Learnings from development of an underground sub-level open stope mine below an open pit

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Abstract

This paper presents the lessons learnt during the transition from an open pit method to a sub-level open stoping method at the Koidu Limited kimberlite pipe. These learnings may assist other mines with planning, both from a practical implementation perspective (layout, charging up challenges), and a method for estimating potential dilution and stability challenges associated with the interaction between the surface and underground excavations. Predictive geotechnical stability modelling was successfully applied to highlight potential failure zones, which allowed modification and enhancement of the mining layouts and quantification of the potential dilution of the kimberlite from indicative instabilities.

Keywords

open pit -underground interaction, geotechnical modelling, sub-level open stoping, dilution, kimberlite

Introduction

Koidu Limited is located in the eastern province of Sierra Leone, approximately 360 km from the capital city, Freetown. The Koidu Limited mine hosts two kimberlite pipes and four major kimberlite dykes, with a number of blow zones (volumetric lenticular enlargements with diameters up to 30 m – appears similar to a kimberlite pipe) – see schematic in Figure 1.

A schematic of the K1 mine design to Level 8, together with the access drives and the K2 mine design, are presented in Figure 2. The K1 kimberlite pipe has been mined down to 280 m during its open pit operations and has since transitioned to an underground mining operation in 2016 as open pit operations became economically unviable due to high stripping ratios (Figure 2). The K1 kimberlite pipe dimensions are 45 m along its N-S axis and 115 m along its E-W axis (at 100 masl). Due to an excessive ore gap during the establishment of the underground mine, it was decided to extend the open pit excavation by an additional 20 m, extending from surface (380 masl) to approximately 300 m below surface. Koidu's weather varies between a wet season of approximately 6-9 months (highest rainfall experienced between June and August months), and a dry season of 3 months from December to February month (winter months). The city records rainfall varying from ~7 mm in December to a ~319 mm in August.

As shown in Figure 2, two access points for the underground mine were decided on, namely a sacrificial adit from the 144 Level switchback located within the K1 open pit (this was changed from the initial 208 Level to further reduce the ore gap, with 208 Level subsequently being used as a pump station), and a permanent decline developed from surface, named the Central Decline (positioned between the K1 Pipe and the K2 Pipe). The main purpose of the 144 Level adit, developed from within the K1 open pit, was to immediately access and establish the underground mine with the knowledge that at some point during its life, it may become unstable due to the impact of the underground mine on the open pit rock walls. The Central Decline intersected the decline developed from the 144 Level adit at ~100 mamsl, whilst the K1 Level 2 stope was being mined. A breakaway was developed from the Central Decline, which provides access to the K2 Pipe, which is located at approximately 255 mamsl.

The blasted material from the first two sublevel open stope (SLOS) levels was hauled through the K1 open pit, after which it was decided to abort hauling via the open pit ramps due to geotechnical stability concerns, i.e., the switchback started collapsing after mining of the Level 1 and Level 2 stopes had been completed. At the same time, the 208 Level pump station was decommissioned (due to stability concerns) and replaced with the 305 Level coffer dam (Figure 2).

All elevation levels are with reference to mean sea level (mamsl) with surface at 380 mamsl.

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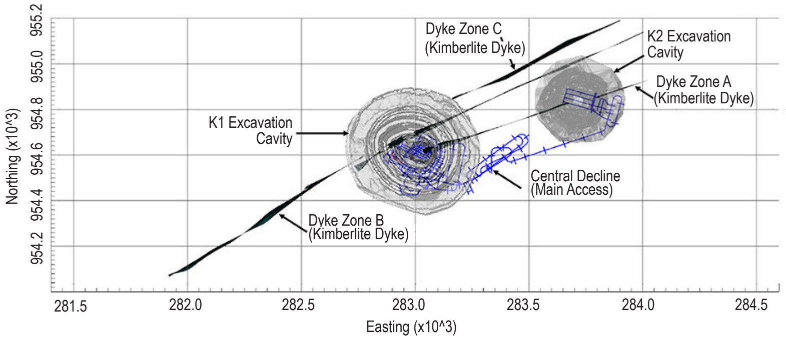

Figure 1—Schematic plan view showing the K1 Pipe, K2 Pipe, and the kimberlite dykes

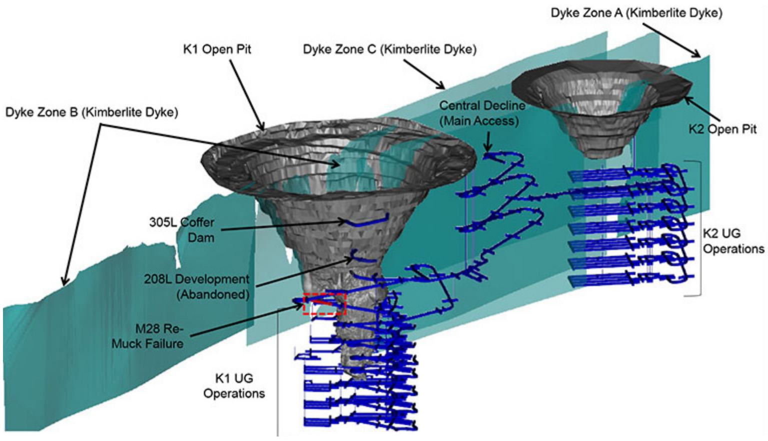

Figure 2—Mine design for K1 sub-level open stope (ground surface is located at 380 m above mean sea level) (Van Eeden, 2024)

This paper presents the stability assessment methods, which informed the decision-making around the placement and the use of the access declines (Central Decline and the sacrificial 144 Level adit) and pumping arrangements, as well as a method for quantification of potential dilution using discrete element modelling, and learnings from a practical implementation perspective.

Photographs showing the tunnelling quality in pre-stoping fresh kimberlite are shown in Figure 3 and in the weathered kimberlite during stoping of the K1 pipe in Figure 4).

Impact of underground mining on the K1 open pit stability

The software, developed by Itasca, Fast Lagrangian Analysis of Continua in 3 Dimensions (FLAC3D) was chosen for this study. The models were initially run applying the elastic constitutive model (prior to mining). The Mohr Coulomb constitutive model was then applied to determine potential tensile and shear fracture zones. Figure 5 presents FLAC3D plots of the open pit and Dyke Zone B (DZB) kimberlite dyke outlines in the model. The initial lithological model based on the exploration drilling information indicated that the kimberlite dykes do not intersect, however it was later established during the underground mining that these dykes do intersect.

Figure 3—Mine design for K1 sub-level open stope (Vermeulen, 2024)

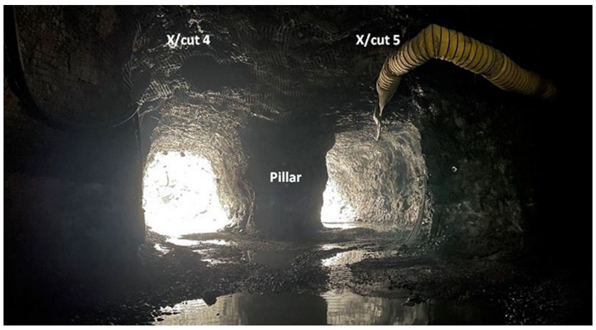

Figure 4—Photograph showing a pillar left between two cross-cuts at an intersection in a large span (Vermeulen, 2024)

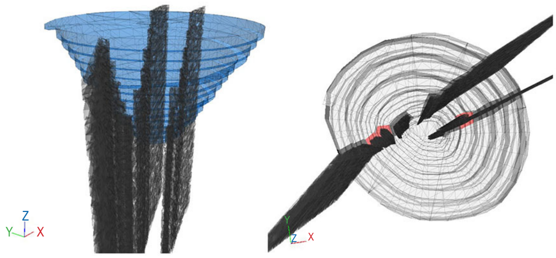

Figure 5—Plots showing the simulated K1 pit shell with kimberlite dykes (the red indicates volumetric enlargements (blow material) within the DZB (kimberlite dyke)

Stability assessment deepening of open pit prior to sub-level cave mining

Based on initial assessment, the potential for failure appears to be associated with the blow material. The potential for failure was predicted from ~240 m above mean sea level, down to the pit floor, which is greater than the failure experienced on the DZB, which extended from ~140 mamsl to ~180 mamsl. This suggested that the rock mass properties assigned to the blow material may be weaker than current conditions and observations. A bridge, which had been constructed after a historical failure along Dyke Zone B during mining of the open pit, is at approximately 160 mamsl. However, the potential exists for deterioration of the blow material, which may lead to potential failure from a depth of ~240 mamsl below surface to the pit floor, which would affect the ramp leading to both the 144 Level adit and the 208 Level pump station. The potential failure depth into the high wall is predicted to be ~50 m and approximately 60 m (measured horizontally) above the elevation of 150 mamsl. Below the elevation of 150 mamsl, the affected area includes virtually all the walls, as predicted by the model. The affected area increases radially throughout the pit, between the pit bottom elevation and 150 mamsl, after extracting the cut between 100 mamsl and 110 mamsl. This indicated the need for improved pre-split drilling and blasting to limit the potential for blast damage and dynamic loading, which may lead to rock falls.

The potential for failure of the granite rock mass was predicted from ~200 mamsl (Figure 6). This meant that the potential failure in the granite was predicted to extend to an elevation of 8 m below the 208 Level pump station, i.e., the bench immediately below 208 Level. The 305 Level coffer dam was predicted to be outside the potential failure zone predicted by the FLAC3D model. The pump station on 208 Level was therefore de-commissioned and replaced by the coffer dam on 305 Level.

Impact of the underground sub-level cave on rock wall stability

Further FLAC3D modelling suggested that, for the conditions assumed in the model (i.e., no water, no deterioration of the rock mass due to weathering, etc.), the secondary deformation in the open pit was more likely to occur during mining of Level 1 Extension as compared to Levels 2 to 4 (Figure 7). The 144 Level switchback was predicted to collapse should the Level 1 extension be mined. Based on the modelling results, it was decided not to mine the Level 1 extension in order to reduce the potential influx of waste material from the potentially large failure predicted. The modelling results further predicted that the planned stope dimensions on Level

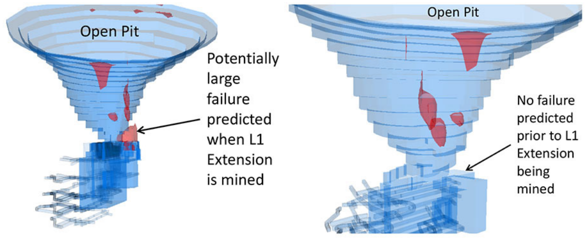

Figure 7—FLAC3D plot showing the zones of potential collapse if the extension stope be mined on Level 1 (red coloured iso-surface plot represents 2.2 mm/m maximum shear strain)

3 (as modelled) should have no negative impact on the stability of the decline ramp, for the middling of ~40 m between the open stope and the decline at Level 3.

Open pit failure events

Failures that have occurred in the open pit are listed below:

- ➤ Failure along the kimberlite dykes (numerous failures have been observed during mining of the open pit, mainly after high rainfall events).
- ➤ Failure of the blow material, which appeared to be as a resultof dynamic loading, possibly due to slip along kimberlite dyke-granite contact (the extent of which was similar to that predicted by the FLAC3D model) – (Shariff, 2018).
- ➤ Undercutting of the jointed granite, which resulted in failure along some of the prominent and persistent joints (as predicted by the distinct element modelling).
- ➤ The K1 open pit shell saw a failure along one of the weathered kimberlite dykes. The failure that extended 30 m into the high walls of the open pit occurred along the intersection of two kimberlite dykes. This failure migrated ~30 m horizontally, initially exposing the M28 re-muck, as per Figure 8 to Figure10. Continuous self-mining of the kimberlite dyke continued to undercut the M28 re-muck, compromising the main access decline. The main access decline ended up being totally undercut and collapsed into the stope. The correct decision was therefore taken to stop all use of the main access decline and to develop a new access decline.
- ➤ Cracks were observed along the 144 Level switchback, after which crack meters were installed to monitor the displacement after the initial failure at M28 re-muck. The crack meter readings showed that no further opening occurred after the collapse of the M28 re-muck, even after completing stoping down to Level 7 had been completed.

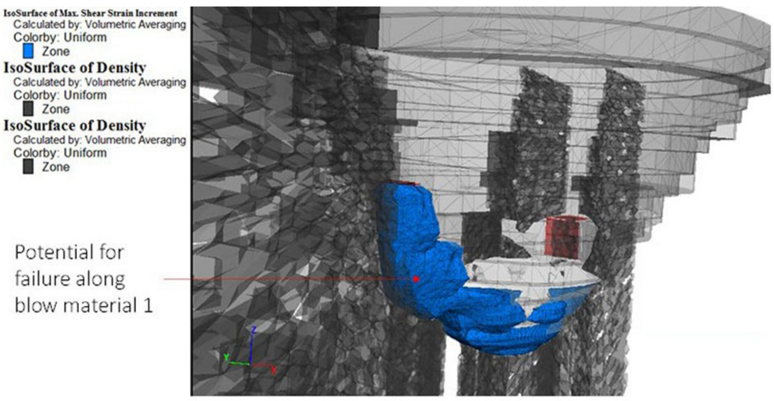

Figure 6—Plot showing rock volume in which 1 mm/m shear strain is exceeded, concentrated around the blow material 1 at pit depth of 100 m

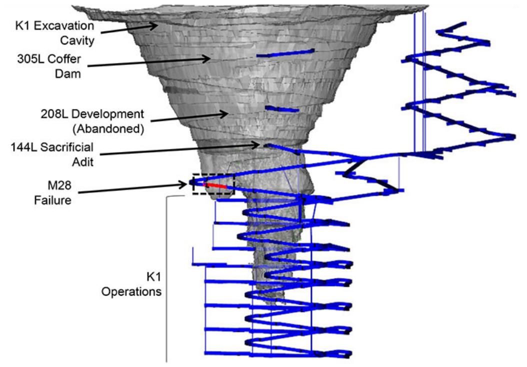

Figure 8—View of the M28 re-muck position relative to the open pit – failure area indicated in red (Van Eeden, 2024)

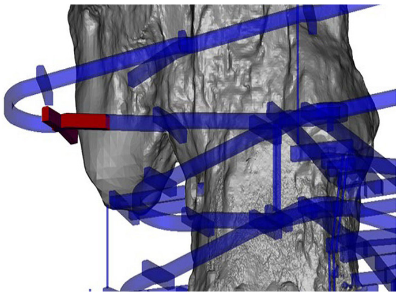

Figure 9—Location of the M28 re-muck position – failure area indicated in red (Van Eeden, 2024)

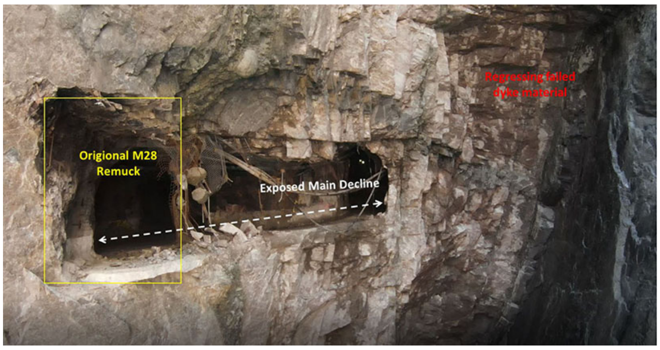

Figure 10—Photograph of the now exposed main decline at the M28 re-muck position

Assessment of potential dilution using a distinct element model

Photographs showing the impact of the different joint sets and dykes on the slope stability in the K1 open pit are presented in Figure 11. The joint set orientation data were used as input into a numerical modelling software to determine the types of failures and extent thereof on the K1 pit (as illustrated in Figure 12). The results indicate that toppling, wedge, and planar failure is predicted (as observed underground). It should be noted that due to the high

Figure 11—Schematic and photo showing exposed joints

wall slope angle, relative to the dip of the joint sets, the effect of the instability is limited to the bench-scale. Although the joints are persistent over more than a single bench height, the following is evident:

- ➤ The undulating and stepped contacts improve the shear strength.
- ➤ Failure may occur where the joint daylights along the slope but requires other planes of weakness to allow dislodging to occur. The bench crests generally have the lowest confinement, which makes dislodging of rocks easier to occur there than from the high wall under confinement.

The mine design for K1 Sub-Level open stopes as simulated using Itasca's 3D Distinct Element Code (3DEC) software is shown in Figure 13.

Three stages of granite collapses are predicted in the distinct element model, which are:

1. Initial collapse – undercutting of the open pit high wall by the underground stope.
2. Subsequent collapse – undercutting of the Level 1 stope by the Level 2 stope.
3. Tertiary collapse – failure of the granite in the open pit, not as a result of undercutting due to mining, but rather by the collapse of rock mass below it.

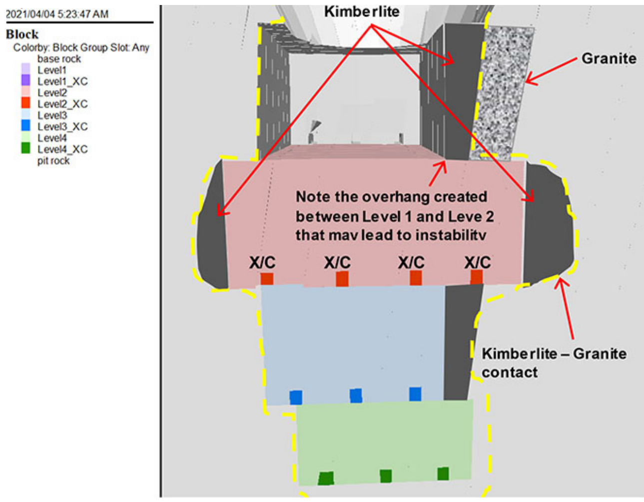

Figure 13—Mine design for K1 sub-level open stopes simulated in 3DEC

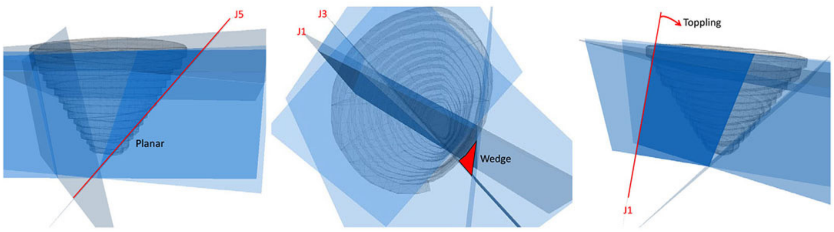

Figure 12—Schematic showing the potential toppling type failure on J1

The distinct element modelling further predicts the following:

1. Kimberlite from the sidewalls of the stopes (and the open pit high walls) is likely to dislodge from the kimberlite-granite contact.
2. Undercutting of granite and kimberlite is likely to result in collapses, due to the failure along joints (overhang conditions created which exposes persistent joints).
3. Both the fresh kimberlite and granite is likely to remain stable if:
   a. No joints (including the contact) in either the fresh kimberlite or granite are present in the overhang rock.
   b. If the joints in the granite dip into the sidewall of the stope. Although toppling failure is possible, the distinct element model mainly indicates opening along the joints. For a collapse to occur, a third joint set or random joint is required).
   c. Even in the absence of joints within the kimberlite, time dependent failure may still occur due to the weathering of the kimberlite, which reduces the overall rock mass strength of the kimberlite and may lead to a collapse due to rock mass failure and not structural failure.
4. Based on the distinct element modelling, the volume of kimberlite rock that will dislodge from the rock walls, is likely to be more than the granite rock that may dislodge (this is mainly due to the volume of kimberlite that has been left against the stope sidewalls). The maximum volume of granite rock mass predicted to collapse into the stope over a cross-cut / mining length of ~30 m (~15 blast rings spaced 2 m apart) is predicted to be ~9,500 m^3, which equates to ~25,650 tonnes. For the same cross-cut / mining length, the maximum volume of kimberlite rock mass predicted to collapse into the stope is predicted to be ~18,000 m3, which equates to ~43,000 tonnes, which is approximately 70% more tonnes and almost double the volume of the granite.
5. The distinct element model predicted that Level 2 would most likely be impacted by the collapse of granite from Level 1 and above. With the mining of cross-cut 4 on Level 2, the model showed that additional kimberlite as well as granite will dislodge from Level 1.
6. The predictive distinct element modelling applied has tracked closely with the collapse of 32 Kt of granite waste rock and the incidental recovery of 81 Kt of additional kimberlite resulting from the collapse.

Practical implementation learnings

Original mine layout

The typical extraction configuration in the main stopes consist of:

- ➤ 4.5 m x 5.0 m crosscuts developed in arch shape.
- ➤ The crosscuts are developed along the E-W axis of the orebody.
- ➤ A slot cut drive is developed perpendicular to the limits of the ore crosscuts on the extremities of the ore-waste contact zone.
- ➤ A circular blind raise, with a diameter of 1.5 m is drilled from the bottom up as an initiating free-face in order to blast a slot cut, which will serve as the free-face for subsequent production blast rings once the stope has been fully established.
- ➤ There are typically 3 draw-points per production level.

Figure 14 shows the mine/stope configuration and Figure 15 shows a plan view of typical drawpoint layouts.

Changes to mine layout

The FLAC3D geotechnical model has significantly guided the design, configuration and placement of draw points and the extraction sequencing. Various zones within the production stopes have been deferred (e.g., Level 1 extension) as they have the potential, according to the geotechnical model, to initiate waste failures and influxes into the stopes, which will have significant adverse financial implications. These deferred zones have been sequenced to be alternatively mined toward the end of the life of the K1 underground mine when such impacts can be managed accordingly through abandonment or selective extraction; this approach will ensure that underlying stopes remain viable due to the current controls on dilution as a result of the predictive insights provided by the FLAC3D modelling. The mine design layouts have been further enhanced by the development of the slot drives in the middle of the ore body, which increases the number of draw points from 3 to 6 with a tramming half- loop, which will have a positive effect on productivities, dilution control, flexibility, and an added layer of safety for personnel and equipment. Placement of all permanent infrastructure (return airways, dams, decline shafts) has also been meticulously informed by the modelling to ensure a safe and efficient underground operation.

Drilling, blasting and charging up

In the planning and execution of drilling, blasting, and charging up operations for the 40-metre vertical stope, a comprehensive approach was adopted, considering various critical factors and

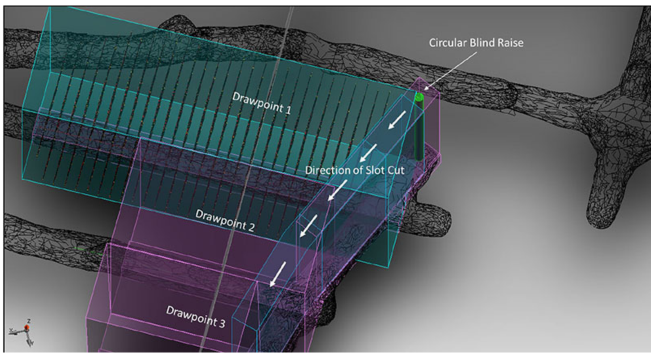

Figure 14—The mine/stope configuration

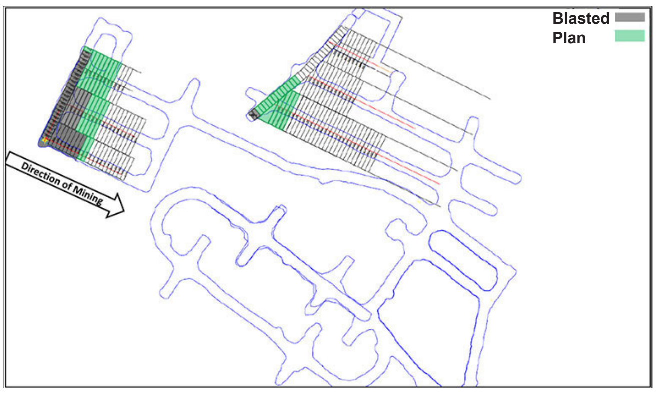

Figure 15—Plan view showing typical draw-point layouts

utilising advanced geotechnical modelling tools such as FLAC3D and 3DEC. These tools provided invaluable insights into the subsurface conditions, enabling the development of effective blind bore slot patterns and stope blast designs prior to commencing any drilling or blasting activities. The detailed blast design for this operation considered the following key elements:

➤ *Blind bore raise positioning*: The positioning of the blind bore raise was carefully planned to ensure long-lasting integrity while establishing the slot. Special attention was given to orient the blind bore raises perpendicular to ore cross-cuts, optimising their effectiveness.
➤ *Blast hole design*: The design of blast holes for slot establishment factored in various critical parameters, including void size (with a specific focus on a 1.5-metre circular blind raise/free face), rock compressive strength, desired fragmentation size, burden and spacing, and the type and quantity of explosives to be employed.
➤ *Stope/cross-cut ring layout*: The layout of the main stope/cross-cut rings was carefully engineered to minimise toe burdens. Consideration was given to avoid crowding of the bottom holes while still achieving the desired fragmentation outcomes.
➤ *Special considerations for waterlogged areas*: Given the challenging waterlogged underground conditions in which mining is conducted, a meticulous approach was taken to the specification of explosives and accessories for blasting operations.
➤ *Sticky emulsion explosives*: In this context, emulsion explosives with adhesive properties were thoughtfully selected. The "sticky" emulsion adheres to the borehole walls, enhancing the efficiency of blasting in wet conditions.
➤ *Innovative solution for groundwater mitigation*: Due to the significant groundwater encountered after blast hole drilling; an innovative in-house solution was developed to prevent emulsion washout from the charged-up holes. Blast balls, in combination with blast hole liners, were strategically inserted into the drilled holes before charging up. This innovation effectively prevents groundwater from diminishing the adhesive properties of the emulsion, thus reducing emulsion losses in the blast holes and ensuring desired blasting outcomes.
➤ *Geotechnical stability and predictive insights:* Throughout the planning and execution phases, geotechnical stability controls were continuously implemented. Insights obtained from advanced FLAC3D modelling contributed significantly to the decision-making process, enhancing the overall viability of the underlying stopes and highlighted the importance of planning a secondary access.

In summary, the drilling, blasting, and charging up operations for the 40-metre vertical stope were conducted with precision and innovation, leveraging geotechnical modelling, meticulous blast design, and adaptive solutions to address the unique challenges posed by waterlogged underground environments. These measures collectively ensure the safety, efficiency, and effectiveness of the blasting operations while optimising the recoveries from the stopes. Figure 16 and Figure 17 illustrate a typical blast ring layout.

Conclusions

Predictive geotechnical stability modelling is invaluable when dealing with complex mining layouts. However, this is dependent on an understanding of the potential failure mechanisms, choosing the correct modelling tools and constitutive model, deriving the representative rock mass strength parameters, which requires mapping data, core logging data, laboratory test data, and instrumentation data (deformation, strain, etc.), continuous back analysis, and carrying out model calibration for application in future (all rock mass failures should be investigated, effectiveness of mining and support strategies should be recorded).

Extensive self-mining of up to 30 m of the kimberlite (because of a combination of weathering due to numerous high rainfall events, and jointed rock mass) in the blow, dykes, and pipes have led to instability of the 144 Level switchback and 144 Level adit, as well as a portion of the main access decline at M28 re-muck, which indicate the importance of compiling three-dimensional structural models together with advanced inelastic predictive modelling that allow simulation of the stability of complex underground excavations.

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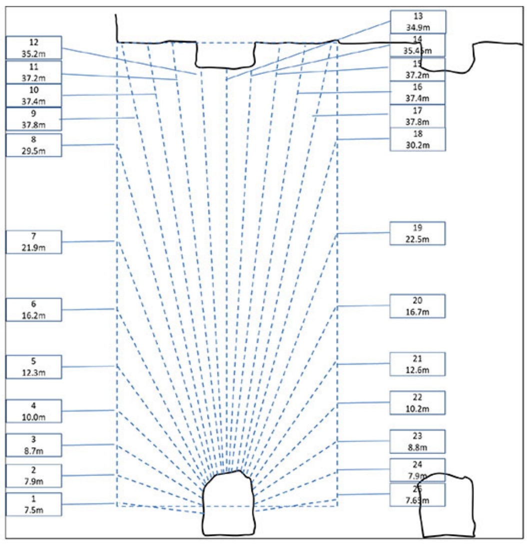

Figure 16—Blast Ring Layout (Kyeramateng, 2024)

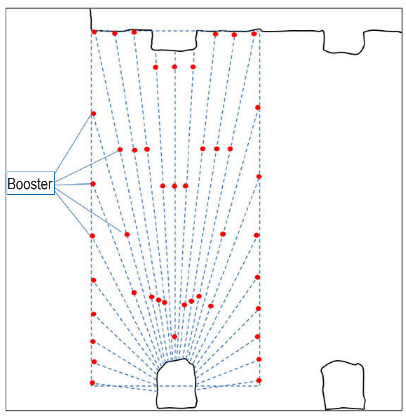

Figure 17—Blast ring layout showing the location of the boosters (Kyeramateng, 2024)