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Volumetric analysis of rock mass instability around haulage drifts in underground mines

Shahé Shnorhokian*, Bryce MacNeil, Hani S. Mitri

Department of Mining and Materials Engineering, McGill University, Montreal, H3A 0E8, Canada

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ABSTRACT

Haulage networks are vital to underground mining operations as they constitute the arteries through which blasted ore is transported to surface. In the sublevel stoping method and its variations, haulage drifts are excavated in advance near the ore block that will be mined out. Numerical modeling is a technique that is frequently employed to assess the redistribution of mining-induced stresses, and to compare the impact of different stope sequence scenarios on haulage network stability. In this study, typical geological settings in the Canadian Shield were replicated in a numerical model with a steeply-dipping tabular orebody striking EW. All other formations trended in the same direction except for two dykes on either side of the orebody with a WNW–ESE strike. Rock mass properties and in situ stress measurements from a case study mine were used to calibrate the model. Drifts and crosscuts were excavated in the footwall and two stope sequence scenarios – a diminishing pillar and a center-out one – were implemented in 24 mining stages. A combined volumetric-numerical analysis was conducted for two active levels by comparing the extent of unstable rock mass at each stage using shear, compressive, and tensile instability criteria. Comparisons were made between the orebody and the host rock, between the footwall and hanging wall, and between the two stope sequence scenarios. It was determined that in general, the center-out option provided a larger volume of instability with the shear criterion when compared to the diminishing pillar one (625,477 m^3 compared to 586,774 m^3 in the orebody; 588 m^3 compared to 403 m^3 in the host rock). However, the reverse was true for tensile (134,298 m^3 compared to 128,834 m^3 in the orebody; 91,347 m^3 compared to 67,655 m^3 in the host rock) instability where the diminishing pillar option had the more voluminous share.

1. Introduction

Haulage networks comprising drifts and crosscuts are the main ore transportation routes used in underground mining, and their continuous stability during ore extraction is a key requirement for the safety and profitability of a mining operation. For thicker orebodies, drifts are usually excavated parallel to the orebody strike while crosscuts are extended perpendicular to it from the footwall side to the hanging wall contact. They are developed previous to, or concurrently with, mining operations in a given ore block depending on the mining method. In the sublevel open stoping method and its variations, the haulage network needs to be in place prior to the commencement of ore extraction, and therefore constitutes a considerable capital expenditure (Lawrence, 1998; Hamrin, 2001; Bullock, 2011). This also implies that it has to remain stable for a longer period of time under continuously changing mining-induced stresses. Sublevel open stoping is a popular method used in hard rock mines within the Canadian Shield due to the presence of favorable conditions there (Mining Sourcebook, 2007).

The initial redistribution of in situ stresses caused by the development of the haulage network – as well as its distance from the orebody – is taken into account when choosing the mining method. Operational and ground control considerations must be addressed when choosing the distance since the best option for one factor might not be desirable for the other. As an example, operational optimization favors a network as close to the orebody as possible to limit the development footage and ore haulage distance. On the other hand, controlling the rock mass behavior and limiting its instability will require a further distance from the stope

* Corresponding author.
E-mail address: shahe.shnorhokian@mail.mcgill.ca (S. Shnorhokian).

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openings. In terms of stress perturbations, the ones generated due to the excavation of the network are relatively minor when compared to the redistribution caused by ore production and extraction of the much larger stopes. Furthermore, different sequence alternatives will induce stress redistributions that can vary to a great extent in terms of magnitude and orientation, rendering them crucial tools for controlling the overall behavior of a rock mass (Villaescusa, 2014). The choice as to the optimum sequence scenario can be made based on ground control (Potvin and Hudyma, 2000; Kaiser et al., 2001; Beck and Sandy, 2003; Villaescusa, 2003; Wiles et al., 2004; Bewick et al., 2009; Trifu and Suorineni, 2009; Cepuritis et al., 2010; Laubscher, 2011; Perman et al., 2011; Sjöberg et al., 2012; Jooste and Malan, 2015), economic (Poniewierski et al., 2003; Manchuk, 2007; Nehring et al., 2012; Bai et al., 2013), and operational (Pelley, 1994; Pareja, 2000) factors and a vast literature exists on guidelines and case studies. In some mining methods, a certain amount of flexibility can be entertained regarding the stope sequence scenario while in others ground conditions, it may present considerable restrictions (Carter, 2011; Stephan, 2011).

The interaction of the local geological settings with different sequence scenarios is best studied using numerical modeling (Board et al., 2001; Jing, 2003; Wiles, 2006; Castro et al., 2012). For an entire ore block or implications on a mine-wide scale, three-dimensional (3D) models are better suited so as to capture induced stress changes along any of the three axis directions. With the development of computer technology and the availability of affordable commercial codes, the comparison of several sequence scenarios can be accomplished within an acceptable period of time as numerous studies have demonstrated (Diering and Laubscher, 1986; Contador and Glavic, 2001; Turner and Beck, 2002; Beck and Sandy, 2003; Wiles, 2005; Villaescusa, 2008; Yao and Moreau-Verlaan, 2010; Perman et al., 2011; Sjöberg et al., 2012; Counter, 2014; Jooste and Malan, 2015). However, it is vital for the model to first be calibrated and validated based on site observations of rock behavior, rock mass properties, and stress measurements (Beck et al., 1997; Diederichs et al., 2002; Wiles, 2007). Variations in properties used as input parameters for the model have impacts on the final results and there are suggested guidelines as to how to address the issue (Wiles, 2006; Bewick and Kaiser, 2009; Idris et al., 2011; Cai, 2011). Once the calibration step has been completed, the model can then be used for the prediction of future stress and deformation developments in the rock mass, and compared to instability criteria to verify if any potential issues are likely to occur.

The main type of instability encountered around haulage networks is dependent on the rock mass properties, the presence of nearby geological structures, and in situ stress regimes. Relaxation or concentration of redistributed stresses at locations around an ore block is a result of interactions between the geological formations and structures, the stope openings, and the principal stresses. For example, gravity-related instability is expected in zones where a drop is recorded in confining stress (Hoek et al., 1995; Hudson and Harrison, 1997; Martin et al., 1999a), compressive or shear failure may occur where the major principal component increases without added confinement (Martin, 1997; Castro et al., 1997; Martin et al., 1999b; Kaiser et al., 2000; Brady and Brown, 2006; Cai and Kaiser, 2014), and tensile conditions result in failure as well as significant rock mass damage (Cai et al., 1998; Diederichs and Kaiser, 1999; Diederichs et al., 2004). Hence, while numerical modeling will provide displacement and stress redistribution contours in the vicinity of an underground haulage network, these should then be compared to instability criteria based on the prevailing geological conditions (Martin et al., 2003) as has been done for numerous case studies (Martin et al., 1999b; Kaiser et al., 2001; Wiles et al., 2004; Zhang and Mitri, 2008; Raju et al., 2015; Cai and Kaiser, 2014; Hoek and Martin, 2014). An excellent summary is provided by Kaiser et al. (2000) regarding potential modes of instability for underground haulage networks under different stress conditions.

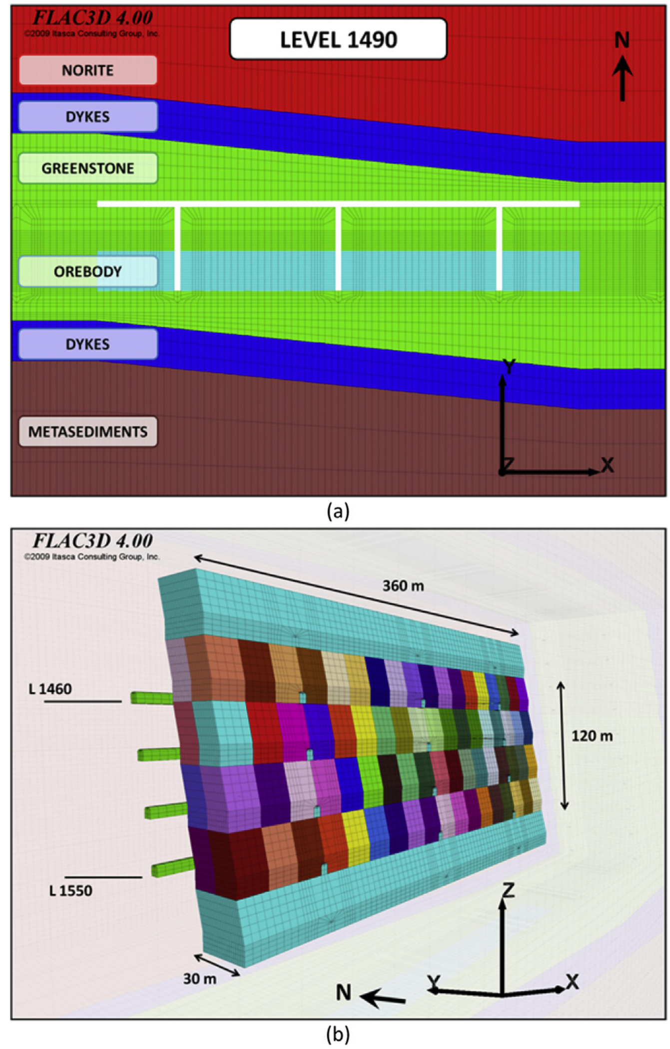

Fig. 1. (a) Plan view of the simplified model showing all geological formations on L 1490; (b) 3D view of the orebody showing the active levels, stopes, and dimensions.

In this paper, a 3D numerical model of a typical hard rock mine in the Canadian Shield was constructed in the finite difference code FLAC^{3D} (Itasca, 2006). The geological formations represented, their rock mass properties, and the premining stress magnitudes and orientations were based on a case study mine. Volumetric analysis was conducted for the three rock mass instability criteria – compression, tension, and shear – to provide a quantitative assessment for two stope sequence scenarios. The focus of the study was the main haulage drift and three crosscuts on one of the levels, as well as the stopes above and below them.

2. Methodology

In order to conduct a realistic study of rock mass instability around a haulage network, a FLAC^{3D} model was constructed with simplified geological formations typically found in the Canadian Shield.

2.1. Model setup

A tabular orebody was constructed that extended 360 m along an EW strike and dipped steeply at 80° to the south. The host rocks

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