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

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Abstract

Joint properties and useful mapping techniques are key to the fall of ground management in underground mining. The study analysed the fall of ground management techniques at the mine with a view to identify the causal factors of the falls of ground. This paper practically demonstrates how two mapping techniques were used to obtain joint data. A brief description of geological discontinuities at the study area is given in the paper. Joint mapping was carried out in both the North and South sections of the mine. Procedures followed when collecting joint data are also provided. The collected joint data 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 1m^3 in size. Results show that larger blocks are more likely to fail through the rotation. Furthermore, to prevent small blocks from falling between support units, areal coverage is suggested in heavily jointed rock masses. Probabilistic analysis can be used to evaluate the probability of rock falls, and support design for stability enhancement. The research noted that at the centre of effective falls of ground management are accurate and precise structural/joint mapping. This research is part of an MSc Engineering study.

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 & Leger, 1988; Leger, 1991; Mark & Iannacchione, 2000; Roberts, et al., 2001; Koldas, 2001; Gumede & Stacey, 2007; Vorster & Franklin, 2008; Ferreira & Minova, 2012). Indeed, there has been a gradual reduction 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 advanced technology within the mining industry (Stacey & Gumede, 2007; Maiti & Khanzode, 2009; Mark, et al., 2011; Teleka, et al., 2012). Furthermore, FOG management is a continuous assessment wherein rock engineering specialists are brought together to come up with solutions regarding FOG. Ryder & Jager (2002) described a rock as a complex engineering material of which its behaviour is influenced by numerous factors. 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 & 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 &

Ryder, 1999; Ryder & 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 characterized 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 fall of ground 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.

Fall of ground 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 & 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 10m x 3m and 3m x 3m, maintaining a 6m 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 (2.1m) spaced at 1.2m x 1.2m whereas the previous support system had 1.8m long roof bolts spaced at 1m x 1m (Chikande & Zvarivadza, 2016). Other examples of fall of ground management strategies are evident in studies by scholars such as Joughin (2008); Vogt, et al., (2010); Joughin, et al., (2012); Esterhuizen (2014); Joughin, et al., (2016); Chikande & Zvarivadza (2018).

In mining, rockfall-related hazards are forever present. The fact that falls of ground management strategies have been put in place means that fall of ground is a major concern. Fall of ground management is an important aspect in mining; hence, 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. The study helps to improve the current system, and consequently, combat falls of ground 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 Layouts

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; and Eales & 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 characterized by geological complexities such as faults, potholes, and intrusions creating dykes and sills (Uken, 1998; Robertson, 1977; and Roberts & 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 & Guler, 1998; and 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 (TMM) in the form of bord and pillar (see Fig. 1). 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 & Wood, 1987; Hoek, et al., 1998). Generally, support design for an underground excavation is summarized 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 stabilize the hanging wall by providing sufficient support resistance to avoid failure (i.e., capacity must be greater than the demand).

3. Research Approaches

The main aim of this study was to identify loopholes within the current ground control system 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 the 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 Falls of Ground (FOG) database of the mine was reviewed in order to develop the 95 percentile FOG empirical design threshold by Jager & 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

In order to understand how a jointed rock mass behaves, it is important to have a comprehensive description of the factors that characterize it. These factors include joint shape, size, orientation, spacing, length, and strength. Joint shape and size are difficult to measure in two-dimension. The most important factors that describe rock joints according to Priest & 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 falls of ground 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.

Joint data can be recovered using numerous techniques. The most common techniques are spot mapping, scanline mapping and cell or area mapping. Orientated drill core can also be interpreted to obtain joint. However, 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 & Brown, 2004). All joints intersecting the tape are mapped (Fig. 2).

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 30m long measuring tape was suspended straight between two points in a working panel mining Southwards (Fig. 3). 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. This is 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 & 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. 4.

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 10m panels and 12m x 12m pillars (Fig. 5). 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 visualize and analyze structural data in 2D using the same technique as that applied in manual stereonet (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 & Stacey (2007).

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 Fig. 6 and Fig. 7.

4. Results And Discussions

Installed support improves safety and stability in excavations, therefore it is critical to understand the required support resistance to stabilize the excavation. This section clearly outlines how the fall out thickness 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 0.95m (Fig. 8). This approach was adopted from Jager & Ryder (1999) who estimated fallout thickness using historical data from falls of ground.

Fallout thickness obtained in both sections is within the 90-100cm thickness range (Figs. 9 and 10). In 2014, the average fallout thickness was 0.7m. However, results show that currently the average fall out thickness is 0.95m. The difference in the two average fallout thicknesses, prove that ground conditions are not always the same throughout the mine. The current ground condition is characterized by many geological discontinuities unlike in 2014. Hence, the current fallout thickness is approximately 1.0m. 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-defined using fallout thickness (Chikande & Zavarivadza; 2018). Furthermore, the criterion used in deducing support resistance is highly dependent on the tributary area as contested by Stacey & 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 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 & 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. 11. Three major joint sets are identified. The dominant joint set (J1) is orientated at $67^{\circ}/107^{\circ}$, joint set 2 (J2) orientated at $83^{\circ}/23^{\circ}$ and lastly (J3) flat dipping at $54^{\circ}/266^{\circ}$. Joint orientation results from the South section also show three joint sets (Fig. 12). The set orientation for J1, J2 and J3 are $61^{\circ}/280^{\circ}$; $85^{\circ}/25^{\circ}$; and $47^{\circ}/114^{\circ}$.

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 summarized in Table 1. The summarized data can now be used to predict rock fall probability together with the evaluation of support.

Table 1
Overall joint data for the two sections

Section	Joint set Number	Dip (deg)	Dip Direction (deg)	Range (deg)	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

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 & Thompson, 1992; Windsor, 1999). Indeed J-Block satisfies both these aspects during 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 2m^3 to 15m^3 . The probability of 1m^3 blocks falling in-between support is 6.3%. For blocks greater than 1m^3 , the probability decreases, but the probability that supports will fail increases (Fig. 13). The probability that supports will fail as the block size increases ranges between 1% and 12%.

Key block size distribution in the South section (Figure. 14) shows that 72% of blocks formed are less than 1m^3 . As the block size increases, the key block size distribution also decreases. The probability of 1m^3 blocks falling in-between support is 77%. As the block size increases, the probability of blocks falling between supports decreases. However, the probability of blocks greater than 1m^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 blocks.

Area outcome distributions show that most block failures are localized in the center of the excavation (Figs. 15 and 16). 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 & 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 section (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 1m^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. 17. The South section is more likely to experience various failure modes, supporting higher failure probabilities mentioned previously. Drop out failure probability is high (80%) for 1m^3 block sizes and decreases with increasing key block size (47–7%). As shown in Fig. 18, 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. 19). The larger the block size, the more likely is to fail by rotation (0–59%).

Stability mode results (Fig. 20) from both sections are similar. All block sizes are stabilized 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 & Swart (2001), support unit length must cover the full fallout thickness, plus 200mm to anchor the parting plane and also incorporate 100mm protruding length. In addition, the total tendon length is maximized if the tendon is installed at right angles to the hanging wall (Stacey & 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 stabilize the excavation. Analyses conducted in preceding sub-sections showed that failure is most likely to take place

in between support units for 1m³ key blocks. This means that support spacing significantly affects the support resistance to stabilize these key blocks.

Table 3 summarizes the effect of support spacing on 1m³ block size failure probability. The same support units (non-grouted 1.5m long roof bolts @ 110kN) 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 3. Support failure may occur if the support capacity is 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 2m x 2m as shown in Table 4. 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. 21).

Table 3
Effect of support spacing on the probability of failure.

Support length (m)	2.0 x 2.0		1.5 x 1.5		1.0 x 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

Table 4
Effect of support length and support capacity on stability.

Effect of Support length and Capacity on 2m x 2m constant spacing									
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		
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

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 & Ryder (1999) was found to be 0.95m. On the other hand, a high percentage (80%) of the overall key blocks formed are 1m³. 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 been documented by several authors (Tyler, et al., 1991; Beauchamp, et al., 1998; Esterhuizen & Streuders, 1998; Dunn, et al., 2008; Gumede & Stacey, 2007; Stacey & Gumede, 2007; Joughin, et al., 2012; Chikande & Zvarivadza, 2016).

4.4. Stability Analysis of the excavation with the change in support spacing

Further 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 utilized, 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. 22a and 23a, wherein an underground excavation without a support system was simulated and the Safety Factor of the wedges across the excavation were ranging between 1.3 to 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. 22b-d and 23b-d and Table 6. 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 1m by 1m 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 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 Safety Factor across the wedges of the excavation has been simulated (see Figs. 24 and 25 and Table 7). 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 Safety Factor of the wedge increases as well as the stability of the excavation (see Figs. 26 and 27). The above results are confirmed by authors such as (Earl, 2007; Dunn, et al., 2008; Stacey & 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 minimizing the instability in 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.

Table 6
Simulated FoS (Safety Factor), 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 x 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 x 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 x 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

Table 7

Simulated FoS (Safety Factor), 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 x 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 x 1.5 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 x 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

5. Conclusions

Fall of ground 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 fall of ground management strategy in place, however, loopholes were identified in the system which were the main contributors to recorded falls of ground. The main aim of this paper was to propose a fall-of-ground management strategy that will minimize the fall of ground incidents experienced in shallow hard rock mining environments.

A review of the current FOG management system was conducted with the aim of identifying the main causes of FOGs at the mine. 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 0.9m and 1.0m. These results conform with joint distribution analysis from J-block wherein about 80% of the blocks formed are less than 1m^3 . The probability of 1m^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 1m^3 and rotation for blocks greater than 1m^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 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 1.0m by 1.0m 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.

Declarations

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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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Tables

Tables 2 and 5 are not available with this version.

Figures

Figure 1

Schematic diagram of chrome mining layout

Figure 2

Scanline mapping (Zhang, 2004)

Figure 3

Scan line tape aligned to the centerline of the panel.

Figure 4

Schematic diagram of window mapping zone

Figure 5
Mine working layout.

Figure 6

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

Figure 7

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

Figure 8

Average fallout thickness estimated from historical FOG data.

Figure 9

Figure 10

Symbol	Feature
○	Pole Vectors

	Color	Dip	Dip Direction	Label
Mean Set Planes				
1m	Green	67	107	
2m	Blue	83	23	
3m	Red	54	266	

Plot Mode	Pole Vectors
Vector Count	52 (52 Entries)
Hemisphere	Lower
Projection	Equal Angle

Figure 11

Overall joint orientation in the North Section

Figure 12

Overall joint orientation in the South Section

Run Heading: NORTH SECTION
 Number of keyblocks generated : 10000
 File: North Secion.JRE

Average hazard per 1000.0 m² : 0.0005
 Failure volume per 1000.0 m² : 1.27
 Area modelled : 13324571.00 m²

Print...

Area results

Failure modes

Stability modes

Slide angles

Save

Quit

Figure 13

Key block size distribution for North

Figure 14

Key block size distribution for South

Figure 15

Area modelled in North Section.

Figure 16

Area modelled in South section.

Figure 17

Block failure modes in the North

Figure 18

Block failure modes in the South

Figure 19

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

Figure 20

Stability modes in North (left) and South (right) section

Figure 21

Effect of support unit capacity on stability

Figure 22

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 2.0m x 2.0m in the opening plane of the excavation (c) an excavation with support units installed on 1.5m x 1.5m in the opening plane of the excavation (d) an excavation with support units installed on 1.0m x 1.0m in the opening plane of the excavation

Figure 23

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 2.0m x 2.0m in the opening plane of the excavation (c) 3D view of an excavation with support units installed on 1.5m x 1.5m in the opening plane of the excavation (d) 3D view of an excavation with support units installed on 1.0m x 1.0m in the opening plane of the excavation

Figure 24

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 2.0m x 2.0m in the opening plane of the excavation (c) an excavation with support units installed on 1.5m x 1.5m in the opening plane of the excavation (d) an excavation with support units installed on 1.0m x 1.0m in the opening plane of the excavation

Figure 25

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 2.0m x 2.0m in the opening plane of the excavation (c) 3D view of an excavation with support units installed on 1.5m x 1.5m in the opening plane of the excavation (d) 3D view of an excavation with support units installed on 1.0m x 1.0m in the opening plane of the excavation.

Figure 26

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 x 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 x 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 x 1.0 m in the opening plane of the excavation.

Figure 27

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 x 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 x 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 x 1.0 m in the opening plane of the excavation.