Microseismicity and its associated rockburst damage constitute a great risk to underground mining operations at Mufulira mine. This mine is located in the Copperbelt province of Zambia, hosting sulphide orebodies, which are rich copper and cobalt mineralization. The rock units are metamorphic sedimentary rocks of the Neoproterozoic Katanga Supergroup. Mufulira mine has been in operation since 1933, mainly in copper production. Serious events of microseismicity and the associated rockburst damage began to be recorded at the mine since the inception of 1970s. In January, 2018, the mine recorded a seismic event of magnitude 2.8, which caused severe rockburst damage in the mining drive at a depth of 1440 meters below surface. The capacity to measure microseismic movements at Mufulira mine was first built in 1991, but the system had low efficiency in providing adequate quantitative parameters of seismic events. Therefore, in December 1994, an integrated seismic system (ISS) was purchased from South Africa and installed to record mining induced seismicity at Mufulira mine. The system deteriorated with time, but in 2006 the mine made efforts to update the system through the addition of new sensors. This paper analyzes microseismicity at Mufulira mine based on trends of seismic events from 2016 to 2018, and proposes possible remediation measures to seismic damages. The overall aim of this study is to provide knowledge for management's decisions to improve safety in seismic prone underground mines, so as to sustain production, and enhance consistent generation of revenue.
1. Introduction

1.1 Definition of rockburst

Rockburst is fracture of rockmass and violent expulsion of rock fragments from the surface of an underground excavation. There are many definitions of rockburst available from various authors. In 2012, Kaiser and Cai proposed that rockburst is damage to an excavation that occurs in a sudden or violent manner and is associated with a seismic event. Sheng-Jun et al. (2016) proposed that rockburst is a sudden brittle failure induced by high stresses. Tian-Hui et al. (2018) proposed that rockburst is a sudden failure of rockmass, which is accompanied by sudden release of strain energy, and he also discussed a number of definitions from various researchers. However, three basic aspects of rockburst can be concluded from these definitions, which are sound, rock fracturing and violent expulsion of rock. Rockburst is a serious danger in underground working environments, having a possibility of leading to loss of production, damage of equipment, injuries and sometimes death of workers. Rockburst events are very common in underground mines, both in hard rock metal mines and coal mines. The trend of rockburst occurrence increasingly becomes common as an underground mine advances to greater depth.

1.2 Classification and mechanism of rockburst

Many contributions have come from various researchers to explain different types and mechanisms of rockburst, and possible causes leading to rockburst. For instance, Manchao et al. (2012) and Muller (1991) mentioned different types of rockburst, which he classified as strainburst, fault-slip burst and pillar burst. Strainburst occurs as a mechanism of relief of strain energy stored in highly stressed zone in the surrounding rockmass environment of an excavation. Fault-slip burst is associated with shear failure along the surface of major discontinuities, while pillar burst is a type of rockburst that occurs in an underground pillar, often influenced by pillar dimensions and stress conditions. According to Ben-Guo et al. (2015), the mechanism of rockburst can correspond to (1) strain relaxation of surrounding rockmass in newly excavated areas, and (2) the dynamic response of rockmass to blast induced waves. Part of the strain energy stored in highly stressed zones is released in form of shear movements along weak planes of discontinuities, while the other part of it is converted into kinetic energy, which eventually leads to rock expulsion from the surface of an excavation. In such case, the position of source triggering rockburst and the position of rockburst damage may be separated by distance and time Fei et al. (2018). Kaiser and Cai (2012) categorized rockburst damage in three classes, (1) bulking due to fracturing, (2) Ejection of rock due to seismic energy transfer, and (3) rockfall induced by seismic vibration. A good understanding of such damage mechanisms could be very helpful when considering mitigation measures for a rockburst prone area.

1.3 Monitoring and prediction of rockburst

Currently there are several methods of monitoring and predicting rockburst occurrence. Research and industrial practice on the estimation and prediction of rockburst mainly focuses on theoretical analysis, laboratory test, numerical simulation and field measurement, as proposed by Fei et al. (2018). Field measurement includes the use of a microseismic monitoring system. In this paper, Displacement Discontinuity Method (DDM), a numerical modeling method was applied to investigate induced stress levels caused by mining extraction at the site of M2.8 rockburst at Mufulira mine site of Mopani Copper Mines in Zambia.

2. Project overview

2.1 Site Description

Mufulira mine is located in the Copperbelt province of Zambia in a town of Mufulira. The mine lies on latitude 12° 32’ 22” South and longitude 28° 14’ 10” East, at an elevation of 1250 m above sea level. The mine is one of the biggest underground mines in Zambia, and its major products are copper and cobalt. The license area for Mufulira mine consists of Mufulira West Portal, Mufulira East Portal and the main Mufulira mine which comprises of the upper, central and deeps section. This research was conducted on the main Mufulira mine in the deeps section.

![Fig. 1. Site layout of main Mufulira mine at 1440 meter level.](image1)

This research began after a rockburst that was associated with a seismic event of magnitude 2.8 occurred at 1440 meter level underground. This incident occurred in the year 2018 on 16 January. The site where this rockburst occurred is indicated by a red dot in Fig. 1 and the picture of this area is as shown in Fig. 2.

![Fig. 2. Site of rockburst at 1440 meter level underground (date taken: 15 March, 2018).](image2)

The mine employs Mechanized Continuous Retreat mining method, which is a variant of Sublevel Open Stoping. Fig. 3 shows a typical layout of Mechanized Continuous Retreat mining method. The method involves fan drilling of long holes from the roof of a sublevel. The sublevels are spaced at intervals of 17 m. The layout of the mine consists of access crosscuts from the decline to the orebody. At the end of these access crosscuts, a footwall drive is developed, as illustrated in Fig. 3(d).
2.1 Typical Mechanized Continuous Retreat Mining method at Mufulira mine

The footwall drive and the mining drive are linked by multiple orebody crosscuts, which are spaced at 50 m apart in the mining level (drilling level) as shown in Fig. 3(b). However, at the drawing level, where blasted ore is collected from, as shown in Fig. 3(d), the orebody crosscuts are spaced at 25 m apart. In terms of dimensions, the mining drive, footwall drive and crosscuts are all 4 m in height and 4 m in width. When fan drilling is performed on two sublevels and the blasted ore is collected from the bottom sublevel referred to as drawing level, the method is known as Mechanized Continuous Retreat 2 (MCR2). The typical layout shown in Fig. 3 is for MCR2. However, when both fan drilling and ore collection are performed only on one sublevel, then the method is known as Mechanized Continuous Retreat 1 (MCR1). The design of the stope leaves a 3 m pillar at the roof as indicated on Fig. 3(a) known as the chain pillar. According to Immanuel and Mutambo (2016), the purpose of the chain pillar is to retain the broken hanging wall material from upper levels and thus prevents ore dilution to the blasted ore.

2.2 Seismic monitoring system

In an effort to predict rockburst events, Mufulira mine employed a microseismic monitoring system in 1994, which has evolved into a number of stages of improvements until today. The primary purpose of this system is to measure microseismicity, and the data obtained is then used to predict potential unstable areas within the mine. These unstable areas are then supported or reinforced to mitigate the risk of damage. Fig. 4 shows the installation units for the microseismic monitoring system at Mufulira mine, which are installed into a network structure from underground to surface. Parts of the results obtained from this system are as shown in Fig. 5, which is a distribution of hypocenters with reference to the mine spatial coordinate system.

2.3 Geological features

Stretching from south-west to north-east of Mufulira mine, the major geological units comprise of a series of the basement complex, the footwall quartzite, the mineralized series, the hanging wall formation, which predominantly contains the lower argillaceous quartzite, and finally the carbonate rocks. The mineralized series hosts three orebodies of Mufulira mine, A, B and C orebody. The orebodies are bedded or of stratiform type (Brandit, 1962), and they all dip at an average of 45°.
3. Rock properties

3.1 Geological and geotechnical mapping

Geological mapping was carried out in the mining drive at 1457 meter level. The location of the rockburst event under study is at 1440 meter level, but at the time of the rockburst it was not safe to carry out any geological or geotechnical mapping of joints. Therefore, an area in close proximity to the rockburst site was mapped, at 1457 meter level, in the mining driving. The results are given in Fig. 6 below.

\[ RQD = 100(\lambda t + 1)e^{-\lambda t} \]  
(1)

Where; \( \lambda \) = Fracture frequency (It is the reciprocal of the mean joint spacing)

\[ t = \frac{2}{(\lambda_{\text{max}} - \lambda_{\text{min}})} \times \ln \frac{\lambda_{\text{max}}}{\lambda_{\text{min}}} \]  
(2)

Hudson and Harrison (1997) proposed a method of calculating RQD based on fracture frequency, as shown by equation (1). The method uses a threshold value \( t \), which is calculated as shown in equation (2). The threshold value makes RQD values more sensitive for large spacing values. Using this method, an RQD value of 78.5 was obtained. The RQD was then used to estimate the modulus of elasticity of the rockmass using the modulus of elasticity of a sample specimen. Zhang and Einstein (2004) explained this method as provided in Fig. 7. Equation (3) was then used to estimate the tangent modulus ratio, shown in Table. 1.

\[ \frac{E_m}{E_r} = 10^{0.10186RQD-1.91} \]  
(3)

\( E_m \): Tangent modulus of rock mass  
\( E_r \): Tangent modulus of rock specimen
Table 1. Estimated RQD and tangent modulus ratio

<table>
<thead>
<tr>
<th>λ</th>
<th>λ&lt;sub&gt;max&lt;/sub&gt;</th>
<th>λ&lt;sub&gt;min&lt;/sub&gt;</th>
<th>t</th>
<th>RQD</th>
<th>E&lt;sub&gt;m&lt;/sub&gt;/E&lt;sub&gt;r&lt;/sub&gt;</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.76</td>
<td>1.67</td>
<td>0.072</td>
<td>3.93</td>
<td>78.5</td>
<td>0.35</td>
</tr>
</tbody>
</table>

3.2 Sample preparation

Cylindrical core rock specimens were prepared from both footwall quartzite and orebody quartzite, having the dimensions of diameter 30 mm and varying length of 60 mm and 30 mm. However, in this paper, only samples with length 60 mm are described. Two weeks before uniaxial compression test was carried out, the samples were kept in containers with 100% relative humidity and a temperature of 295K. This arrangement is as shown in Fig 8.

![Fig. 8. Arrangement for rock specimens, (a) inside the container, (b) sealed container with 100% humidity.](image)

3.3 Uniaxial Compression Test

Two weeks after samples were kept at 100% relative humidity and a temperature of 295K, uniaxial compression test was conducted on the specimens (Figure 9) at a strain rate of 10<sup>-4</sup> s<sup>-1</sup>, using an Instron loading frame. The results of this test are provided in Table 1 and Table 2. However, the axial strain (ε') was corrected as follows (Alam et al., 2014).

\[
\varepsilon' = \varepsilon - D\sigma
\]

\[
\varepsilon = \varepsilon - \varepsilon_{50} + \varepsilon_{50}, \sigma \geq \sigma_{50}
\]

\[
D = \frac{1}{E_{50}} - \frac{1}{E_{50}}
\]

where σ is axial stress, E<sub>50</sub> is the stroke-based 50% tangent modulus, E<sub>50</sub> is the strain gauge-based 50% tangent modulus, ε<sub>50</sub> is ε at the 50% stress level, and ε<sub>50</sub> is ε at the 50% stress level.

![Fig. 9. Experiment set up for UCS test, (a) specimen and extensometer arrangement, (b) Instron loading frame.](image)

Table 2. Physical properties of orebody quartzite

<table>
<thead>
<tr>
<th>OBQ Sample No.</th>
<th>UCS (MPa)</th>
<th>Tangent modulus (GPa)</th>
<th>Poisson's ratio</th>
<th>Critical compressive strain (10&lt;sup&gt;-2&lt;/sup&gt;)</th>
<th>Critical extensile strain (10&lt;sup&gt;-2&lt;/sup&gt;)</th>
</tr>
</thead>
<tbody>
<tr>
<td>3</td>
<td>259</td>
<td>69.2</td>
<td>0.192</td>
<td>0.419</td>
<td>-0.193</td>
</tr>
<tr>
<td>4</td>
<td>223</td>
<td>70.9</td>
<td>0.280</td>
<td>0.340</td>
<td>-0.315</td>
</tr>
<tr>
<td>7</td>
<td>265</td>
<td>69.8</td>
<td>0.216</td>
<td>0.491</td>
<td>-0.254</td>
</tr>
<tr>
<td>11</td>
<td>263</td>
<td>67.9</td>
<td>0.276</td>
<td>0.434</td>
<td>-0.177</td>
</tr>
<tr>
<td>Average</td>
<td>253 ± 20</td>
<td>69.5 ± 1.3</td>
<td>0.241 ± 0.044</td>
<td>0.421 ± 0.062</td>
<td>-0.235 ± 0.063</td>
</tr>
</tbody>
</table>

![Fig. 10. Typical stress-strain curves, (a) for an orebody quartzite sample, (b) for a footwall quartzite sample.](image)
4. Numerical analysis

Changes in stress levels around the rockburst site were investigated using the Displacement Discontinuous Method (DDM). This method was originally developed by Crouch (Crouch & Fairhurst, 1973, Crouch, 1976). It is a boundary element method that was specially developed for tabular mining problems. At Mufulira mine, the thickness of the orebody is rather thicker if compared to usual cases in tabular mining, such as longwall coal mining. However, DDM can still be used to estimate stress redistribution at a wider scale mine. More detailed conceptual equations for this method can be found in Fujii et al. (1997) and Fujii et al. (2001). The vertical component of stress $\sigma_v$ was assumed to be overburden pressure, and was calculated by equation (7). The horizontal component was however calculated by equation (8).

$$\sigma_v = \gamma h$$  \hspace{1cm} (7)

$$\sigma_h = \sigma_r + \frac{V}{1-V}$$  \hspace{1cm} (8)

$$\sigma_1 = q_u + C\sigma_r$$  \hspace{1cm} (9)

$$C = \frac{(1 + \sin \Phi)}{(1 - \sin \Phi)}$$  \hspace{1cm} (10)

where $\gamma$ is unit weight of rock, $h$ is depth of the overburden, $V$ is Poisson's ratio, $\sigma_v$ is vertical stress component, $\sigma_H$ and $\sigma_h$ are maximum and minimum horizontal stress component, $q_u$ is uniaxial compressive strength, $C$ is cohesion, $\sigma_1$ is maximum principal stress, $\sigma_3$ is minimum principal stress, $\Phi$ is angle of internal friction.

4.1 Numerical procedure

The input model was used as represented in Fig. 11. This model was divided into square elements, each element represented by either the number zero (0) or one (1) as shown in the figure. The number 1 represents unmined areas, while 0 represents mined out areas. The length of the model consists of 90 elements, and its width consists of 24 elements. The size of each element is 3.33m in length and 3.33m in width. This model is based on the mine layout at Mufulira mine as of February, 2018. The model is extracted from 1407 meter level to 1457 meter level underground in depth, and along mining drives from mining block 62 to mining block 65 of the main mine, in the deeps section. Table 4 shows the input parameters. From the table, the value of orebody tangent modulus of elasticity was obtained by using equation (3).

![Input model for the numerical analysis](image)

Fig. 11. Input model for the numerical analysis [layout in form of numbers; 0 representing mined out areas, and 1 representing unmined areas]

Table 4. Input parameters for numerical analysis

<table>
<thead>
<tr>
<th>Density (N/m^3)</th>
<th>Poisson's ratio</th>
<th>Orebody tangent modulus of elasticity (GPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>27000</td>
<td>0.22</td>
<td>24.3</td>
</tr>
</tbody>
</table>

4.2 The effect of face advance

As mine extraction progresses, induced stresses begin to increase around the excavation. It was observed that stress levels at the site of the rockburst increased with face advance (Fig. 12). The rockburst event took place after a distance of 90 m was extracted along the mining drive at 1440 meter level. The highest stress concentration can be seen in the chain pillars between 1423 meter level and 1457 meter level.

![Changes in distribution of normal stress due to face advance](image)

Fig. 12. Changes in distribution of normal stress due to face advance [A: normal stress above the roof at rockburst site, B: normal stress below the floor at rockburst site].
Field investigations revealed a larger number of random joints, which were mainly caused by blasting. Very high stresses were observed in the chain pillars and low stress concentration at the rockburst site during initial stages of mining, but later stress levels gradually increased with mining, which could be what caused the rockburst. Further investigation will be carried out to predict rock fracturing and identify prevention measures of rockburst occurrence in future.

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References


