

Substantiation of stope extraction parameters for thin gold ore bodies based on failure-zone assessment

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Abstract

Purpose. To establish the relationships governing the influence of the Geological Strength Index (GSI) and the dip angle of a thin ore body on the size of the potential failure zone in the stope roof, and to substantiate an approach for using this zone when selecting technological parameters of stope extraction.

Methods. The study was conducted using two-dimensional finite element modelling in RS2 according to a full 4×4 factorial design with GSI values of 35, 45, 55, and 65 and ore-body dip angles of 20, 25, 30, and 35°. The model depth was 400 m, the stope width was 12 m, the stope height was 4 m, and the inter-stope pillar width was 8 m. The potential failure zone was defined as a contiguous near-contour region with a Strength Factor ≤ 1.0.

Findings. A consistent reduction in the potential failure zone was established with increasing rock-mass quality. As GSI increased from 35 to 65, the mean z value decreased from 3.48 to 1.71 m, corresponding to a reduction of 50.8%. For individual dip angles, the reduction in failure-zone size ranged from 48.5 to 52.5%. The maximum value, $z = 3.65$ m, was obtained at GSI = 35 and a dip angle of 20°, whereas the minimum value, $z = 1.59$ m, was obtained at GSI = 65 and a dip angle of 25°. Within the investigated range of 20-35°, the influence of dip angle was substantially weaker than that of rock-mass quality and exhibited a non-monotonic pattern. Descriptive decomposition of the numerical matrix showed that 97.6% of the total variation in z was associated with differences between GSI levels, 2.3% with changes in dip angle, and approximately 0.1% with the combined and residual component. The mean relative disturbance coefficient K_z decreased from 0.871 at GSI = 35 to 0.429 at GSI = 65.

Originality. The dominant role of rock-mass quality in the formation of the potential failure zone in the stope roof during the extraction of thin ore bodies has been quantified. Relationships between GSI, ore-body dip angle, and the size of the potentially unstable near-contour region have been established.

Practical implications. The results can be used for the preliminary selection of measures to adjust stope and pillar dimensions, exposed span, and perimeter blasting parameters. Direct prediction of dilution requires site-specific calibration considering the actual thickness and morphology of the ore body and field-monitoring data.

Keywords: *thin ore body; gold ore deposit; failure zone; numerical modelling; ore dilution*

1. Introduction

The development of thin gold ore deposits is characterized by a high sensitivity of extraction performance to the geometry of the ore body and the stability of its contacts with the host rocks. Where the ore body is of limited thickness, even localized instability of the stope roof or sidewalls may result in overbreak and the incorporation of waste rock into the blasted ore mass. This leads to increased dilution, reduced valuable component content, and deterioration of the technical and economic performance of mining operations [1]. At the same time, excessive reduction in stope dimensions or the retention of oversized pillars decreases the completeness of reserve extraction and may increase mining losses. Therefore, the substantiation of stope parameters for thin ore bodies requires the combined consideration of technological and geomechanical factors [2].

In a broader context, the selection of the mining method and its parameters should be based on an integrated assessment of geological conditions, ore-body characteristics, technological solutions, and potentially critical states of the natural-technical system [3]-[6]. In turn, changes in the mining layout and the method of managing the mined-out space can substantially affect stress redistribution, deformation of the near-contour rock mass, and the stability of adjacent underground workings [7].

One of the key factors governing the stability of the near-contour rock mass is its structural disturbance and overall quality, which can be integrally characterized using the Geological Strength Index (GSI) [8], [9]. The importance of rock-mass structure is also confirmed by studies of the interaction between natural fractures and elements of the mining system

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during the extraction of steeply dipping ore deposits, where the degree and pattern of fracturing directly affect the interaction between the host rock and backfill masses [10]. Deposit geometry, particularly the dip angle, is also of considerable importance, as it determines the spatial orientation of the stope and ore-body contacts relative to the prevailing stress field [11]. The combined influence of these factors governs stress redistribution and the extent of potentially unstable zones around stopes [12], [13].

A reliable representation of rock-mass structural characteristics is essential for assessing the stability of underground mining structures. The spatial orientation and parameters of fracture systems determine the nature of localized rock weakening and can significantly influence the formation of potentially unstable zones. Modern investigation methods involve instrumental fracture mapping, processing of spatial data, and the development of three-dimensional models of underground mining structures, thereby enabling a more comprehensive consideration of rock-mass structural heterogeneity in subsequent geomechanical analyses [14]–[16].

Numerical modelling combined with engineering-geological observation data makes it possible to investigate stress and deformation redistribution as mining conditions evolve, identify potentially hazardous areas, and assess the development of disturbed zones around underground excavations. This approach is particularly important for fractured rock masses, where changes in the stress-strain state may be accompanied by localized deformation and dynamic manifestations. The integrated use of numerical, structural, and monitoring data provides a basis for substantiating excavation parameters and improving the reliability of rock-mass stability predictions [17]–[20].

In current studies of the underground extraction of thin and narrow-vein ore bodies, the problem of ore losses and dilution is considered in relation to hangingwall and footwall stability, stope geometry, deposit dip angle, and rock-mass quality [21]. Numerical modelling of the stress-strain state makes it possible to assess the formation of near-contour zones susceptible to failure and to predict the conditions under which overbreak may occur, with overbreak being one of the principal sources of external dilution [22], [23]. Study [24] demonstrated the influence of deposit geometry and stope-block parameters on ore losses and dilution. Study [25] established a relationship between the predicted dilution of thin ore bodies, GSI, and ore-body thickness. The practical effectiveness of geomechanical control of host-rock stability for limiting dilution has also been confirmed by field investigations of thin ore bodies [26].

For thin ore deposits, this problem is particularly important because even relatively limited overbreak may be comparable to the thickness of the extracted ore body and can substantially alter the proportion between ore and the admixed host rock. However, most studies focus either on the geomechanical assessment of stope stability or on the direct calculation of technological indicators of ore losses and dilution. The quantitative transition from a numerically predicted zone of potential rock-mass failure to the substantiation of stope extraction parameters remains insufficiently addressed.

Additional uncertainty arises from the fact that the maximum depth of a potentially unstable zone cannot be directly equated with actual overbreak, caving, or dilution. Such a transition requires determination of the geometry and kinematically feasible portion of the failure zone, estimation of the volume of host rock capable of collapsing into the stope, and

subsequent comparison of this volume with the geometry and thickness of the ore body. Therefore, an intermediate geomechanical criterion is required to quantitatively characterize the scale of potential rock-mass disturbance and to use the results of numerical modelling as a basis for subsequent assessment of overbreak, Equivalent Linear Overbreak/Slough (ELOS), ore losses, and dilution.

The purpose of this study is to establish the relationships governing the influence of the Geological Strength Index (GSI) and the dip angle of a thin ore body on the size of the potential failure zone in the stope roof, and to substantiate an approach for using this zone when selecting technological parameters of stope extraction.

To achieve this purpose, the following tasks were addressed: to numerically investigate the stress-strain state of the rock mass by varying GSI and the dip angle of the ore body; to determine the dimensions of potential failure zones in the stope roof for the investigated combinations of factors; to quantitatively assess the relative influence of GSI and dip angle on changes in the size of the failure zone; to determine the relative extent of the potentially unstable region; to develop a procedure for transitioning from geomechanical assessment of the failure zone to the determination of overbreak, ELOS, and predicted dilution indicators; and to substantiate directions for adjusting stope extraction parameters for different levels of rock-mass quality.

2. Methods

2.1. Geomechanical characteristics of the rock mass

The strength condition of the rock mass was assessed using the Geological Strength Index (GSI), which characterizes rock-mass structure, blockiness, degree of fracturing, and the condition of fracture surfaces. The use of GSI enables the transition from the strength characteristics of laboratory specimens to the equivalent properties of a fractured rock mass and allows calculation cases representing different degrees of rock-mass disturbance to be compared.

The Geological Strength Index is used in conjunction with the generalized Hoek-Brown criterion to determine the equivalent strength and deformation characteristics of the rock mass. The GSI concept was proposed by E. Hoek [27] and was subsequently further developed and quantitatively formalized [28]. GSI is a dimensionless parameter ranging from 0 for extremely disturbed rock masses to 100 for practically undisturbed rock masses.

GSI can be quantitatively estimated using fracture-condition and rock-mass quality parameters [28]:

$$GSI = 1.5 \cdot JCond_{89} + \frac{RQD}{2}, \quad (1)$$

where:

$JCond_{89}$ – joint-condition rating according to the RMR₈₉ classification;

RQD – rock Quality Designation, %.

The joint-condition rating $JCond_{89}$ is determined as the sum of individual fracture characteristics:

$$JCond_{89} = JA_{41} + JA_{42} + JA_{43} + JA_{44} + JA_{45}, \quad (2)$$

where:

JA_{41} – joint-roughness rating;

JA_{42} – joint-length rating;

JA_{43} – joint-aperture rating;

JA_{44} – joint-infill characteristic;

JA_{45} – degree of weathering of joint walls.

The Rock Quality Designation (RQD) is generally determined from the analysis of drill-core material obtained from engineering-geological boreholes. In the absence of direct core-sampling data, RQD can be approximately estimated from fracture intensity using the relationship applied in [29]:

$$RQD = 100e^{-0.1\lambda} (0.1\lambda + 1), \quad (3)$$

where:

λ – average number of fractures per metre of rock-mass length.

For visual estimation of GSI, the Hoek-Brown classification chart is also used, taking into account the rock-mass structure and the condition of fracture surfaces. The chart makes it possible to relate the degree of rock-mass blockiness, the spatial arrangement and interlocking of structural blocks, and the quality of discontinuity surfaces to the corresponding GSI range [28].

GSI accounts for the geological characteristics and structure of the rock mass, as well as a set of geomechanical and geometric characteristics of its fracture network. The combined use of GSI and the Hoek-Brown criterion makes it possible to determine the equivalent rock-mass properties required for numerical modelling of its stress-strain state.

In the present study, GSI was considered an independent variable factor. For the parametric analysis, GSI values of 35, 45, 55, and 65 were adopted, representing a progressive change in rock-mass quality from disturbed to relatively stable conditions. These values were not calculated using Eqs. (1)–(3) for a specific deposit but were specified as factor levels in a full factorial experiment. Equations (1)–(3) define the procedure for determining GSI during subsequent site-specific calibration of the model based on engineering-geological mapping, drill-core investigation, and assessment of rock-mass fracturing.

2.2. Numerical model and conditions of the numerical experiment

The stress-strain state of the rock mass was investigated using the finite element method (FEM), which makes it possible to account for the geometry of the mined-out space, the mechanical properties of the rock mass, and the prescribed boundary conditions. The numerical approach was selected because several calculation cases had to be compared while systematically varying GSI and the dip angle of the ore body. The finite element method is widely used for parametric analyses of stope stability and prediction of overbreak [30]–[33], as well as for investigating the stress-strain state of the rock mass surrounding underground excavations [34]. In ore-deposit mining, numerical assessment of the stress-strain state can be supplemented by field geomechanical observations and analytical calculations, making it possible to identify potentially unstable zones and substantiate the parameters of underground mining structures [35]–[37]. For fractured rock masses, deterministic geomechanical assessment may additionally be combined with probabilistic analysis when substantiating support parameters and the stability of underground excavations [38].

Numerical modelling was performed using RS2 software (Rocscience Inc., Canada), which is designed for two-dimensional finite element analysis of the stress-strain state of rock masses [39]. The numerical model represented a homogeneous equivalent rock mass containing a system of stopes 12 m wide. The stopes were located at a depth of 400 m, with a stope height of 4 m and an inter-stope pillar width of 8 m.

A stope height of 4 m was adopted to simulate the extraction of a thin ore body. The obtained results are parametric in nature and require subsequent adaptation to the actual thickness, morphology, and physical and mechanical properties of a specific gold ore deposit. The main geometric and geomechanical parameters of the numerical model are presented in Table 1.

Table 1. Input parameters for numerical modelling

Parameter	Adopted value
Geological Strength Index (GSI)	35; 45; 55; 65
Ore-body dip angle (α), degree	20; 25; 30; 35
Stope depth (h), m	400
Stope width (B), m	12
Stope height (H), m	4
Inter-stope pillar width (b), m	8
Rock unit weight (γ), MN/m ³	0.027
Uniaxial compressive strength of intact rock (σ_{ci}) MPa	100
Intact rock parameter (m_i)	10
Rock-mass disturbance factor (D)	0.25
Poisson's ratio (ν)	0.30
Rock-mass Young's modulus (E), MPa	Determined as a function of GSI
Lateral stress coefficient (K_0)	1.0
Strength criterion	Generalized Hoek-Brown criterion
Material behaviour	Elastic
Finite element type	Six-node triangular elements

The strength condition of the rock mass was described using the generalized Hoek-Brown criterion. The rock-mass parameters mb , s , and a were determined in RS2 based on the specified GSI values, the intact rock parameter m_i , and the disturbance factor D . As GSI varied, the corresponding equivalent strength and deformation properties of the rock mass changed accordingly. All other input parameters were kept constant for all calculation cases.

The model domain was selected to be sufficiently large to eliminate the influence of the external boundaries on stress redistribution in the vicinity of the stopes. The external boundaries were located at a distance of at least five characteristic excavation dimensions from the contour under investigation.

The model domain was discretized using an automatically refined finite element mesh composed of six-node triangular elements. Local mesh refinement was applied near the stope boundaries and inter-stope pillars to provide a more detailed representation of changes in the stress state within the near-contour region.

Zero normal displacements were prescribed along the external boundaries of the model. The lateral boundaries restrained horizontal displacements, whereas the bottom boundary restrained vertical displacements. After excavation was simulated, the stope boundaries were assumed to be free of external loads.

The initial stress state was defined considering the self-weight of the rock mass at a depth of 400 m. The ratio between the horizontal and vertical components of the in situ stress field can substantially influence the deformation behaviour and localization of potentially unstable zones around underground excavations [40]. For the parametric comparison of the calculation cases, the ratio of the horizontal to vertical components

of the in situ stress field was assumed to be $K_0 = 1.0$. The same model-domain dimensions, boundary conditions, and discretization parameters were used in all calculation cases.

The stope geometry and finite element mesh of the numerical model are shown in Figure 1.

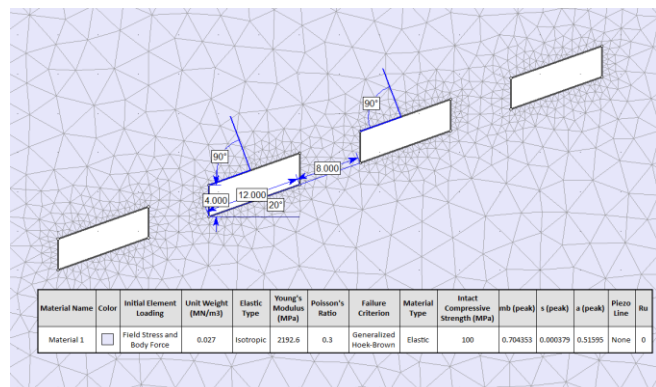


Figure 1. Geometry and finite element mesh of the numerical model at $\alpha = 20^\circ$

The size of the potential failure zone in the stope roof, z , was adopted as the stability indicator. Since an elastic material behaviour model was used, the boundary of the potential failure zone was identified based on the distribution of the Strength Factor. A contiguous near-contour region in which the Strength Factor did not exceed the critical value of 1.0 was considered potentially unstable.

The failure-zone size z was defined as the maximum distance from the designed stope-roof boundary to the outer boundary of the contiguous region with Strength Factor ≤ 1.0 . A single z value was recorded for each calculation case. Identical model-domain settings, boundary conditions, and finite element mesh parameters ensured the comparability of the results and made it possible to relate changes in z to the effects of the varied factors – GSI and the ore-body dip angle.

2.3. Processing of modelling results

The numerical design represented a full 4×4 factorial experiment. Each of the four GSI values – 35, 45, 55, and 65 – was combined with each of the four ore-body dip angles – 20, 25, 30, and 35°. Thus, a total of 16 calculation cases were generated. The stope geometry, depth, inter-stope pillar dimensions, boundary conditions, and all other input parameters remained unchanged when comparing the cases. This design of the numerical series made it possible to separately assess the effects of GSI and dip angle on the size of the potential failure zone.

For each combination of factors, a single calculated value of the maximum failure-zone size in the stope roof, z , was recorded. The resulting values were arranged in a 4×4 matrix, with rows corresponding to GSI levels and columns corresponding to ore-body dip angles. This matrix was used for comparative and descriptive quantitative analysis.

For each GSI level, the mean, minimum, and maximum values of z were calculated across the four dip angles. In addition, the absolute range of variation in z , the difference between the extreme GSI levels, and the relative reduction in the mean failure-zone size compared with GSI = 35 were determined. An analogous comparison was performed for each dip angle as GSI varied from 35 to 65. To assess the relative extent of the disturbed near-contour region, the following dimensionless relative disturbance coefficient was used:

$$Kz = \frac{z}{H}, \quad (4)$$

where:

z – maximum size of the potential failure zone in the stope roof;

H – stope height adopted in the model, equal to 4 m.

The coefficient Kz indicates the fraction of the stope height represented by the calculated depth of the potentially unstable region. It was used to compare calculation cases with the same stope height and was not treated as a direct measure of dilution, since it does not account for the actual failure area and volume, ore-body thickness, or rock density.

For a quantitative comparison of the effects of the investigated factors, a descriptive decomposition of the total variation in the z matrix was performed. The total sum of squared deviations was calculated relative to the mean value for all 16 calculation cases. The contribution of GSI was determined from the deviations of the row means from the overall mean, whereas the contribution of dip angle was determined from the deviations of the column means. The difference between the total variation and the contributions of the two factors was treated as the combined and residual component. The share of each component was expressed as a percentage of the total sum of squared deviations.

This decomposition was used exclusively to describe the structure of variation in the results within the adopted numerical matrix. No statistical significance testing of the factors was performed because only one deterministic calculated value was obtained for each combination of GSI and dip angle, with no replicate simulations.

Based on the processed results, relationships between z and dip angle were plotted for different GSI levels, together with the ranges of minimum, mean, and maximum z values. The z values were rounded to 0.01 m, Kz to 0.001, and relative changes and shares of variation to 0.1%.

3. Results and discussion

3.1. Influence of GSI and dip angle on the size of the failure zone

The results of numerical modelling of potential failure zones for the extreme rock-mass quality levels, GSI = 35 and GSI = 65, at ore-body dip angles of $\alpha = 20, 25, 30$, and 35° are presented in Figure 2.

Figure 2 presents eight representative calculation cases and allows the formation of potential failure zones to be compared at the extreme rock-mass quality levels of GSI = 35 and GSI = 65. At GSI = 35, the obtained z values were 3.65, 3.35, 3.52, and 3.41 m for $\alpha = 20, 25, 30$, and 35° , respectively. At GSI = 65, the failure-zone sizes decreased to 1.88, 1.59, 1.72, and 1.67 m. Thus, for the same dip angles, an increase in GSI from 35 to 65 reduced the failure-zone depth by 1.77 m (48.5%) at 20° , 1.76 m (52.5%) at 25° , 1.80 m (51.1%) at 30° , and 1.74 m (51.0%) at 35° . The mean z value decreased from 3.48 to 1.71 m, corresponding to a reduction of 1.77 m, or 50.8%.

The quantitative analysis confirms the visual pattern shown in Figure 2. Variation in GSI primarily governs the extent of the disturbed region, whereas changes in dip angle within the investigated range affect its configuration and position relative to the stope boundary. At GSI = 35, the range of z values across the four dip angles was 0.30 m, from 3.35 to 3.65 m, whereas at GSI = 65 it was 0.29 m, from 1.59 to 1.88 m.

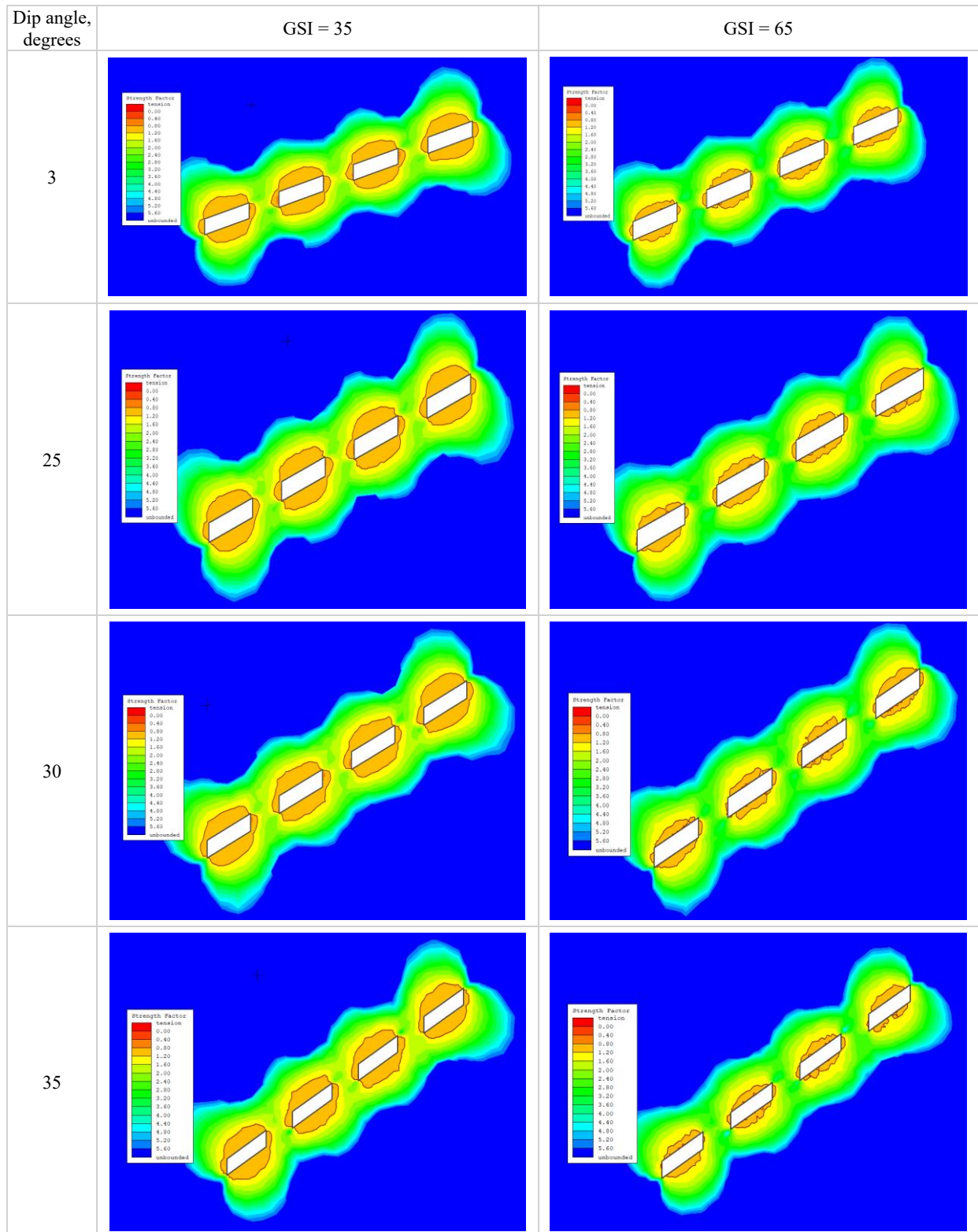


Figure 2. Changes in potential failure zones at different dip angles for GSI = 35 and GSI = 65

In both cases, the minimum value was obtained at $\alpha = 25^\circ$ and the maximum at $\alpha = 20^\circ$, indicating that the influence of dip angle is non-monotonic. With a constant stope height of $H = 4$ m, the relative failure depth z/H ranged from 0.84 to 0.91 at GSI = 35 and from 0.40 to 0.47 at GSI = 65. Thus, an increase in rock-mass quality approximately halves the relative depth of the potentially unstable near-contour region.

For the most unfavourable case considered – GSI = 35 and $\alpha = 20^\circ$ – the failure zone reached 3.65 m, corresponding

to 91.3% of the stope height. For the most favourable case – GSI = 65 and $\alpha = 25^\circ$ – its size was 1.59 m, or 39.8% of the stope height. The difference between these extreme cases was 2.06 m, meaning that the maximum calculated depth of the disturbed region decreased by a factor of 2.30. This provides a geomechanical basis for considering low GSI values as a factor associated with an increased risk of overbreak and host-rock entrainment during the selective extraction of a thin ore body.

The generalized relationships between the size of the roof failure zone and the ore-body dip angle at different GSI values are presented in Figure 3.

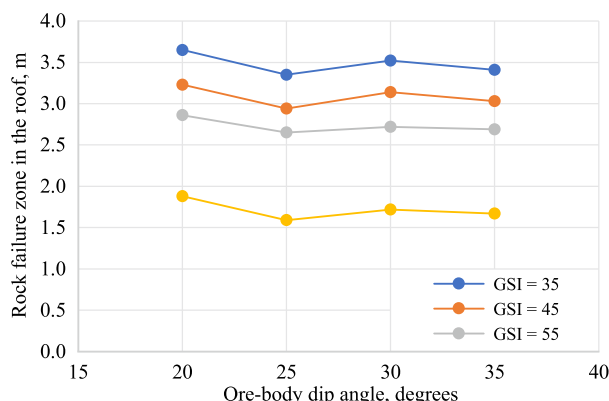


Figure 3. Relationship between the size of the roof failure zone and the ore-body dip angle at different GSI values

As shown in Figure 3, the maximum failure-zone size was 3.65 m at GSI = 35 and a dip angle of 20°, whereas the minimum value was 1.59 m at GSI = 65 and a dip angle of 25°. Within the investigated range, variation in rock-mass quality had a more pronounced effect on the failure-zone size than variation in dip angle.

The obtained results confirm that GSI is the primary factor governing the extent of roof failure, whereas the dip angle affects the redistribution of the disturbed rock zone and modifies its size within each rock-mass quality level. From the standpoint of selective extraction, the most unfavourable cases are those with lower GSI values, since enlargement of the disturbed near-contour region creates conditions for host-rock entrainment. Therefore, when designing stope extraction for thin gold ore bodies, stope and pillar parameters should be selected with consideration of the predicted position of the failure-zone boundary rather than solely on the basis of ore-body geometry.

The mean failure-zone size across the four dip angles decreased from 3.48 m at GSI = 35 to 3.09 m at GSI = 45, 2.73 m at GSI = 55, and 1.71 m at GSI = 65. The reduction in the mean value between the extreme GSI levels was 50.8%.

For individual dip angles, the reduction in failure-zone size as GSI increased from 35 to 65 ranged from 48.5 to 52.5%. The generalized quantitative indicators characterizing changes in the mean, minimum, and maximum failure-zone sizes at different GSI values are presented in Table 2.

Table 2. Quantitative assessment of the effect of GSI on failure-zone size

GSI	Mean, m	Minimum, m	Maximum, m	Reduction relative to GSI = 35, %
35	3.48	3.35	3.65	0.0
45	3.09	2.94	3.23	11.4
55	2.73	2.65	2.86	21.6
65	1.71	1.59	1.88	50.8

The obtained results confirm that, under the considered modelling conditions, GSI is the dominant factor. Variation in the dip angle from 20 to 35° at a fixed GSI causes non-monotonic local changes in the failure zone, whereas an increase in GSI results in a consistent reduction in its size. A similar influence of rock-mass quality and stope geometry on external dilution has been reported in [25], [30]–[32].

The established relationships provide a geomechanical basis for controlling ore losses and dilution: as the predicted failure zone increases, the stope parameters, extraction sequence, and dimensions of the retained rock-mass elements should be adjusted accordingly. However, the numerical modelling performed in this study has several limitations: the stope height was kept constant at 4 m, the rock mass was represented as an equivalent continuum, and ore-loss and dilution indicators were not calculated directly in the current series of models. Therefore, the established relationship with mining-quality indicators reflects geomechanical risk and should be complemented by further calculations for different ore-body thicknesses, stope and pillar dimensions, as well as by validation against field data. For the practical interpretation of the established relationships and the selection of appropriate directions for adjusting technological parameters, the obtained results are summarized in Table 3.

Table 3. Interpretation of numerical results for selecting technological solutions

GSI	Mean failure-zone size, m	Geomechanical assessment	Recommended direction for adjustment
35	3.48	Unfavourable	Reduce the stope span or extraction step; strengthen control of perimeter blasting; retain protective rock-mass elements
45	3.09	Increased risk	Limit exposed span; adjust stope and pillar parameters; reduce damage to the near-contour rock mass
55	2.73	Moderate risk	Select parameters based on the predicted failure boundary; monitor actual overbreak
65	1.71	Relatively favourable	More complete extraction may be considered, provided that the results are confirmed by field monitoring and stability control

Table 3 is intended as an engineering interpretation of the numerical results and does not replace a detailed design calculation. Its purpose is to demonstrate how the established geomechanical relationship can be translated into a technological decision aimed at limiting external dilution. The final stope and pillar dimensions, as well as drilling-and-blasting parameters, should be refined for a specific deposit considering the actual ore-body thickness, in situ stress field, and field-monitoring results.

3.2. Quantitative assessment of the effects of GSI and dip angle

The results of the descriptive decomposition of the total variation in failure-zone size between the investigated factors are presented in Table 4. The descriptive decomposition of the sum of squares for the balanced 4×4 numerical matrix shows that 97.6% of the total variation in z is associated with differences between GSI levels, whereas 2.3% is attributable to changes in the ore-body dip angle.

Table 4. Descriptive decomposition of variation in failure-zone size

Factor	Share of total variation, %	Interpretation
GSI	97.6	Primary source of variation
Dip angle α	2.3	Secondary source of variation
Residual variation	0.1	Combined and residual component

Approximately 0.1% corresponds to the combined and residual component. This distribution confirms that, within the investigated factor space, rock-mass quality has a substantially more pronounced effect on the failure-zone size than the ore-body dip angle. The reported values characterize only the investigated numerical matrix and do not represent the results of statistical significance testing, since no replicate simulations were performed for each combination of GSI and dip angle. Therefore, the obtained shares should be regarded as a descriptive quantitative characterization of the parametric modelling results.

The calculated values of the relative disturbance coefficient Kz for all 16 calculation cases are presented in Table 5.

Table 5. Relative disturbance coefficient Kz for the calculation cases

GSI	Kz , 20°	Kz , 25°	Kz , 30°	Kz , 35°	Mean Kz
35	0.912	0.838	0.880	0.853	0.871
45	0.807	0.735	0.785	0.757	0.771
55	0.715	0.662	0.680	0.672	0.682
65	0.470	0.398	0.430	0.417	0.429

The mean Kz value consistently decreased from 0.871 at GSI = 35 to 0.771 at GSI = 45, 0.682 at GSI = 55, and 0.429 at GSI = 65. Thus, increasing GSI from 35 to 65 approximately halves the relative extent of the disturbed near-contour region.

The maximum value, $Kz = 0.912$, was obtained at GSI = 35 and $\alpha = 20^\circ$, whereas the minimum value, $Kz = 0.398$, was obtained at GSI = 65 and $\alpha = 25^\circ$. At each fixed GSI value, variation in dip angle resulted in relatively small and non-monotonic changes in the coefficient. At the same time, increasing GSI produced a consistent decrease in Kz for all considered dip angles, which is consistent with the results obtained for the absolute failure-zone size.

Therefore, the coefficient Kz can be used for comparative assessment of the relative extent of the potentially unstable region at a constant stope height. However, it is not a direct measure of dilution because it does not account for the actual volume of collapsed rock, ore-body thickness, or the geometry of the extraction block. The transition from the calculated z and Kz values to predicted dilution indicators is considered in the following subsection.

3.3. Relationship between the failure zone, predicted dilution, and technological decisions

The failure-zone depth alone is insufficient for the direct calculation of dilution. Such a calculation requires determination of the area or volume of actual overbreak, ore-body thickness, extraction-block geometry, as well as the densities of the ore and host rock. According to the published methodology for thin ore bodies, dilution is predicted after determining the area of potential caving around the stope and comparing the volume of entrained host rock with the volume of extracted ore [25].

Therefore, in the present study, hypothetical dilution percentages were not calculated solely on the basis of the parameter z . The obtained failure-zone values are regarded as a numerical basis for the subsequent stage – relating them to the actual thickness and morphology of the gold ore deposit.

To formalize the transition from the geomechanical failure-zone indicator to the technological dilution indicator, the Equivalent Linear Overbreak/Slough (ELOS) can be used. ELOS represents the ratio of the overbreak volume to the stope surface area [25], [30]:

$$ELOS = \frac{V_{over}}{S_{st}}, \quad (5)$$

where:

V_{over} – volume of host rock involved in overbreak, m^3 ;
 S_{st} – stope surface area, m^2 .

When the ore-body thickness m is known, predicted dilution can be estimated using the relationship between ELOS and ore-body thickness [25]:

$$r_d = \frac{ELOS}{m} \cdot 100\%, \quad (6)$$

where:

r_d – predicted dilution, %;
 m – true ore-body thickness, m.

The failure-zone depth z calculated in the present study is not equivalent to ELOS. Application of Eqs. (5) and (6) requires determination of the actual area or volume of the portion of the failure zone that is kinematically capable of collapsing into the stope.

Therefore, the presented relationships are considered a framework for subsequent site-specific verification rather than a basis for determining dilution from the parameter z alone. The input parameters required for direct prediction of ore losses and dilution when adapting the model to a specific deposit are presented in Table 6.

Table 6. Input parameters required for direct prediction of ore losses and dilution

Parameter	Symbol	Determination method	Role in the calculation
True ore-body thickness	m , m	Geological and mine-surveying data	Determines the volume of in-situ ore reserves and sensitivity to host-rock dilution
Potential failure area	A_f , m^2	Numerical modelling and digitization of the failure-zone boundary	Determines the volume of potentially entrained host rock
Length and height of the calculation section	L , H , m	Stope-block parameters	Enable conversion of the failure area into volume
Ore and host-rock density	ρ_o , ρ_r , t/m^3	Laboratory and production data	Required to establish the mass balance between ore and diluting host rock
Actual overbreak	ELOS, m	Laser scanning, CMS, or mine surveying	Used to verify the calculated failure zone
Ore losses	Pl, %	Surveying of residual ore reserves and sampling	Characterize extraction completeness when stope parameters are modified

Table 6 shows the additional data that must be obtained when adapting the model to a specific thin gold ore deposit. Thus, two levels of investigation are distinguished: the present study substantiates a geomechanical criterion for dilution risk, whereas direct prediction of technological indicators should be performed after geometric calibration of the failure zone and determination of the actual ore-body thickness.

The obtained failure-zone range of 1.59-3.65 m should be considered in the context of published studies on thin ore deposits. For thin, steeply dipping ore bodies, Imashev et al. [25] showed that decreasing GSI is accompanied by an increase in the potential failure area and predicted dilution. For ore-body thicknesses of 1-3 m, substantially different dilution values were obtained depending on GSI, confirming the appropriateness of using GSI as one of the principal factors governing geomechanical risk.

Additional practical evidence of the relationship between near-contour rock-mass stability and the quality of extracted ore was obtained by Mussin et al. [26] during the mining of thin ore bodies. Artificial support of the hangingwall using cable bolting in steeply dipping sections reduced dilution from 62.8 to 48.7%. For inclined deposits with dip angles of up to 40°, the effectiveness of this support was substantially lower. This finding is important for interpreting the results of the present study, which considered dip angles of 20-35°. Within this range, the principal potential for reducing risk should be associated primarily with accurate formation of the excavation contour and the rational selection of drilling-and-blasting parameters, including the characteristics of the explosives used and the

calculation of drilling and blasting parameters [41], [42], as well as with the exposed span and the geometry of stopes and pillars [22], [25], [43], [44]. The practical implementation of technological measures involving adjustment of drilling-and-blasting parameters also imposes requirements on the operational durability of drilling tools, the rational selection of operating modes, and the reduction of adverse effects from dynamic loading, particularly when drilling hard rocks [45]-[48].

Based on the calculated z and Kz values and their comparison with published data, a matrix of technological decisions was developed (Table 7). It is intended for the preliminary selection of optimization measures prior to site-specific calibration of the model.

The proposed matrix does not establish the final dimensions of stopes and pillars or the explosive charge parameters. Its purpose is to define the decision-making sequence: first, rock-mass quality is classified according to GSI; then, the predicted failure zone is determined; subsequently, the direction of technological adjustment is selected, and its effectiveness is assessed through repeated numerical modelling.

For conditions corresponding to $GSI = 35-45$, limiting overbreak is the priority because the mean size of the disturbed zone ranges from 3.09 to 3.48 m. At $GSI = 65$, the mean failure-zone size decreases to 1.71 m, creating conditions for increasing extraction completeness without a proportional increase in the risk of host-rock entrainment. However, the corresponding technological decision should be confirmed by field monitoring. The sequence for geomechanical substantiation of technological parameters is presented in Table 8.

Table 7. Geomechanically substantiated matrix of technological decisions

GSI	Mean z , m	Kz	Risk level	Priority technological decision
35	3.48	0.871	High	Minimize exposed span; apply controlled or perimeter blasting; reduce the extraction step; retain protective rock-mass elements; implement enhanced monitoring
45	3.09	0.771	Increased	Adjust stope span and pillar dimensions; reduce explosive charge near ore-rock contacts; monitor actual overbreak
55	2.73	0.682	Moderate	Select stope parameters considering the predicted failure-zone boundary; provide local support for potentially unstable areas
65	1.71	0.429	Relatively low	Consider more complete reserve extraction where stability is confirmed; monitor contact conditions and ore losses

Table 8. Sequence of geomechanical substantiation of technological parameters

Stage	Action	Result
1	Engineering-geological mapping; determination of GSI, RQD, fracture parameters, and rock properties	Design rock-mass characteristics
2	Refinement of ore-body thickness, dip angle, and morphology	Geometric model of the ore deposit
3	Numerical modelling of stope, pillar, and blasting-parameter variants	Failure-zone boundaries and Kz values
4	Determination of A_f , V_{over} , and ELOS for the kinematically feasible portion of the failure zone	Predicted volume of entrained host rock
5	Calculation of r_d and ore losses for the considered mining-layout variants	Comparable extraction-quality indicators
6	Selection of the preferred variant based on minimizing dilution and ore losses while ensuring stability	Recommended technological parameters
7	Field monitoring of overbreak using CMS data and determination of actual dilution	Model calibration and refinement

The proposed sequence establishes a link between the results of geomechanical modelling and the selection of technological solutions. In this case, stope extraction parameters are evaluated not only in terms of the stability of exposed rock surfaces but also with consideration of their expected influence on ore losses and dilution.

The applicability of the obtained quantitative results is limited to the conditions of the performed parametric experiment.

The results apply to a two-dimensional RS2 model with $GSI = 35-65$, dip angles $\alpha = 20-35^\circ$, a depth of 400 m, a stope height of 4 m, a stope width of 12 m, and an inter-stope pillar width of 8 m. All cases were calculated using identical boundary conditions, $K_0 = 1.0$, and a consistent discretization approach. Therefore, the identified differences in z primarily reflect the effects of the varied factors, GSI and α .

The model represents the fractured rock mass as an equivalent continuum and does not reproduce the individual geometry of discrete fractures, localized weakened ore-host rock contacts, spatial variability in ore-body thickness, or three-dimensional end effects. In site-specific assessments of structurally disturbed rock masses, additional consideration of the spatial orientation of fracture sets and kinematically feasible block-failure mechanisms can improve the reliability of stability predictions [49]. The numerical series also did not vary the blasting sequence, blast-induced dynamic damage to the rock mass, or possible changes in the in situ stress field with depth. Therefore, the calculated failure zone characterizes a potential region of instability but should not be automatically equated with the actual volume of collapsed rock or the magnitude of dilution.

To assess the robustness of the established relationship within the investigated range, the variation in z caused by changes in dip angle at a fixed GSI was additionally considered. For the four GSI levels, the absolute ranges $z_{\max} - z_{\min}$ were 0.30, 0.29, 0.21, and 0.29 m, respectively. These values are substantially smaller than the change in the mean z between GSI = 35 and GSI = 65, which was 1.77 m, confirming the robustness of the main conclusion regarding the dominant role of rock-mass quality. At the same time, the non-zero range indicates that the influence of dip angle should be considered in site-specific design. Figure 4 presents the calculated range of variation in failure-zone size as a function of GSI.

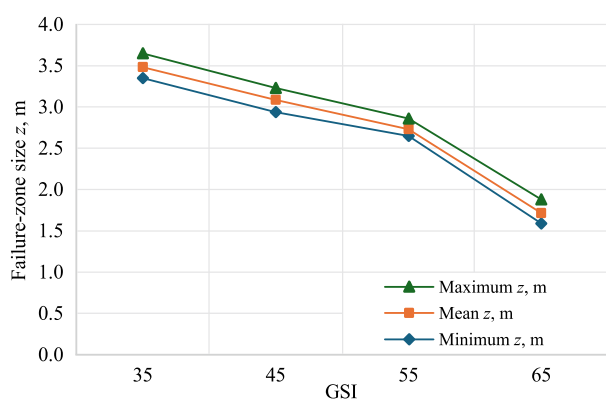


Figure 4. Range of variation in failure-zone size at $\alpha = 20\text{--}35^\circ$ for different GSI values

The lower and upper curves correspond to the minimum and maximum z values, respectively, among the four investigated dip angles, whereas the middle curve represents the mean value. The convergence of the three curves with increasing GSI confirms the overall reduction in the extent of failure, while the vertical width of the range characterizes the residual sensitivity of the results to ore-body geometry. Figure 4 can be used for preliminary engineering assessment only under the adopted initial conditions.

Application of the obtained results to a specific thin gold ore deposit requires site-specific calibration. The minimum required input dataset should include the actual ore-body thickness and morphology, GSI, RQD and fracture parameters, contact strength, deformation modulus, Poisson's ratio, as well as the measured or otherwise substantiated in situ stress field. Additional information on structural heterogeneities and deformation manifestations may be obtained using geophysical methods for rock-mass investigation [50] and integrated geodetic monitoring [51], [52]. After the numerical analysis, the predicted failure-zone boundary should be compared with

the results of mine surveying, CMS or laser scanning of actual overbreak, and measured dilution values. Only after such back-analysis can the established relationships be used to determine specific stope and pillar dimensions, as well as drilling-and-blasting parameters.

4. Conclusions

Within the investigated factor space, increasing GSI from 35 to 65 reduced the mean size of the potential failure zone in the stope roof from 3.48 to 1.71 m, corresponding to a decrease of 50.8%. At a fixed GSI, variation in the ore-body dip angle within the range of $20\text{--}35^\circ$ caused substantially smaller and non-monotonic changes in z . Therefore, for the adopted numerical scheme, rock-mass quality is the primary factor governing the extent of near-contour failure.

Descriptive analysis of the 16 calculation cases showed that, within the investigated matrix, 97.6% of the total variation in z was associated with differences between GSI levels, 2.3% with changes in dip angle, and approximately 0.1% with the combined and residual component. The mean relative disturbance coefficient K_z decreased from 0.871 to 0.429 as GSI increased from 35 to 65, confirming a substantial reduction in the relative extent of the potentially unstable near-contour region.

The practical outcome of the study is represented by the sequence "GSI and ore-body geometry – failure zone – ELOS and overbreak volume – predicted dilution – adjustment of technological parameters". For rock masses with GSI = 35–45, the priority measures are to reduce contour damage during drilling and blasting, limit the exposed span, and adjust stope and inter-stope pillar dimensions. At the same time, the calculated failure-zone depth z is not a direct measure of dilution. Direct calculation of dilution should be performed only after determining the kinematically feasible failure area or volume and the actual ore-body thickness.

The established relationships apply to a two-dimensional RS2 model at a depth of 400 m, with a stope height of 4 m, a stope width of 12 m, an inter-stope pillar width of 8 m, GSI = 35–65, and dip angles of $20\text{--}35^\circ$. For practical application to a specific thin gold ore deposit, the model should be calibrated using actual data on ore-body thickness and morphology, the physical and mechanical properties of the rocks, fracture characteristics, and the in situ stress field. The numerical results should also be compared with CMS or laser-scanning data and actual dilution. Further research should focus on establishing quantitative relationships between failure zones, ore losses, and dilution while varying ore-body thickness and stope and pillar dimensions.

Author contributions

Conceptualization: AS, IA, AK; Data curation: AM; Formal analysis: NS; Funding acquisition: AK; Investigation: AM, MI; Methodology: AM, IA; Project administration: AS; Resources: NS; Software: AM; Supervision: AM; Validation: AS, AK; Visualization: AM, IA; Writing – original draft: NS; Writing – review & editing: AM, MI. All authors have read and agreed to the published version of the manuscript.

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Conflicts of interest

The authors declare no conflict of interest.

Data availability statement

The original contributions presented in the study are included in the article, further inquiries can be directed to the corresponding author.

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Обґрунтування параметрів очисного виїмання малопотужних золоторудних тіл на основі оцінювання зон руйнування масиву

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Мета. Встановити закономірності впливу геологічного індексу міцності GSI та кута падіння малопотужного рудного тіла на розмір зони можливого руйнування в покрівлі очисного простору та обґрунтувати підхід до використання цієї зони під час вибору технологічних параметрів очисного виїмання.

Методика. Дослідження виконано методом двовимірною скінченно-елементного моделювання в RS2 за повним факторним планом 4×4 при значеннях GSI = 35, 45, 55 і 65 та кутів падіння рудного тіла 20, 25, 30 і 35°. Глибина розташування очисного простору становила 400 м, його ширина – 12 м, висота камери – 4 м, ширина міжкамерного цілика – 8 м. Зону можливого руйнування визначали як зв'язну приконтурну область зі значенням Strength Factor (SF) ≤ 1.0 .

Результати. Встановлено стійке зменшення зони можливого руйнування зі зростанням якості масиву. При підвищенні GSI від 35 до 65 середнє значення z зменшилося з 3.48 до 1.71 м, тобто на 50.8%. Для окремих кутів падіння зменшення розміру зони руйнування становило 48.5-52.5%. Максимальне значення $z = 3.65$ м отримано при GSI = 35 і куті падіння 20°, а мінімальне $z = 1.59$ м – при GSI = 65 і куті падіння 25°. У дослідженому діапазоні 20-35° вплив кута падіння був суттєво слабшим за вплив якості масиву та мав немономонний характер. Дескриптивне розкладання варіації розрахункової матриці показало, що 97.6% загальної варіації z пов'язано з відмінностями між рівнями GSI, 2.3% – зі зміною кута падіння та близько 0.1% – із сумісною та залишковою складовою. Середній коефіцієнт відносної порушеності K_z зменшився з 0.871 при GSI = 35 до 0.429 при GSI = 65.

Наукова новизна. Кількісно встановлено домінуючу роль якості масиву у формуванні зони можливого руйнування в покрівлі очисного простору під час розробки малопотужних рудних тіл. Встановлено взаємозв'язок між GSI, кутом падіння рудного тіла та розміром потенційно нестійкої приконтурної області.

Практична значимість. Результати можуть бути використані для попереднього вибору напрямів коригування розмірів камер і ціликів, величини відкритого відслонення та параметрів контурного підривання. Безпосередній прогноз збіднення потребує об'єктного калібрування з урахуванням фактичної потужності та морфології рудного тіла й даних натурного моніторингу.

Ключові слова: малопотужне рудне тіло; золоторудне родовище; зона руйнування; чисельне моделювання; збіднення руди

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