Investigation of Rockburst in Deep Underground Mines,  
A case study of Mufulira mine, Copperbelt, Zambia

Sinkala, P.  
Faculty of Engineering, Hokkaido University, Sapporo, Japan  
Nishihara, M., Fujii, Y., Fukuda, D. and Kodama, J.  
Faculty of Engineering, Hokkaido University, Sapporo, Japan  
Chanda, E.  
Mopani Copper Mines Plc, Mufulira Mine site, Mufulira, Zambia

ABSTRACT: Mufulira mine has been in operation since 1933. The mine is situated in the Copperbelt region of Zambia, which is predominantly rich in copper and cobalt mineralization. Since the beginning of 1970s, the mine has been recording incidents of rockbursts and applying various efforts to find mitigation measures for rockbursts. Recently, an M2.8 rockburst occurred in the mining drive at 1440 meter level underground on 16 January, 2018. In order to understand the mechanism of the rockburst, three major steps were taken. These were field geotechnical investigation, followed by uniaxial compression tests, and finally stress analysis. Under field investigations, scan-line mapping of joints indicated few major joint sets in the surrounding rock mass to the rockburst location. Stress analysis showed very high stresses 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. It was therefore concluded that fracturing of the relatively intact rock mass around the mining drive under gradual stress increase by mining could be the cause of the rockburst.

1. INTRODUCTION

1.1. Definition of rockburst

Rockburst is fracture of rock mass 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 rock mass, 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 environment, 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 rock mass 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 rock mass in newly excavated areas, and (2) the dynamic response of rock mass to blast induced waves. Part of the strain energy stored in highly stressed zones is released in the 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 (Ben-Guo et al., 2015; Ortlepp and Stacey, 1994). 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 of rockburst-prone sites
Currently there are several methods of monitoring and identifying potential rockburst-prone zones. Research and industrial practice on the estimation and identification of rockburst-prone zones 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.

This paper summarizes three major steps that were applied in order to understand the mechanism of an M2.8 rockburst at Mufulira mine. These were field geotechnical investigation, which was followed by uniaxial compression tests, and finally stress analysis by Displacement Discontinuity Method (DDM). DDM is a numerical modeling method which was applied to investigate induced stress levels caused by mining extraction. This method was further applied to simulate the rockburst.

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. 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, deeps section.

Fig. 1. Site layout of main Mufulira mine at 1440 meter level.

This research began after a rockburst that was associated with a seismic event of magnitude 2.8 occurred at 1440 meter level underground on January 16, 2018 (Figs. 1 and 2).

Fig. 2. Location of rockburst at 1440 mL underground (date taken: 15 March, 2018).

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.

![Diagram of Continuous Retreat mining method](image)

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). 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. If fan drilling is performed on two sublevels and the blasted ore is collected from the bottom sublevel, which is referred to as drawing level, then the method is known as Mechanized Continuous Retreat 2 (MCR2). The typical layout for MCR2 is shown in Fig. 3. However, if 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 in Fig. 3(a), which is 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. Geological features

The major geological units comprise of a series of five different units. Stretching from south-west to north-east of Mufulira mine, these units are: basement complex, footwall quartzite, mineralized series, the hanging wall formation 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). Location of the M2.8 rockburst is in the C orebody, which ranges from 8 m to 10 m in thickness, and has a strike length of 1,200 meters. All the three orebodies dip at an average angle of 45°. The footwall ranges in thickness from 0 to 150 m.

In order to understand the geological characteristic of the rock type in the study area, Fig. 4 shows microscope images of rock samples from the C orebody and the footwall. From the images, the C orebody quartzite showed coarse-grained (arkosic) sandstone severely disturbed and poorly sorted, with the ore forming minerals identified as chalcopyrite, chalcopyrite and Bornite. The footwall quartzite showed characteristics of granular to grained (arkosic) sandstone, which is poorly sorted with weak contacts that are metamorphosed. Some of the minerals observed in the footwall quartzite included quartz, orthoclase, plagioclase, spinel and granite. In both the orebody quartzite and footwall quartzite, sericite and anhydrite were identified as secondary minerals.
Fig. 4. Microscope images of Orebody Quartzite in C-orebody (a, b, c, d) and Footwall Quartzite (e, f, g, h).

2.3. Seismic monitoring system
In an effort to identify seismic hazard, 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 record microseismicity, and the data obtained is then used to identify potential unstable areas within the mine. These unstable areas are either supported or reinforced to mitigate the risk of damage. Fig. 5 shows the installation units for the microseismic monitoring system at Mufulira mine, which are connected to a network structure that is installed from underground to surface. The data obtained from this system is as shown in Fig. 6, which is a distribution of hypocenters with reference to the mine spatial coordinate system. Using Eq. (1), a Gutenberg-Richter relation for the seismic events was represented as shown in part (d) of Fig. 6.

\[
\log (N) = a - bM
\]

where \( M \) is Magnitude of seismic events, \( N \) is the number of seismic events equal or greater than magnitude \( M \). The constant \( b \) is the slope of the relation, and \( a \) is a constant related to the amount of seismic events. In this study, \( b \) was estimated at 1.02, and \( a \) was estimated at 3.57.

Fig. 5. Installation units of the seismic monitoring system [a: Digital Subscriber Line Modem (DSL), b: Analogue Time Unit (ATU), c: Uninterrupted Power Supply (UPS), d: Net Signal Processor (netSP), e: Net Analogue Digital Converter (netADC), f: Digital Subscriber Line Modem (DSL), g: Geophone].

Fig. 6. Distribution of hypocenters at Mufulira mine (from the year 2016 to 2018).
3. FIELD GEOTECHNICAL INVESTIGATION

Scan-line mapping was carried out in the mining drive at 1457 meter level. The M2.8 rockburst occurred 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 at 1457 meter level in the mining drive, which is close to the location of rockburst, was mapped. Results obtained from this mapping are summarized in the stereographic projection shown in Fig. 7. The figure shows approximately four major joint sets and some random fractures. From the mapping exercise, it was concluded that the rock mass surface had fewer major joint sets.

Fig. 7. Stereographic projection of major joint sets at 1457 meter level in the mining drive, from block 62 panel 5 to block 63, mining drive, east-north side.

Core logging was also conducted, using two borehole cores from 1423 and 1457 meter levels. Boreholes for the two cores were drilled horizontally, at 0° dip angle toward the hanging wall. As shown in Fig. 8, EDZ (Excavation Damaged Zone) at 1457 m level is around 10 m, because the level was almost virgin. However, it extends to more than 80 m at 1423 m level due to disturbance associated with mining.

4. ROCK PROPERTIES

4.1. Sample preparation

Cylindrical core rock specimens were prepared from both footwall quartzite and orebody quartzite, having dimensions of diameter 30 mm and length 60 mm for one set of samples. 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 was done in order to keep the specimens in consistency with in-situ conditions of the rock mass. The arrangement described above for rock specimens is as shown in Fig. 9.

Fig. 9. Arrangement for rock specimens, (a) inside the container, (b) sealed container with 100% humidity.

4.2. Uniaxial Compression Test

Uniaxial compression test was conducted on the specimens at a strain rate of 10^-4 s^-1, using an Instron loading frame (Fig. 10). The results of this test are provided in Tables 1 and 2. Fig. 11 shows the typical stress-strain curves obtained from this test. The axial strain ($\varepsilon'$) was corrected as follows (Alam et al., 2014):

$$\varepsilon'' = \varepsilon' - D\sigma \quad (2)$$

$$\varepsilon'' = \varepsilon''_{S0} + \varepsilon_{S0}\sigma \geq \sigma_{S0} \quad (3)$$

$$D = \frac{1}{E_{S0}} - \frac{1}{E_{S0}} \quad (4)$$

where $\sigma$ is axial stress, $E_{S0}$ is the stroke - based 50% tangent modulus, $E_{S0}$ is the strain gauge-based 50% tangent modulus, $\varepsilon''_{S0}$ is $\varepsilon''$at the 50% stress level, and $\varepsilon_{S0}$ is $\varepsilon$ at the 50% stress level. Correction on the UCS for specimen size was applied, using Eq. (5) as proposed by Hoek and Brown (1980).

$$\frac{\sigma_{S0}}{\sigma_a} = \left( \frac{D}{50} \right)^{0.18} \quad (5)$$

where $\sigma_{S0}$ is the UCS of a 50mm diameter specimen, $\sigma_a$ is the UCS of a specimen with different diameter, and $D$ is diameter of the specimen. The tangent modulus was corrected in the same way ($E_{S0}$).

Fig. 8. Variation of RQD with horizontal depth, (a) at 1423 meter level, (b) at 1457 meter level.
Fig. 10. Experiment set up for UCS test, (a) specimen and extensometer arrangement, (b) Instron loading frame.

Fig. 11. Typical stress-strain curves, (a) for orebody quartzite sample, (b) for footwall quartzite sample

Table 1. Mechanical properties of orebody quartzite

<table>
<thead>
<tr>
<th>OBQ Sample No.</th>
<th>( \sigma_n ) (MPa)</th>
<th>( \sigma_{50} ) (MPa)</th>
<th>( E_r ) (GPa)</th>
<th>( E_{50r} ) (GPa)</th>
<th>Poisson's ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>3</td>
<td>259</td>
<td>236</td>
<td>69.2</td>
<td>63.1</td>
<td>0.192</td>
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<tr>
<td>4</td>
<td>223</td>
<td>203</td>
<td>70.9</td>
<td>64.7</td>
<td>0.280</td>
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<tr>
<td>7</td>
<td>265</td>
<td>242</td>
<td>69.8</td>
<td>63.7</td>
<td>0.216</td>
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<tr>
<td>11</td>
<td>263</td>
<td>240</td>
<td>67.9</td>
<td>61.9</td>
<td>0.276</td>
</tr>
<tr>
<td>Average</td>
<td>253 ± 20</td>
<td>230 ± 18</td>
<td>69.5 ± 1.3</td>
<td>63.3 ± 1.1</td>
<td>0.241 ± 0.044</td>
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</table>

Table 2. Mechanical properties of footwall quartzite

<table>
<thead>
<tr>
<th>FWQ Sample No.</th>
<th>( \sigma_n ) (MPa)</th>
<th>( \sigma_{50} ) (MPa)</th>
<th>( E_r ) (GPa)</th>
<th>( E_{50r} ) (GPa)</th>
<th>Poisson's ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>4</td>
<td>275</td>
<td>251</td>
<td>61.3</td>
<td>55.9</td>
<td>0.241</td>
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<tr>
<td>5</td>
<td>254</td>
<td>232</td>
<td>53.0</td>
<td>48.3</td>
<td>0.188</td>
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<td>9</td>
<td>245</td>
<td>223</td>
<td>62.7</td>
<td>57.2</td>
<td>0.244</td>
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<tr>
<td>10</td>
<td>283</td>
<td>258</td>
<td>60.0</td>
<td>54.7</td>
<td>0.247</td>
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<tr>
<td>14</td>
<td>271</td>
<td>247</td>
<td>51.0</td>
<td>46.5</td>
<td>0.188</td>
</tr>
<tr>
<td>Average</td>
<td>266 ± 16</td>
<td>242 ± 14</td>
<td>57.6 ± 5.3</td>
<td>52.5 ± 4.8</td>
<td>0.222 ± 0.031</td>
</tr>
</tbody>
</table>

5. NUMERICAL ANALYSIS

Changes in stress levels around the rockburst site were investigated using the Displacement Discontinuity 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 when compared to usual cases in tabular mining, such as longwall coal mining. However, DDM can still be used to roughly but rapidly estimate the variation in stress distribution with the mining progress at a wider scale. More detailed conceptual equations for this method can be found in Fujii et al. (1997) and Fujii et al. (2001).

5.1. Numerical procedure

The input model was used as represented in Fig. 12. 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, deeps section.

The hydrostatic stress state was assumed as described in Eqs. (6) and (7).

\[
\sigma_v = \gamma h \quad (6)
\]

\[
\sigma_H = \sigma_h = \sigma_v \quad (7)
\]

where \( \gamma \) is unit weight of rock, \( h \) is depth of the overburden, \( \sigma_v \) is vertical stress component, \( \sigma_H \) is maximum horizontal stress, and \( \sigma_h \) is minimum horizontal stress.

The unit weight of rock mass was assumed to be 27 kN/m$^3$. The average Poisson's ratio for footwall quartzite was used, and a tangent modulus of 10.5 GPa was set.
This tangent modulus \( (E_m) \) was determined using Eq. (8), assuming an average RQD of 64.5% for 1423 and 1457 m levels. In this equation, the value of \( E_r \) used was the corrected tangent modulus of rock specimen \( (E_{50r}) \) for footwall quartzite.

\[
\frac{E_m}{E_r} = 10^{0.0186 \times RQD - 1.91}
\]  

(8)

where \( E_m \) is tangent modulus of rock mass, and \( E_r \) is tangent modulus of rock specimen.

5.2. Stress change due to face advance

As mine extraction progresses, induced stresses increase around the excavation (Fig. 13). From the analysis, the highest stress concentration was observed in the chain pillars between 1423 meter level and 1457 meter level. It was observed that stress levels at the site of the rockburst increased with face advance (Fig. 14). The rockburst event took place after a distance of 90 m was extracted along the mining drive at 1440 meter level.

Fig. 14. Changes in distribution of normal stress due to face advance. See Fig. 13 for A and B.

Fig. 13. Changes in distribution of normal stress on the ore body due to face advance from mining block 64 Panel 3 (64P3) to mining block 65 Panel 1 (65P1), covering a distance of 90 m in total (\( d \): face advance).
5.3. Stress levels and failure criterion

The rock mass above and below the ore deposit was divided into cubic rock elements. The stress values for each element were determined and compared with the Coulomb's failure criterion with a tension cutoff;

\[ \sigma_1 = q_u + C\sigma_3 \]  \hspace{1cm} (9)

\[ C = \frac{(1 + \sin \phi)}{(1 - \sin \phi)} \]  \hspace{1cm} (10)

\[ -\sigma_3 \leq T_0 \]  \hspace{1cm} (11)

where \( q_u \) 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 and \( T_0 \) is the tensile strength. The elements which first satisfy the failure criterion for the face advance of 63 m are shown in Fig. 15. One element is located close to the front of the rockburst site.

The assumed value of 36.5 MPa for the uniaxial compressive strength was selected because it gave the best results within several assumed values. The value is much lower than that of the intact specimen and might seem unreasonable. This difference, however, reflects the effects of rock mass fractures. For example, 45.1 MPa is obtained after applying the correction described by Eq. (8) on the strength of the intact rock specimen for the ore body. The assumed strength of 36.5 MPa is just slightly lower than the corrected value.

![Fig. 15: Elements which first satisfy the failure criterion for the 63 m face advance.](image)

6. DISCUSSION

It was confirmed that the rock mass at Mufulira mine can be classified as very hard rock (Tables 1 and 2). However, the rockburst did not occur at a very high stress concentration in the chain pillar or the mining face. This would be because the rock mass was sequentially fractured due to mining, as suggested by the wide EDZ at 1423 m level. Instead, the rockburst occurred in a diminishing pillar in the mining drive at 1440 m level. At this level, the rock mass could be relatively intact because blasting had only been done initially for developing the mining drive, as the case at 1457 m level. The level of stress concentration was not high before mining the stope D in Fig 15. The relatively intact rock mass would satisfy the criteria for rockburst due to the face advance.

7. CONCLUSION

In order to understand the mechanism of the rockburst with magnitude 2.8 at 1440 meter level in the mining drive at Mufulira mine, field geotechnical investigation was conducted, which was followed by uniaxial compression tests, and finally stress analysis was carried out.

Stress analysis showed very high stresses 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. It was therefore concluded that fracturing of relatively intact rock mass around the mining drive under gradual stress increase by mining could be the cause of the rockburst.

Prevention measures of rockburst occurrences will be further investigated, and the simulation method will be improved.

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