole stoping in kimberlite at Diavik Diamond Mine

JM Burns *Rio Tinto, Canada*

Abstract

Diavik Diamond Mine has used blasthole stoping (BHS) for the extraction of ore from the A154N orebody since 2010. As the only diamond mine globally to use this mining method for kimberlite extraction, there are several unique challenges that Diavik had to overcome with regards to ground support. Geotechnical variability of the kimberlite, mine sequencing, stope sizing and production output requirements have led to the development of a variety of ground support designs to overcome these challenges.

A change from a vertical walled, rectangular stoping geometry to a geometry requiring partial undercutting of the stope crown led to a transformation in the overcut ground support requirements. New support elements including, pre-tensioned cable bolts and resin rebar, have been integrated into the designs to account for the change in stope geometry.

Concurrently, higher-capacity inflatable friction bolt support has been required to account for the geotechnical variability of the kimberlite and the transition from primary stoping to secondary stoping.

This paper describes the various ground support designs at Diavik Diamond Mine used for BHS in kimberlite and how these designs have evolved over the life of the mine in response to changing ground conditions and production requirements.

Keywords: *open stoping, ground support, cable bolts, kimberlite, blasthole stoping*

1 Introduction

Diavik Diamond Mine is located on the east island of Lac de Gras in the Northwest Territories, Canada, approximately 350 km northeast of the city of Yellowknife. The kimberlite pipes mined at Diavik were originally under the waters of Lac de Gras and highly engineered water-retaining dykes were constructed to allow access to the kimberlite. Mining operations began in 2003, with the initial mining of three orebodies; A154N, A154S and A418. The initial mining was undertaken via two open pits. During the mining of the open pits, construction of an underground mine commenced in 2005 to ensure the continuation of mining of the three orebodies. Underground production began in 2010, and by 2012, open pit mining in the two pits was completed, transitioning Diavik to a fully underground mine. Diavik remained a fully underground mining operation until 2018 when a fourth kimberlite pipe began open pit production. Mining at Diavik is planned to complete in 2026.

During initial studies for the underground mining of A154N, A154S and A418, blasthole open stoping (BHS) and underhand cut-and-fill were evaluated as the primary options to be used. However, during the open pit mining and early underground construction, the methods were re-evaluated, and BHS was retained for the A154N orebody while sublevel retreat (SLR) was selected for the A418 and A154S orebodies (Lewis et al. 2018).

The primary constraint on the decision to use BHS for A154N was the proximity of the orebody to the north dyke where the orebody is within approximately 200 m laterally of the dyke. This was deemed too close, and if SLR was to be used, it could induce instability within the dyke. For this reason, transverse blasthole open stoping with backfill was chosen. It was determined that by mining the A154N using this method it would significantly reduce the risk to the dyke structure.

2 Kimberlite stoping at Diavik

The A154N orebody is a vertical, cylindrical-shaped volcanic pipe varying in diameter from 50–150 m made up of multiple kimberlite units with varying geotechnical characteristics (Figure 1). It is mined using transverse open stopes. The orebody has been broken up into four major stoping blocks (A, B, C and D) separated by in situ kimberlite sill pillars. Each block is between 100 and 125 m in height and contains between five and six development levels, resulting in four to five stoping levels, respectively. Each level, including the sill pillars, has a 25 m vertical separation, sill-to-sill and contains up to 21 vertical slices or stoping lines through a primary/secondary stoping sequence.

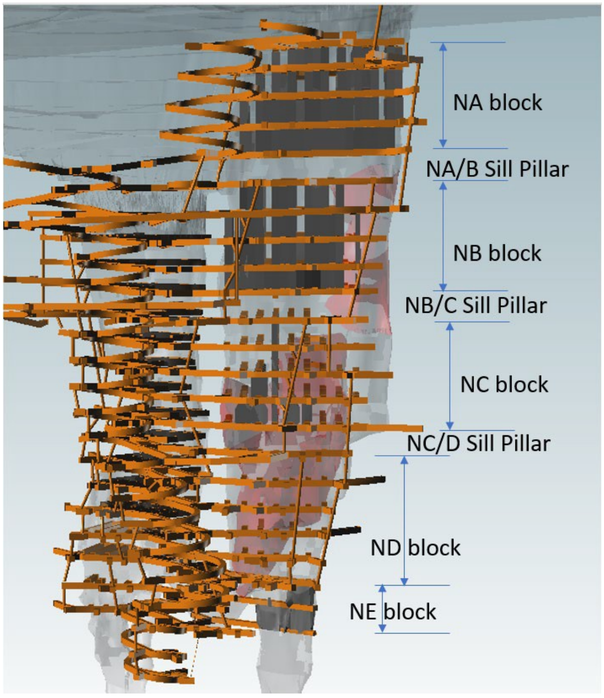

Figure 1 Isometric view of A154N orebody

2.1 A and B block stoping

A and B blocks are situated in the upper part of the A154N kimberlite pipe and are from levels 9275–9175 and 9150–9050, respectively. The development and stoping designs for these two upper blocks have been divided into 7.5 m wide stoping lines and alternate in a primary/secondary sequence. This has allowed for 7.5 m wide vertical stopes between levels and result in a 30 m high stope (including the overcut and undercut development excavations). The primary stopes, which are now completed, were backfilled with cemented rockfill (CRF) to allow for secondary stopes to be developed between them. Figure 2 depicts an idealised illustration of this sequence. Figure 3 shows an in-field photo and subsequent cavity monitoring system (CMS) of an open 7.5 m wide stope at Diavik.

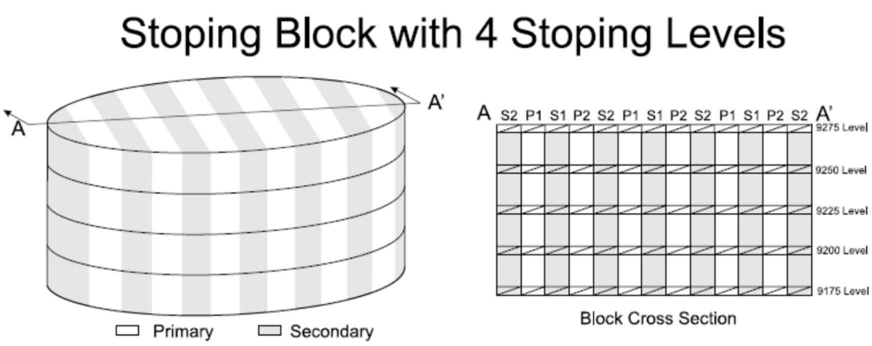

Figure 2 Stoping block example (Lewis et al. 2017)

(a) (b)

Figure 3 (a) An open B block secondary stope; (b) CMS (Archibald 2023)

2.1.1 A and B block stoping sequencing

In the initial mine design for A154N, stope sequencing recommendations for A and B block were laid out as:

- A level-by-level sequence from centre out for both primary and secondary stopes.
- Secondary mining on the lowest level starts almost as soon as possible, upon completion of primary stopes on the level above.
- Active production mining on up to three consecutive levels.
- No constraint was placed on sequencing and if one stope line had to be finished before mining could commence on a consecutive line.
- No minimum number of pillars between open stopes recommended.

This initial idealised sequence would have resulted in a relatively wide and low mining front advancing from centre out with only three active levels per block. In practice, it would have led to a limited number of faces at the start of the sequence, congestion if actively mining on more than one stope line at a time and narrow pillars (7.5 m, nominally) between stope lines, which would become problematic, especially in secondary mining with CRF pillars.

In order to address these problems and avoid thin pillars between active and open development on adjacent stope lines, the following sequencing rules and practices were established:

- Active stope lines were to be spaced minimum 30 m centre-to-centre, leading to primary 1 (P1) – primary 2 (P2) and secondary 1 (S1) – secondary 2 (S2) principle. On every level, all P1 or S1 stope lines were to be mined first, before advancing to P2 or S2 lines. This resulted in minimum 22.5 m (nominal) pillars between open stopes (Figure 2, block cross-section).
- When actively mining on adjacent P1(S1) or P2(S2) lines, a minimum distance of 30 m was to be maintained between closest open stopes.
- No limit was placed on the number of active levels.
- No requirement as to starting or to maintaining any particular shape of the mining front.

These sequencing rules and practices ensure a 1:1 ratio between open stope height and centre-to-centre distance between two adjacent stope lines. To this date, maintaining this ratio has proved effective in preventing stope wall instabilities and minimising the chance of an active stope caving into another active stope.

2.2 C and D block stoping

C and D blocks are situated in the lower part of the A154N kimberlite orebody and are from levels 9025–8925 and 8900–8775, respectively. These two stoping blocks are relatively new in the mine life with C block only coming into full production in 2018 and D block in 2021. With the increase in production output requirements, the decision was made to change the stoping geometry from the 7.5 m wide vertical walled stope designs implemented in A and B blocks to a 15 m wide design (Figure 4). This diamond-shape stope geometry, or a variation of this geometry, is commonly used in hard rock stoping due to what can effectively be drilled and, therefore, effectively blasted, whilst trying to recover the reserve in the most effective manner whilst maintaining stability. Due to Diavik currently being the only diamond mine to use blasthole open stoping as a mining method (Jakubec 2020), no directly comparable analogy was available. Experience and information gathered from other hard rock operations had to be utilised in conjunction with trialling of designs in the initial stages.

(a) (b)

Figure 4 (a) An open C block secondary stope; (b) LiDAR scan (Archibald 2023)

2.2.1 C and D block stoping sequencing

Two overarching mining sequences were proposed for C and D blocks based on learnings from the A and B block mining experience. Pillarless single chevron and pillarless double chevron fronts were proposed and extensive numerical modelling undertaken on both options to assess the advantages and disadvantages. Double chevron was considered because of its operational advantage of providing more headings and higher production early in the sequence.

A single chevron mining front is geotechnically superior; however, in practice, it would be difficult to apply in its pure form because of the limited number of faces in the early stages of mining and some legacy issues, such as development, already being in place. The C and D block sequences currently being used is a compromise between single chevron and the double chevron.

At the stope-extraction scale, the change in stope sizing has resulted in the removal of the P2 and S2 stope lines and yields a P1–S1 sequencing profile for C and D blocks to achieve the overall chevron advancing front. The 30 m offset distances between open stopes still applies and can periodically place interaction constraints on stoping; especially in the deeper parts of the kimberlite pipe where the orebody footprint decreases.

3 Ground support for stoping

Ground support for blasthole stoping at Diavik has undergone many iterations to account for the varying conditions that occur within the orebodies due to geotechnical variability of the kimberlite, mine sequencing, stope sizing and production output requirements. The initial ground support designs outlined in the original mine design document and ground control management plans are vastly different to what is used today. The current ground support requirements have been developed using a combination of:

- Empirical guidelines using estimates of the anticipated range of rock mass quality and the excavation support ratio.
- Kinematic stability analysis.
- Practical experience gained at Diavik and other Canadian hard rock mines.
- Practical operational considerations, including readily available ground support elements and equipment, need for standardisation over the range of ground conditions and full mining cycle of ore-extraction drifts (e.g. need for brow support in future drawpoints).
- Back-analyses of falls of ground at Diavik.

3.1 Ground support for A and B bock stoping

Initially, ground support for primary stoping in A and B blocks utilised the support patterns that were used for the development headings in these areas. These ground support requirements were largely based on an empirical ground support design methodology adapted from Barton (Grimstad & Barton 1993) using the relevant support chart presented in that paper which relates opening size and support requirements to estimate rock mass quality (Q'). Limited numerical modelling was also undertaken at this initial stage with inputs coming from site investigations and the in-field observations of performance of the minimal open ground available at the time. No additional secondary support was installed specifically for the stoping cycle.

The support pattern remained relatively unchanged through the primary stoping of A and B blocks. With secondary development and stoping beginning in late 2014, the original ground support design was modified and called for the CRF walls of the development to no longer require the pattern bolting and mesh outlined in the primary development standard but instead be shotcreted and spot bolted in-cycle (Lewis et al. 2017). Pattern bolting of the walls was not considered at this point because it was initially believed that the stresses from the weight of the remaining kimberlite would transfer to the stiffer waste rock resulting in minimal loading on the CRF and the CRF would be self-supporting. The intent of the shotcrete was to prevent the CRF from breaking apart and unravelling, with the spot bolting added to tie the shotcrete to the CRF wall. Table 1 outlines the initial support requirements for these parts of the mine.

Table 1 Initial minimum ground support requirements for secondary blasthole stope overcut and undercut drifts up to 7.5 m width

Rock quality	Bolt length (m)	Bolt spacing (m)	Mesh	Shotcrete (mm)	Comments	Typical round length for drill and blast (m)
BHS support categories – primary support for secondary overcut and undercut drifts.						
Good rock	2.4 m Swellex® Pm12	1.2 × 1.2 in the back only (square pattern)	Back only	50 mm walls and shoulders only	Shotcrete on walls to extend down to the sill. Shotcrete on shoulders to overlap with outside row of bolts in the back.	4
Fair rock	2.4 m Swellex® Pm12	1.2 × 1.2 in the back only (dice-5 pattern)	Back only	50 mm walls and back	Shotcrete to extend to the sill.	3
Poor rock	2.4 m Swellex® Pm12	1.2 × 1.2 in the back only (dice-5 pattern)	Back only	50 mm walls 100 mm back	Shotcrete to extend to the sill. Sill conditions may require placement of crushed rock for trafficability. Spiling may be required to maintain a consistent profile.	2
Secondary support for secondary overcut and undercut drifts.						
Secondary back support				• Primary bolt spacing and shotcrete requirements are dictated by the support category for the lowest quality rock in the drift. • Secondary bolting shall consist of 3.6 m long Swellex® Pm24 (Super Swellex®) bolts in the back. Secondary bolt spacing is 1.8 × 1.8 m. • For 7.5 m wide drifts, secondary support must be installed to within 4 m of the working face. • Intersection and excavations wider than 7.5 m require secondary support designed case-by-case.		

This design worked well during initial secondary stoping. However, as secondary stoping matured and stress redistributions occurred, the design was no longer effective. The long-term loading and stress redistribution on the CRF resulted in partial deformation of the primary CRF pillars. This deformation, in conjunction with the stiffness disparity between the CRF and shotcrete surface support, resulted in the shotcrete and CRF separating causing slabbing of the shotcrete and a loss of confinement on the CRF walls that the spot bolting was not able to overcome. The resulting rehabilitation programs drove the changes in the primary support system that is currently being used.

3.1.1 Current ground conditions for A and B block stoping

Currently, A and B blocks are a mixture of backfill filled primary stopelines and kimberlite pillars. The kimberlite within this part of the mine can generally be described as a strong, moderately jointed rock mass with joint spacing greater than 0.3 m. Slight alteration with non-softening joint infill is also observable.

The kimberlite in this section of the pipe also has a low clay/mudstone content and has a low susceptibility to slaking. Typical Q values are between four and 40. However, with the extent of the extraction in both A and B blocks, the stability in the development and stoping in these areas are now primarily controlled by the backfill quality. The CRF parameters were initially developed based on numerical and empirical analyses, and experience at other similar operations and only considered short-term single stope opening stability. The CRF has a design target strength of 1.5 MPa and can be described as a highly jointed fill mass. The CRF design has undergone minimal changes in its design parameters and essentially remains the same since its inception.

3.1.2 Current ground support for A and B block stoping

With the current and remaining ground conditions in A and B blocks, the ground support has had to change to accommodate the stress redistribution, squeezing and variable CRF quality exhibited in the development through backfilled areas and stope walls. As the stope dimensions and mining plan have not changed from the initial plan laid out at the beginning, original development support patterns continue to be used as the stoping support patterns.

Swellex® rockbolts continue to be utilised extensively for the primary support in A and B blocks as they have proven to be effective in CRF. The irregularity of the holes drilled in CRF appears to have made the bond between the bolts and the CRF stronger when compared to other types of ground support trialled. As the Swellex® inflates, it is able to conform to the irregular shape of the drill holes during installation. The smaller pre-inflation diameter also eases installation of the rockbolt in the irregular CRF.

The significant changes from the initial secondary stoping support design have been that of pattern bolting, meshing of the walls and removal of the shotcrete from the walls. There are isolated instances where shotcrete is still applied to the CRF walls. However, this is undertaken on a case-by-case basis. Table 2 outlines the current support requirements for these parts of the mine. As a result of this updated support design, the site saw greatly reduced rehabilitation requirements pre and post stoping, and the design has proved capable of maintaining the pillars while resisting the squeezing ground to a large extent.

Table 2 Current minimum ground support requirements for rectangular flat back drifts up to 7.5 m wide in kimberlite – 154N

<table>
<tr><th>Rock quality</th><th>Bolt length back and walls (m)</th><th>Bolt spacing (m)</th><th>Mesh</th><th>Shotcrete (mm)</th><th>Comments</th><th>Typical round length for drill and blast (m)</th></tr>
<tr><td colspan="7">BHS support categories – primary support for secondary overcut and undercut drifts.</td></tr>
<tr><td>Good rock</td><td>2.4 m Swellex® Pm12</td><td>1.2 × 1.2 (square pattern)</td><td>Yes</td><td rowspan="2">50 mm to back and shoulders, extended by 1 m down the walls.</td><td>Bolts and mesh to be installed to within 1.5 m of sill.</td><td>4</td></tr>
<tr><td>Fair rock</td><td>2.4 m Swellex® Pm12</td><td>1.2 × 1.2 (square pattern)</td><td>Yes</td><td>Bolts and mesh to be installed as close as possible to sill (<0.5 m).</td><td>2–3</td></tr>
<tr><td colspan="7">Secondary support for secondary overcut and undercut drifts.
• First pass bolt spacing and shotcrete requirements are dictated by the support category for the lowest quality rock in the round.
• Second pass bolting shall consist of 3.6 m Pm24 Swellex® bolts in the back. Secondary bolt spacing is 1.8 × 1.8 m.
• Secondary support must be installed to within 4 m of the working face.
• Lower wall screen/bolts allowed to lag face between 16 and 36 m.</td></tr>
</table>

3.1.2.1 Support for stopes with kimberlite skins in A and B block

There are, however, locations within the current mining sequence in A and B blocks where the updated support design cannot be utilised predominantly due to the condition of the hosting CRF pillars. Some of the secondary development must be developed at 5 m wide along one CRF pillar rather than the standard 7.5 m wide in these mining blocks. This leaves an approximate 2.5 m kimberlite skin on one wall of the development heading resulting in a potentially undercut wall when the stope below is mined (Figure 5).

Supporting these kimberlite skins is problematic, as they become fully unconfined when production blasting occurs. The production blasting undercuts these skins, which then fail into the stope requiring extensive and time-consuming rehabilitation during the stope-filling cycle. They are also unable to be supported and pinned to the adjacent CRF, which they themselves are supporting, as the CRF is unable to support a bolting pattern and the associated weight of the skin when it becomes unconfined.

To eliminate this potential, slashing of the kimberlite skins and subsequent supporting of the wider span and CRF wall occurs immediately prior to the mining of the stope below. This reduces the exposure time and potential for loss of confinement in the development and subsequent stope wall, in turn reducing the potential for stope wall failure.

The support pattern, at this point, requires the stope overcut to be brought to the same minimum ground support requirements as if the heading had been originally excavated at the full 7.5 m width (Table 2).

When kimberlite skins are required, the removal of these skins is the preferred option as outlined above. However, this may not be possible in some instances and specific secondary stoping support will be required to reduce the potential for the skin to fail into the stope during mining. Installation of Pm24 Super Swellex® or grouted rebar, on a 1.2 × 1.2 m square pattern, embedded at least 1.2 m into competent backfill, is specified.

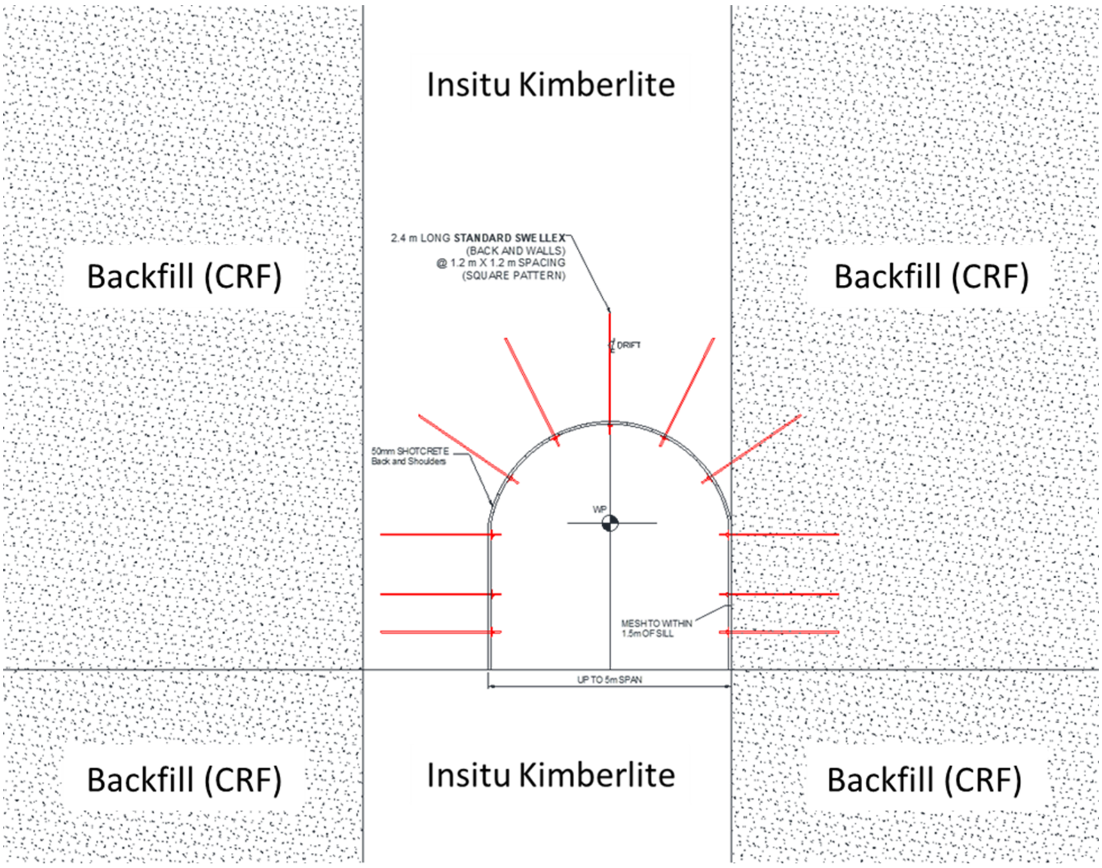

Figure 5 Cross-section through kimberlite skin development

3.2 Ground support for C and D block stoping

As described in Section 2.2, C and D block stoping is significantly different from that undertaken in A and B blocks due to the production output requirements at this stage of the mine life. The change was further driven by the poorer geotechnical characteristics of the kimberlite in the deeper parts of the mine and their effect on development stability. There were also pre-existing constraints that the stope dimensions had to adhere to, including existing development and infrastructure. These parameters meant that few options were available for stope geometry and the ground support design would be dictated by this geometry. With the change in stope geometries, the previously utilised ground support designs for stoping in A and B block were not transferable and an overall new approach had to be taken.

3.2.1 Ground conditions for C and D block stoping

C and D blocks are dominated by two kimberlite units within the orebody. Over the vertical extent of the mining blocks, these two units have great geotechnical variability which provides challenges for both the development and stoping support design.

The first of the two kimberlite rock masses within this part of the mine can generally be described as moderately strong to strong, moderately jointed to blocky, with joint spacing greater than 0.3 m. Slight alteration with non-softening joint infill is also observable. Clay/mudstone is very rare in this unit; however, it does have a high susceptibility to slaking. Q values range between 8 and 30.

The second of the two dominant kimberlite rock masses within this part of the mine can generally be described as a weak to moderately strong and massive to well stratified. Joint spacing is variable and ranges from less than 0.2 m to greater than 1.0 m. Softening and low friction mineral coatings are observable throughout the rock mass, both within any observable structure but also as thin laminar beds. Clay/mudstone is very common. Q' values range between two and 43.

Commonly used rock mass values that are used for design can be seen in Tables 3 and 4

Table 3 Representative geotechnical rock mass parameters for dominant kimberlite units in C block

Kimberlite units	RQD	Fracture spacing	J_r	J_a	$J_n^{(4)}$	J_{CR}	Intact strength (MPa)	E_i (GPa)	Poisson's ratio	m_i	Q'
Unit 1	75	1–3 m	1.5	1	9	20	59	22	0.25	11	12.5
Unit 2	90	1–3 m	1.5	1	9	20	32	16	0.23	6	15

Note: intact strength is best-fit σ_{ci} from the collected data.

Table 4 Representative geotechnical rock mass parameters for dominant kimberlite units in D block

Kimberlite units	RQD	Fracture spacing	J_r	J_a	$J_n^{(4)}$	J_{CR}	Intact strength (MPa)	E_i (GPa)	Poisson's ratio	m_i	Q'
Unit 1	75	1–3 m	2	2	9	17	59	22	0.25	11	8
Unit 2	90	>3 m	2	3	9	15	32	16	0.23	6	7

Note: intact strength is best-fit σ_{ci} from the collected data.

3.2.2 Ground support for C and D block stoping

Unlike in A and B blocks, the stoping geometry in C and D blocks required the installation of specific secondary support beyond that which is required for development. However, this secondary support must work in conjunction with the primary support to achieve the overall stability required by the stoping excavation.

Initial designs for stoping support were developed using the information presented by Hutchinson & Diederichs (1996) (Figure 6), the information presented by Wyllie & Mah (2004) on the use of reinforcement with tensioned anchors and the principle demonstrated in Lang's 1961 self-supporting bolted gravel model (Lang 1961, in Hoek et al. 2000) (Figure 6).

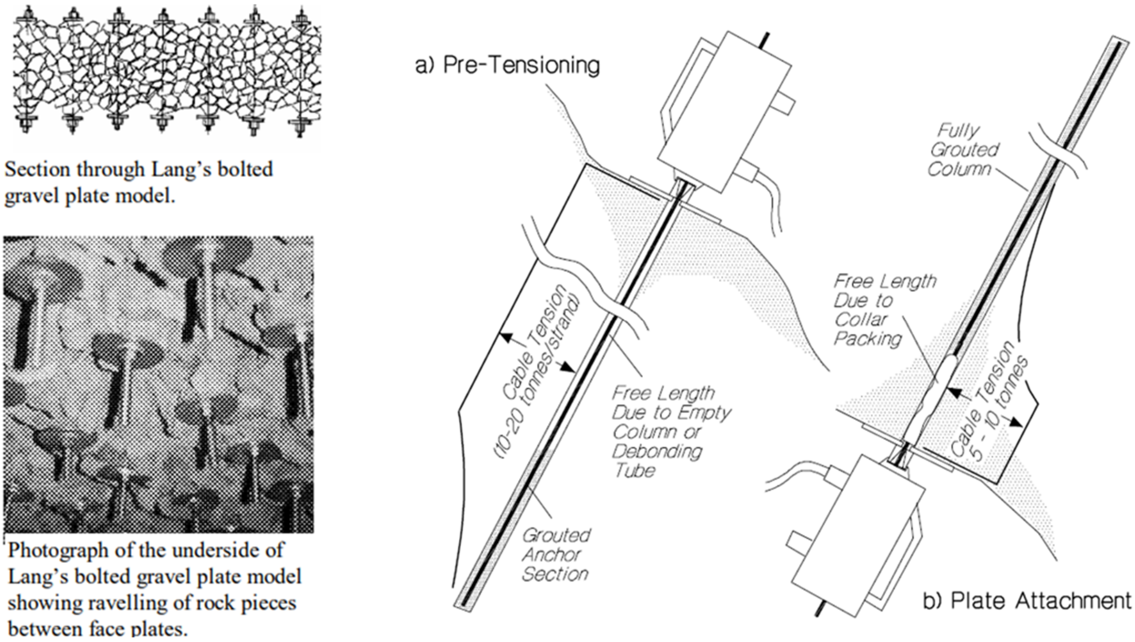

Figure 6 Lang's bolted gravel plate model (Hoek et al. 2000) and plating pre-tensioned cables (Hutchinson & Diederichs 1996)

This, in conjunction with the author's practical experience in the use of tensioned cable bolts on other job sites, lead to the development of a stoping support system using both fully grouted cables and pre-tensioned cables.

3.2.2.1 Initial cable bolt support designs for C and D block stoping

The initial design took into consideration that a 5 m section under each of the development walls would be undercut during stoping and that these walls would be in a tensile state once the stoping excavation was opened as illustrated in Figure 7.

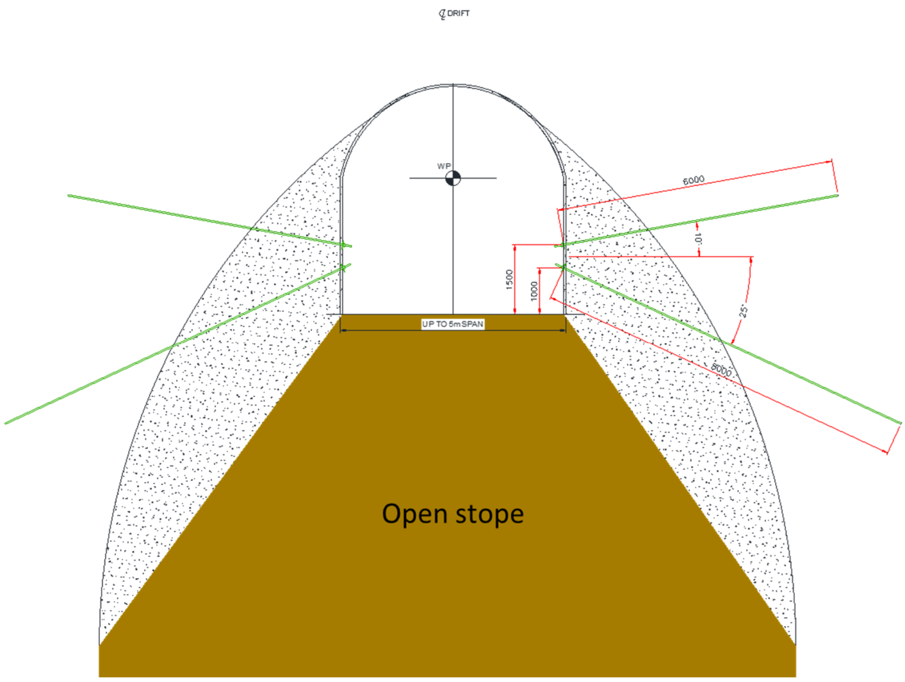

Figure 7 Initial cable bolt secondary support design

The initial design had two 8 m pre-tensioned cables installed 25° below the horizontal, two twin strand fully grouted 6 m cables installed at 10° above the horizontal and two or three fully grouted 5 m cables installed in the crown. The 8 m bolts were designed to increase the clamping forces in the undercut portion of the wall whilst the twin strand cables were designed to increase the shear capacity in this section of the wall.

The pre-tensioned cables had a 2 m bond zone outside of the predicted failure envelope, expressed as the hatched zone in Figure 7. The cables were loaded up to 150 kN, ~60% of their minimum breaking strength prior to their second stage grouting and plating.

As there was no analogue for this type of support in kimberlite, the initial design was trialled for several stopes prior to its full implementation. Once a sufficient number of stopes had been mined, a back-analysis was able to be undertaken on the performance of the ground support. The back-analysis showed that the design had mixed success as the various rock masses were encountered. In areas where the stope had overbroken in the crown, the kimberlite had unravelled around the cable bolts rather than bolts yielding. Quality assurance (QA) results also indicated installation was not to design specifications, including plating and loading of the pre-tensioned cables. These issues, along with the production output requirements from this area of the mine, drove the second iteration of the ground support design.

To combat the kimberlite unravelling around the cable bolts, the next iteration of support separated the twin strand cables in the wall to be two individual single strand cables, decreasing the bolt spacing and assisting in distributing the load over a greater area when interacting with the surface support. This change can be seen in Figure 8. In addition to this, the QA procedures were updated to include greater emphasis on plating and tensioning methods to resolve the installation issues.

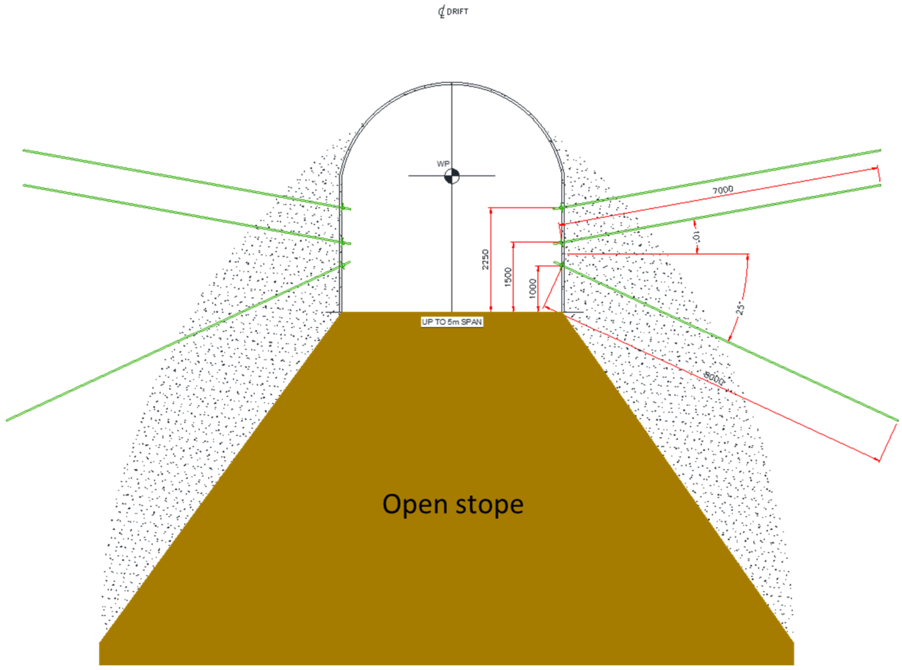

Figure 8 Second iteration of cable bolt secondary support design

This iteration of the separated cables underwent several small modifications, such as placement of the cables in the wall, dice-5 pattern, spacing, and elevation changes but ultimately remained the same. The success of this design was short lived as no increase in performance was realised. A third iteration was developed which started to utilise the tensile capacity of the cables.

The third iteration, prior to moving to the support design that is currently used, increased the angle of the uppermost wall cable to be at 25° above the horizontal. This design attempted to begin to mobilise the tensile capacity of the cable whilst still supporting the idealised failure envelope (Figure 9).

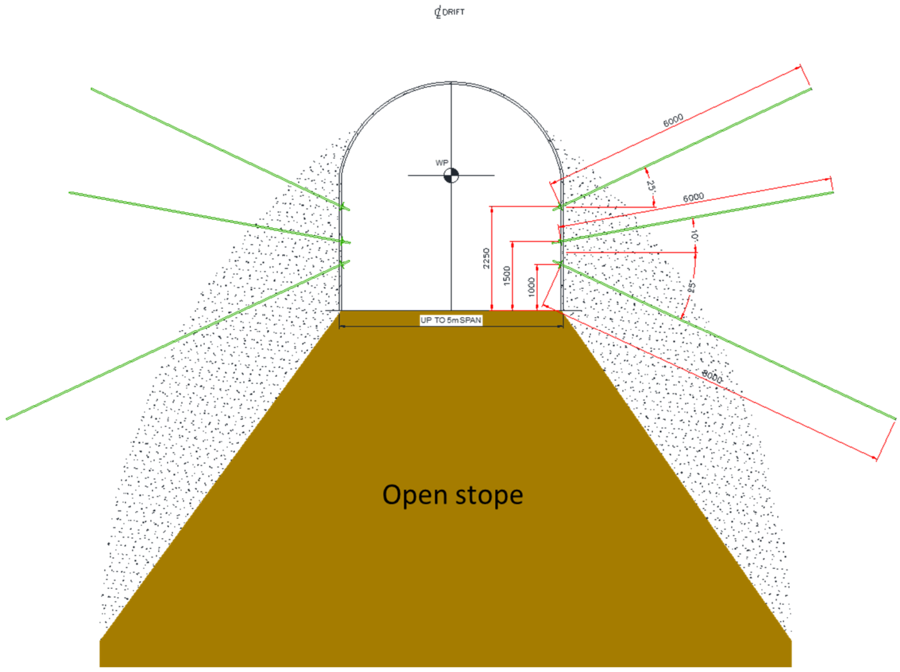

Figure 9 Secondary cable support design with increase in cable angle

3.2.2.2 Current cable bolt support design for C and D block stoping

The current cable bolt design was the result of continued underperformance of the midwall low-angle cable which had been part of the design since its inception. For this current design, both wall cables were moved further up the wall and into the shoulder. The same theory of increasing the use of the cables' tensile strength was the primary driver for the new cable placement. The lower wall pre-tensioned cables have remained as part of the support design for the primary stope lines. However, as more secondary stoping lines come into production, this design is modified to remove the pre-tensioned cables. Figure 10 outlines the current cable bolt support design used for primary stopes.

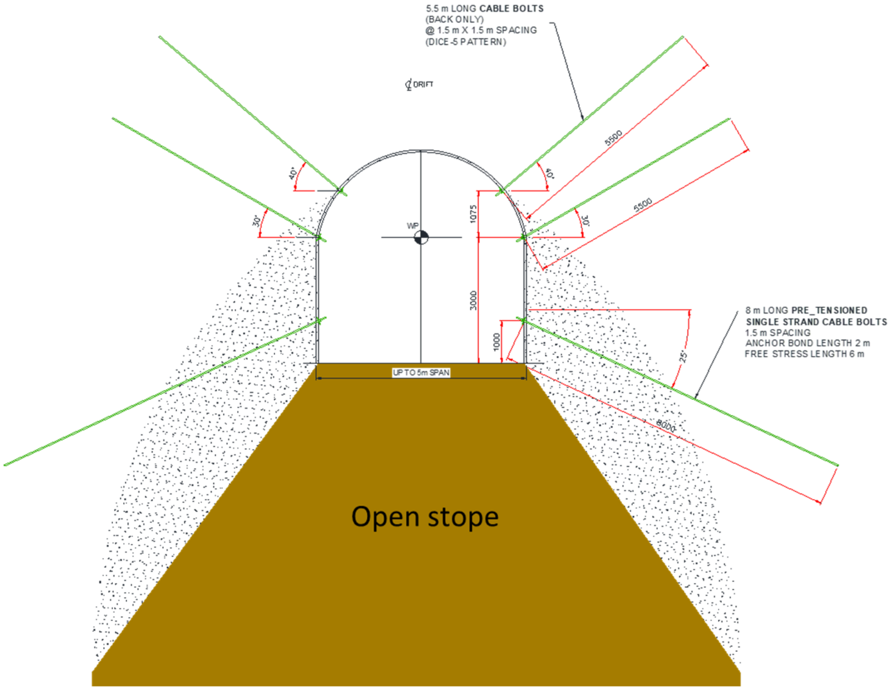

Figure 10 Current cable bolt secondary support design for primary stopes

This design, in conjunction with other changes made to the stoping processes, has been exhibiting promising results in maintaining the stability of the stope crown. The reason for removal of the pre-tensioned cables in the secondary stopes is that since the bond zone required for tensioning cannot be anchored within the CRF of the adjacent backfilled primary stope line and, therefore, cannot be anchored outside of the predicted failure envelope. Once the anchor point was unable to be anchored beyond the predicted failure envelope, this specific cable could not perform as designed.

3.2.2.3 Addition of resin grouted rebar to the support design for C and D block stoping

During the evolution of the cable bolt secondary support design, resin grouted rebar was added to the secondary support design. Rebar was added in response to modelled and subsequently observed stress changes as the stoping front began to develop. The original Swellex® bolts installed during the development cycle were observed to fail in shear in localised areas once stoping began. This was controlled by adding rebar to the secondary support design. Figure 11 outlines the resin grouted rebar design which is installed concurrently with the cable design.

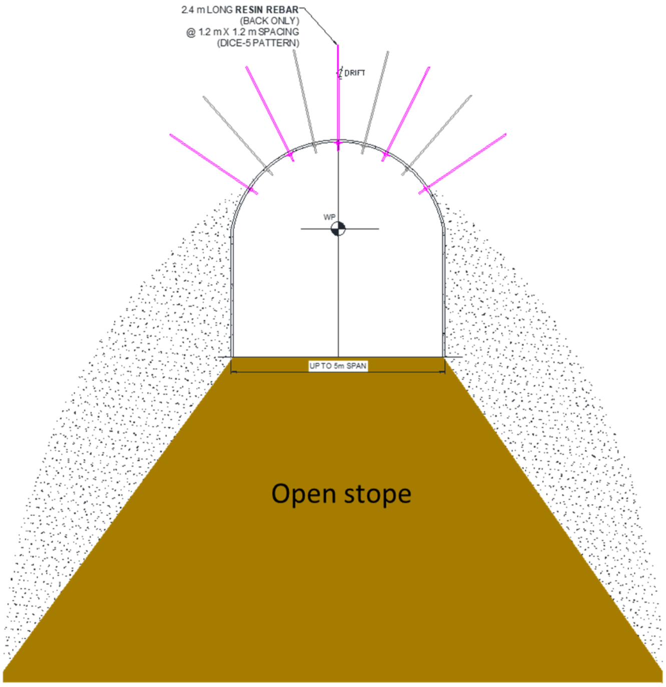

Figure 11 Resin rebar secondary support design

3.2.2.4 Addition of high-capacity support elements to the support design for D block stoping

Unlike C block, a higher stress to rock mass strength ratio is observed in D block, resulting in plastic deformation seen in the development. This typically presents as squeezing in the lower rib of development or loading of the rockbolts in the shoulders but can periodically manifest as large wall movements. Figure 12 is an example of some large wall movement due to squeezing in the right wall of the development. The depth of the deformation was greater than the length of the tendon support.

Figure 12 Large-scale plastic deformation in development at Diavik

The deformation in D block has had a negative impact on stoping due to rework and rehabilitation, and required a change in the primary support design to complement the secondary support design. The support required an increase in the tendon support capacity, as well as an increase in the surface support capacity. The tendon support was required to be upgraded from 2.4 m Pm12 Swellex® to 2.4 m Pm24 Swellex®. There was also a requirement to increase the surface support. The original prescribed support was for an initial placement of 50 mm of 35 MPa strength shotcrete, then no# 6 welded wire mesh with tendon support.

The upgraded support added an additional 50 mm of 35 MPa strength shotcrete on top of the installed welded wire mesh, embedding the mesh between the two shotcrete layers. This allowed for greater surface bearing capacity and for the cable loading to be distributed over greater surface area when the cables are installed. This design change has been able to manage the plastic deformation seen in D block development and subsequently helped maintain stability in the stope crown.

4 Conclusion

Diavik's path to optimising its stoping support kept safety and stability as the primary drivers. An iterative approach was taken, trialling and refining various support designs based on the response of the rock mass being supported throughout the mine. As the rock mass characteristics have had significant variability in some locations, Diavik has had to adapt its ground support designs to this variability to achieve its ore recovery targets and reduce production disruptions.

As further demonstrated in this paper, the support designs adopted have had to evolve in parallel with the changing approach to stope design and production output requirements. Changing from a vertical stoping geometry seen in the higher parts of the mine to a trapezoidal shape geometry required a similarly iterative approach to that undertaken to address the rock mass variability.

As Diavik progresses towards closure, the lessons learned throughout this evolutionary process can be applied to other operations with similar geotechnical conditions, mine sequencing, stope sizing and production output requirements.

References

Archibald, A 2023, 'Application of geotechnical numerical modelling to aid in operational decision making', *Proceedings of MTL 2023*, Canadian Institute of Mining, Metallurgy and Petroleum, Westmount.

Grimstad, E & Barton, N 1993, 'Updating the Q-system for NMT', in R Kompen, OA Opsahl & KR Berg (eds), *Proceedings of the International Symposium on Sprayed Concrete – Modern Use of Wet Mix Sprayed Concrete for Underground*, Norwegian Concrete Association, Oslo, pp. 46–66.

Hoek, E, Kaiser, PK & Bowden, WF 2000, *Support of Underground Excavations in Hard Rock*, 1st edn, CRC Press, London.

Hutchinson, DJ & Diederichs, MS 1996, *Cable Bolting in Underground Mines*, BiTech Publishers Ltd, Richmond

Jakubec, J 2020, 'Mining for diamonds – history and present', paper presented at MassMin2020, Santiago, 9–11 December 2020.

Lewis, PA, Auld, CP & Karami, A 2017, 'A comprehensive approach to geotechnical instability management in an open stoping mining block', paper presented at CIM Convention 2017, Montreal.

Lewis, PA, Clark, LM, Rowles, SJ, Auld, CP, Petryshen, CM & Elderkin, AP 2018, 'Sublevel retreat mining in the subarctic: a case study of the Diavik Diamond Mine', in Y Potvin & J Jakubec (eds), *Caving 2018: Proceedings of the Fourth International Symposium on Block and Sublevel Caving*, Australian Centre for Geomechanics, Perth, pp. 57–72, https://doi.org/10.36487/ACG_rep/1815_02_Lewis

Wyllie, DC & Mah, CW 2004, *Rock Slope Engineering*, 4th edn, CRC Press, London.