Geotech Geol Eng (2022) 40:2231–2254
https://doi.org/10.1007/s10706-021-02023-3

ORIGINAL PAPER

Fall of Ground Management Through Underground Joint Mapping: Shallow Chrome Mining Case Study

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Received: 3 May 2021 / Accepted: 16 November 2021 / Published online: 27 November 2021

Abstract The use of joint properties and joint mapping techniques are key for fall of ground management in underground mining. This paper outlines the use of probabilistic design approach in addressing potential falls of ground, based on identified keyblocks. A shallow chrome mine was used as a case study to identify potential causes of falls of ground with the aim of improving the existing fall of ground management system. The existing fall of ground management system comprises of visual observations, Ground Penetrating Radar (GPR) scanning and Borehole camera inspections. The mine is characterised by geological structures such as faults and joints, hence the system leans towards structural analysis. Joint mapping was carried out in the North and South sections of the mine using window mapping and scanline mappint techniques. The collected joint data from each section was used to evaluate rock fall probability. Rockfall probabilistic analysis carried out in the study indicates that about 80% of all key blocks formed are 1 m^3 in size. Results show that larger blocks are more likely to fail through rotation whereas small key blocks are most likely to fall in-between support units. Further stability analysis was conducted through simulation of the effect of change in support spacing on excavation stability. Indeed support spacing plays a critical role in the overall stability of the excavation as opposed to the length and capacity of the support unit. This conclusion was drawn based on the improvement of the factor of safety during the simulation exercise. This research is based on an MSc Engineering study.

Keywords Fall of ground · Scan line mapping · Window mapping · Risk management · Probabilistic design

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1 Introduction

Fall of ground has been considered to be the most common problematic issue faced by underground mining in decades. This elusive challenge has resulted in an increase in injuries as well as fatalities within underground mining working places (Eisner and Leger 1988; Leger 1991; Mark and Iannacchione 2000; Roberts et al. 2001; Koldas 2001; Gumede and Stacey 2007; Vorster and Franklin 2008; Ferreira and Minova 2012). Indeed, there has been a gradual rededication in accidents associated with falls of ground (FOG) from the early 90s to date (Anon 2020). It is anticipated that this gradual reduction in FOG incidents is rejuvenated by the advances in technology and research within the mining industry (Koldaş 2003; Stacey and Gumede 2007; Maiti and Khanzode 2009; Mark et al. 2011; Teleka et al. 2012; Zhang et al. 2017; Adoko et al. 2019). Furthermore, FOG management is a continuous assessment wherein rock engineering specialists are brought together to come up with solutions regarding FOG. Ryder and Jager (2002) described a rock as a complex engineering material of which its behaviour is influenced by numerous factors such as strength, density, permeability, porosity etc. The environment in which the mining takes place cannot be changed (Yilmaz 2015) as a result, the stability of underground excavations is a key concern. The focus is mainly on stability enhancement by means of excavation support designs and monitoring of ground conditions. Mining activities such as drilling, and blasting change the stress environment in the periphery of the excavation. Owing to these changes, risk assessments and close monitoring of the exposed rock mass are considered to be essential (Walke and Yerpude 2015).

The term fall of ground (FOG) is used to classify incidents related to unexpected rock mass movement or the uncontrolled release of rock in excavations due to gravity, pressure, or rockburst. There are numerous ground monitoring systems used to manage falls of ground in the mining industry. The depth of mining has a degree of influence on the choice of fall of ground monitoring and management system approach. Deep level mines are prone to seismicity and rockburst due to high stresses, as opposed to shallow mining environments which are prone to falls of ground and minimal stresses (ISRM 1978; Jager and Ryder 1999; Ryder and Jager 2002). According to Ozbay et al. (1995), shallow hard rock mining environments are usually associated with joints and bedding planes that weaken the hanging wall strata. The hanging wall rock mass at this depth is characterised by well-defined discontinuities subjected to deadweight tension (Ozbay et al. 1995). The occurrence of planes of weakness in the hanging wall strata is a major factor for excavation stability in underground mining environments (Adoko et al. 2017). This is because the interaction of these planes may result in unstable blocks of rocks with the potential to fall under the influence of gravity. Therefore, deep level ground monitoring and management systems will lean towards seismicity, and shallow mining ground monitoring and management systems will lean towards structural analysis (Parkasiewicz et al. 2017; Mishra et al. 2017; Xia et al. 2018; Malinowska et al. 2019; Yang et al. 2019; Rahimi et al. 2020; Mondal et al. 2020; Małkowski et al. 2020). An ideal FOG management system incorporates visual observations, the use of ground monitoring equipment with the capability to give warning and identify structures with the potential to cause damage at an early stage. The above-mentioned combination is critical for support design and decision-making.

FOG management is critical for excavation stability enhancement. The larger the excavation, the more the geological structures are exposed, which consequently influence the stability of mine excavations and the support system required thereof. The above discussion can be crystallized by an example from a study by Chikande and Zvarivadza (2016), where a platinum room and pillar mine with intense faulting and jointing of the rock mass, resulted in poor ground conditions. At times, poor ground conditions may necessitate a different mining layout as a way of managing and preventing FOGs. To enhance stability in this example, a different layout was designed for that specific ground condition, wherein two-pillar sizes were designed which are 10 m × 3 m and 3 m × 3 m, maintaining a 6 m board width. Besides, a new support system was also designed for this area to help improve safety and production, as the previous support system was not adequate for such poor ground conditions. The new system included the use of longer roof bolts (2.1 m) spaced at 1.2 m × 1.2 m whereas the previous support system had 1.8 m long roof bolts spaced at 1 m × 1 m (Chikande and Zvarivadza 2016). Other examples of fall of ground management strategies are evident in studies by scholars cuch as Joughin (2008), Vogt et al. (2010), Joughin et al. (2012), Esterhuizen (2014), Joughin et al. (2016), Chikande and Zvarivadza (2018).

In mining, rockfall-related hazards are forever present. The fact that FOGs management strategies have been put in place means that FOG is a major concern. It is critical for a mine to implement a strategy that will help combat rock fall-related hazards. This study is conducted to identify loopholes within the current fall of the ground management system at a shallow chrome mine. This study helps to improve the current system, and consequently, combat FOGs at the mine. The study focuses firstly on determining the mode of rock failure attributing to geological structures present in the mining environment. Secondly, the study helps to design an empirical support system based on a probabilistic approach using numerical modeling software packages.

2 Geological Setting and Mining Layout

The mining is taking place in the eastern Bushveld Complex (BC), in South Africa. Amongst others, the Rustenburg Layered Suite of the BC stratigraphically comprises a Marginal zone, Lower zone, Critical zone, Main zone, and Upper zone. The Upper Critical zone contains both the Merensky Reef and the UG2 reef (PGEs) (Kruger 1990; Uken 1998; Eales and Cawthorn 1996). The Critical zone found in both the eastern and western limbs hosts the largest stratified chromitite reserves in the world (Kinnaird 2005).

The eastern Bushveld Complex is characterised by geological complexities such as faults, potholes, and intrusions creating dykes and sills (Uken 1998; Robertson 1977; Roberts and Clark-Moster 2010). In addition, geological discontinuities (fractures, joints, and parting planes) related to rock mechanics are also present in the area (Nong 2010). As a result, the continuity and consistency in the width of chrome bearing layers are disturbed by the occurrence of geological structures. The behavior of rock mass around mine excavations is highly dependent on the presence of geological structures, pre-existing stresses, and mining-induced stresses (Quaye and Guler 1998; Adoko et al. 2016). Therefore, the knowledge of geological structures plays a vital role in the overall design of a mine.

The mine uses a trackless mechanized mining method (TMMM) in the form of bord and pillar. This mining method already provides regional support by means of pillars. Underground excavations are supported for three main reasons which are: (1) to ensure the safety of the working places, (2) to prevent falls of the ground by preventing key blocks from falling, and lastly (3) to control the movement of large blocks or fragments in the periphery of the excavation. Owing to that, local support is deemed essential (Hoek and Wood 1987; Hoek et al. 1998). Generally, support design for an underground excavation is summarised into two aspects namely demand and capacity, both measured in kN/m^2. Demand, in this case, refers to load generated by the hanging wall (to be supported by one support unit) and capacity is the resistance or load generated by one support unit over a specific tributary area. If the demand exceeds the capacity, then failure occurs. The aim is to stabilise the hanging wall by providing sufficient support resistance to avoid failure (i.e., capacity must be greater than the demand).

The FOG management system at the mine comprises of three techniques namely: (1) visual observations, (2) Ground Penetrating Radar (GPR) scanning and (3) Borehole camera inspections. Visual observations are conducted and panels are rated based on the complexity of the geology and their support compliance. Secondly horizontal and subhorizontal parting planes within the hanging wall are scanned for, using GPR. In cases where parting planes are detected within the hanging wall when scanning, at least three borehole camera holes are drilled for further inspection of the detected structure. The mine has however reported numerous FOG casesregardless of the FOG management system in place.

3 Research Approaches

The main aim of this study was to identify loopholes within the current ground control system (which comprises of visual observations, GRP scanning and Borehole camera Inspections) in order to know how it can be improved to be more efficient. This was achieved by looking at the current FOGs management technique used at the mine together with the historical fall of ground database to identify the main causes of FOGs. The approaches used are explained in detail in the following subsections.

3.1 Determination of the Fallout Thickness Across the Mine

The historical FOG database of the mine was reviewed in order to develop the 95 percentile FOG empirical design threshold by Jager and Ryder (1999). In this regard, a histogram of previous rock falls measured was plotted to obtain the average fallout thickness. Two approaches were used to determine fall-out thickness. Firstly, fall out thickness from previous FOGs was used and secondly, brow thicknesses were measured for the same exercise. The fallout thickness, in this case, is based on historical FOG thicknesses for both North and South sections of the chrome mine combined. Brow thickness data was collected for each section. This method was conducted to confirm whether the fallout thickness from historical data would correlate with that obtained using the brow thickness method. For this method, brow thicknesses were measured in both the North and South sections.

3.2 Joint Mapping Techniques

The most important factors that describe rock joints according to Priest and Samaniego (1983) are orientation, spacing, length and shear strength of the joint. However, in this study, the interest is in the three geometrical properties namely: length, spacing, and orientation. Predominantly, the joint properties of interest in a project are influenced by the nature of the problem and its objectives. Since this study is based on FOGs and identifying the potential unstable key blocks, the joint properties of interest were selected based on their ability to define a block of rock in a rock mass. The three joint properties of interest are joint orientation, joint spacing, and joint length. For this study, only two underground joint mapping techniques are discussed, these are scanline mapping and window mapping.

(a) *Scanline mapping*

Scanline mapping is the most common technique used in underground mining. This technique requires a measuring tape to be extended over a distance on the surface of a rock mass. The tape can be pinned on both ends in order to keep it in place for the duration of mapping (Brady and Brown 2004). All joints intersecting the tape are mapped (Fig. 1).

For this study, the scanline mapping technique was the least used to collect joint data due to limiting factors such as time and accesibility. A 30 m long measuring tape was suspended straight between two points in a working panel mining Southwards (Fig. 2). The tape was looped over roof bolts in order to keep it in place avoiding sagging. After the tape was successfully put in place, all joints intersecting the tape were mapped. The mapping process involved measuring and recording the following:

(a) The distance along the tape where discontinuities are intersecting the tape.
(b) Type of discontinuity.
(c) Dip angle and dip direction of each discontinuity.
(d) Spacing and trace lengths of discontinuities.

(b) *Window mapping*

Window mapping is sometimes referred to as cell or area mapping because of the nature of mapping. The technique involves a systematic manner of dividing the area to be mapped (face, hanging wall, or sidewall) into zones of equal size. Depending on the mine, the zone shapes will vary. Some may be rectangular while some are square. According to Nicholas and Sims (2000), these zones are usually square dimensions. Visual observations are carried out in each window, and joint properties (orientation, spacing, length, etc.) are recorded. An example of a rectangular window is shown in Fig. 3.

The window mapping technique was conducted in both sections of the mine. The technique is fast, thus more panels were mapped using this technique. The mining layout consists of 10 m panels and 12 m $\times$ 12 m pillars (Fig. 4). The mapping zone in this case was rectangular. After defining the mapping window, discontinuities observed within the window were mapped. The following was measured and recorded:

(a) Type of discontinuity.
(b) Dip angle and dip direction of the discontinuity.
(c) Spacing and trace length of discontinuities.

3.3 Numerical Simulation on Joints Distribution

The collected joint data was analyzed using DIPS 7.0 Rocscience software. Dips software is designed to interactively analyze geological discontinuities data orientation. The software also helps to visualise and analyze structural data in 2D using the same technique as that applied in manual stereonets (Rocscience 2002). Joint orientation is defined by the dip and dip direction angle. This joint property is regarded as one of the most important parameters since it neutralizes the effect of other properties when oriented favorably as stated by Gumede and Stacey (2007).

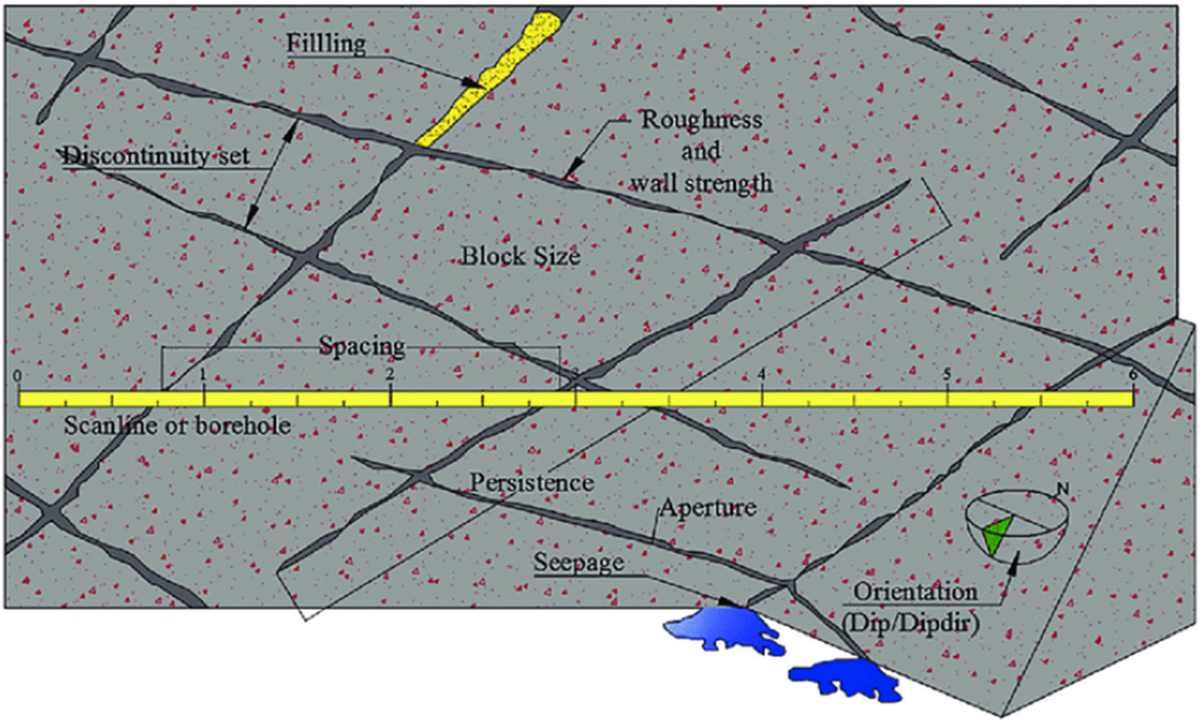

Fig. 1 Scan line mapping (Monsalve et al. 2018)

3.4 Numerical Simulation of the Impact Support Spacing on the Stability of the Excavation

In order to understand the best situation in which the bolt can be installed, numerical simulation with a varying spacing of the bolt was performed. The numerical simulation consisted of several input parameters including rock properties, joint sets, principal stresses, excavation dimensions, etc. Nonetheless, the simulation was also used to validate the effectiveness of the bolts in the varying orientation of the bolt installation. It is important to indicate that the simulation was largely focusing on simulating Factor of Safety (FoS) of wedges revolving around the excavation. Three joint sets were used to simulate underground excavation at a shallow mine. A stereonet of the simulated excavation is shown in Figs. 5 and 6.

4 Results and Discussion

Installed support improves safety and stability in excavations, therefore it is critical to understand the required support resistance to stabilise the excavation. This section clearly outlines how the fall out was estimated and how joint data was used to identify the modes of failure and ultimately design support according to the formed wedges in the excavation.

4.1 Fallout Thickness Evaluation

Based on the historical data from the Chrome mine, the average fallout thicknesses at 95% cumulative frequency were found to be 0.95 m (Fig. 7). This approach was adopted from Jager and Ryder (1999) who estimated fallout thickness using historical data from FOGs.

Fallout thickness obtained in both sections is within the 90–100 cm thickness range (Figs. 8 and 9). In 2014, the average fallout thickness was 0.7 m. However, results show that currently the average fall out thickness is 0.95 m. The difference in the two average fallout thicknesses, prove that ground conditions are

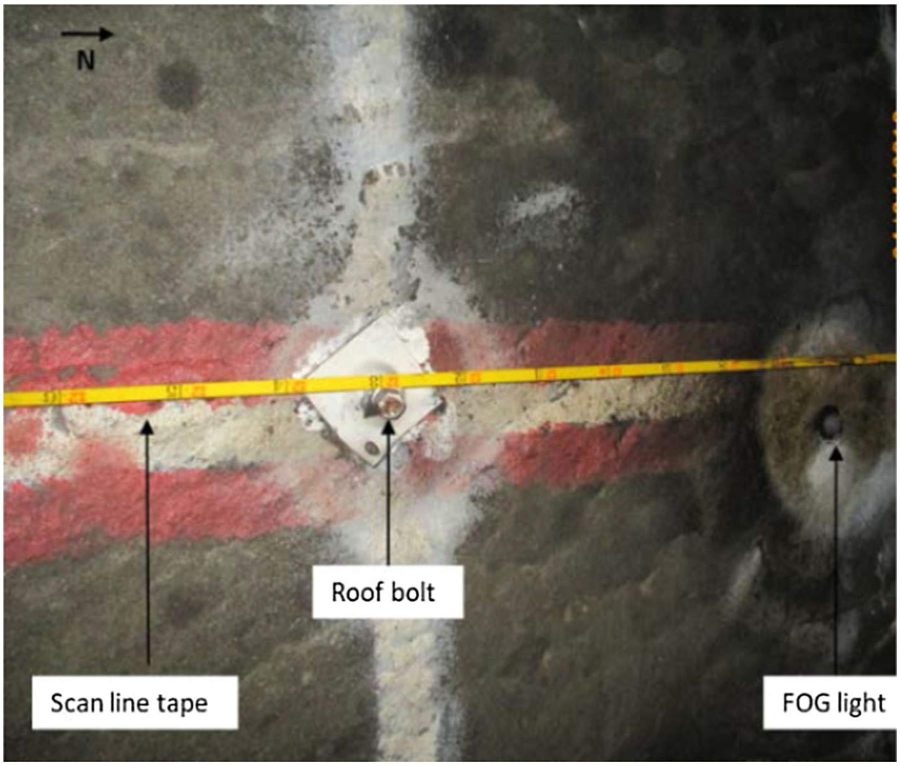

Fig. 2 Scan line tape aligned to the centerline of the panel

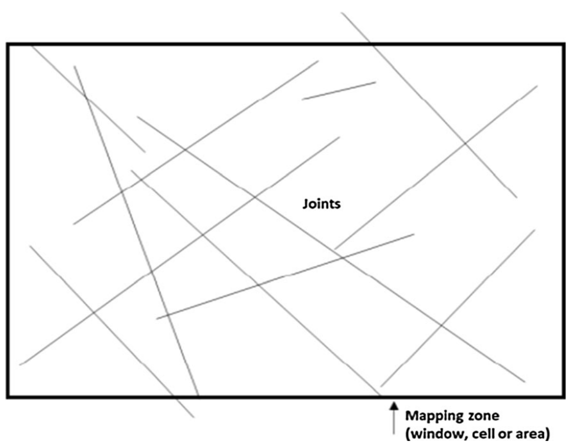

Fig. 3 Schematic diagram of window mapping zone

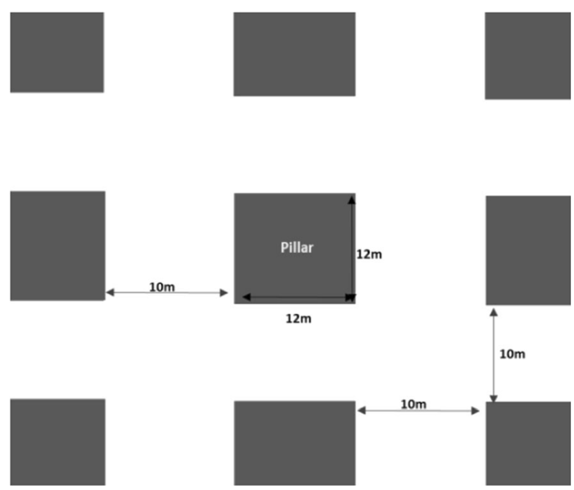

Fig. 4 Mine working layout

not always the same throughout the mine. The current ground condition is characterised by many geological discontinuities unlike in 2014. Hence, the current fallout thickness is approximately 1.0 m. Although results from the two methods are similar, caution should be exercised with roof bolts installation angle. As soon as the angle of installation gets shallow, the support unit length becomes insufficient to support 100% of the estimated fallout thickness. The fallout thickness evaluation is a critical part of this study because it helps deduce the possible potential height of unstable wedges formed from joint distribution across the excavation. Indeed support resistance is well-

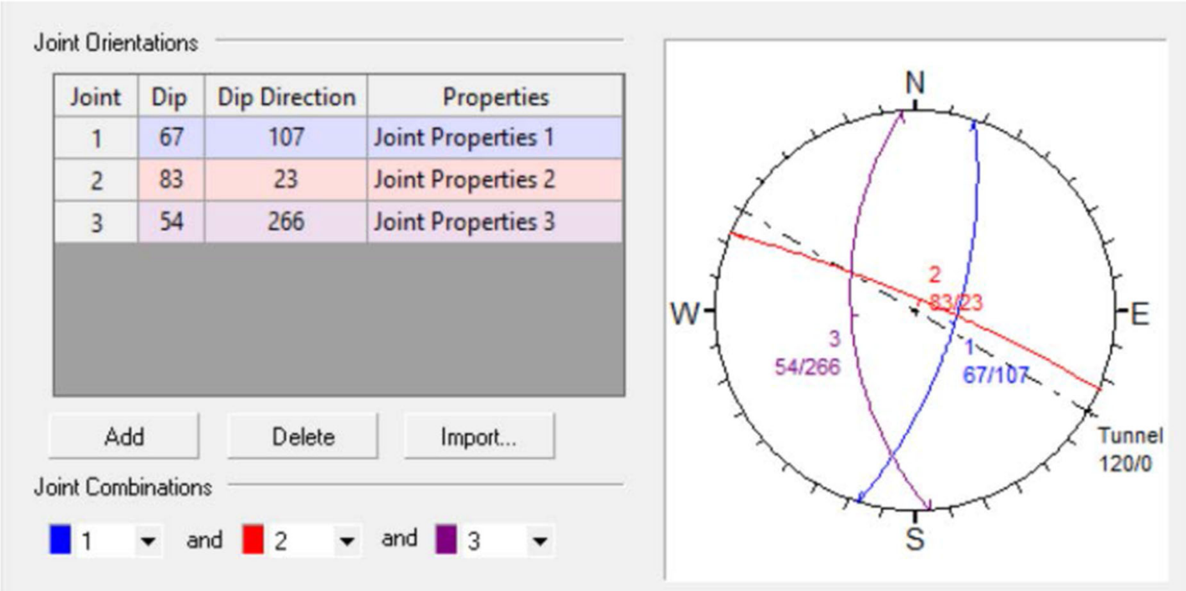

Fig. 5 North-Stereonet of the simulated underground situation using Unwedge (Finite Element Method)

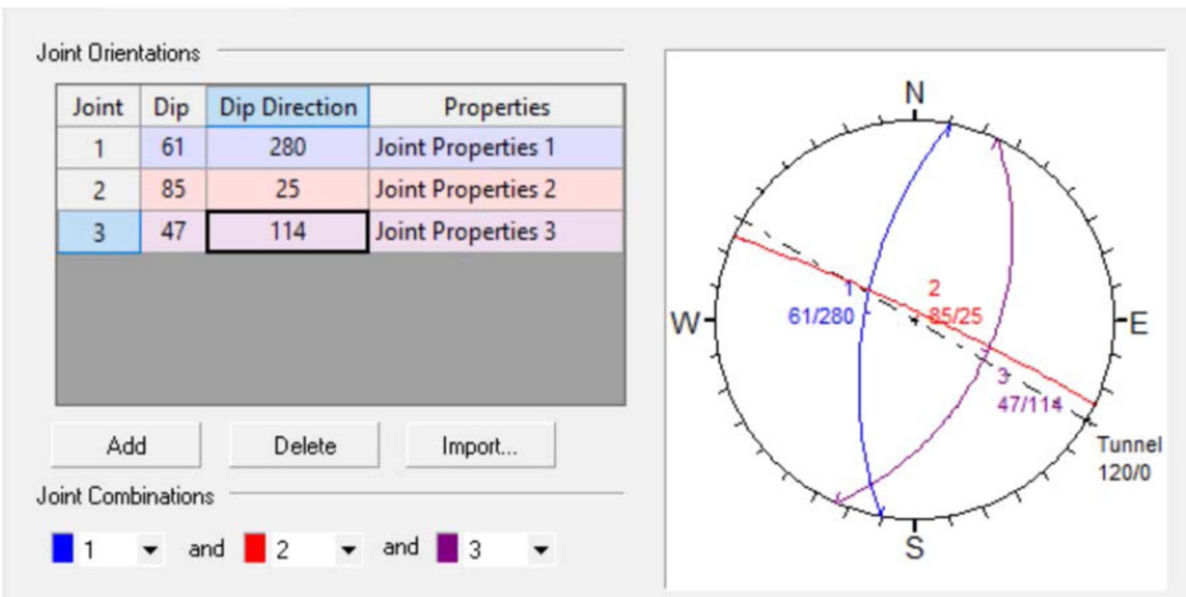

Fig. 6 South-Stereonet of the simulated underground situation using Unwedge (Finite Element Method)

defined using fallout thickness (Chikande and Zavarivadza 2018). Furthermore, the used in deducing support resistance is highly dependent on the tributary area as contested by Stacey and Swart (2001).

4.2 Joints Distribution Across the Mine

The use of joint data to predict rock fall probability has been demonstrated using J-Block software. J-Block is a well-known rock engineering software designed to evaluate the probability of potential rockfalls in mining excavations. Underground mapping of

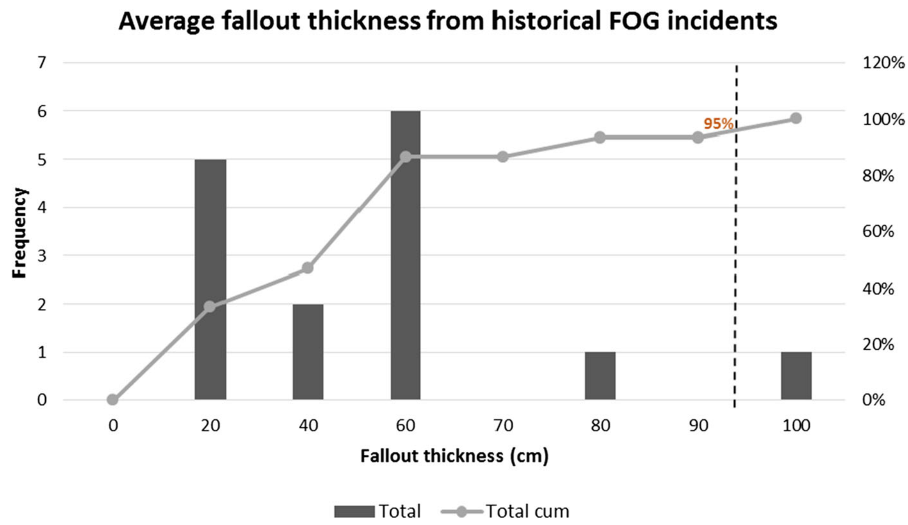

Fig. 7 Average fallout thickness estimated from historical FOG data

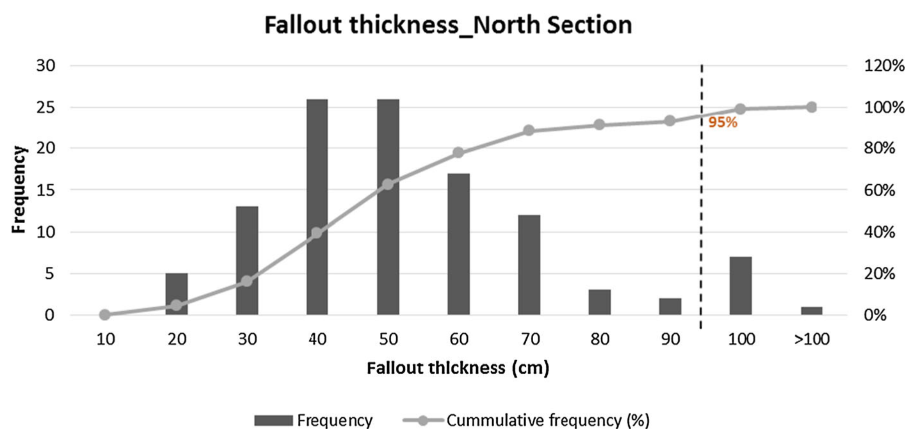

Fig. 8 Fallout thickness _North section

individual stress fractures and joints is not completely practical. Owing to this limitation, the software simulates blocks in the hanging wall formed by jointing. The simulation is completed by means of statistical methods in order to identify potential unstable key blocks (Esterhuizen 2003). J-Block only simulates one surface at a time; thus, corners are not simulated as they are made of more than one surface (Grenon and Hadjigeorgiou 2003).

4.2.1 Stereonets Plots on Joint Sets Distribution

A graphic illustration of joint orientation results in the North section is given in Fig. 10. Three major joint sets are identified. The dominant joint set (J1) is orientated at 67°/107°, joint set 2 (J2) orientated at 83°/23° and lastly (J3) flat dipping at 54°/266°. Joint orientation results from the South section also show three joint sets (Fig. 11). The set orientation for J1, J2 and J3 are 61°/280°; 85°/25°; and 47°/114° respectively. Although three joint sets were identified in each section, there is a discrepancy in the dip and dip direction of these sets.

Overall, joint data analysis results are summarised in Table 1. The summarised data can now be used to predict rock fall probability together with the evaluation of support.

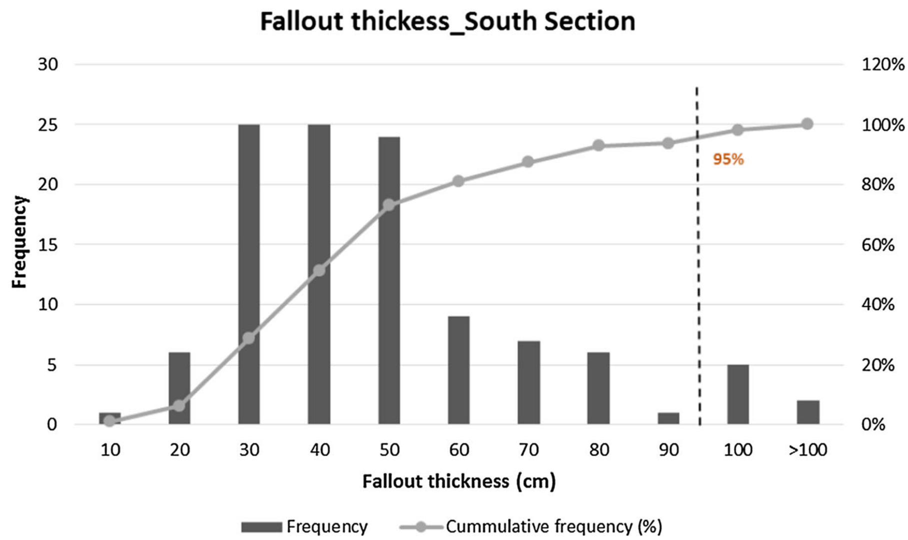

Fig. 9 Fallout thickness _South section

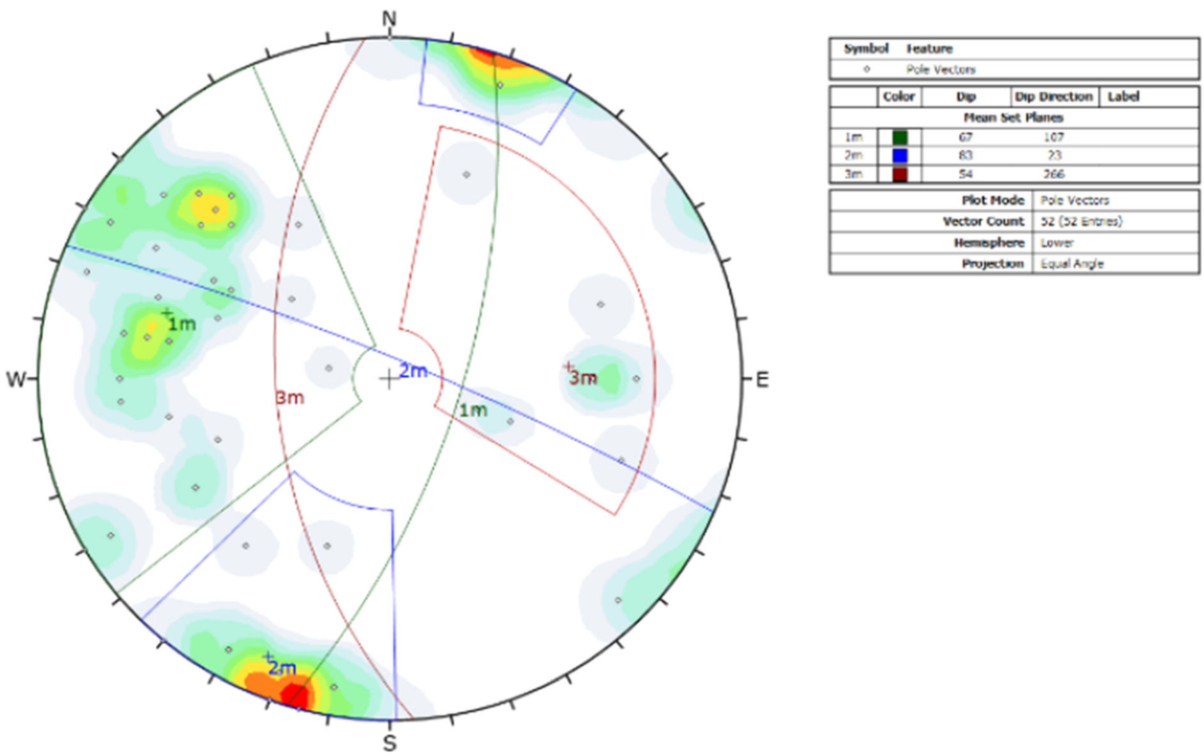

Fig. 10 Overall joint orientation in the North Section

4.2.2 Key Block Size Distribution

Results obtained from the simulations include key block size distribution, distribution of failure probabilities, failure and stability modes. In addition, support layout analysis was also carried out. Block size and shape are critical aspects in determining the possible fallout thickness and required support capacity (Windsor and Thompson 1992; Windsor 1999). Indeed J-Block satisfies both these aspects during

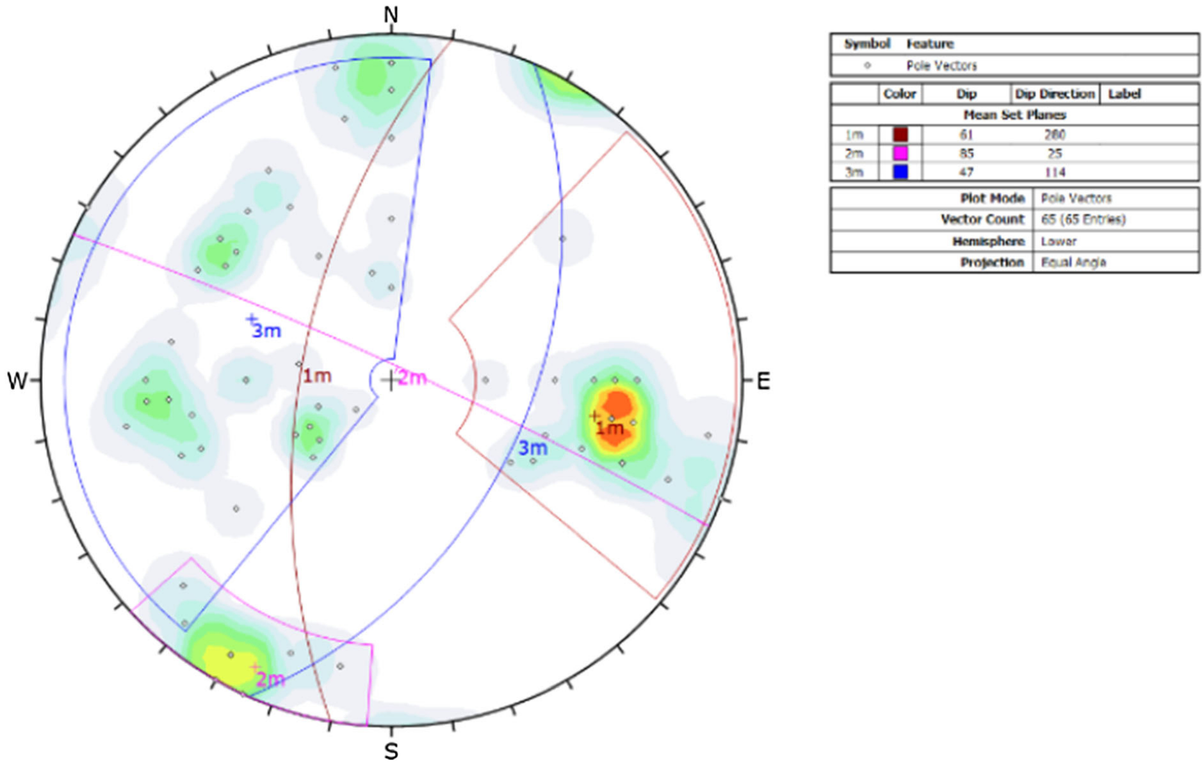

Fig. 11 Overall joint orientation in the South Section

Table 1 Overall joint data for the two sections

Section	Joint set number	Dip (°)	Dip direction (°)	Range (°)	Spacing (m)			Length (m)		
					Mean	Min	Max	Mean	Min	Max
North	J1	67	107	10	2.5	0.2	6	13	5	20
	J2	83	23	10	2.5	1.5	4	16	10	20
	J3	54	266	10	2.4	1	4	14	5	22
South	J1	61	280	10	2.9	0.8	7	8.3	2	20
	J2	85	25	10	3	1	5	10	1.5	20
	J3	47	114	10	2.6	0.5	5	9.5	2	20

simulation, hence outcomes are key block sizes with the potential to fall in between support and support failure.

Simulation of joint sets in the North section shows that 61% of key blocks formed are less than one cubic meter. The key block size distribution probability ranges between 1 and 9% as the block size increases from 2 to 15 m^3. The probability of 1 m^3 blocks falling in-between support is 6.3%. For blocks greater than 1 m^3, the probability decreases, but the probability that supports will fail increases (Fig. 12). The probability that supports will fail as the block size increases ranges between 1 and 12%.

Key block size distribution in the South section (Fig. 13) shows that 72% of blocks formed are less than 1 m^3. As the block size increases, the key block size distribution also decreases. The probability of 1 m^3 blocks falling in-between support is 77%. As the

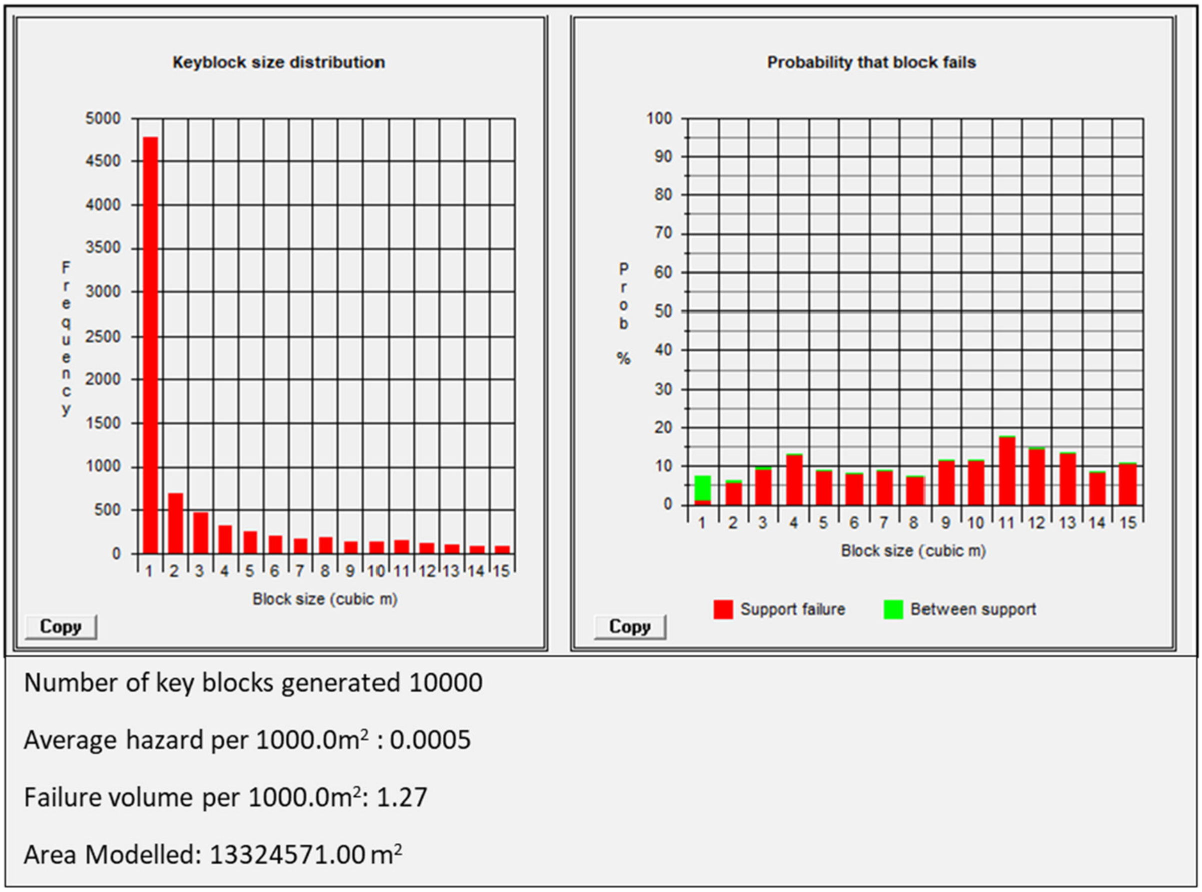

Fig. 12 Key block size distribution for North

block size increases, the probability of blocks falling between supports decreases. However, the probability of blocks greater than 1 m^3 falling due to support failure increases (6–18%). Failure probabilities in the North section are less than those in the South. This is because of varying joint set orientation. Although both sections have three joint sets, the South joint sets are flatter than those of the North. This means that joint geometries in the South section have the potential to form more key bocks.

Area outcome distributions show that most block failures are localized in the center of the excavation (Figs. 14 and 15). This may not completely be the case as J-Block simulates one surface at a time. Since edges are of more than one surface, corners were not considered in the simulation. Nevertheless, Esterhuizen and Streuders (1998) contested that unstable blocks with high potential to fall are defined by the geological structures and are usually found on the hanging wall of the excavation. This, however, does not imply that key blocks on the face and sidewall cannot fall. The J-Block outcome shows that there is 8.7% and 26.6% fall percentage for North and South sections respectively. Based on the fall frequency distribution, the South Sect. (198 blocks) is most likely to experience rockfalls in the center of the excavation when compared to the North (124 blocks).

4.2.3 Block Failure Modes and Stability Modes

Rock failure modes detected by J-Block include single plane, double plane, drop out and rotation. In the North section, there is 93% probability of 1 m^3 key blocks to fail by drop out. This is because small blocks are likely to fall in between support units. As key block size increases, rotation failure mode probabilities also increase (53–100%) as shown in Fig. 16. The South section is more likely to experience various failure

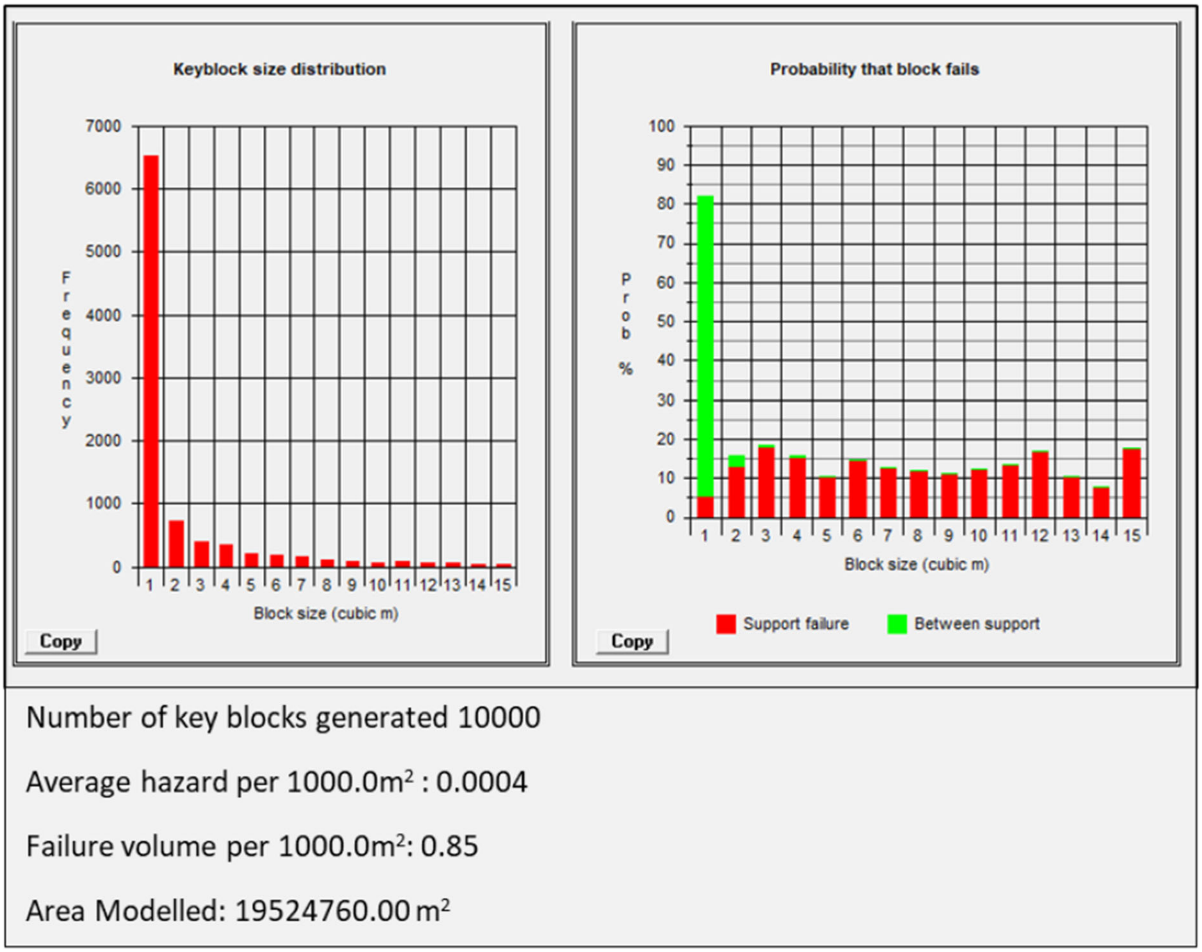

Fig. 13 Key block size distribution for South

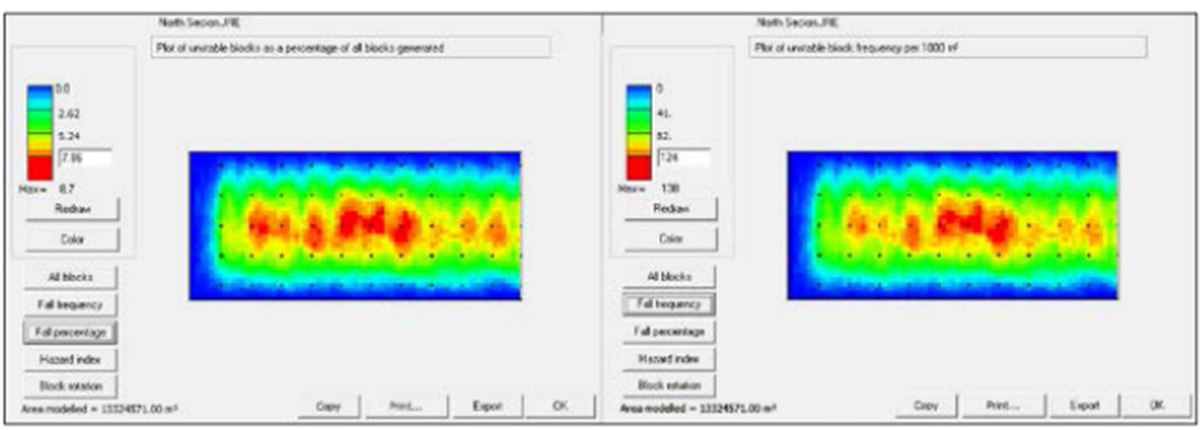

Fig. 14 Area modelled in North Section

modes, supporting higher failure probabilities mentioned previously. Drop out failure probability is high (80%) for 1 m^3 block sizes and decreases with increasing key block size (47–7%). As shown in

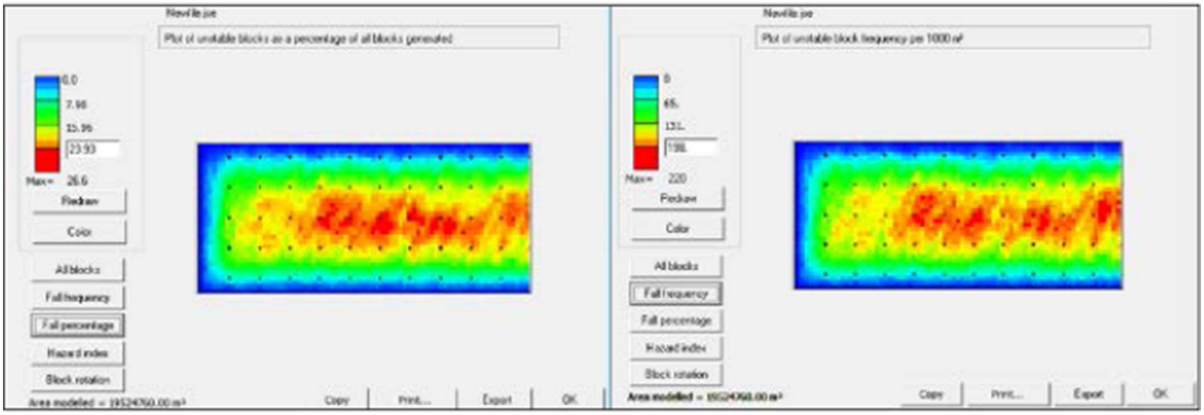

Fig. 15 Area modelled in South section

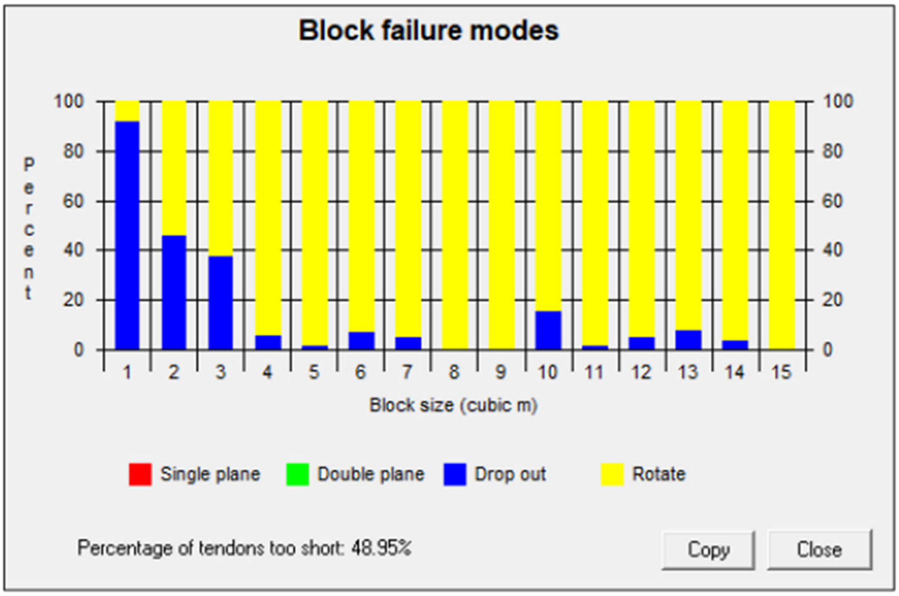

Fig. 16 Block failure modes in the North

Fig. 17, single plane failure mode is more likely to occur due to sliding along 50° dipping planes. On the other hand, sliding planes for key blocks in the North section are steeper, ranging between 60° and 80° (Fig. 18). The larger the block size, the more likely it is to fail by rotation (0–59%).

Stability mode results (Fig. 19) from both sections are similar. All block sizes are stabilised by friction as opposed to supporting. However, there are 48.95% and 57.48% that tendons are too short for North and South sections, respectively. According to Stacey and Swart (2001), support unit length must cover the full fallout thickness, plus 200 mm to anchor the parting plane and also encoporate100mm protruding length. In addition, the total tendon length is maximised if the tendon is installed at right angles to the hanging wall (Stacey and Swart 2001).

4.3 Support Analysis

Installed support improves safety and stability in excavations. Support layout and capacity will influence excavation stability. If the weight of generated key blocks exceeds the support capacity, failure is deemed to take place. Therefore, support systems employed must be adequate enough to stabilise the

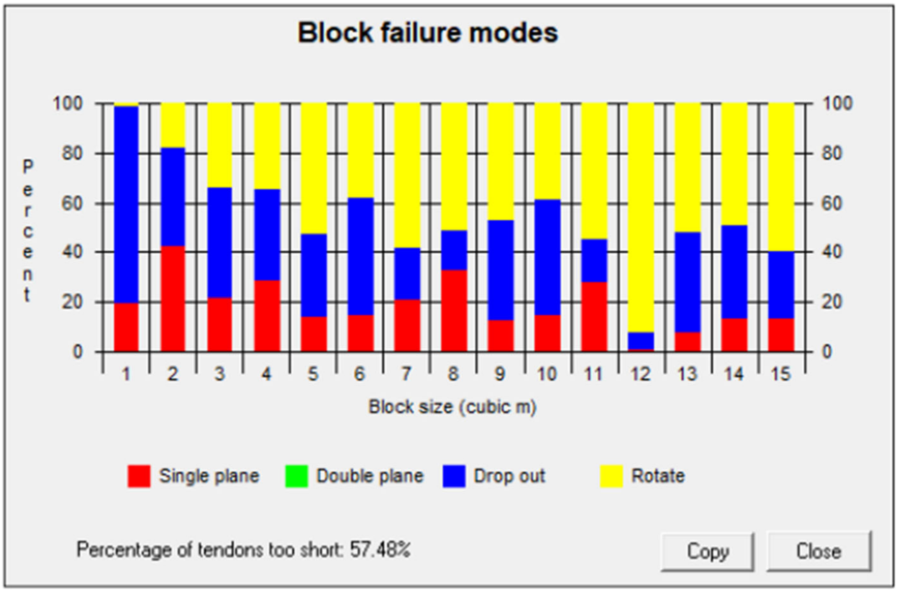

Fig. 17 Block failure modes in the South

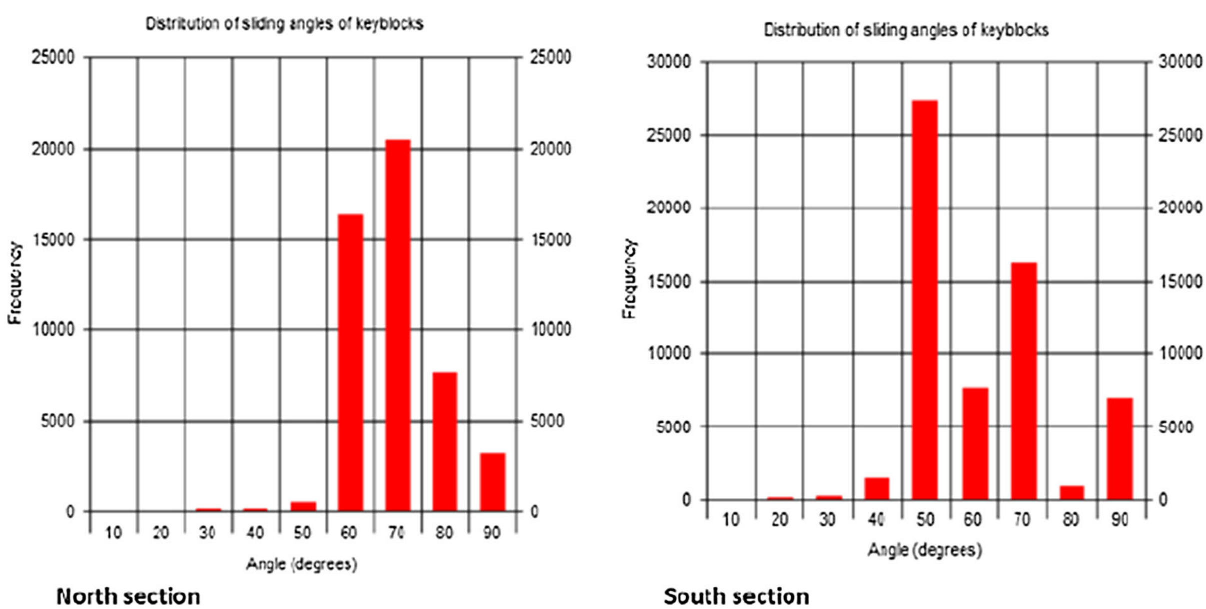

Fig. 18 Distribution of sliding angles of key blocks and stability modes in North and South section

excavation. Analyses conducted in preceding sub-sections showed that failure is most likely to take place in between support units for 1 m^3 key blocks. This means that support spacing significantly affects the support resistance to stabilise these key blocks.

Table 2 summarises the effect of support spacing on 1 m^3 block size failure probability. The same support units (non-grouted 1.5 m long roof bolts @ 110 kN) were evaluated over three different support spacing layouts. Reducing the support spacing decreases the probability of small key blocks falling between support units. Nevertheless, reducing support spacing will increase the support density and cost thereof. Closely spaced support units also increase the support failure probability as shown in Table 2. Support failure may occur if the support capacity is

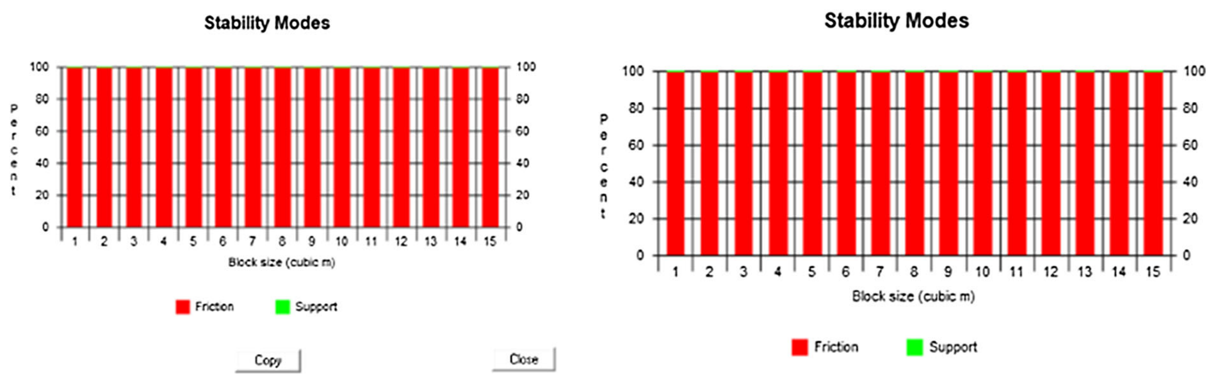

Fig. 19 Stability modes in North (left) and South (right) section

Table 2 Effect of support spacing on the probability of failure

Support length (m)	2.0 × 2.0		1.5 × 1.5		1.0 × 1.0	
Failure type and Block size (m^3)	% prob of support failure	% prob of fall between support	% prob of support failure	% prob of fall between support	% prob of support failure	% prob of fall between support
1	1,3	6,3	1,5	4,9	2,1	4,1
2	5,9	0,3	8,4	0,1	2,8	0,0
3	9,5	0,2	12,5	0,0	5,2	0,0
4	13,1	0,0	13,6	0,0	9,6	0,0
5	8,9	0,0	18,6	0,0	6,8	0,0
6	8,2	0,0	23,2	0,0	19,4	0,0
7	8,8	0,0	19,5	0,0	17,9	0,0
8	7,4	0,0	23,4	0,0	26,7	0,0
9	11,7	0,0	18,6	0,0	13,1	0,0
10	11,6	0,0	25,0	0,0	30,7	0,0
11	17,7	0,0	18,3	0,0	38,1	0,0
12	14,8	0,0	27,8	0,0	30,0	0,0
13	13,7	0,0	8,8	0,0	22,3	0,0
14	8,7	0,0	23,0	0,0	25,5	0,0
15	11,0	0,0	21,6	0,0	28,8	0,0

smaller than the required resistance. To remedy the probability of support units being too short, models were also run with varying support lengths and capacity keeping a constant spacing of 2 m × 2 m as shown in Table 3. Based on these results, it can be concluded that the probability of support failure decreases with increasing support capacity for the same support length and spacing (Fig. 20).

The use of joint data to predict rock fall probability has been demonstrated using J-Block software. The probability of key block size formed confirms the overall fallout thickness evaluated. The average fallout thickness of the mine estimated through an approach by Jager and Ryder (1999) was found to be 0.95 m. On the other hand, a high percentage (80%) of the overall key blocks formed are 1 m^3. Probabilistic analysis can be used to evaluate the probability of rock falls, and support design for stability enhancement. To support this statement, the use of the probabilistic approach in support design of jointed rock mass has

Table 3 Effect of support length and support capacity on stability

Capacity (KN)	110			150			200		
Support length (m)	1,5	2	2,5	1,5	2	2,5	1,5	2	2,5
Block size (m^3)	% prob of support failure			% prob of support failure			% prob of support failure		
Effect of Support length and Capacity on 2 m × 2 m constant spacing									
1	1,3	1,1	1,2	1,1	1,2	1,3	1,1	1,2	1,0
2	5,9	6,2	6,3	5,4	5,3	6,1	3,4	4,7	5,0
3	9,5	10,7	9,3	8,7	10,4	10,0	5,8	8,3	8,0
4	13,1	12,3	8,7	10,4	11,2	12,5	7,7	7,3	8,8
5	8,9	11,0	8,7	11,9	10,2	6,4	11,2	8,1	9,7
6	8,2	9,2	15,8	9,8	7,7	7,6	10,6	11,0	7,8
7	8,8	14,3	11,7	11,9	18,6	15,2	10,9	7,6	13,1
8	7,4	10,5	16,4	12,4	13,1	12,3	12,9	6,0	13,5
9	11,7	10,9	16,6	11,8	11,8	10,6	9,8	13,3	11,1
10	11,6	13,6	23,1	15,4	16,3	18,3	8,9	16,2	8,0
11	17,7	12,0	17,0	8,5	14,6	13,9	13,8	12,2	12,6
12	14,8	6,9	10,5	9,4	4,5	9,0	6,8	7,2	10,0
13	13,7	18,2	12,9	9,6	15,1	11,6	11,9	11,7	22,2
14	8,7	14,1	24,3	9,2	12,2	16,2	14,4	11,4	14,1
15	11,0	12,8	16,8	9,6	10,8	8,0	19,3	7,4	8,9

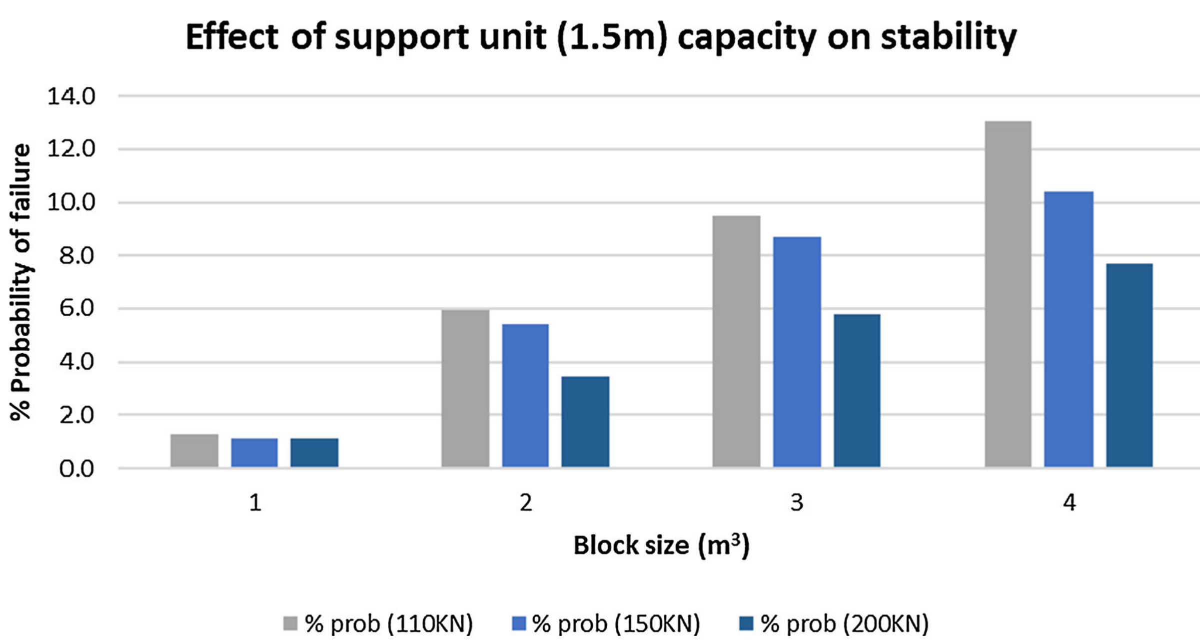

Fig. 20 Effect of support unit capacity on stability

been documented by several authors (Tyler et al. 1991; Beauchamp et al. 1998; Esterhuizen and Streuders 1998; Dunn et al. 2008; Gumede and Stacey 2007; Stacey and Gumede 2007; Joughin et al. 2012; Chikande and Zvarivadza 2016).

4.4 Stability Analysis of the Excavation with the Change in Support Spacing

A three-dimentional stability analysis and visualisation software pregramme from RocScience known as UnWedge, was used for further stability analysis. This analysis was conducted with the purpose of simulating the effect of change in support spacing on the stability of the excavation. Indeed, the previous results on structural mapping outlining the joint sets detected in the field were utilised, in generating the wedges across the excavation. As already denoted, the task was to see which support spacing will provide long-term stability of the excavation. For argument's sake, the stability of the underground excavation deteriorates with time and is quicker when there is no support system holding the rock units (Sandrone et al. 2007; Sandrone 2008). Indeed it can be seen from the first simulation Figs. 21a and 22a, wherein an underground excavation without a support system was simulated and the FoS of the wedges across the excavation were ranging between 1.3 and 0.8. In this regard, it can be deduced that the surrounding wedges will eventually be dislodged from the excavation boundaries into the excavation. However, further analysis entailed installation of a support system across the excavation with varying support spacing as shown in Figs. 21b–d and 22b–d; Table 4. The results of the simulation have shown some greater improvement with the FoS on the wedges across the excavation as the support spacing reduces. In simple terms, when the support spacing reduces the density of installed support tendons increase, which alone increases the stability of the excavation.

The simulation confirms that in jointed rockmass, short spacing or spacing of 1 m by 1 m could be the best which could be able to provide long-term stability of the excavation. The simulation also confirms the observation and the previous discussion on the block falling and stability of the excavation. It was also noticed that support spacing is the crucial part of stope stability analysis, while the change in tendon length and capacity play some certain roles, but their roles are very limited in this regard.

On the other hand, the northern side of the mining was also simulated, through the integration of the

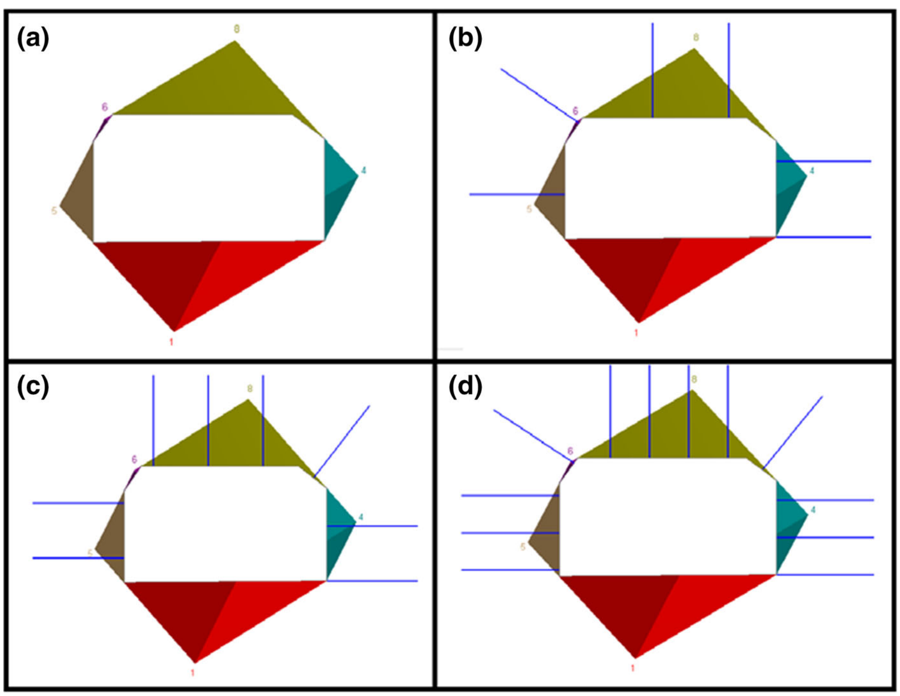

Fig. 21 2D distribution of support units across the excavation on the Southern side of mining **a** an excavation without support unit installed, **b** an excavation with support units installed on 2.0 m × 2.0 m in the opening plane of the excavation **c** an excavation with support units installed on 1.5 m × 1.5 m in the opening plane of the excavation **d** an excavation with support units installed on 1.0 m × 1.0 m in the opening plane of the excavation

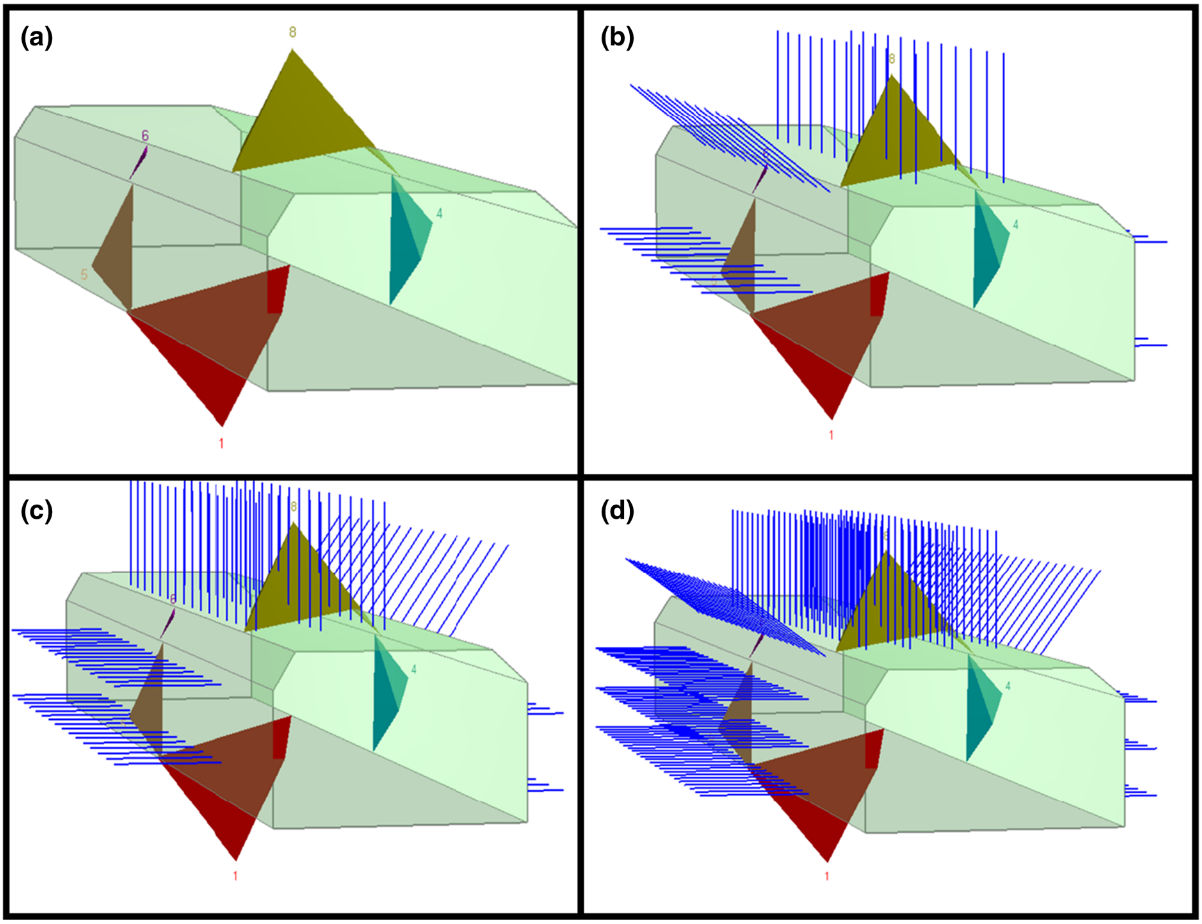

Fig. 22 3D distribution of support units across the excavation on the Southern side of mining **a** 3D view of excavation without support unit installed, **b** 3D view of an excavation with support units installed on 2.0 m × 2.0 m in the opening plane of the excavation **c** 3D view of an excavation with support units installed on 1.5 m × 1.5 m in the opening plane of the excavation **d** 3D view of an excavation with support units installed on 1.0 m × 1.0 m in the opening plane of the excavation

information and finding the correlation among the section of the mines. Indeed, as the field observation and other techniques denoted that the northern side of the mine compose of several geological features which generated large wedges, as such low FoS across the wedges of the excavation has been simulated (see Figs. 23 and 24; Table 5). In simple terms, FoS ranging from 1.7 to 1.02 were simulated, however, the FoS of the mining section do not differ greatly they are indeed closely related. A similar situation has been observed that as the spacing of the support units reduces, the stability of the excavation increases as well.

Furthermore, simulations were done to perform sensitivity analysis on wedge FoS based on the bolt distribution along with the wedge. It was noted that as the spacing of the support units reduces, the number of support units installed across the wedge increases, and as a result the FoS of the wedge increases as well as the stability of the excavation (see Figs. 25 and 26). The above results are confirmed by authors such as (Earl 2007; Dunn et al. 2008; Stacey and Gumede 2007). In this regard, a similar task was conducted in the Northern and Southern sections of the mining.

Based on the results of the simulation, it can be deduced that support spacing has many effects as compared to other aspects associated with the support system along with the wedges. Therefore, the support design is crucial in minimising the instability in

Table 4 Simulated FoS, Apex, and mode of failure of the wedges across the stope/excavation in the Southern side of the mine

Spacing of bolts	Criteria	Roof Wedge (8)	Lower Right Wedge	Lower Left Wedge
No Support	Factor of Safety	1.334	0.893	0.900
	Weight of the wedge (MN)	0.255	0.037	0.030
	Apex Height (m)	1.83	0.82	0.80
	Mode of failure	Falling Wedge	Falling Wedge	Lifting Wedge
1.0 m × 1.0 m	Factor of Safety	1.461	1.035	1.086
	Weight of the wedge (MN)	0.255	0.037	0.030
	Apex Height (m)	1.83	0.82	0.80
	Mode of failure	Falling Wedge	Falling Wedge	Lifting Wedge
1.5 m × 1.5 m	Factor of Safety	1.394	0.964	0.974
	Weight of the wedge (MN)	0.255	0.037	0.030
	Apex Height	1.83	0.82	0.80
	Mode of failure	Falling Wedge	Falling Wedge	Lifting Wedge
2.0 m × 2.0 m	Factor of Safety	1.368	0.929	0.974
	Weight of the wedge (MN)	0.255	0.037	0.030
	Apex Height	1.83	0.82	0.80
	Mode of failure	Falling Wedge	Falling Wedge	Lifting Wedge

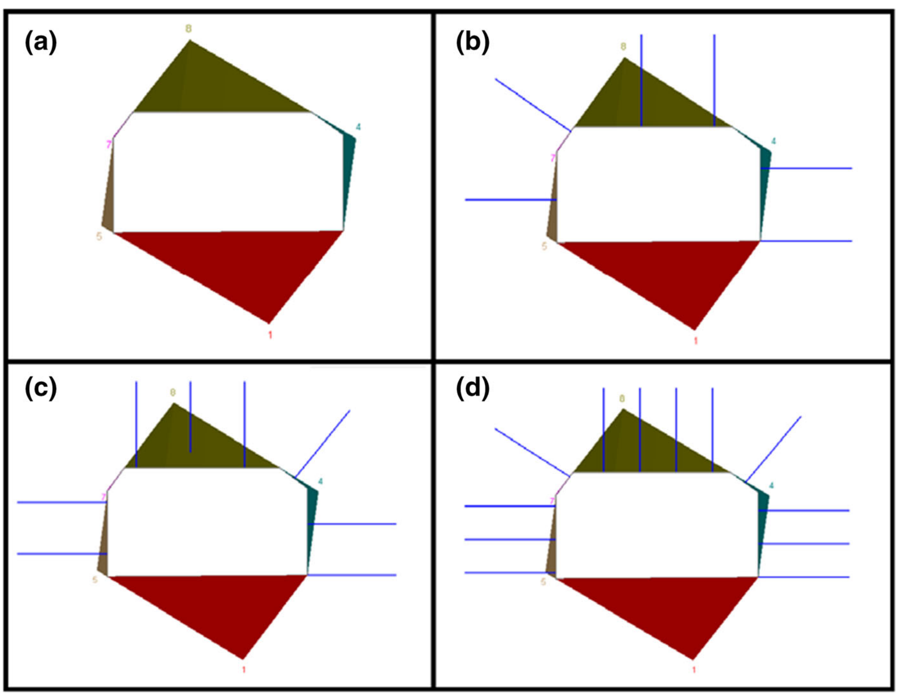

Fig. 23 2D distribution of support units across the excavation on the Northern side of mining **a** an excavation without support unit installed, **b** an excavation with support units installed on 2.0 m × 2.0 m in the opening plane of the excavation **c** an excavation with support units installed on 1.5 m × 1.5 m in the opening plane of the excavation **d** an excavation with support units installed on 1.0 m × 1.0 m in the opening plane of the excavation

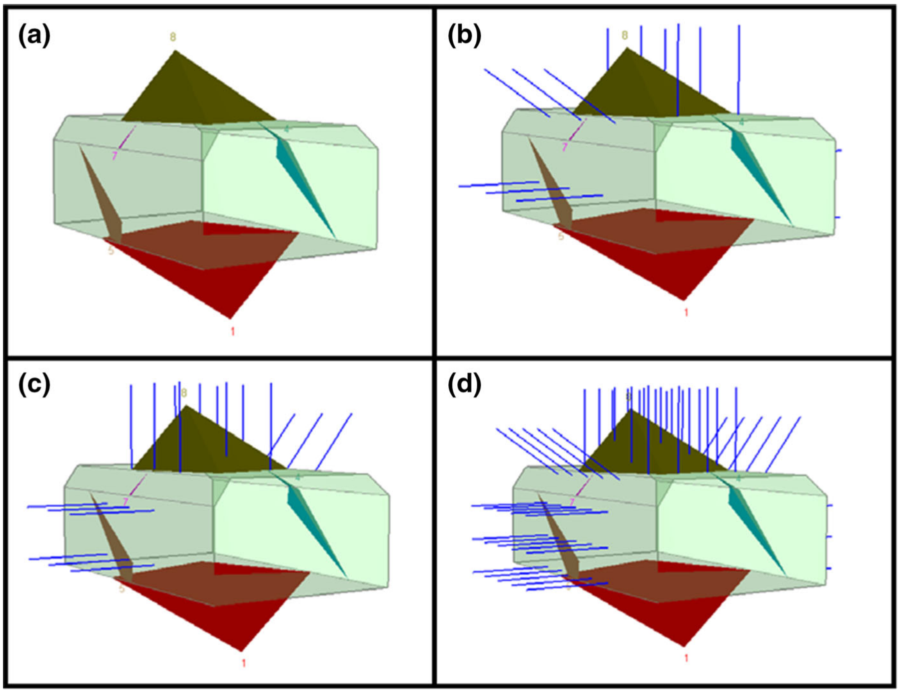

Fig. 24 3D distribution of support units across the excavation on the Northern side of mining **a** 3D view of excavation without support unit installed, **b** 3D view of an excavation with support units installed on 2.0 m × 2.0 m in the opening plane of the excavation **c** 3D view of an excavation with support units installed on 1.5 m × 1.5 m in the opening plane of the excavation **d** 3D view of an excavation with support units installed on 1.0 m × 1.0 m in the opening plane of the excavation

Table 5 Simulated FoS, Apex, and mode of failure of the wedges across the stope/excavation in the Northern side of the mining

Spacing of bolts	Criteria	Roof wedge (8)	Lower right wedge	Lower left wedge
No Support	Factor of Safety	1.734	1.075	1.027
	Weight of the wedge (MN)	0.095	0.037	0.002
	Apex Height (m)	1.89	0.82	0.29
	Mode of failure	Falling Wedge	Falling Wedge	Lifting Wedge
1.0 m × 1.0 m	Factor of Safety	1.848	1.131	1.154
	Weight of the wedge (MN)	0.095	0.037	0.002
	Apex Height (m)	1.89	0.82	0.29
	Mode of failure	Falling Wedge	Falling Wedge	Lifting Wedge
1.5 m x1.5 m	Factor of Safety	1.791	1.075	1.090
	Weight of the wedge (MN)	0.095	0.037	0.002
	Apex Height	1.89	0.82	0.29
	Mode of failure	Falling Wedge	Falling Wedge	Lifting Wedge
2.0 m × 2.0 m	Factor of Safety	1.751	1.075	1.057
	Weight of the wedge (MN)	0.095	0.037	0.002
	Apex Height	1.89	0.82	0.29
	Mode of failure	Falling Wedge	Falling Wedge	Lifting Wedge

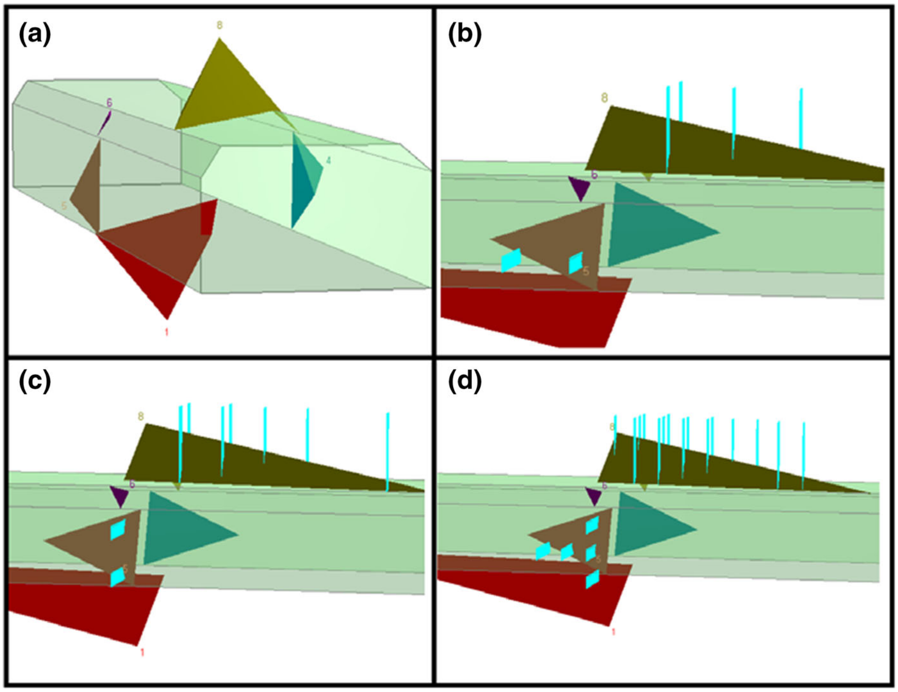

Fig. 25 3D distribution of support units which anchor the wedges across the excavation on the Southern side of mining **a** 3D view of an excavation all wedges denoted and without support, unit installed, **b** 3D view of an excavation with support units which anchored the wedges when installed on 2.0 m × 2.0 m in the opening plane of the excavation **c** 3D view of an excavation with support units which anchored the wedges when installed on 1.5 m × 1.5 m in the opening plane of the excavation **d** 3D view of an excavation with support units which anchored the wedges when installed on 1.0 m × 1.0 m in the opening plane of the excavation

underground excavation. It is also important to note that structural mapping is the critical part of these assessments, therefore the structural mapping data has led to the general conclusion drawn in this study.

5 Conclusions

Fall of ground (FOG) is one of the major hazards in underground mining. Fatalities and injuries in the mining industry are largely due to rock-related hazards. The chrome mine had a FOG management strategy in place comprising of visual observations Ground Penetrating Radar Scanning and Borehole camera inspections. However, loopholes were identified in the system which were the main contributors to recorded FOGs. The main aim of this paper was to propose a FOG management strategy that will minimise the FOG incidents experienced in shallow hard rock mining environments. Two methods were used to determine the overall fallout thickness at the mine. In addition, two joint mapping techniques were used to collect joint data which was later used to determine the joint distribution and failure modes of the wedges formed. Stability analysis was conducted through simulation of a factors of safety for both South and North mining sections.

The fallout thickness evaluation from both the historical data and brow thickness ranges between 0.9 and 1.0 m. These results conform with joint distribution analysis from J-block wherein about 80% of the blocks formed are less than 1 m^3. The probability of 1 m^3 failing in between support is high in both the North and South sections. As the block size increases, the probability of support failure also increases. Block failure modes detected in the North section were dropout for blocks less than 1 m^3 and rotation for blocks greater than 1 m^3. In contrast, the South section modes of failure detected were single plane, drop out, and rotation. The difference in block failure modes is due to the varying joint properties in each section. From the results, it can be seen that support

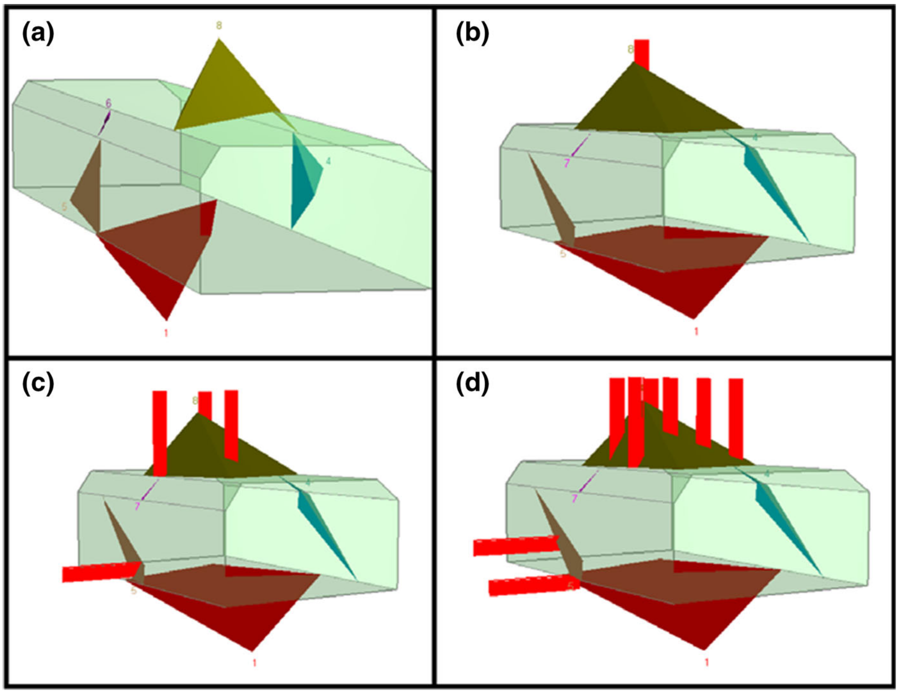

Fig. 26 3D distribution of support units which anchor the wedges across the excavation on the Northern side of mining **a** 3D view of an excavation all wedges denoted and without support, unit installed, **b** 3D view of an excavation with support units which anchored the wedges when installed on 2.0 m × 2.0 m in the opening plane of the excavation **c** 3D view of an excavation with support units which anchored the wedges when installed on 1.5 m × 1.5 m in the opening plane of the excavation **d** 3D view of an excavation with support units which anchored the wedges when installed on 1.0 m × 1.0 m in the opening plane of the excavation

spacing is a critical aspect of excavation safety and stability.

Further analysis was conducted with the purpose of simulating the effect of change in support spacing on the stability of the excavation. Indeed, support spacing is important and the simulation assisted in outlining and identifying the support spacing that will provide long-term stability of the excavation based on the FoS. The results of the simulation have shown some greater improvement with the FoS on the wedges across the excavation as the support spacing reduces. When the support spacing reduces, the density of installed support units increases, consequently resulting in improved excavation stability. Simulation results showed that 1.0 m × 1.0 m support spacing is ideal for these ground conditions.

The structural/joint mapping is the critical part of all evaluations carried out in this study, and should therefore be conducted with precision and accuracy. The analysis of the joint mapping data has led to the general conclusion drawn in this study. Probabilistic analysis can be used to evaluate the probability of rock falls, and support design for stability enhancement. Support system evaluation is deemed essential for excavation stability. Large key blocks require support units with resistance greater than their weight. It is advised that the mine introduce probabilistic analysis to improve the FOG management system and also design support for highly jointed ground conditions.

Acknowledgements The authors would like to say rest in peace to those who lost their lives to falls of ground. This work is based on a project study that was submitted to the University of the Witwatersrand, Johannesburg in partial fulfillment of a Master of Science in Engineering. The authors would like to thank Dr. Fhatuwani Sengani for his efforts, inputs, knowledge, comments, suggestions and advice given throughout the writing process of this paper.

Declarations

Conflict of interest The authors wish to confirm that there are no known conflicts of interest associated with this publication. Moreover, no financial support was offered to influence the outcome of this work.

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