Stope stability assessment by the Mathews–Potvin method: a case-study of open stoping in salt rock mass under conditions of secondary stress field

Ch. V. Khazhyylai 1, M. A. Kosyrev 1, V. A. Eremenko 1,*, A. R. Umarov 1

1National University of Science and Technology MISIS, Leninsky Av. 4, 119991, Moscow, Russia

*Corresponding author: prof.eremenko@gmail.com

Abstract: Predicting the stability of open stopes is a challenging task for underground mine engineers. The introductory part of this paper presents main issues related to stability and safety of soil and rock mass in course of large earthworks and underground works. On the basis of former studies, this paper reviews the research findings on stope stability by the methods of Mathews and Potvin in the stoping design for rock salt mines operating at depths greater than 1 km under conditions of secondary stress fields, as well as the rock mass quality assessment by Barton’s method. The conservative stope design has been performed for a few scenarios of design parameters. In conclusions, it is underlined by the stability graph at the given parameters, that in the first case-study scenario, HR is 4.5 m and the Mathews–Potvin stability number is 68 m. It is concluded that the stopes will be stable and preserve their shapes. On the basis of presented case study (stopes stability assessment in the test halite–polyhalite–polymineral salt deposit was carried out for a few scenarios at maximal effective stresses and under conditions of jointing), the authors provide the general provisions on salt rock mass assessment at great depths.

1. Introduction – the goal of the study
The problem of stability of soil and rock masses in course of earthworks and mining production is a vital issue in contemporary mining and civil engineering industry. The authors of reference [1] emphasise that the stability graph method has been used for decades as the first step of open stope design around the world. Through decades, it could be observed that there are some shortcomings of this method. Stability graph method does not account for the relaxation zones around the stopes. Another limitation of the stability graph is that this method cannot to be used to evaluate the stability of the stopes with high walls made of backfill materials [2, 3], that are likely to use in order to reduce the spoil amount and environmental impact of mining production [4]. Several analytical and numerical methods that can be used to overcome these limitations were published last years. Problems of stability are however, due to extremely high risk and cost of eventual accident, widely described in the technical literature, starting from simplest cases of deep excavations [5], up to the complicated analyses of shaft protection pillars in terms of reliability index [6]. In this study, the authors propose to assess stope stability under conditions of secondary stresses induced in salt rock mining at depths greater than 1 km using the stope design approaches of K. Mathews [7] and Y. Potvin [8, 9]. Experiences from using Mathews Method juxtaposed in references [10-17] were considered in the current study.
Original experience and research are also developed on Russian universities. Works [18-23] summarize those developments. The studies into the secondary stress fields induced by different systems of mining are carried out within Russian Science Foundation R&D project No. 19-17-00034.

2. Mathews method of the stability analysis

In 1981 K. Mathews put forward stability number N to assess stope stability as function of properties, strength and jointing of rocks and depth of mining [7]. The stability estimates were obtained from the in-situ research in underground mines in hard rocks at depths greater than 1 km.

The results were used to improve Barton’s method (Q-system) with purpose to update the influence of rock mass jointing on stope stability. The stability number N is determined using four criteria: Q, A, B and C. Mathews modified the rock mass rating as follows on equation (1):

\[
N = \frac{RQD}{J_n} \cdot \frac{J_r}{J_a} \cdot A \cdot B \cdot C
\]

where:
- \(RQD\) is the Rock Quality Designation;
- \(J_n\) is the joint set number;
- \(J_r\) is the roughness number;
- \(J_a\) is the joint alteration number;
- \(A\) is the strength-to-stress ratio (compression strength/shear stress);
- \(B\) is the factor of fracture orientation relative to exposed hanging wall;
- \(C\) is the factor of the incline (slope) of an exposed surface.

\(RQD/J_n\) takes into account the rock mass structure; \(J_r/J_a\) is the estimate of shear strength of rocks. The stope stability estimates have been obtained for different mining scenario, maximal effective stresses and nature of sets of joints [5-11]. The calculations represented a conservative approach, with a single joint set, although jointing was left unrecorded in core tests from 12 boreholes drilled in the halite–polyhalite-compound salt deposit selected as a subject of the research.

It is determined for the open stopes in the test salt deposit that \(RQD=75\) and \(J_n=3\) (one set of joints and a random joint are recorded). The numbers \(J_r\) and \(J_a\) equal 4 and 2, respectively, as the joints are discontinuous, extensile and altered. The factor \(A\) takes into account the ratio of strength to maximal effective stresses \(\sigma_{\text{max}}\). It replaces Barton’s SRF for the more accurate assessment of effective stresses in rock mass surrounding mine stopes at great depths. Moreover, this factor includes the uniaxial compression strength \(UCS\) of rock salt. The maximal effective stresses at the boundaries of stopes were determined from the numerical modeling in Map3D environment. Determination of the factor \(A\) needs two parameters (Figure 1):
- maximal effective stress in the center of a test stope in the secondary shear stress field;
- uniaxial compression strength.

\[\begin{align*}
\text{Figure 1.} & \quad (a) \text{ Determination of factor } A \text{ by Mathews’s Method; (b) maximal effective stress in the center of the test stope in the secondary shear stress field [2].}
\end{align*}\]

The maximal effective shear stress was found from numerical modeling of the stress state of salt rock mass. At the designed mining depth of 1 km below ground surface, the uniaxial compression strength of salt is \(\sigma_{\text{UCS}} = 33.07 \text{ MPa}\) and the maximal effective stress is \(\sigma_{\text{max}} = 40 \text{ MPa}\). The factor \(B\) takes into
account the orientation of joints relative to a stope in rock mass (Figure 2). The angle between the joint and the stope, the number of the join sets, as well as the dip and strike angles of joints were determined in DIPS.

For the test stopes at depths greater than 1 km, we assumed a conservative scenario: \(\alpha = 56^\circ\) in the sidewalls of stopes at 90° and \(\alpha = 56^\circ\) for horizontal stopes.

The factor C takes into account the slope of the exposed surface. In case of damage by gravity, with sloughing of salt, the factor C can be given by formula 2:

\[
C = 8 - 6 \cos(\alpha) \tag{2}
\]

where \(\alpha\) is the slope of the exposed surface.

In the case under study, \(C = 8\) in the vertical stopes.

The calculation of \(C\) needs to:

- Determine the mechanism of structural damage in rock mass during qualitative and quantitative assessment of rock mass quality (Figure 3);
- Determine \(C\) by the angles of stopes and angles of joints, i.e., to find the mechanism of structural damage by gravity and sloughing

![Figure 2. Determination of factor B by Potvin’s Method [2].](image)

**Figure 2.** Determination of factor B by Potvin’s Method [2].

![Figure 3. Mechanisms of structural damage in rock mass [2].](image)

**Figure 3.** Mechanisms of structural damage in rock mass [2].
The graph in Figure 4 was plotted by Yves Potvin in 1989 [8] using the data from 180 case histories of measurements in unsupported stopes; it represents two parameters:

- hydraulic radius of stope, m;
- modified dimensionless stability number N as a characteristic of stope stability under certain stresses.

It is seen from the graph that each exposed surface belongs to one of three zones:

- Stability zone—no damages, or small damages are observed/recorded during unsupported life, i.e. the stope preserves its shape. Local sloughing of stope walls results in dilution of ore to 5%.

- Instability zone—local damages in roof and sidewalls and rock falls in areas to 30% of the stope. Possible falls of rocks from the roof or hanging wall, within the dome of natural equilibrium (a stable dome is formed finally).

- Caving zone—prevailing damages. Stopes require support installation.

3. Results of the case study

For unsupported stopes, the same graph was plotted by Y. Potvin in 1988 and was updated jointly with Nickson in 1992 using the data from 112 stopes with cable bolting. According to the graph, at the modified stability number N 300, a stope is stable at any value of the hydraulic radius. Supported stopes can also be either stable, unstable or caved [22], and have the same characteristics as unsupported stopes.

Geometry of a stope is described with a hydraulic radius HR which is a ratio of the exposed area to the exposed perimeter, i.e. an equivalent exposed span (Figure 5). The perimeter of the exposed surface should be measured precisely and correctly. In stope without cable bolting, it is recommended to select HR for sidewalls using the criterion which says, “Minimum N should never cross the Transition zone and the Stable Unsupported zone boundary” (marked by points in Figure 6).

![Figure 5. Determination of hydraulic radius HR](image)

![Figure 6. Hydraulic radius for sidewalls in stopes without cable bolting](image)
The selection criterion of HR for a stope roof as the same as for sidewalls, with small permissible intersection of the boundary between the unsupported transition zone and unsupported stable zone if the central drilling drift is reinforced with rock bolts. The stable-to-unstable transition zone was greatly diminished. In 1989 Yves Potvin added the graph with a zone within which cable bolting was an efficient support.

The stope stability assessment in the test halite–polyhalite–polymineral salt deposit at the depth greater than 1 km was carried out for a few scenarios, at certain maximal effective stresses and under certain conditions of jointing. In the first scenario, HR is 4.5 m and the Mathews–Potvin stability number is 68 m. It is seen in the stability graph that at the given parameters, the stopes will be stable and preserve their shapes (red point in Figure 7).

![Stability graph](image)

**Figure 7.** Stability graph for stopes in the test halite–polyhalite–polymineral salt deposit at the mining depth greater than 1 km.

### 4. Discussion and conclusions

Design and predicting the stability of open stopes is a challenging task for underground mine engineers. On the basis of former studies, this paper reviewed the research findings on stope stability by the methods of Mathews and Potvin in the stoping design for rock salt mines operating at depths greater than 1 km under conditions of secondary stress fields, as well as the rock mass quality assessment by Barton’s method. The conservative stope design has been performed for a few scenarios of design parameters. On the basis of presented case study (stopes stability assessment in the test halite–polyhalite–polymineral salt deposit was carried out for a few scenarios at maximal effective stresses and under conditions of jointing), the authors provided the general provisions on salt rock mass assessment at great depths.

**Funding information**

The studies into secondary stress field induced by mining with different system are supported by the Russian Science Foundation R&D Project No. 19-17-00034.

**References**

[1] Sepehri M, Apel D, Liu W 2017 *Archives of Mining Sciences* 62(3) pp 653-669
[2] Drusa M and Giang N 2004 *Komunikacie* 6(3) pp 5-9
[3] Ivannikov A L, Kongar-Syuryun C, Rybak J, Tyulyaeva Y 2019 *IOP Conf. Ser.: Earth Environ. Sci.* 362(1) 012130
[4] Kongar-Syuryun C, Tyulyaeva Y, Khairutdinov A M, Kowalik T 2020 *IOP Conf. Ser.: Mat. Sci. Eng.* 869(3) 032004
[5] Rybak J, Ivannikov A, Kulikova E, Żyrek T 2018 MATEC Web Conf. 146 02012
[6] Bauer J, Pula W, Wyjadłowski M 2015 Georisk 9(4) 242-249
[7] Mathews K E, Hoek E, Wyllie D C, Stewart S B V 1981 CANMET Report DSS Serial No. OSQ80-00081
[8] Potvin Y 1988 PhD Thesis Dept. Mining and Mineral Processing University of British Columbia
[9] Hadjigeorgiou J, Leclair J G, Potvin Y 1995 97th CIM-AGM Rock Mechanics and Strata Control Session
[10] Bewick R P and Kaiser P 2009 Proceedings of the 3rd CANUS Rock Mechanics Symposium 01/2009
[11] Hoek E, Kaiser P K, Bawden W F 2001 Support of underground excavations in hard rock
[12] Steward S B V and Forsyth W W 1995 CIM Bull. 88 pp 45-53
[13] Trueman R, Mikula P, Mawdesley C, Haries N 2000 CIM Bull. 93 162-167
[14] Mawdesley C, Trueman R and Whiten W 2001 Trans. Instn Min. Metall (Sect. A: Min. Techno) 110 A27-A39
[15] Hustrulid W A and Bullock R L 2001 SME 718 p
[16] Suorineni F T 2010 Reclamation and Environment 24(4) pp 307-339
[17] Wang J, Milne D, Wegner L Reeves M 2006 Int. Journal of Rock Mechanics and Mining Sciences 44 pp 289-298
[18] Galchenko Yu P, Eremenko V A, Kosyreva M A, Vysotin N G 2020 Eurasian Mining 1 pp 9-13
[19] Eremenko V A, Galchenko Yu P, Kosyreva M A 2020 Journal of Mining Science 56(3) (in press)
[20] Eremenko V A, Galchenko Yu P, Vysotin N G, Kosyreva M A, Yakusheva E D 2020 IOP Conf. Ser.: Earth Environ. Sci. 523 012030
[21] Yu L, Ignatov Y, Ivannikov A, Khotchenkov E, Krasnoshtanov D 2019 IOP Conf. Ser.: Earth Environ. Sci. 324(1) 012004
[22] Batugin A, Kolikov K, Ivannikov A, Ignatov Y, Krasnoshtanov D 2019 SGEM 19(1.3) pp 717-724
[23] Eremenko V A 2020 Gornyi Zhurnal 1 pp 67-73