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Support Design of Underground Cut and Fill Mine by using Hybrid Numerical Empirical Model

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Abstract

A detailed study to propose new design support systems for underground mine was carried out in this paper. The depth was increased to develop the production in accordance with the vein. The cut and fill mining method was selected for this mine in line with the geometry of vein, topographic specifications, and annual production rates. Excavationsize, distance between stope and decline, depth, and the impact of fault governedthe induced stresses. Different depths in multiple excavations were analyzed in this study especially on decline area. The rock properties of investigated areawere defined from the field and laboratory test for each rock type. Support systems for the decline were designed based on the suggestions of the Rock Mass Rating (RMR) system and the Q system for rock mass classification. Induced stress on different levels were calculated to define Stress Reduction Factor (SRF) and applied to predict the support system by Q system. New method namely Hybrid Numerical Empirical Model was developed to support requirement analysis in cut and fill mining method. The support system requirement influenced by depth. The shotcrete will be thicker and rock bolt space more closely as increasing the depth.

Keywords: Support system;Underground mine; Cut and Fill; Rock mass; Hybrid numerical empirical model

Introduction

Obtaining a stablethe underground mining isone of the main concernsforanunderground mine engineer. A stable underground mining is determined by the suitability and economical support system with a convenient excavation method.

Rock mass classifications are traditionally used to group areas with similar geomechanical characteristics. This classifications is expected to provide guidelines for stability performance, and to select appropriate support. The rock mass classifications have been successfully applied to many tunnel construction designs. Two widely-used rock mass classifications in underground mines are Rock Mass Rating (RMR) and Norwegian Geotechnical Institute (NGI, Q-system).



J Geol Geosci ISSN: 2329-6755 JGG, an open access journal The main objective of this paper is to predict the required support systems for cut and fill underground mine which focus on decline main ways. In this study, support system requirements were calculated based on rock mass classification systems, and considered of induced stresses. This research is important to propose support systems for underground mine openings on different level activities.

Kirsch, was the first publisher for the induced stresses distributions of a circular tunnel [1].Hoek and Brown, developed and ditional tool that can be predicting the induced stresses distributions [2]. The impact of induced stresses on stope stability was applied to Mathews empirical stability graph method [3,4]. Rock massclassification, *Qsystem* is proposed for predicting underground support system [5,6]. The effects of induced stresses are determined by comparing the maximum tangential stresses (σ_{θ}) to the intact uniaxial compressive strength (σ_{c}) in a similar manner to the Mathews' method. Wattimena, 2003 applied the Mathews'methodon block caving mines in Australia and proposed a Hybrid NumericalEmpirical Model (HNEM) to predict induced stresses and to calculate the *Q* system for estimating support requirement on undercut and production levels [7].

Geology

The field site for this study is Cibaliungunderground gold minewhichlocated in the western part of Java Island, Indonesia (Figure1). The resource is estimated to be approximately 1.5 million wmt gold with grade is estimated be 9.8 ppm. The type of ore is vein with low sulphidation. Geological mapping and geotechnical description were conducted in the field.

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Figure 2: Regional geology of Cibaliung (source: CSD).

Parameter	Brecci	a Smect	ite (hw)	Brecci	a Chlorit	e (fw)	Quartz	
	min	max	mean	min	max	mean	(Stope)	
σ_{c} (MPa)	2	68	24	5	153	51	74	
E (GPa)	1.38	37	21	3.21	97	57	2.52	
v	0.25	0.62	0.36	0.17	0.25	0.21	0.35	
C (MPa)	0.2	19	11	0.44	35	22	0.345	
φ (°)	35.5	41	38	39	40.5	33	38.6	
σ_t (MPa)	0.004	3.5	2.45	0.018	9	6.71	0.01	
p(ton/m ³)	2.50	2.51	2.505	2.56	2.60	2.58	2.70	

Table 1: Mechanical and physical properties of Cibaliung rock materials.

The host rock of Cibaliung deposit is in Honje Formation consists of andesite, basaltic andesite volcanic and volcanic breccia intercalated with tuffaceous sediment [8]. Andesite rock types, consists of andesite, andesite breccia, polymictic and monomictic breccia. Those host rocks are altered by chlorite-adularia and smectite-illite. The type of ore is a vein with low sulphidation epithermal deposit dominated by quartz vein. This vein follows the Cibaliung fault within NNW-striking/ENEdipping, with two shoot target Cikoneng and Cibitung ore shoot. The Cikoneng ore geometry is 250 m length, 2-10 m width and 200 m depth, whereas Cibitung shoot geometry is 150 m length, 2-15 m width and 300 m depth with dip of ore is 80°. Some minor faults occur throughout the hanging wall excavation with strike NE-SW dip 75°. Angel of this fault with axis of tunnel is 21°.The geology of Cibaliung can be seen in Figure 2 [9].

Three main rock types, quartz vein as ore body, andesite-breccia altered by chlorite-adularia on footwall abundance, and andesitebreccia altered by smectite-illite dominant on hanging wall rock are observed in the study area.

Engineering Properties

The rock mass rating (RMR) system [10,11] and the Qsystem developed by the Norwegian Geotechnical Institute [5] were used to classify rock masses. The physical and mechanical properties of the rocks were investigated based on rock mass classifications from the field and laboratory studies on intact rock samples.

These tests coveran evaluation of uniaxial compressive strength (σ_c), tensile strength (σ_c), Young's modulus (*E*), Poisson's ratio (ν), density (ρ), internal friction angle (ϕ), and cohesion (*c*). Uniaxial compressive strength, triaxial compressive strength, Brazilian and density tests were conducted in accordance with the ISRM standard. The values of minimum and maximum UCS varies, with the lower mean of UCS is in hanging wall rock. The results of rock properties tests are summarized in Table 1.

In this study, RMR and Q-system classifications were applied rock mass characterization. Some parameters were determined for RMR, includes uniaxial compressive strength (UCS), Rock Quality Designation (RQD), discontinuity spacing, discontinuity conditions, and groundwater conditions. The results of this study are shown in Table 2 to Table 4 for RMR.RQD, number of joint sets (J_n), joint surface roughness (J_r), joint weathering and alteration (J_a), joint water reduction factor (J_w), and stress reduction factor (SRF) are parameters for Q to quantify rock mass. RMR rating can be determined from the classification proposed by Bieniawski, [10]. Meanwhile, the Q value of a rock mass can be calculated by the equation given below, as determined in the classification proposed by Barton et al. [5].

$$Q = \frac{RQD}{J_n} x \frac{J_r}{J_a} x \frac{J_w}{SRF}$$
(1)

Parameters	Description	Rating	Description	Rating
Location(chainage)	565-568		591-593	
σ _c , Mpa	2-68	2-7	2-68	2-7
RQD (%)	55	13	55	13
Spacing of discontinuous (cm)	15-60	8	15-60	8
Condition of discontinuous	Slickenslide to slightly rough surfaces	10	Slickenslide to slightly rough surfaces	19
Groundwater	Damp to dry	10	Dry	15
Effect discontinuous	Very Unfavourable	-12	Unfavourable	-10
Total RMR		30-31		48
Classification		Poor		Fair

 Table 2: The rock mass classification of breccia smectite (hanging wall) based on the RMR.

Parameters	Description	Rating	Description	Rating
Location (chainage)	791-792		814-817	
σ _c , Mpa	5-153	2-12	5-153	2-12
RQD (%)	60	13	65	13
Spacing of discontinuous (cm)	20-60	10	20-60	10
Condition of discontinuous	Slickenslide to slightly rough surfaces	20	Slickenslide to slightly rough surfaces	20
Groundwater	Wet to damp	8	Damp to dry	13
Effect discontinuous	Unfavourable	-8	Unfavourable	-1
Total RMR		48		61
Classification		Fair		Good

 $\ensuremath{\text{Table 3:}}$ The rock mass classification of breccia chlorite (footwall) based on the RMR.

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Parameters	Description	Rating	Description	Rating
Location (chainage)	Xcut 4 1.6		Xcut 4 12.5	
σ _c , Mpa	74	7	74	7
RQD (%)	63	13	70-77	13-17
Spacing of discontinuous (cm)	20-60	10	>200	20
Condition of discontinuous	Slickenslide to slightly rough surfaces	19	Slickenslide to slightly rough surfaces	19
Groundwater	Wet to damp	10	Wet to damp	10
Effect discontinuous	Fair to unfavourable	-5	Fair to unfavourable	-5
Total RMR		54		64
Classification		fair		Good

Table 4: The rock mass classification of quartz vein (stope) based on the RMR.



Dis	stance	5		10		20		30		40	
	depth	hw	fw								
0	88	3.201	3.830	2.509	2.708	2.274	2.376	2.167	2.246	2.132	2.216
side	150	5.378	6.383	4.210	4.483	3.747	3.920	3.630	3.770	3.604	3.737
vall	200	7.139	8.440	5.602	5.882	4.985	5.167	4.871	5.054	4.749	4.963
>	300	10.681	12.518	8.389	8.728	7.429	7.614	7.254	7.472	7.155	7.396
	88	5.292	6.237	5.832	6.015	6.228	6.253	6.319	6.349	6.306	6.256
oť	150	9.551	10.508	10.044	10.849	10.904	11.382	11.265	11.168	10.828	11.700
2	200	11.904	13.149	13.649	13.866	13.986	14.577	15.135	14.001	14.382	15.382
	300	18.477	21.166	20.536	21.367	20.504	20.734	22.066	21.576	22.836	22.741

Table 5: The predicted tangential stress (MPa) in hanging wall (hw) and footwall (fw) declines from numerical modeling.

Design of Support Systems

Design of support systems was carried out by considering the in situ stress, stress around the openings, and evaluation of failure conditions. In situ stress was calculated from eq. (2) for different levels within each geological formation.

$$\sigma_{v} = \gamma . h = \rho . g . h \tag{2}$$

Where:

where.

- σ_{v} = vertical stress (MPa)
- γ = specific weight of the rock properties (MN/m³)
- h =depth of overburden (m)
- ρ = average density of rock properties (ton/m³)
- g = acceleration due to gravity (m/s²)

Hoek and Brown, proposed equations to calculate tangential stress around the roof and the wall of the openings on eq. (3) and eq. (4), respectively. The constants *A* and *B* are taken as 3.2 and 2.3 for a horseshoe-shaped gallery[2].

$$\sigma_{\theta r} = \sigma_{v} (B - k) \tag{3}$$

$$\sigma_{\theta_W} = \sigma_v (Ak - 1) \tag{4}$$

Since the influence of multiple excavations could not be predicted by the abovementioned equations, numerical modeling analysis is a proposed as a solution. Therefore Phase 2* software is useful to predict induced stress on two dimensions.

The tangential stresses are determined near the opening on the roof and wall side. Figure 3 shows the initial condition for numerical modeling. The results of tangential stresses by numerical modeling are given in Table 5andFigure4 shows the graphic of induced stress on different levels.

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The value of the ratio $\sigma_c/\sigma_{_{\Theta}}$	Description
>100	Near surface, low stress, open joints
100-3	No rock spalling/stable/medium, favorable stress condition/No rock stress induced instability
3-2	Low rock spalling/minor spalling/high stress, very tight structure/high stress, slightly loosening
2-1.5	Moderate rock spalling/severe spalling/moderate slabbing after 1 h/light rock burst or spalling
1.5-1	High rock spalling/heavy support required/slabing and rock burst/heavy rock burst
<1	Heavy rock burst/severe rock burst

Table 6: Rock burst classifications by Grimstad& Barton [6].

Position	Depth		Distance from HW decline to stope					Distance from FW decline to stope			
		5	10	20	30	40	5	10	20	30	40
0	88	7.50	9.57	10.55	11.08	11.26	13.32	18.83	21.46	22.71	23.01
side	150	4.46	5.70	6.41	6.61	6.66	7.99	11.38	13.01	13.53	13.65
vall	200	3.36	4.28	4.81	4.93	5.05	6.04	8.67	9.87	10.09	10.28
>	300	2.25	2.86	3.23	3.31	3.35	4.07	5.84	6.70	6.83	6.90
	88	4.54	4.12	3.85	3.80	3.81	8.18	8.48	8.16	8.03	8.15
oť	150	2.51	2.39	2.20	2.13	2.22	4.85	4.70	4.48	4.57	4.36
2	200	2.02	1.76	1.72	1.59	1.67	3.88	3.68	3.50	3.64	3.32
	300	1.30	1.17	1.17	1.09	1.05	2.41	2.39	2.46	2.36	2.24

Table 7: The value of ratio $\sigma_c / \sigma_{\theta}$ for calculating rock burst.

Tangential stresses at the boundaries increase as increasing depth of excavations. In general, the tangentialstresses at the boundaries of both hanging wall and footwall decline excavations in the sidewall decreased with the distance from the decline to the stope. Significant tangential stresses decreases was observed when the distance changed from 5 m to 10 m. The tangential stresses are relatively constant when the distance between declines and stope up to 30 m on the hanging wall decline and 20 m on the footwall decline.

The tangential stresses in the roof increase with increasing of depth. The trend of tangential stress for both hanging wall and footwall declines in the roofwas found to be relatively constant as increasing distance between declines and stope. These results are the basis data to predict damage and calculate the support requirement. Tangential stress on the roof is higher than that on the wall and considered to support analysis.

Prediction of Damage

The relation between intact rock material strength and tangential stress can be used to predict rock burst in underground. A high value for tangential stresses is a good indicator for rock burst. Rock burst potentially damage classified by Grimstad and Barton [6]. The classifications are summarized inTable 6. Calculated values between intact rock and tangential stress to predict rock burst potential are given in Table 7. The intact uniaxial strength of hanging wall decline was 24 MPa and footwall decline was 51 MPa.

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Location	RMR	Width (m)	Rock load	Span (m)	Rock density	Safety	Rock bolt	Numbor	Rock bolt grouted	Numbor
			noight (iii)		((())))))))))))))))))))))))))))))))))))	luotoi	capacity (ton)	(pieces)	capacity (ton)	(pieces)
Llonging well	30	4.2	2.94	2.4	2.505	1.5	7	15.91	15	7.42
Hanging wai	48	4.2	2.184	2.4	2.505	1.5	7	11.81	15	5.51
Footwall	48	4.2	2.184	2.4	2.58	1.5	7	12.17	15	5.68
Footwall	61	4.2	1.638	2.4	2.58	1.5	7	9.13	15	4.26

Location	Geometry (m)	RMR	Rock load pressure (ton/m²)	USB shotcrete (ton/m²)	Safety factor	Rock density (ton/m ³)	Shotcrete thickness (m)
Llanging wall	4.2 x 4.8	30	7.36	2500	1.5	2.505	0.040
Hanging wai	4.2 x 4.8	48	5.47	2500	1.5	2.505	0.030
Footwall	4.2 x 4.8	48	5.63	2500	1.5	2.580	0.031
	4.2 x 4.8	61	4.22	2500	1.5	2.580	0.023

Table 8: The rock bolt requirement.

Table 9: The shotcrete thickness.

Based on the relation between intact rock and tangential stress, the damage potential in the roof is higher than that in the wall sides. Given the condition in the hanging wall, a rock damage will be more likely to occur in the hanging wall when compared with the footwall excavations. The results suggest that there will be minor rock spalling when the activities below 300 m depth in the wall sides on hanging wall. However, low to moderate rock spalling will take place in the roof when the mine activities occurs under 150-200 m depth. At the depth lower than 300 m, high rock spalling is predicted to occur and necessitating a heavy support. Although the condition on the footwall was observed to be relative stable, low rock spalling could occur at the depth below 300 m.

RMR Support Design

Bieniawski, proposed guidelines for excavation and support of 10 m span rock tunnels in accordance with the RMR system. The current support system in Cibaliung underground gold mine determined by using RMR. The support systems of individual openings were designed according to the surrounding rock mass characteristic. The equations proposed by Barton et al. [12] and Unal[13], which includes rock mass classifications, excavation width, span excavation, and material rock density was used to determine the splitset number and shotcrete thickness to support requirements [13]. The results from eq. 5 to eq.8 are showninTable 8 and Table 9.

$$L = \frac{2 + 0.15B}{ESR} \tag{5}$$

$$P_{RMR} = \frac{100 - RMR}{100} B.\rho \tag{6}$$

$$n = \frac{B.ht.c.\rho.SF}{R_{\max}}$$
(7)

$$t = 0.434 \frac{P_{RMR}.B}{\tau} \text{ and } \tau = 0.2 \frac{LB}{SF}$$
(8)

Where:

L = rock bolt length (m)

B = excavation width (m)

ESR = Excavation Support Ratio

n = number of rock bolt (pieces)

ht = rock-load height (m)

c = span excavation (m)

 ρ = rock density (ton/m³)

SF = safety factor

 R_{max} = bearing capacity (ton)

LB = UCS shotcrete (ton/m²)

t =shotcrete thickness

 τ =shotcrete shear stress (ton/m²)

 P_{RMR} = rock-load pressure (ton/m²)

Based on the equations above, the hanging wall with poor rock mass needs 16 rock bolts and only 12 rock bolts for fair rock mass. In additional, some rock bolt with grouted are also needed to support the systems. Eight grouted rock bolts and six grouted rock bolts are required for poor and fair rock masses, respectively. Shotcrete will be used as the support system. The capacity of shotcrete proposed around 25 MPa for all area withthe minimum thickness of shotcrete is 45 mm for hanging wall poor rock type and 35 mm for fair rock type.

However, the capacity of rock boltandshotcrete for hanging wall and footwall are identical, the numbers of rock bolt on footwall are less than the hanging wall due to the difference rock masses and rock properties. The numbers of rock bolt for footwall are 13 and 10 for fair and good rock mass, respectively. Six grouted rock bolts are needed for fair rock mass, and five grouted rock bolts for good rock mass. The thickness of shotcrete for fair rock mass in footwall is similar with shotcrete on hanging wall around 35 mm, and for good rock mass the thickness of shotcrete around 25 mm for the footwall.

Q System Support Design

In 2003, Wattimena, etalhasproposed a Hybrid Numerical-Empirical model to predict the support requirements in block caving mining method [7]. This model is the combination of numerical modeling and empirical analysis to design the support requirements. From numerical modeling, induced stress on complex design of mine method can be predicted, and the result is applied to the empirical Q system to design Citation: Purwanto, Wahyudi S, Shimada H, Sasaoka T, Wattimena R, et al. (2014) Support Design of Underground Cut and Fill Mine by using Hybrid Numerical Empirical Model. J Geol Geosci 3: 148. doi: 10.4172/2329-6755.1000148

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	Description	σ,/σ,	σ_{e}/σ_{c}	SRF (Barton et al. 1974 Grimstad and Barton, 1993)
Н	Low stress, near surface, open joints	>200	<0.01	2.5
J	Medium stress, favourable stress condition	200-10	0.01-0.3	1
к	High stress, very tight structure. Usually favourable to stability, may be unfavourable for wall stability	10-5	0.3-0.4	0.5-2
L	Moderate slabbing after > 1 hour in massive rock	5-3	0.5-0.65	5-50
м	Slabbing and rock burst after a few minutes in massive rock	3-2	0.65-1	50-200
N	Heavy rock burst and immediate dynamic deformations in massive rock	<2	>1	200-400

Table 10: Classification of SRF parameter used in Q system (after Barton et al. [5] and Grimstad and Barton [6]).

Position			Distance bet	ween HW de	cline to stope		Distance between FW decline to stope				
	depth	5	10	20	30	40	5	10	20	30	40
0	88	0.13	0.10	0.09	0.09	0.09	0.08	0.05	0.05	0.04	0.04
side	150	0.22	0.18	0.16	0.15	0.15	0.13	0.09	0.08	0.07	0.07
/all	200	0.30	0.23	0.21	0.20	0.20	0.17	0.12	0.10	0.10	0.10
5	300	0.45	0.35	0.31	0.30	0.30	0.25	0.17	0.15	0.15	0.15
	88	0.22	0.24	0.26	0.26	0.26	0.12	0.12	0.12	0.12	0.12
oť	150	0.40	0.42	0.45	0.47	0.45	0.21	0.21	0.22	0.22	0.23
ē	200	0.50	0.57	0.58	0.63	0.60	0.26	0.27	0.29	0.27	0.30
	300	0.77	0.86	0.85	0.92	0.95	0.42	0.42	0.41	0.42	0.45

Table 11: Ratio between tangential stress and intact rock strength values.

$\sigma_{e'}/\sigma_{c}$	SRF (Barton et al. 1974 Grimstad and Barton, 1993)	Location
0.04 - 0.3	1	Hanging wall: Wallsides, depth 88 m- 200 m. Roof; depth 88 m. Footwall: Wallsides; all area. Roof; depth 88 m-200 m.
0.3 – 0.47	0.5 – 2	Hanging wall: Wallsides; depth 300 m. Roof; depth 150 m. Footwall: Roof, depth 300 m.
0.5 - 0.63	5 – 50	Hanging wall: Roof, depth 200 m
0.77 – 0.95	50 – 200	Hanging wall: Roof, depth 300 m

Table	12:	SRF	classification.
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Location	chainage	RQD	Jn	Jr	Ja	Jw	Q values on depth (m)			
							88	150	200	300
Hanging wall	565-568	55	9	2	6	1	2.04	1.02	0.04	0.01
	591-593	55	12	2	3	1	3.06	1.53	0.06	0.02
Footwall	791-792	60	9	3	3	0.66	4.40	4.40	4.40	2.20
	814-817	65	6	3	3	0.66	7.15	7.15	7.15	3.58

Table 13: Q values in the roof.

support system. To calculate the Q values as eq. (1), induced stress is important as parameter for SRF. To describe SRF, Barton et al. [5] and Grimstad and Barton [6] proposed the relation between tangential induced stress and intact rock uniaxial strength[5,6]. Table 10 shows the classification of SRF, and the ratios between tangential stress and intact rock uniaxial strengthare given in Table 11.

By using Q system to design the underground support, the influence of induced stress on differentlevels can be predicted. Based on SRF values, we can obtain minimum and maximum range of SRF as input to calculate the Q values. Based on the values of ratio between tangential stress and intact rock strength, the research area could be classified into four classes of SRF. The classification of the SRF is given in Table 12. To determine Q values, other factorsas seen in Table 13 should be considered as well. The results of Q values are shown in Table 13.When the SRF classification of Barton et al. [5] and Grimstad and Barton [6] was applied, the range of Q values are 0.01 - 2.04 and 0.02 - 3.06 (extremely poor to poor). Meanwhile, the Q values are between

2.20–4.40 and 3.58 – 7.15 which mean poor to fair. Based on Table 13, it can be seen that the lowest Q values based on calculation using eq. (1) is located on the roof. Therefore the roof condition will be considered to analyze the support requirements.

The Qvalue is then used to predict the supportsystem requirement. Two additional parameters should be considered in using the Q system for predicting the support systemfor an underground excavation are Excavation Support Ratio (ESR) and equivalent dimension.

Barton et al. [5] determined equivalent dimension (D_{e}) from the following equation:

$$D_e = \frac{span \ or \ Height(m)}{ESR} \tag{6}$$

ESR values are taken as 1.60 for permanent decline, ramp, and shaft that are expected to serve for the whole production life of mine. The height of the hanging wall and the footwall declines are 4.8 m. The value of D_e is 3 mwhen the ESR for permanent decline. The support system can

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	Location	Support	Depth							
	Ch.		88 m	150 m	200 m	300 m				
Hanging wall	565-568	Rock bolt length	1.7 m	1.7 m	1.7 m	1.7 m				
		Bolt spacing	1.6 m	1.7 m	1.2 m	1.0 m				
		Shotcrete	unreinforced shotcrete, 40-50 mm	unreinforced shotcrete, 40- 50 mm	fibre reinforced shotcrete, 50-90 mm	fibre reinforced shotcrete, 120-150 mm				
	591-593	Rock bolt length	1.7 m	1.7 m	1.7 m	1.7 m				
		Bolt spacing	1.4 m	1.3 m	1.2 m	1.0 m				
		Shotcrete	unshotcrete	fibre reinforced shotcrete, 90-120 mm	fibre reinforced shotcrete, 50-90 mm	fibre reinforced shotcrete, 120-150 mm				
Footwall	791-792	Rock bolt length	1.7 m	1.7 m	1.7 m	1.7 m				
		Bolt spacing	1.6 m	1.6 m	1.6 m	1.4 m				
		Shotcrete	unshotcrete	unshotcrete	Unshotcrete	unshotcrete				
	814-817	Rock bolt length	1.7 m	1.7 m	1.7 m	1.7 m				
		Bolt spacing	1.5 m	1.5 m	1.5 m	1.4 m				
		Shotcrete	unshotcrete	unshotcrete	Unshotcrete	unshotcrete				

Table 14: Support system recommended based on Q system.

be determined by plotting the D_e values against Q that was proposed by Barton et al. [5]. The sample of the support requirement estimation is given in Figure 5.

For the hanging wall decline when the activities up to 88 m, the Q values are between 2.04–3.06. As a conservative approach, Q values of 2.04 is used to support system analysis. The support requires includeshotcrete 4-5 cmthickness, and bolt with length 1.7 m and spacing 1.8 m. The support required will be increased with the increasing depth. This can be seen from the bolt spacing. For 150 m depth, the Q values are between 1.02-1.53. Within this range of Q value, the supportsystem requiresshotcrete with 4-5 cmthickness, and bolt with length 1.7m and spacing 1.7 m. The support system needs

shotcrete 9-12 cm thickness, and bolt with length 1.7 m and spacing 1.2 m when at 200 m depth. The support system for activities up to 300 m needs rock bolt with 1.7 m length and 1.0 m spacingtogether with shotcrete 12-15 cmthickness.

For the footwall decline, the Q values are higher than that on the hanging wall decline. Similar with the hanging wall decline, the highest tangential stresses occur in the roof. The stress was observed to increase with increasingdepth. For example, when the activities took place at depth \leq 200 m, the Q value is estimated to be in the range of 4.40-7.15 and classified under faircategory. The proposed support system requiresbolt length 1.7 m with space 1.6 m. and shotcreteis not necessary. When the activities increase up to 300 m, the support system includes shotcrete 4-5 cm, bolt length 1.7 m and bolt space 1.8 m. Detail predicted supports are given in Table 14.

Conclusion

Recommended support systems for different levels (88 m, 150 m, 200 m and 300 m depth) were designed using a new method; Hybrid Numerical Empirical Model.

The following results were obtained from damage potential analysis. Damage potential will increase with increasingthedepth. Low to moderate rock spalling will occur for hanging wall on150-200 m depth in the roof, and there will be high rock spalling when the mine activities below 300 m depth. In contrast, footwall will start encounter low rock spalling when the activities below 300 m.

It was recommended by the RMR system classification to use rock bolts and shotcrete. In hanging wall decline, minimum rock bolts for poor rock mass are 16 pieces and8 pieces of rock bolts grouted. The shotcrete is required for this condition with minimum thickness is recommended 45 mm and the capacity minimum 25 MPa.However, footwall classified as good rock mass, the support system is needed. For the good condition of rock mass 10 pieces of rock bolt and 5 pieces of rock bolt grouted are required with minimum 25 mm thickness of shotcrete.

Based on Qsystem, which is analysed by using Hybrid Numerical Empirical Model the requirement of rock bolts length aresame for all rock mass classifications that are 1.7 m. Recommended support system by Q classification influenced by depth. The shotcrete will be thicker and rock bolt space more closely as increasing the depth. When the activities at 88 m depth, the support system requires shotcrete 4 - 5 cm

thickness and bolt spacing 1.8 m.Meanwhile, the shotcrete is 12 – 15 cm thickness and bolt spacing is 1.0 m are required for depth up to 300 m. From this study the advantage to calculate support requirements using SRF is influence of induced on different levels could be predicted.

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