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Optimization of Information Point Distances in the Estimation and Classification of Coal Resources Using Geostatistical Methods Compared to SNI 5015:2019 for Moderate Geological Conditions

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ABSTRACT

The reference currently used in estimating and classifying coal resources in Indonesia is SNI 5015:2019, which refers to the Australian Guidelines for Estimating and Reporting of Inventory Coal, Coal Resources, and Coal Reserves, 2003 Edition. However, several changes have emerged with the issuance of The JORC Code, 2012 Edition and Australian Guidelines for The Estimation and Classification of Coal Resources, 2014 Edition. One of the changes is in calculating geostatistical aspects in the estimation and classification of coal resources. In this study will be discussed about the need for the use of geostatistical methods and evaluation of SNI 5015:2019. The purpose of this study is to determine the optimal borehole spacing, determine the classification and estimation of resources using the geostatistical method, and compare it with SNI 5015:2019. The method used is the kriging relative error method and global estimation variance. The two methods give different results from SNI 5015:2019. This thing exactly gives different resource estimation results. This difference indicates the need to evaluate the classification and estimation system of SNI 5015:2019, especially related to the use of geostatistical methods accompanied by geological interpretations that describe the actual state of the research location.

INTRODUCTION

In the coal resources classification, Indonesia still refers to SNI 5015:2019 [2], which only considers aspects of geological conditions in the classification system while other factors such as quality continuity such as inherent moisture, volatile matters, calory value, and others are not particular aspects of the assessment in determining the distance of information points (area of influence). The Australian Guidelines for the Estimation and Classification of Coal Resources 2014 Edition [4] document itself has used geostatistical aspects in specific considerations regarding the determination of the distance of information points for resource classification, where this will undoubtedly make a justification in the estimation and classification of resources relevant and closer to the actual situation. In response to this, this study was carried out to optimize the distance of information points in the estimation and classification of resources in an area with the Global Estimation Variance (GEV) method in order to obtain the optimal distance of information points and under the characteristics of data distribution and actual geological conditions and compare the results with SNI. 5015:2019.

LITERATURE REVIEW

Univariate statistics is a statistical method used to analyze the relationship between each data from a population regardless of the location of the data. It consists of two types, namely measures of central tendencies (central tendency) such as mean, median, and data variance measures (scatter) such as standard deviation variance, slope (skewness), and coefficient of variance (covariance). Meanwhile, bivariate statistics aims to see whether or not there is a correlation between parameter Y and parameter X, and this determines whether or not data accumulation is necessary due to similar data characteristics [9]. In carrying out geostatistical analysis, especially in semivariogram fittings, the univariate and bivariate statistical analysis must be carried out first to see each data's characteristics and correlations and not produce biased data and seem overconfident vice versa. Meanwhile, in SNI 5015:2019, the classification of resources is divided based on different geological conditions with Diehl and David [7] (Table 1) or de Souza et al. [6] (Table 2), who classify coal resources based on the level of error (relative error). it considers aspects of spatial data correlation in its analysis and decision making.

Table 1. Resource Classification Based on Error Value and Confidence Level (Diehl & David 1982)

<table>
<tr><td rowspan="4">Recources</td><td colspan="3">Identified</td><td colspan="2">Undiscovered</td></tr>
<tr><td colspan="2">Demonstrated</td><td colspan="3" rowspan="2"></td></tr>
<tr><td>Measured</td><td rowspan="2">Indicated (possible)</td></tr>
<tr><td>proved probable</td><td>Inferred</td><td>Hypothetic</td><td>Specul</td></tr>
<tr><td>Error tolerance</td><td>± 10% ±20%</td><td>± 40%</td><td>±60%</td><td></td><td></td></tr>
<tr><td>Confidence level</td><td>>80% 60-80%</td><td>40-60%</td><td>20-40%</td><td>10-20%</td><td><10%</td></tr>
<tr><td></td><td colspan="3">Economically significant resources</td><td colspan="2">Resources base</td></tr>
</table>

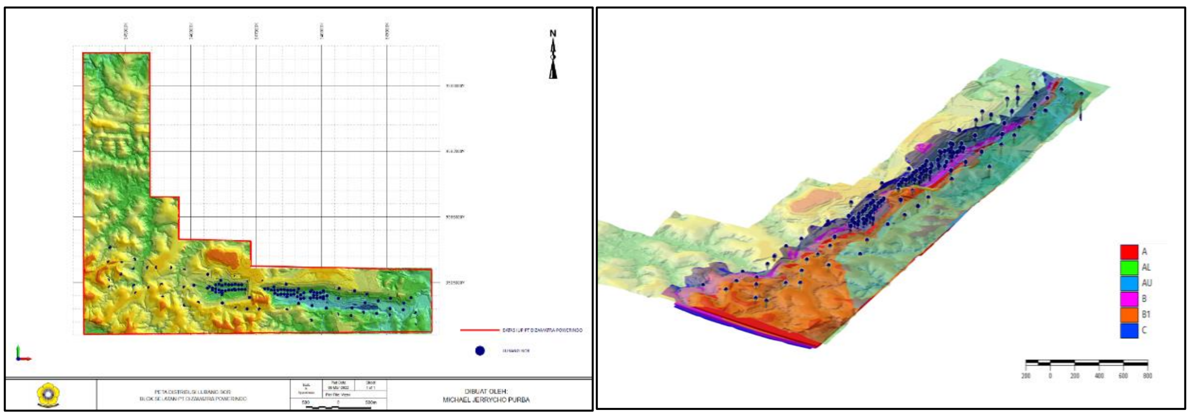
Table 2. Resource Classification Based on Error Value (de Souza dkk. 2004)

Resource Classification	Maximum Estrapolation Distance	Maximum Spce Of Information Points	*Error tolerance*
Measured	500 m	+1 km:<500m	0-10%
Indicated	1000 m	+2 km:<1 km	10-20%
Inferred	2000 m	+4 km	>20%

RESEARCH METHODOLOGY

The coal seams studied in this study were A, B, and C seams in a mine in the Lahat area, South Sumatra. The initial input data prepared in this study were:

- Model of coal deposit form in the form of Seam Block Model
- Exploration data such as data collar, quality, and lithology of coal deposits
- Concession map data (IUP) of the area under study
- Geological maps and geological characteristics of the area under study



(a) (b)

Figure 1. Distribution of Boreholes in Study Area (a) The Model of Coal Deposit in The Study Area (b)

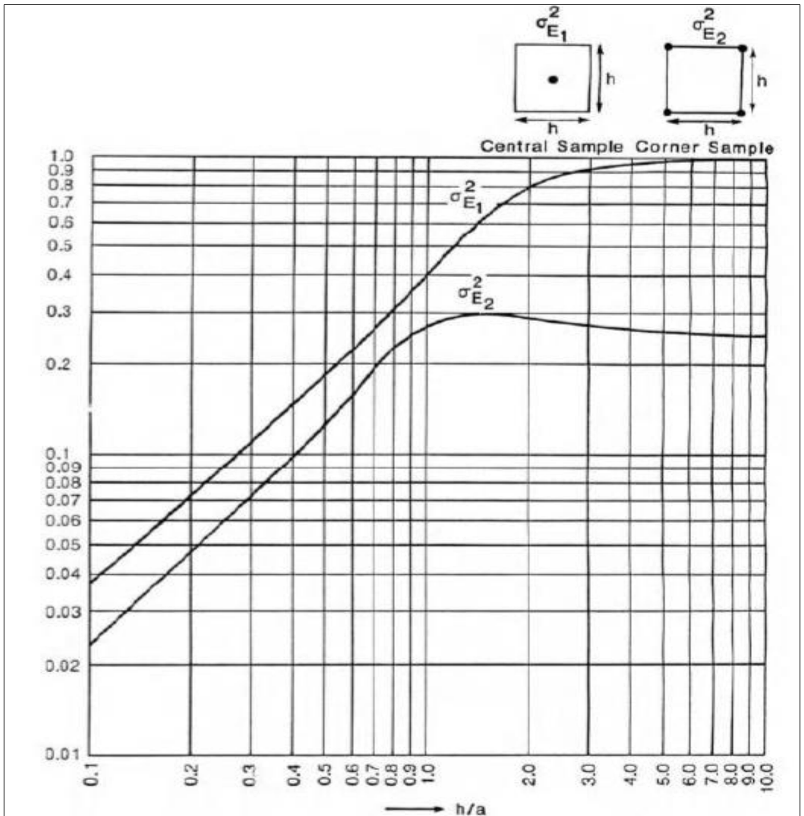
Through the calculation of Global Estimation Variance (GEV) will be obtained relative error values [8] which obtained from several processes ranging from data preparation using univariate and bivariate statistical analysis, fitting the variogram, determining the estimated variance (σ_E^2) with an extension variance nomogram (estimation of variance).) with nugget 0 and sill [1] (Figure 2), adjust the nugget and sill for each parameter using and generate the estimated variance for that parameter then (σ_r^2) (1) determine the global variance of the study area (σ_R^2) (2) and ends with the determination of the relative error (3)[5].

After determining the relative error, the Drill Hole Spacing Analysis (DHSA) will be carried out on each seam and obtain results in the form of information point distances for each coal resource classification. The end of the research process is to obtain tonnage by estimating and classifying resources using the relative error method and SNI 5015:2019.

$$\sigma_E^2 r = C0 + (C\ x\sigma_E^2\ ) \quad (1)$$

$$\sigma_E^2 R = \frac{\sigma_E^2 r}{N} \quad (2)$$

$$Relative\ Error = \pm\ 1.96\ .\ \sigma_E\ .\frac{100\ \%}{Mean} \quad (3)$$



Gambar 3.12 Nomogram Nilai Varian Ekstensi/Estimasi (σ_e^2) Titik Terhadap Bidang Bujur Sangkar untuk Model *Spherical* (Annels, 1991)

Figure 2. Nomogram of Extension Variant Value Towards Square Field with Spherical Variogram Model

FINDING AND DISCUSSION

Descriptive Statistics Analysis

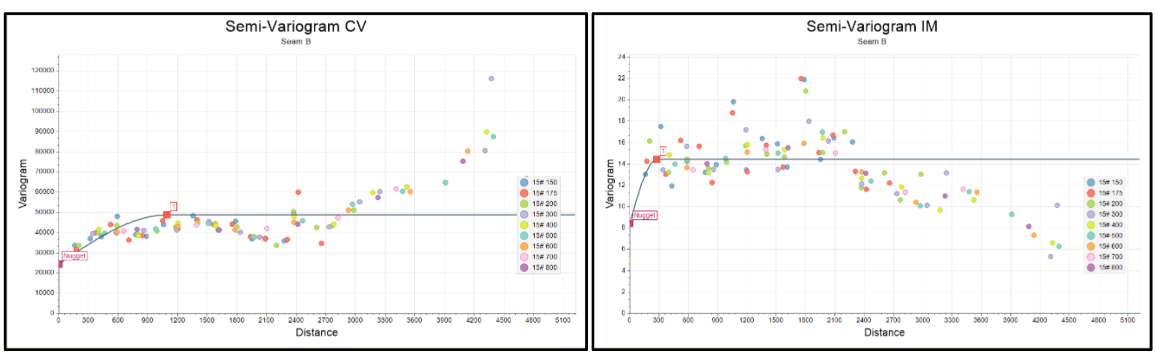
This analysis was carried out on seam A, seam B, and seam C and based on coal quality data, including volatile matters, inherent moisture, fixed carbon, and calories (calory value). And the geometry is the thickness and relative density of each coal seam at the research location. The quality and thickness data for seam A amounted to 140 data, for seam B totaled 173 data, and seam C, totaled 74 data. Univariate statistical analysis is intended to consider the value of random or random data regardless of the location of a sample. The results of descriptive statistical analysis for each seam's quality and geometry data can be seen in Table 3. The first parameter that will be the center of the analysis is the coefficient of variation. From the numerical series, all parameters on all seams have a coefficient of variation smaller than 1.5, so it can be said that the existing data is still within normal limits, so there is no need to cut outliers for the upper and lower limits of descriptive statistical data for each coal seam, and this can also be a justification that all data can be used in the further analysis [8]. From the results of the descriptive statistics of Seam A, it can be seen that there is a reasonably good distribution of data on each parameter that can reflect the level of data continuity. This is indicated by the coefficient of variation of each parameter, where the value is still less than 0.5. As is known, the coefficient of variance can indicate the condition of the distribution (variability) of the existing data. A high coefficient of variance indicates a wide distribution of data.

Table 3. Descriptive Statistics Seam A, B, and, C

Quality	Seam	Minimum	Maximum	N	Mean	Variance	Std Dev	Weighted Mean	Coeff. of Variation	Mode	Median
CV	A	4,015.00	5,119.00	140	4,559.0	40,174.34	200.44	4,565.05	0.044	4,582.00	4,563.50
FC	A	33.14	43.15	140	38.20	5.09	2.26	38.16	0.059	36.60	38.16
IM	A	9.14	25.03	140	17.90	13.80	3.72	17.97	0.208	19.41	18.47
RD	A	1.03	1.27	140	1.16	0.01	0.08	1.16	0.065	1.27	1.14
VM	A	32.48	47.07	140	39.06	6.14	2.48	38.99	0.063	38.72	38.71
THICKNESS	A	4.27	16.20	140	13.50	2.08	1.44	13.50	0.107	14.19	13.70
CV	B	3,843.00	5,117.00	173	4,618.0	44,731.52	211.50	4,616.44	0.046	4,627.04	4,638.00
FC	B	32.13	44.15	173	37.12	7.41	2.72	37.05	0.073	35.20	36.58
IM	B	9.99	24.94	173	17.52	13.43	3.67	17.56	0.209	19.81	18.38
RD	B	1.05	1.34	173	1.17	0.00	0.06	1.17	0.053	1.25	1.17
VM	B	2.52	47.36	173	38.26	46.38	6.81	38.30	0.178	39.52	39.30
THICKNESS	B	10.67	20.00	173	17.56	2.12	1.46	17.56	0.083	18.60	17.60
CV	C	4,053.00	5,359.00	74	4,635.0	54,290.20	233.00	4,636.89	0.050	4,583.04	4,628.00
FC	C	25.73	42.81	74	37.15	6.28	2.51	37.17	0.067	35.42	37.25
IM	C	9.67	23.88	74	18.59	9.90	3.15	18.49	0.169	13.80	18.91
RD	C	1.02	1.24	74	1.12	0.00	0.05	1.12	0.042	1.14	1.12
VM	C	34.78	43.79	74	38.47	3.10	1.76	38.54	0.046	36.10	38.50
THICKNESS	C	4.15	9.05	74	7.44	1.35	1.16	7.44	0.156	7.90	7.90

Variogram and Geostatistics

The statistical description presented in the previous discussion is a description that is only based on the results of the quality analysis, without regard to the position (location) of the distribution of the data [9]. To find out the pattern of data distribution, geostatic analysis can be used so that the direction and variation of the data distribution (anisotropy/isotoropy) can be known. The importance of the geostatistical method is because it also considers the position of the distribution of the data so that the direction and variation of the data distribution can be known. In contrast to the results of statistical analysis, which only describes data based on the quality analysis results only. The analyzed variables are geometry data and quality data, ignoring the borehole position that does not have quality or geometry data.



(a) (b)

Figure 3. Distribution of Boreholes in The Study Area (a) The Model of The Coal Deposit in The Study Area (b)

Table 4. Omni-Directional Semivariogram Parameters

Seam	Parameter	Model	Nugget	Sill	Range	CoV
A	CV	*Spherical*	3264.029514	51804	708.125	0.04
A	FC	*Spherical*	4.156196218	5.36265	168.643	0.06
A	IM	*Spherical*	5.55740048	15.1882	311.481	0.21
A	RD	*Spherical*	0	0.006486648	429.289	0.06
A	VM	*Spherical*	2.953119975	6.63481	323.921	0.06
A	THICKNESS	*Spherical*	0.585466039	2.4972	273.651	0.11
B	CV	*Spherical*	24485.42142	48725.2	1092.743	0.05
B	FC	*Spherical*	4.216893915	7.91191	160.397	0.07
B	IM	*Spherical*	8.439269816	14.429	278.923	0.21
B	RD	*Spherical*	0.000899054	0.004246	644.398	0.05
B	VM	*Spherical*	0	58.5223	286.071	0.18
B	THICKNESS	*Spherical*	1.254148568	2.34073	160.397	0.08
C	CV	*Spherical*	761.7659045	85524	1447.216	0.05
C	FC	*Spherical*	0	6.998202211	521.619	0.07
C	IM	*Spherical*	0	11.2450298	541.003	0.17
C	RD	*Spherical*	0.00045992	0.00236249	309.441	0.04
C	VM	*Spherical*	0	3.37091	478.947	0.05
C	THICKNESS	*Spherical*	0.271259246	1.74266	1372.898	0.16

fitting process is carried out in the azimuth direction of N 0°E, dip 0°, and angle tolerance of 90° (Omni-directional). The maximum distance is ± 800 m with a lag distance of 50-100 m (based on data distribution and the average drill hole distance), and the number of lags varies to facilitate the variogram fitting. From the results of the variogram fitting, the range, sill, and nugget variance values will be obtained.

Global Estimation Variance dan Relative Error

The global estimation variance (GEV) obtained from calculations based on the nomogram model is then used to estimate the relative error value. Next, the plotting between the relative error values and the borehole spacing is carried out to create a Drill Hole Spacing Analysis (DHSA) graph. From the DHSA graph, it is then known that the area of influence is for measured, indicated, and inferred resources, with the relative error values being less than 10% (measured), 10-20% (indicated/indicated), and 20-50. % (inferred). According to Bertolli [3], the borehole spacing reaches the optimum point when the relative error value is exactly ±10% for the measured, ±20% for indicated, and ±50% for inferred. The geostatistical parameters for the nugget variance (c0), sill (c), and range (a) values obtained from the variogram fitting results will be used in the following process to determine the optimum drill distance, which is entered in the calculation table (Table 5) with the Global Estimation method. Variance (GEV) calculates the relative error value and determines the optimum drill spacing. The description of table 5 below is the mean value taken from descriptive statistics for each parameter of seam A, seam B, and seam C, while h and l are drill spacings which are added up in multiples of 100 m assuming h & l are the same areas. The value of _X is the difference between the maximum X and minimum X coordinates divided by the borehole spacing, and _Y is the difference between the maximum Y and minimum Y coordinates divided by the drill spacing. The value of N is the product of _X and _Y.

Furthermore, the parameters obtained from the variogram fitting results are entered, namely range (a), nugget variance (C0), and sill (C). Furthermore, the extension/estimated variance is obtained by reading the drill spacing (h)/range (a) on the nomogram, as shown in Figure 2. Furthermore, the value of the variance of the point estimate to the planned plane of the drill spacing projection (r) must be adjusted again with the nugget variance and sill values for each parameter. After that, looking for the global estimation variance with the ratio between the point estimate variance value to the field and the amount of data (N) will produce a global estimate variance (R). Then perform calculations on the value of the global standard deviation, which is the square root of the global estimated variance (R). The last calculation stage is to find the relative error value by multiplying the confidence level, which is a constant 1.96, then multiplied by the standard deviation divided by the average value of the quality or thickness parameter obtained from descriptive statistical calculations, so in the end you will get the relative error value for each distance multiple. 100 meters. The results of the calculation of the relative error value will then be made a graph of the Drill Hole Spacing Analysis (DHSA) graph in logarithmic form using Microsoft Excel software based on the theory of Bertolli et al. [3] and Cornah et al. [5] as in Figure 4 for DHSA seam A.

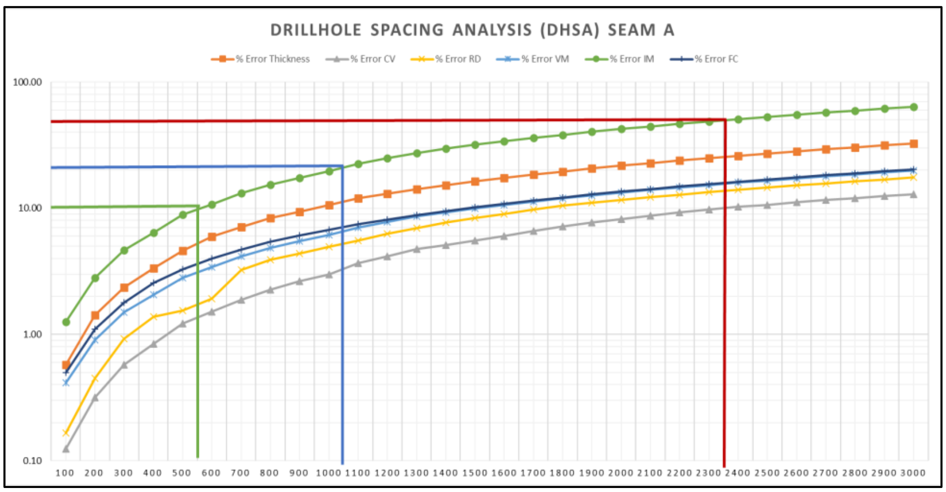


Figure 4. Seam A Drillhole Spacing Analysis Chart

Drillhole Spacing Analysis Chart

The second stage is creating a DHSA chart that also uses excel by reading the drill spacing values and the relative error of the four quality parameters and one thickness parameter. To draw drill spacing lines based on relative error based on Bertolli's theory [3] by reading the distance reached by the relative error value when it reaches the values of 10%, 20%, and 50%, then that is the optimum distance. This graph was created using Microsoft Excel software by reading the drill spacing values (x-axis) and relative error (y-axis) of the four quality parameters and one geometry parameter. To draw drill spacing lines based on a relative error, namely based on Bertolli's (2013) theory, namely reading the distance reached by the relative error value when it reaches the value of 10%, 20%, and 50%, then the distance is the optimum for each resource. From the graph in Figure 4, we get the optimum distance of information points (drill hole sampling) for seam A, for resource classification measured at a distance of 500 m, indicated at a distance of 1000 m, and inferred at a distance of 2300 m. From the graph, it can also be concluded that the low the distance radius of the information point (range) owned by IM is in line with the high CoV (coefficient of variation) of the inherent moisture itself, which is shown in table 4, which is indeed the highest CoV in seam A, it can be justified that the range will be shorter with increasing the value of the coefficient of variation (CoV) of a parameter.

Table 5. The Example of Global Estmation Variance and Relative Error Seam A (Thickness)

Mean	h	l	_X	_Y	N	a	C0	C	h/a	i/a	Varian Ekstensi	[illegible] (r)	[illegible] (R)	[illegible] ((R)	% Relative Error
13.50	100	100	46.9083	11.7147	549.5167	273.651	0.585466	2.4972	0.37	0.37	0.110	0.860158	0.001565	0.039564	0.574323598
13.50	z	200	23.45415	5.85735	137.3792	273.651	0.585466	2.4972	0.73	0.73	0.290	1.309654	0.009533	0.097638	1.417345809
13.50	300	300	15.6361	3.9049	61.05741	273.651	0.585466	2.4972	1.10	1.10	0.410	1.609318	0.026357	0.16235	2.356729453
13.50	400	400	11.72708	2.928675	34.34479	273.651	0.585466	2.4972	1.46	1.46	0.500	1.834066	0.053402	0.231088	3.354555727
13.50	500	500	9.38166	2.34294	21.98067	273.651	0.585466	2.4972	1.83	1.83	0.650	2.208646	0.100481	0.316988	4.601512432
13.50	600	600	7.81805	1.95245	15.26435	273.651	0.585466	2.4972	2.19	2.19	0.800	2.583226	0.169233	0.411379	5.971727615
13.50	700	700	6.701186	1.673529	11.21463	273.651	0.585466	2.4972	2.56	2.56	0.850	2.708086	0.241478	0.491404	7.133403724

Comparison of Information Point Distance (Area Of Influence) and Tonnage Relative Error Method VS SNI 5015:2019

With different total tonnages, this difference can be used as an evaluation of SNI 5015:2019 in order to use geostatistical considerations and provide recommendations for the distance of information points, not only to focus on geological conditions and complexity. It is proven by a geostatistical study in the form of relative error, it can be obtained that the distance of the information point is farther away than the classification based on SNI 5015: 2019, but the relative error method itself is susceptible to the high and low variability of the data so that data preparation is needed first before processing the data so that The data is typically distributed and can be estimated or processed at a different level according to the information needs to be obtained. Meanwhile, the polygon method used by SNI 5015:2019 is a resource estimation method that assumes a value (such as a thickness value) as the average value of a specific block size. For certain conditions, where the deposit has a good distribution of data, this method will give good results and vice versa; sometimes, the relationship between the data held in a place is farther than the distance set by SNI 5015:2019 itself, and that is what we can see in the results of this study where the distance of influence calculated geostatistically using a coal quality and thickness database in the research location has a more extended range than the classification using only geological complexity.

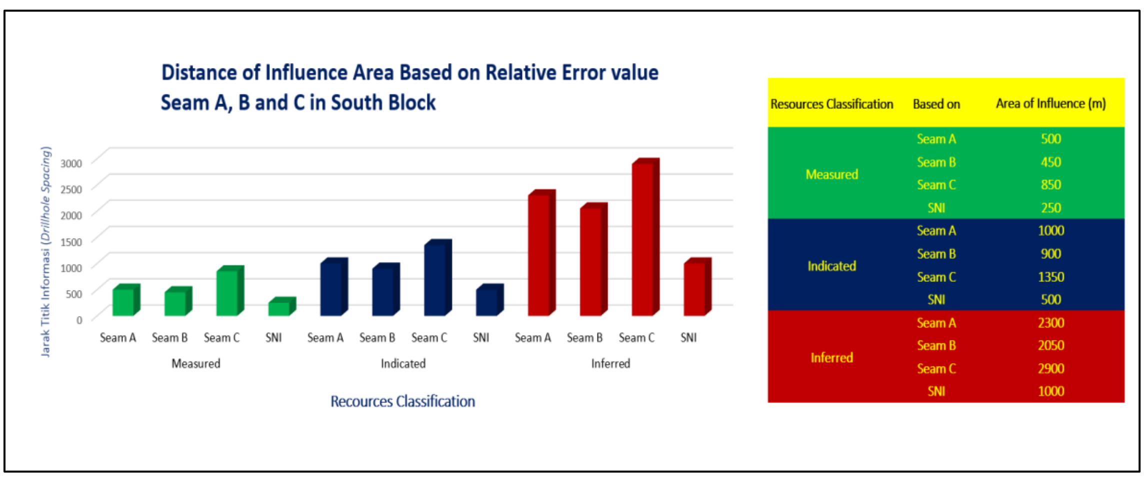


Figure 5. Distance of Influence Area Based on Relative Error value Seam A, B and C

The total accumulation of measured and indicated resources using the relative error method is 4,434,692 million tons more than using the SNI 5015:2019 polygon method. This difference occurs because there are inferred resources in the resource classification using the SNI 5015:2019 method. Meanwhile, the estimated total coal resources in the research area are 134,681,294 million tons, with the surface model limit for deposits is topography limit end of mine 2021.

Table 6. Resource Tonnages Classification SNI 5015:2019 vs *Relative Error*

Methods	Classification				Total Sumberdaya
	Measured	Indicated	Inferred	Measured and Indicated	
Relative Error	129,080,731	5,600,563	0	**134,681,294**	134,681,294
SNI 5015:2019	104,644,384	25,602,218	4,434,692	**130,246,602**	134,681,294

CONCLUSION AND FURTHER RESEARCH

Based on the Drill Hole Spacing Analysis (DHSA) with the Global Estimation Variance (GEV) method, the optimal distance information points (borehole spacing) for each seam are:

a) Seam A with measured resources of 500 m at 10% relative error, indicated at 1000 m at 20% relative error, and 2300 m inferred at 50% relative error.

b) Seam B with measured resources of 400 m at a relative error of 10%, indicated by 900 m at a relative error of 20%, and inferred by 2050 m at a relative error of 50%.

c) Seam C with measured resources of 850 m at a relative error of 10%, indicated by 1350 m at a relative error of 20%, and inferred by 2900 m at a relative error of 50%.

Based on statistical and geostatistical analysis, which was also carried out by considering the degree of confidence (confident level) based on the amount of data compared to other seams, the drill hole spacing was chosen from seam B as a recommendation for optimal borehole spacing for the continuous exploration drilling process, with a distance of 450 m for the measured resource class, 900 m for the indicated resource class and 2050 m for the inferred resource class. From the comparisons made, it is known that there are differences in the total accumulation of measured and indicated resources; in the indicated and measured resource relative error methods, the relative error method is 4,434,692 million tons more than the SNI 5015:2019 polygon method with moderate geological conditions, this difference occurs due to differences in the range of point distances. Information from the two methods so that there is an inferred resource in the resource classification with the SNI 5015:2019 method due to the smaller distance of the information point in that method. An evaluation of SNI 5015:2019 is needed to accommodate other important aspects such as geostatistics in making information point distance determinations in the coal resource classification system. The combination of geostatistical methods accompanied by the interpretation of the relevant geological models in the estimation and classification of resources is very necessary so that the classification and estimation results obtained follow the conditions in the field. It is not necessary to add drill holes to increase the data density because the distance of the existing drill holes, which is on average at a distance of 150 - 300 meters, is closer than the most optimal drill hole distance classification of 450 m with a radius of 225 m based on the relative error value from the analysis. Drillhole Spacing Analysis (DHSA). Based on this justification, the distance between drill points should be expanded to save drilling costs in the research area.

It is advisable to do further geostatistical research in the research area regarding other quality parameters such as total sulfur and ash content which in this research area has relatively poor data regularity and has a high variance value.

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The Determination of River Order Classification with ArcGIS Application

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ABSTRACT

River is a natural or artificial water channel or container in the form of a water drainage network and the water in it, starting from the upstream to the estuary, where based on this understanding the river has a discharge and size dimensions. Based on the discharge and size dimensions, rivers can be used for human needs and a place for the development of aquatic life so that it can affect the ecosystem and must be protected and not be disturbed. However, not all rivers can be used for human needs and a place of life. This study will discuss the types of rivers and the classification of stream orders based on the laws and regulations in Indonesia. Therefore, decision can be made whether the rivers that located in densely populated areas and areas of economic facilities can be modified in form and location according to human needs or must be maintained based on the original nature shape.

INTRODUCTION

Watershed is a land area located on the right and left of the river that follows the pattern of river flow from upstream to downstream, and it also functions as a rain catchment area where all rainwater that falls in the watershed area will flow to fill the river. (Saidi et al., 2018; Sobatnu et al., 2017). The watershed is an integral part of the river and its tributaries and becomes the habitat of living things that are closely related to their environment (Centeno, 2012; Hakim et al., 2019).

The types of rivers are divided into several types, such as: 1) Types of rivers based on the direction of flow; and 2) Types of rivers based on their geological structure. Types of rivers based on the direction of flow are divided into several types (Figure 1), such as: 1) Consequent rivers, its flow direction is in accordance with the slope; 2) Subsequent rivers, it flows perpendicular to the consequent river; 3) Obsequent river, which is a sub-sequence tributary which has direction opposite to the consequent river; 4) Resequent river, which is a sub-sequence tributary that flows parallel to the consequent river; and 5) Insequent rivers, which flow direction is irregular and not bound by the slopes of the plains. Meanwhile, the river types based on their geological structure are divided into several types, such as: 1) Superposed

River, which is a river that has a transverse position and has a flow direction according to or following its geological structure; and 2) Antecedent rivers, which is rivers that maintain the direction of the water flow even though there is a transverse geological structure he stream is the rank of the branching arrangement of the river channel which consists of the main river and its tributaries (Murtiono, 2001; Nurfaika, 2015). Stream orders are classified through several methods, such as the Strahler, Shreve, Horton, and Scheidegger method (Nurfaika, 2015; Pattiselanno, 2017). The Strahler method is the most commonly used method and is integrated with Geographic Information System (GIS) applications (Denaswidhi, 2020; Nurfaika, 2015; Pattiselanno, 2017; Stenger-Kovács et al., 2014).

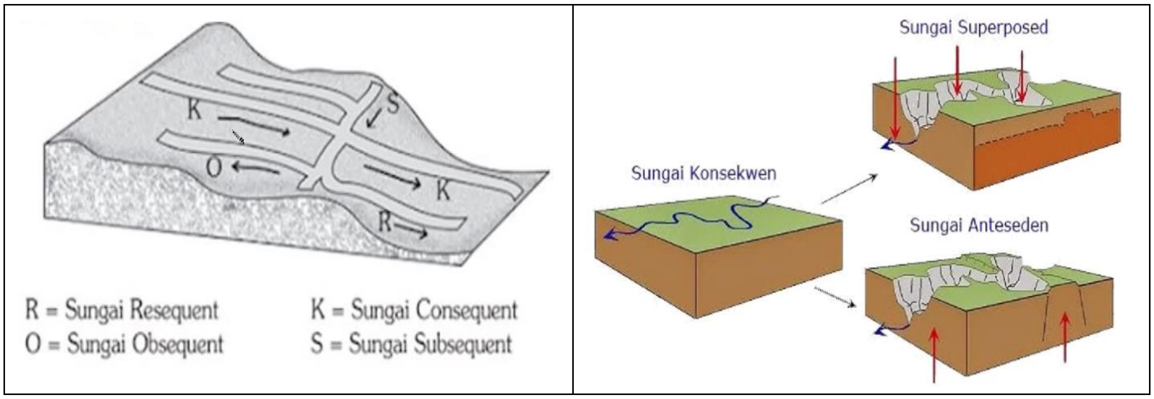

Figure 1. Types of Rivers Based on the Direction of Flow and Geological Structure

Based on the Strahler method, tributaries that are in the upstream position are classified into first order (order 1). Furthermore, the meeting of the same branch is classified into second order (order 2), and the meeting of the different order will not change the stream order. This continues until the river branches meet at the main river with the order of the largest order as shown in the figure below. (Ningkeula, 2016; Nurfaika, 2015).

Figure 2. Determination of stream with Strahler Method (Purwanto, 2013)

Watershed is the most vulnerable area according to the negative impact resulted from settlement development activities that follow the pattern of river flow (Hakki, 2015), as well as other economic activities that do not pay attention to environmental aspects (Ningkeula, 2016). This can lead to a decrease in watershed potential in some areas characterized by flooding, landslides, erosion, sedimentation and drought (UU No. 41; 1999). Therefore, it is necessary to manage the watershed according to the watershed classification determined based on the area of the watershed. Peraturan Direktorat Jendral Bina Pengelolaan DAS dan Perhutanan Sosial (2013) divide the watershed into 5 types, as can be seen in the table below.

Tabel 1. Watershed Classification Based on Watershed Area

No.	Area of Watershed (Ha)	Classification of Watershed
1.	> 1.500.000	Very Large
2.	500.000-1.500.000	Large
3.	100.000-500.000	Medium
4.	10.000-100.000	Small
5.	<10.000	Very Small

METHODOLOGY

Research Time and Location

This research was conducted by paying attention to the watershed in the Kebur Village area, West Merapi District, Lahat Regency, South Sumatra Province as shown in Figure 3. The area that becomes the research center is divided into 3 regions, which is Region A, Region B, and Region C. Overall there are 6 (six) tributaries in the three research areas.

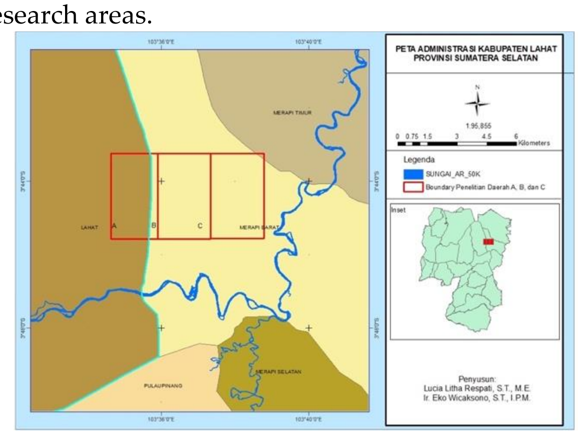

Figure 3. The Administration Map of Lahat Regency

Data Collection

This research was conducted in several stages, such as preparation, data collection, data analysis, and research writing. At the preparatory stage, the author develops a framework of thought. At the data collection stage, the authors collect secondary data and data entry. In the data analysis stage, the authors process the secondary DEM data into a river flow map at the research location, and analyze the method of classification of river orders. At the research writing stage, the author summarizes the results of the analysis from ArcGIS 10.2 software and displays a map visualization for later discussion regarding environmental impacts.

The data collected includes:
1. Watershed area and watershed classification
2. Stream order

Data Processing and Data Analysis

Watershed modelling and river order classification were carried out with ArcGIS 10.2 software. The data used was DEM (digital elevation model) which accessed from the DEMNAS website. This website managed by the government, which is tanahair.indonesia.go.id.

RESULT

Watershed Area and Watershed Classification

The calculation of the watershed area was obtained from the calculation in the attribute table of the defined research area boundaries. Watershed modelling was done by processing Basin data by using software ArcGIS 10.2 as shown in Figure 4. The area of the watershed in Regions A, B and C can be explained from the table below. Research area A has a watershed area of 1,001 Ha. Meanwhile, the area of the watershed in Region B was 1,138 Ha and in Region C was 1,138 Ha.

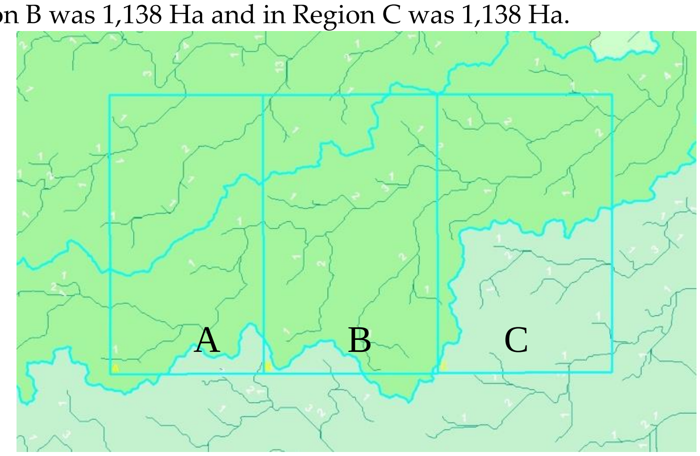

Figure 4. River Basin Modelling

Table 2. Watershed area of Region A, Region B and Region C

Boundary Penelitian Daerah A, B, dan C

FID	Shape *	Id	Luas
0	Polygon	1	10,012,036.3299
1	Polygon	2	11,375,499.6772
2	Polygon	3	11,381,882.2373

Based on the calculation of the watershed area that has been obtained, the classification of the watershed based on the area size in Region A, Region B and Region C was classified as a Very Small. The classification was made due to the watershed area in those three regions were less than 10,000 Ha.

Stream Order

From the results of monitoring in research areas A, B, and C, there were 6 (six) tributaries that classified as upstream rivers. Region A has 3 (three) tributaries with classification of stream order 1 and 2. Region B had 2 (two) tributaries with classification of stream order 1, and 2. While Region C, there were 3 (three) tributaries with classification of stream order rivers 1 and 2.

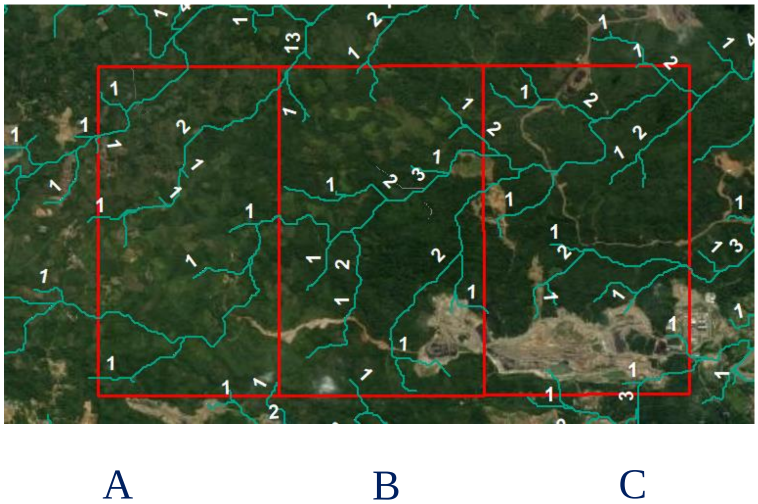

Figure 5. Stream Order in Region A, Region B and Region C

CONCLUSION

Based on the results of research conducted in Regions A, B, and C, it can be concluded as follows:

1. The area of the watershed in area A was 1,001 Ha, so classified as Very Small Watershed. The watershed area in Region B was 1,138 Ha, so classified as Very Small Watershed. Meanwhile Region C also had watershed area of 1,138 Ha, therefore classified as Very Small Watershed.
2. The stream order in the research area was divided into stream orders 1 (one) and 2 (two). In Region A, there were 3 (three) tributaries that classified as stream orders 1 and 2. In Region B, there were 2 (two) tributaries that classified as stream orders 1 and 2. And in Region C, there were 3 (three) tributaries that classified as stream orders 1 and 2.

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Empirical and numerical method approaches in determining the value of rock slope safety factors at JJLS Gunung Kidul, Yogyakarta

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ABSTRACT

Limestone is one of the most numerous sedimentary rock groups, limestone consists of non-clastic limestone and clastic limestone. The research slope is on the Southern Cross Road (JJLS) in Gunung Kidul LOT 4 (Legundi-Plajan) which has length of 4.7 KM, where the rock slopes are in a location that is busy with traffic and close to where residents live. The purpose of this study is to determine the slope safety value with empirical and numerical approaches using RMR and RS 2. In this study, there are several types of data used, namely field data including megascopic rock descriptions at the research location, and laboratory data, namely UCS data. From this study, it was found that the formation of slopes 1 and 2 was included in the category of rock mass quality, while slope 3 was good. RMR values that are not much different do not make the slopes have the same weighting, because the slope geometry and the discontinuity geometry plane. Based on the value of slope stability using phase 2 software to get stable FK, there is a significant difference between slope 1,2 and slope 3, where the difference in safety factors can be influenced by discontinuity conditions, discontinuity orientation and activities around the slope.

INTRODUCTION

Limestone is one of the most numerous sedimentary rock groups, limestone consists of non-clastic limestone and clastic limestone. Clastic limestone is the result of the breakdown of non-clastic types of limestone through the process of erosion by water, transportation, sorting and sedimentation. Therefore, during the process other types of minerals also follow which are impurities and give color to the limestone. Meanwhile, non-clastic limestones are colonies of starfish, namely Coelenterata, Mollusca, Protozoa and Foraminifera (Sukandarrumidi, 2009). In this study, the limestone slopes are composed of crystalline limestone with brownish yellow color, non-clastic texture, massive structure with mineral conditions of calcite, carbonate, and reef limestone with bright white color, weathered yellowish brown color. According to Sustriani (2012) the geological structure can affect the stability of the slope, where the geological structure will find a weak field that has the potential as a slip plane if it is in the direction of the slope. Unstable slopes are very dangerous to the surrounding environment, therefore slope stability analysis is very necessary. Instability on slopes can also be caused by geological structural conditions, the direction of discontinuities in rocks such as joints, fractures, planes, faults and other types of cracks in rock, physical properties mechanics of slope-forming rock, groundwater pressure, and slope geometry. Thus it can be said that the fundamental behavior of rock mass is strongly influenced by its discontinuities (Endaryanto 2007).

The research slope is on the Southern Cross Road (JJLS) in Gunung Kidul LOT 4 (Legundi-Plajan) see Figure 1 which has a length of 4.7 KM, where the rock slopes are in a location that is busy with traffic and close to residents' residences, there are three research slopes. namely the first slope at STA 3+000, the second slope at STA 1+250 and the last slope at STA 3+300. Therefore, this research is focused on empirical and numerical approach to slope stability, using RMR (Rock Mass Rating) and kinematic analysis using RS2 to determine the FK (Safety Factor) value of the slope.

RESEARCH METHODS

The research begins with calculating the stiffness in the field, then the next stage is the data collection and processing stage. In this study, there are several types of data used, namely field data and laboratory data and the next stage is data analysis. Field data includes megascopic rock descriptions at the research location, such as discontinuity areas, discontinuity length, discontinuity position, discontinuity position (strike, dip and dip direction), discontinuity openings, fill material, and water conditions. Laboratory data, namely data from the test results of rock strength values based on the UCS (uniaxial compressive strength) test. Rock samples used are rock samples taken directly in the field, then preparation is carried out before testing.

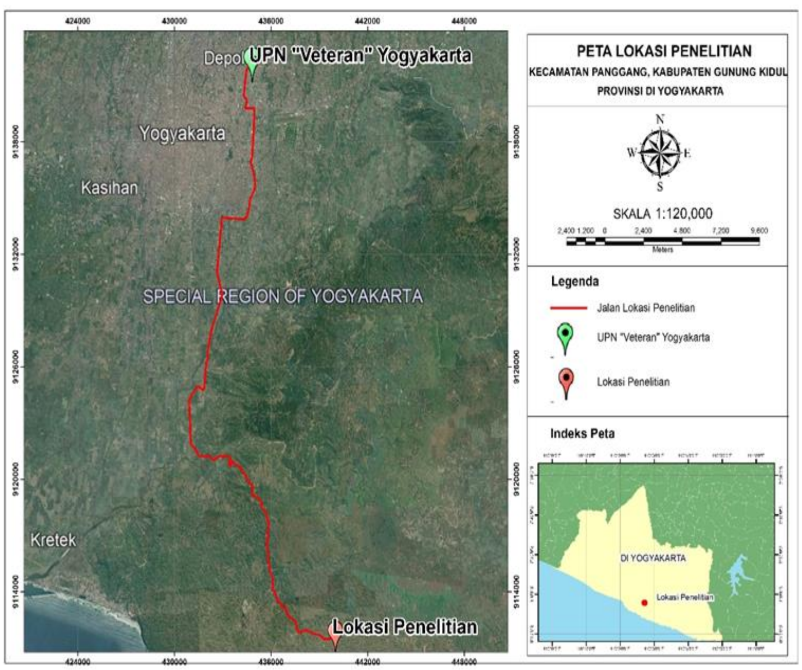

Figure 1. Research site map

Analysis stage

After all the data is complete, the next stage is kinematic analysis, rock mass classification and calculation of the safety factor value.

Kinematic analysis

This analysis was carried out based on field data, namely joint distance, joint conditions and groundwater conditions. Where data retrieval is done based on the scanline method. The analysis was carried out by entering slope geometry data, discontinuity data on the observation slope with the help of Phase 2 software.

Rock mass analysis

Rock mass classification is carried out based on the parameters in the RMR taken from three research slopes and UCS tests in the laboratory, aiming to determine the condition of the rock mass. In this study, using the RMR table developed by Bieniawski (1987), so that the weights of the three slopes are obtained and determine the rock mass class.

Calculation of safety factor value

Calculation of the value of the factor of safety (FK) is carried out to determine the comparison of the value of the resisting force with the driving force on a slope, which aims to determine the condition of the slope in a stable or unstable condition. Determining the FK value using Phase 2 software, which uses the Mohr-Coloumb criteria by entering the shear strength test data, namely the value of cohesion, internal shear angle and the proton ratio. From these data, the results of the safety factor of the three slopes and also the rock mass class at the study site were obtained.

RESULTS AND DISCUSSION

Based on the calculation of RMR in the three research sites, it was found that all research sites were composed of rocks with RMR values ranging from 51-62 which were included in class II (good) and III (moderate). The RMR value on the 1st research slope at STA 3+000 is 54 in class 3 which is moderate, the 2nd slope at STA 1+250 with an RMR value of 51 which is class 3 moderate and the 3rd slope STA 3+300 with an RMR value his 62 are in class 2 which is good. The difference in RMR values at each study site is relatively small, due to the similarity of lithology. The RMR value is determined based on the parameters in the RMR including rock strength data (UCS), RQD (Rock Quality Designation) data, distance data between discontinuity planes, discontinuity plane conditions and general groundwater conditions. The following is the weighting of the three slopes.

Table 1. RMR value on slope 1

No.	Parameter	Weight
1.	Compressive	4
2.	strength	20
3.	RQD	10
4.	Sturdy distance	10
5.	Strong condition	10
	Groundwater	
Total		**54**

Table 2. RMR value on slope 2

No.	Parameter	Weight
1.	Compressive	4
2.	strength	20
3.	RQD	10
4.	Sturdy distance	10
5.	Strong condition	7
	Groundwater	
Total		**51**

Table 3. RMR value on slope 3

No.	Parameter	Weight
1.	Compressive strength	7
2.	RQD	20
3.	Sturdy distance	8
4.	Strong condition	20
5.	Groundwater	7
Total		**62**

The following graph shows the relationship between classification parameters and RMR weighting.

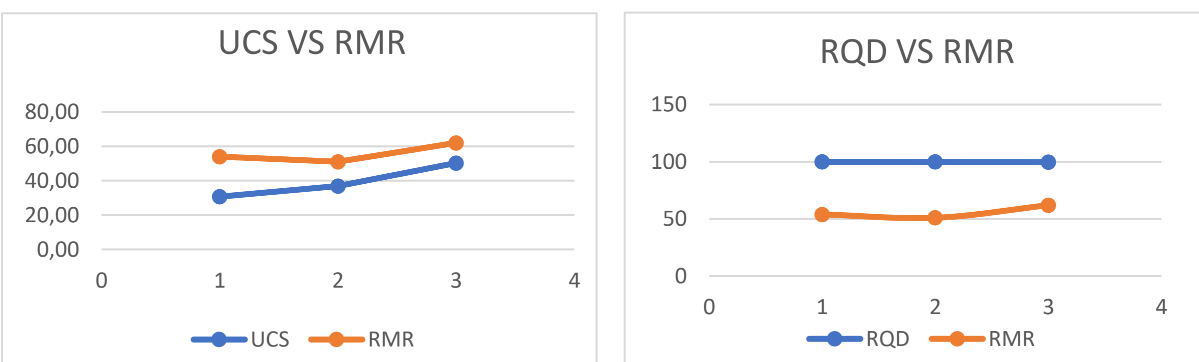

Figure 2. (a) Relationship between UCS VS RMR (b) Relationship between RQD vs RMR

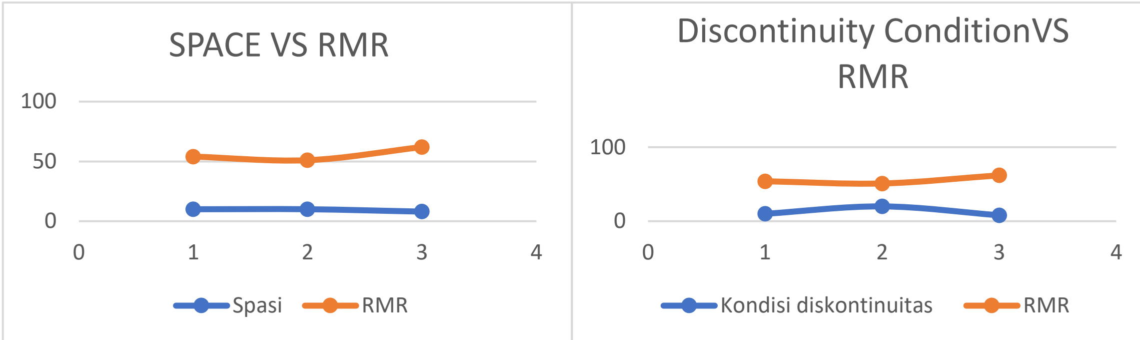

Figure 3. (a) Relationship between discontinuity spaces VS RMR (b) The relationship between discontinuity conditions VS RMR

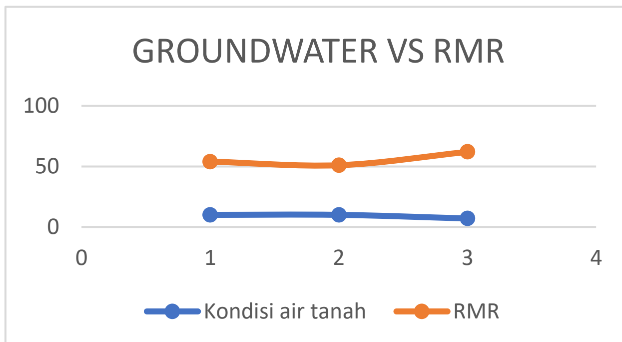

Figure 4. Relationship between groundwater VS RMR

SAFETY FACTOR

Slope conditions can be reviewed through the value of the safety factor. The factor of safety is the ratio between the resisting force and the driving force. Based on the calculation results of the FK (Safety Factor) it is found that the relationship between the RMR value of the constituent rocks and the FK is directly proportional, so the higher the RMR value, the higher the FK value. To determine the relationship between slope stability parameters, it is necessary to analyze the actual conditions of the slopes to be analyzed. The model was obtained after analyzing the slope conditions. Calculation of the safety factor is done by entering the data from the shear strength test. For FK on each slope, the 1st slope of STA 3+300 is 8.48, the 2nd slope of STA 1+250 is 7.79 and the 3rd slope of STA 3+300 is 5.26.

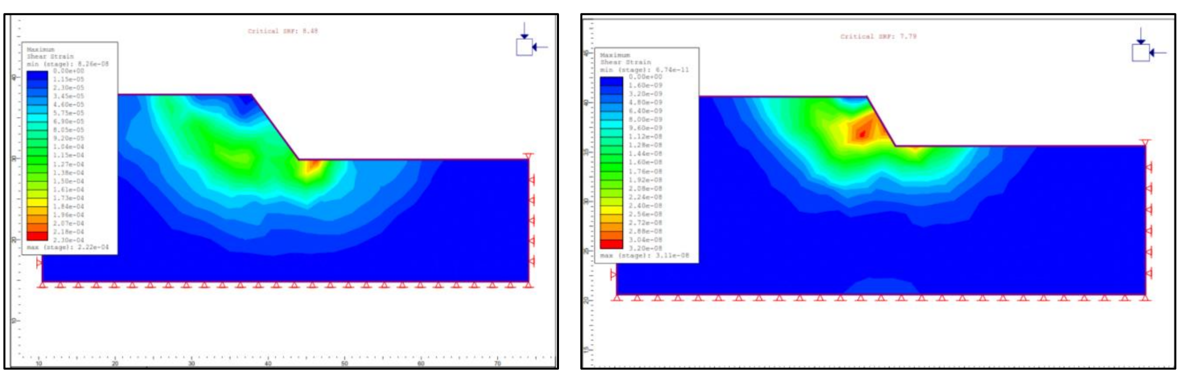

Figure 5. (a) Slope condition with FK 8.48 (b)Slope condition with FK 7.79

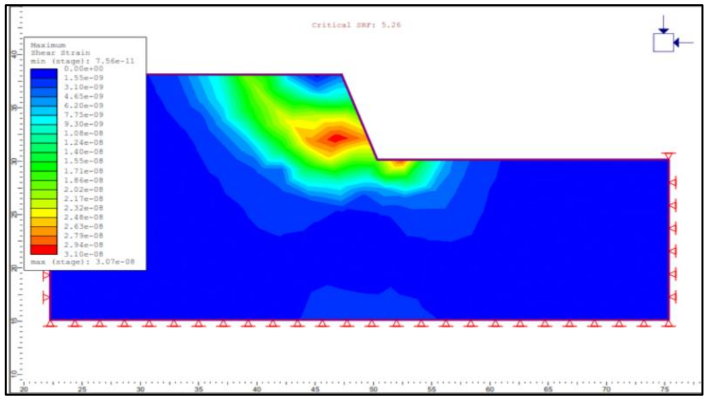

Figure 6. Slope conditions with FK 5.26

Table 4. Safety factor of each slope

No	Slope	Safety factor
1	I	8,48
2	II	7,79
3	III	5,26

CONCLUSION

Based on empirical and numerical calculations at the research site, it can be concluded that the rock mass classification (RMR) shows that the formation of slopes 1 and 2 is in the category of moderate rock mass quality and the third slope is in the good category. In weighting the RMR values, the three slopes have weights that are not much different, but do not make the three slopes have the same rock mass class due to slope geometry and discontinuity geometry. Calculation of the value of slope stability using software phase 2 obtained stable FK and there is a significant difference between slope 1,2 and slope 3, where the difference in FK can be influenced by discontinuity conditions, discontinuity orientation and activities around the slope.

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Impact of Open Pit Coal Mine on Groundwater Recharge: Alternative Method for Environmental Assessment

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ABSTRACT

Coal mines are widespread in Indonesia. It is the primary energy source for Indonesian electricity, which also contributes to the national revenues even during the Corona Virus Disease (COVID-19) pandemic. On the other hand, groundwater is also one of the important resources in Indonesia. It is commonly utilized for domestic water supply including drinking water, irrigation, municipalities, and industries. Mining with an open-pit system or surface mining is regarded as an activity that affects environmental deterioration. The impact on groundwater including a decrease in the quantity of groundwater is a common and significant issue concerned. According to environmental regulations in Indonesia, each mining company is obliged to submit Environmental Impact Assessment (EIA) documents before starting the mining production to protect and manage the groundwater in mine areas. The objective of this study is to analyze the impact of coal mining on groundwater recharge using a water balance approach as a part of EIA. Water balances, before and during the mining operation, should be evaluated based on natural hydrological conditions in the land around open pit coal mine areas. Hydrologic data, such as precipitation (P) and temperature, combined with topographic data, were collected to calculate the evaporation-transpiration (ET) and run-off (Ro) values. Then, groundwater recharge (U) was determined by a water balance equation (U=P–ET-Ro). The estimation of runoff coefficient before and during mining operation were used to predict the value of runoff, controlling the estimation of recharge before and during mining operation by water balance equation. The results of this study showed that the groundwater recharge before mining operation was 659 mm. Meanwhile, during mining operation, the recharges were 321 mm/year at land clearing stage, 152 mm/year at open pit mining, 321 mm/year after backfilling stage, and 557 mm/year after re-vegetation stage. Decreasing the recharge value during mining operation would influence the total amount of groundwater in the aquifer storage around the mine area. Based on this study, it can be concluded that runoff coefficient determination before and during mining operations could be an alternative to assess the impact of open pit coal mines on groundwater quantity.

INTRODUCTION

Coal is recognized as the primary energy resource for electricity in Indonesia as about 80% of domestic coal is used for power generation. In addition, with specific low ash and low sulfur, Indonesian coal become a favorite in China and India as the dominant export target of the Indonesian coal industry. Around 80% of Non-Tax State Revenue of the Indonesian mining sub-sector comes from the coal industry. This factor is positive leading to constant global market demand for Indonesian coal despite the price being volatile, especially during Corona Virus Disease (COVID-19) pandemic. Indonesian coal reserves are 38.84 billion tons with average coal production of 600 million tons per year (Ministry of Energy and Mineral Resources of the Republic of Indonesia, 2021). As one of the largest coal producers and exporters in the world, coal mining companies are widespread in Indonesia. One area where many coal mines are located in Kalimantan Island, in which 62.1% of the Indonesian coal reserves (25.84 billion tons) and resources (88.31 billion tons) are located.

Mining, both with the open pit system (surface mining) and underground system, has been regarded as activities that impact the surrounding environment (Haq et al, 2016). Direct impact on groundwater, including a decrease in the quantity of groundwater, is the particular issue concerned in this study. Mining activities, such as land clearing and soil removal, are considered to reduce the recharge in the vicinity of mine area.

Naturally, water balance is closely related to the hydrologic cycle. According to Freeze & Cherry (1976), Schwartz & Zhang (2002), Todd & Mays (2005), the hydrologic cycle is a continuous process of water circulation on the Earth. Water evaporates from the ocean and land surfaces and becomes water vapor in the atmosphere. The water vapor condenses and precipitates as rainfall or snow on the land and ocean. On the land, some portion of precipitated water may be absorbed by vegetation, infiltrate into the ground and percolate to recharge groundwater. Some other portions of precipitated water may flow into streams as a run-off and then back to the ocean. Due to elevated temperatures, evapotranspiration will increase in the land area. Evapotranspiration is the term for both the direct return of surface water to the atmosphere by evaporation and its indirect return through the leaves of plants (Pipkin et al., 2005).

Thus, based on environmental regulations in Indonesia, each mining company is obliged to submit Environmental Impact Assessment (EIA) documents to the government, before operation. The purpose of this study is to estimate the impact of coal mining on groundwater recharge using a water balance approach as a part of the EIA. The study of groundwater is a complex process related to natural systems and processes involved. Therefore, natural hydrological conditions around open pit coal mine areas should be understood to determine the water balance in mine areas.

METHODOLOGY

Research Area

One of the coal mining concessions located in Barito Timur, Central Kalimantan was selected for this study. The company has a concession covering an area of 2,000 – 3,000 ha. The coal target is about 400,000 – 700,000 tons per month with the open pit system. Geologically, the mining site is located in the Warukin Formation, lithologically dominated by silt and clay material (see Fig. 1). According to Asminco (1996), Warukin Formation consists of three parts, namely upper Warukin, middle Warukin, and lower Warukin. Upper Warukin is dominated by a coal layer of 30 – 40 m in thickness and a clay layer. Middle Warukin is classified into upper sandstone and lower sandstone. Meanwhile, lower Warukin is relatively dominated by claystone.

Data Collection

Primary data were collected from field investigations, such as groundwater tables and river water level measurements. Water levels in dug wells and boreholes were measured to estimate the direction of the groundwater flow pattern. Primary data also include a detailed investigation of geological conditions in the study area. Secondary data were also collected from various information sources, such as a topographic map from the National Land Affairs Department of Indonesia, regional geological maps from the Research and Development Center of Geology in Bandung, meteorological data from the Department of Meteorological Climatological and Geophysics in Buntok collected between 2010 to 2019, and mining plan design from a mining company in the study area.

Estimation of Water Balance Before and During Mining Operation

Water balance in the natural condition was estimated based on hydrological conditions. Meanwhile, meteorological data, including rainfall and temperature, were interpreted to understand the hydrological conditions such as evapotranspiration, run-off, and recharge. Evapotranspiration was estimated from an empirical equation from Turc (1954, in Putra 2013)

$$ET_r = \frac{P}{\sqrt{0.9 + \frac{P^2}{\left(300 + 25 \cdot T_m + 0.05 \cdot {T_m}^3\right)^2}}} \tag{1}$$

Where,

ETr : Annual Evapo-transpiration (mm/year)

P : Annual Precipitation (mm/year)

Tm : Annual temperature (^{O}C)

Surface run off is part of the rainfall that flows over the land surface to rivers, lakes, and the sea. The flow occurs because the rainwater that reaches the ground surface is not infiltrated due to the intensity of the rain exceeding the infiltration capacity or other factors, such as the slope, the shape and compactness of the soil surface and vegetation. In addition, rainwater that has entered the ground then comes out again to the ground surface and flows to the lower part. Sharma method (in Putra, 2013) was used to obtain the run-off value. This method requires annual temperature (Tm), annual rainfall (P), and area of watershed (A).

$$Ro = \frac{1.511 x P^{1.44}}{T_m^{1.34} x A^{0.0613}} \tag{2}$$

Where,

Ro : Run-off (cm/year)

P : Precipitation (cm/year)

Tm : Annual temperature (^{O}C)

A : Area of watershed (km^2)

Recharge values were calculated by water balance concept, described by this equation:

$$U = P - ET - Ro \tag{3}$$

Where,

P : Annual Precipitation (mm/year)

Ro : Annual Run-off (mm/year)

ET : Annual Evapo-transpiration (mm/year)

U : Annual Recharge (mm/year)

Land clearing and natural landscape degradation due to mining operation were regarded to increase the run-off value. Increasing the run-off value would impact the decrease in the recharge value. Therefore, prediction of the recharge value during mine operation could be made by comparing run-off value on the natural condition and during mining operation. In this study, run-off coefficients from the Sivanappan classification (1992) were applied (Table 1). The runoff coefficient is the ratio between the peak velocity of runoff to the rainfall intensity which is influenced by the rate of soil infiltration, vegetation cover, and rainfall intensity.

Table 1. Run-off coefficients from Sivanappan Classification (1992)

Vegetation and Topography	Material		
	Sandy clay	Dusty silt and clay	Dusty Silt
1. Forest			
Flat (slope <5%)	0.1	0.30	0.40
Bumpy (5-10%)	0.25	0.35	0.50
Hilly-montainous (>25%)	0.30	0.50	0.60
2. Reed			
Flat (slope <5%)	0.1	0.30	0.40
Bumpy (5-10%)	0.16	0.36	0.55
Hilly-montainous (>25%)	0.22	0.42	0.60
3. Agriculture			
Flat (slope <5%)	0.30	0.50	0.60
Bumpy (5-10%)	0.40	0.60	0.70
Hilly-montainous (>25%)	0.52	0.72	0.82

RESULTS AND DISCUSSION

Hydrologic Setting of Study Area

Based on rainfall data obtained from the Meteorological and Geophysics Station of Buntok (2021), annual precipitation in 2010-2019 varies between 1,499 mm/year and 3,350 mm/year, with an average of 2,798 mm/year. The highest precipitation occurred in 2010 with an amount of 3,350 mm, while the lowest precipitation was in 2019 with an amount 1,499 mm/year (Tabel 2).

Table 2. Precipitation in study area period 2010 - 2019

Month	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019
January	288.4	453	479.9	310.7	284.8	404.3	383.5	311.4	113.8	175
February	170.6	199.2	385.3	377.4	169.2	252.4	336.9	221.7	399	149
March	303.7	211.4	260.7	362.1	305.2	405.1	392.4	410.7	302.5	172
April	319.5	320.4	294.5	345.3	266.1	253.6	257.9	224.7	168.2	177
May	280.3	283.4	232.6	390.8	352.1	116.7	330.9	240.9	123.9	51
June	263.8	117	84.3	95.1	214.3	198.4	112.6	187.3	72.6	148

Continued from **Table 2.**

Month	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019
July	290.5	132.2	179.8	341.9	129.8	66.8	342.7	133.1	111.6	48
August	134.9	106.8	114	81.9	229.1	13.9	112.9	209.6	44.7	128
September	243.8	79.8	155.6	184.5	63.9	-	186.1	84.8	107.7	20
October	409.6	171.8	157.8	202.9	57.3	-	386.7	128.3	244.8	66
November	334	316	292.2	247.8	315	433.6	585	294.4	492.9	106
December	310.2	423.1	281.7	340.9	363.7	204.2	319.1	337.8	310.3	259
Annual	3,349.3	2,814.1	2,918.4	3,281.3	2,750.5	2,349	3,746.7	2,784.7	2,492	1,499

Source: Meteorological Station Buntok, 2022

In addition, the average monthly temperature in the research area varied between 26.6 °C and 27.4 °C. The annual temperature is 27° C. By substituting precipitation and temperature values of Turc (Equation 1), the average annual evapotranspiration (ET) was 1,630 mm/year. According to annual temperature and annual precipitation, an area of 20 km^2, the run-off value was calculated by the Sharma method (Equation 2) at 50.7 cm/year (507 mm/year).

The rational method (US Soil Conservation Service, 1973) is also an approach to predict runoff through mathematical calculations through several assumptions to simplify the calculation, involving rainfall intensity and area of the watershed. Rainfall with intensity occurs continuously, then the direct runoff rate increases for some periods until when the watershed has contributed to the flow at the outlet. This method commonly obtains reasonable results and is considered accurate for estimating surface runoff in Indonesia. One of the substantial parameters of the rational method is the intensity of rain (mm/hour) in research area. It is characteristic of rain events that are expected to occur in the future. However, the analysis of rainfall intensity requires a series of detailed measurement data at rainfall stations over a certain period, for example, maximum rainfall in one day and duration of rainfall. Therefore, in some cases like this study, when the detailed rainfall data is not available, an alternative method (i.e., Sharma) was necessary. In addition, the runoff value of Sharma (507 mm/year) was acceptable because it is not higher than the precipitation in the research area. By substituting precipitation, run-off, and evapotranspiration values on water balance (Equation 3), the recharge value in the research area was 659 mm/year.

Impact of Coal Mining Activity on Groundwater Recharge

Most coal mining companies in Indonesia applied open-pit coal mining method, which includes in surface mining system. It is economically favorable when the coal seams are located near the surface (less than 200 m). The operations of open pit coal mine begin with land clearing to remove growing plants in the working area covering the mine area, disposal or dumping area, topsoil stockpile, mining road, settling pond, and other supporting facilities. The land clearing process is carried out using human power and/or heavy equipment. Wood plants with a diameter of more than 30 cm were cut using chainsaws, while the smaller plants were uprooted using bulldozers. The timbers are collected and stacked in a place that does not interfere with the further mining process and can be used for construction purposes.

After clearing the land, the main open pit activity is conducted by (i) stripping the overlying rock strata and (ii) excavating the coal seams. The former includes the removal of top soil and other material called overburden. Topsoil is material with high nutrients and is indispensable for restoring soil fertility. It is generally moved into some spaces for conservation as a planting medium during reclamation and post mining. Overburden is stripped by making slopes, in which the geometry of the slope is determined based on geotechnical analysis to estimate slope stability. Then after removing top soil and overburden, the exposed coal deposits are excavated and transported to the coal stockpile Raw of Material (ROM). Backfilling method is a mandatory practice within open-pit coal mines, though it is noticed as costly and time-consuming. This process consists of returning overburden material that was removed during excavation into the mined-out area (i.e., the area where coal reserves have been excavated).

The impact of mining activities on the quantity of groundwater could be in the form of a decrease in groundwater level to below the operational limits of the mining pit (Hamilton & Wilson, 1977; Libicki, 1982; Erbele & Razem, 1985; Morris et al, 2003). Coal mining activities may be located in the aquifer layer so that the decrease in groundwater level can be caused by the mining activity itself or the dewatering activities carried out (Morris et al, 2003). According to Libicki (1982), groundwater subsidence caused by mining activities is a function of several factors, among these factors are the depth of groundwater subsidence, geological structure, infiltration coefficient & specific yield, and time. The impact in the form of a decrease in the groundwater level can occur in residential areas around the mining area. This is because groundwater subsidence is not limited only by mining areas, but is limited by geological and hydrogeological conditions in an area (Haq, 2015).

The quantity of groundwater is influenced by the water supply from the surface infiltrating the subsurface. Open pit mine would lead to a disturbance on the natural land surface due to the removal of top soil and overburden, and excavation of coal. This directly increases the surface runoff, and subsequently, affects the hydrologic – budget in research area which was estimated based on the water balance equation stating that groundwater inflow minus outflow equals the change in storage. Therefore, a change in the value of recharge could be predicted by comparing the run-off value before and during mining operation.

Factors that affect runoff can be grouped into (i) factors related to climate such as rainfall, and (ii) factors related to watershed characteristics including topography, geology, and land use (e.g., vegetation type and density). Vegetation can slow the rate of runoff and increase the amount of infiltration water on the ground due to surface detention, while a high slope (>15%) could increase the velocity of runoff and decrease the infiltration (Fig 1). The increase and decrease in the rate and volume of runoff are related to changes in the value of the runoff coefficient (C) expressed with a value of 0 to 1, which is the ratio between the amount of runoff and rainfall. It is the comparison value between the input rate and the peak discharge rate.

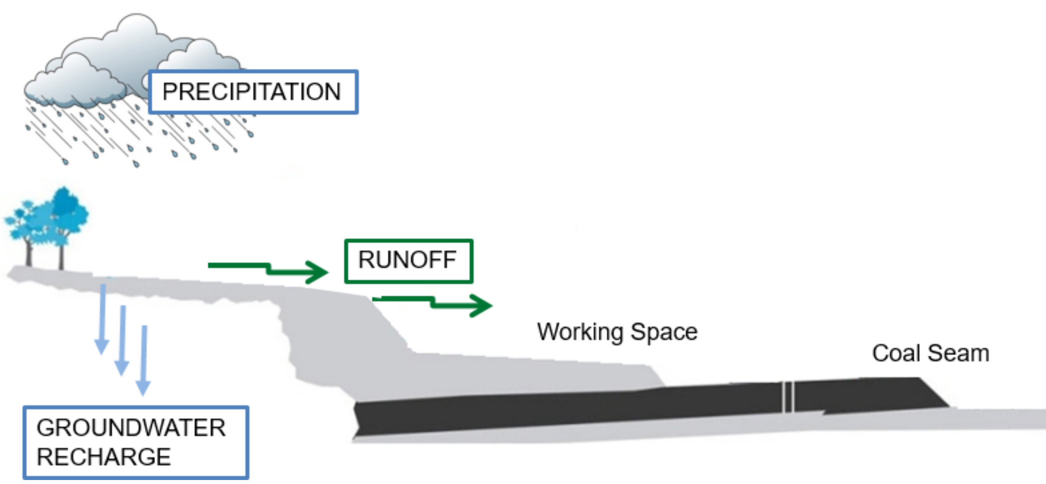

Figure 1. Hydrological setting during open pit coal mine. Land clearing and natural landscape degradation were regarded to increase the run-off. Increasing in the run-off would impact on decrease of the recharge

Although the runoff value in this study (507 mm/year) was estimated by the Sharma method which does not require run off coefficient for the calculation, change in the runoff coefficient before and during mining operation could be an alternative to predict the value of runoff before and during mining operation. For an instant, the natural condition of the research area is a flat forest with levels of slope <5%, and the dominant material are silt and clay. According to the classification of Sivanappan, (1992) (Table 1), the run-off coefficient before mining operation (i.e., natural condition) was 0.30. On the other hand, the condition during mining operation was a bumpy slope of 5-10%, and no forests and reeds. As a result, the run-off coefficient amounted to 0.60 or twice higher than the coefficient of natural run-off. The watershed characteristic and hydrological estimation before and during mining operation are resumed in Table 3.

Table 3. Hydrological setting before (natural condition) and during mining operation

Parameters	Natural Conditions	Mining Operation
Vegetation	Forest	No Forest
Morphology	Average slope <5%	Average slope of 5-10%
Land Soil/Material	Silt and clay	Silt and clay
Runoff Coefficient	0.3	0.6

The runoff coefficient in Table 3 indicates that the run-off value during operation was predicted to be 1014 mm/year. The runoff in each stage during mining operation involving land clearing, open pit mining, after backfilling, and after re-vegetation could be predicted by using same methodology as shown in Table 4. During land clearing and after backfilling, the research area is predicted to be flat with no forest with levels of slope 5 – 10 %, and the runoff coefficient and value were 0.5 and 844 mm/year, respectively. Then after re-vegetation, the research area is predicted to be flat with reed and levels of slope <5%, and the runoff coefficient and value were 0.36 and 608 mm/year, respectively.

Based on the water balance equation, the recharge values during land clearing, open pit mining, after backfilling, and after re-vegetation stages were 321, 152, 321, and 557 mm/year, respectively. During land clearing and after backfilling, the recharge value decrease to 57.4% compared to natural condition. The most significant impact was during open pit mining with the lowest recharge value, about 36% compared to natural conditions. Afterwards, the recharge would recover after re-vegetation stage to 87.2% compared to natural conditions. Groundwater recharge estimation in each stage during open pit coal mine operation is illustrated in Fig 2.

Table 4. Run-off estimation before and during mining operation

Mining Stages	Vegetation and Topography	Runoff coefficient*	Runoff (mm/year)
Natural condition (before mining)	Flat Forest (slope <5%)	0.3	506
Land clearing	Flat with no forest (slope 5-10%)	0.5	845
Open pit mining	Bumpy area with no forest (slope 5-10%)	0.6	1014
Open pit mining after backfilling	Flat area with no forest (slope <5%)	0.5	845
Open pit mining after revegetation	Flat area with reed (slope <5%)	0.36	608

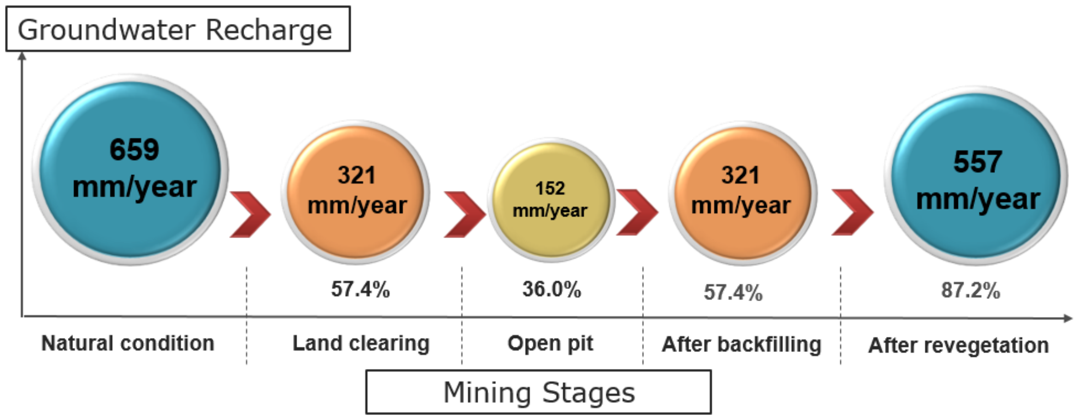

Figure 2. Groundwater recharge estimation in open pit coal mine operation

CONCLUSION

Estimation of the runoff coefficient before and during mining operation could be an alternative to predict the value of runoff before and during mining operation. Thus, the recharge value around the mine area could be estimated by the water budget concept. Land clearing and soil removal during mining operations would cause an increase in the run-off value, leading to a decrease in recharge value. The run-off value of 1,013 mm/year was predicted during open pit coal mine, twice higher than that of the natural condition (506 mm/year). The recharge value decreased from 659 mm/year before mining operation into152 mm/year during open pit mining. Therefore, the impact of coal mining activity may be significant. An understanding of natural hydrological systems is an important step in the groundwater modeling process. This is because a conceptual model of hydrogeology in coal mining areas can be developed to obtain accurate and under the actual situation in the field. This would result in an appropriate hydrogeological conceptual model, promoting the realistic prediction in the EIA document.

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Rock Mass Classification and Probability of Failure in Determining Slope Stability

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ABSTRACT

Scientific research on slopes is always evolving, alongside the development of science itself. In many cases, slope instability is a problem in the field. Most of the roads have a rock slope, which can be unstable because of the rock mass conditions and external factors such as water and seismic activity. The purpose of this research is to analyze slope stability using two methods: rock mass characterization and numerical modeling to calculate safety factor and probability of failure. As a result of this study, inclination 1 is more stable than inclination 2 with each value of 6.03 and 2.02 for each failure probability of 0 per cent and 0.48 per cent. The result of numerical modeling is directly proportionate to the characteristics of the stone's mass using RMR and GSI, and the rock's mass is in the appropriate state for the slope 1, and the stone's mass is classified in the appropriate state for the slope 2. The reasons for the differences in stability on the two slopes will be discussed further in this paper.

INTRODUCTION

The rock slope present on most roadways, particularly in hilly places, frequently has instability issues caused by the rock mass characteristics around the slope, as well as external variables such as water and seismic activity [9]. Internal variables influencing slope stability include frequency and discontinuity plane features, as well as the physical and mechanical qualities of the rock mass. Aside from internal considerations, slope geometry, such as slope height and slope angle, plays a vital influence in slope stability. Rainfall and earthquake activity are two exogenous elements that have an impact [5].

Researchers are occasionally concerned about slope stability. A number of approaches for evaluating slope stability have been developed. Kinematic analysis, boundary equilibrium, numerical modeling, and empirical approaches are divided into four groups [8]. The focus of this paper's study is on empirical techniques and numerical approaches using a probability of failure approach (RS2). The empirical technique is a valuable instrument that is frequently used to examine the early behavior of rock masses [1]. While the numerical technique was established to confirm the empirical method's first evaluation, the calculation results are more accurate and indicative of field settings.

The breccia andesite slopes in the two research locations have two conditions: the first in the agricultural area is fresh, and the second is weathered on the edge of the village road. The presence of these two slopes prompted the authors to do more research on the stability of the slopes in each site in order to identify possible hazards to inhabitants and road users near the slopes. Rock Mass Rating (RMR) and Geological Strength Index are the methodologies used to characterize rock masses (GSI). Meanwhile, the numerical technique employs RS2 software to compute SRF (Strength Reduction Factor) and Failure Probability (PoF).

RESEARCH SITES

The research is being conducted in two locations: Gedangsari Districts, Gunung Kidul, and DI Yogyakarta. The first location is in Jatigulung, Hargomulyo Village, at 7o49'34"S and 110o35'33"E, on a slope above the locals' rice fields. The second place is at Buyutan, Ngalang Village, with coordinates 7o51'31"S and 110o35'6"E, which is a roadside hillside. The two places have breccia andesite rock lithology.

Regionally, it is part of the Southern Mountain range, and geologically (FIGURE 1), it lies in the overlap region of the Kebo-Butak Formation (Tomk) and the Semilir Formation (Tms). The Kebo Butak Formation (Late Oligocene age) is the oldest formation exposed in Gunung Kidul Regency, consisting of layered sandstone, siltstone, claystone, shale, tuff, and agglomerates, with locally andesite fractured basalt and andesite breccia at the top. The Semilir Formation originated in the Early Miocene, overlaying harmoniously above the Kebo Butak formation, which was comprised of tuff, tuffaceous sandstone, and shale [3].

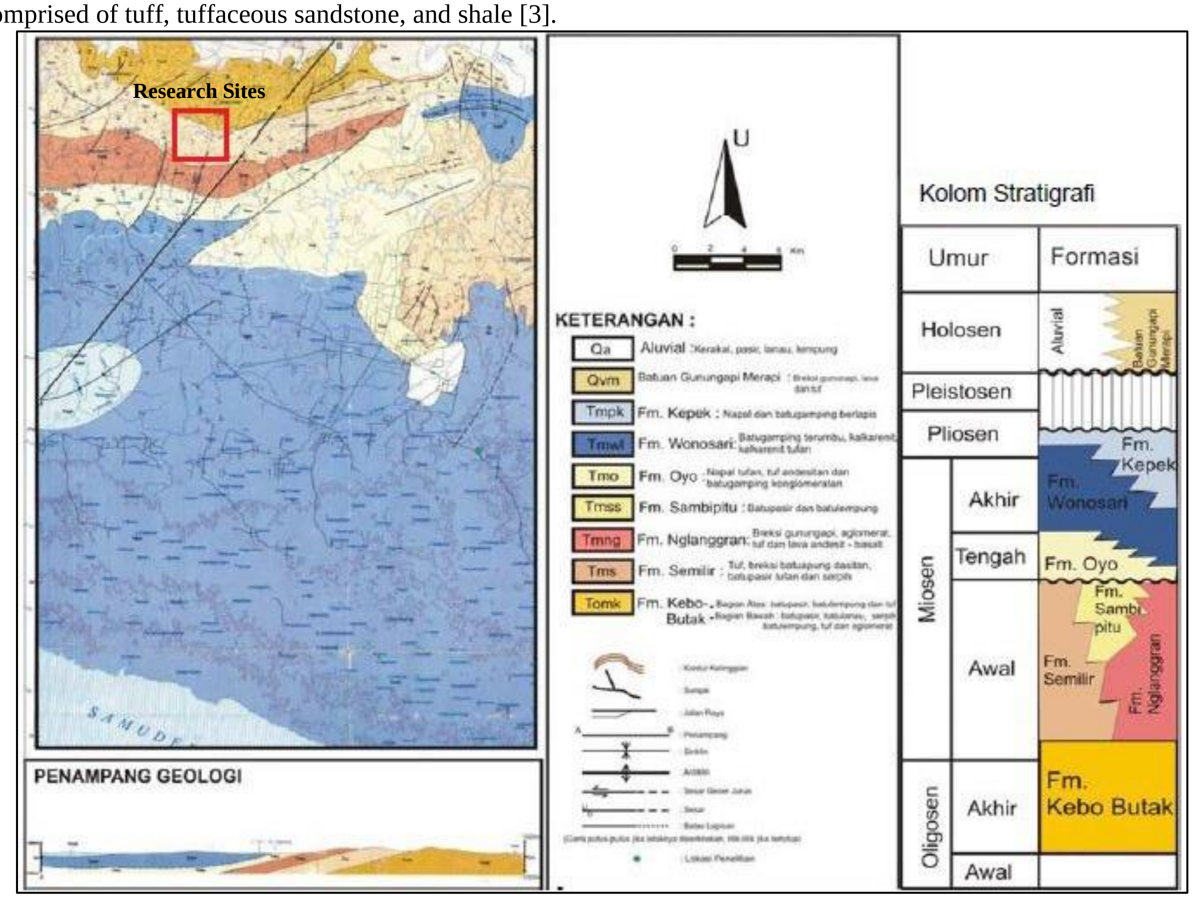

FIGURE 1. Regional geological map and stratigraphic column of research area

LITERATURE REVIEW

The technique of categorizing rock masses by making observations on joint geometry and joint circumstances is known as rock mass characterization. Joint geometry comprises joint orientation, joint spacing, and joint continuity measurements. While joint roughness, joint wall strength, joint opening width, joint filling, weathering, and groundwater discharge in joints are all considered joint conditions [12].

Rock Mass Rating [10, 13] is a categorization system for rock masses developed by Bieniawski (1973-1989) to assess the quality of a rock mass. RMR is made up of five basic characteristics that define rock mass conditions and discontinuities: (1) compressive strength of intact rock (UCS), (2) rock quality designation (RQD), (3) distance between discontinuities/joints, (4) discontinuous/joint condition, and (5) ground water condition. Tables 1 and 2 show the weighting of each parameter and the assessment of rock quality using the RMR classification.

Tabel 1. Parameters of Rock Mass Classification and Weighting

	Parameter		**Rating**						
1	Strengh of intact rock material	PLI (Mpa)	>10	4-10	2-4	1-2	For low compressive strength (UCS)		
		UCS (MPa)	>250	100-250	50-100	25-50	5-25	1-5	<1
	Rating		**15**	**12**	**7**	**4**	**2**	**1**	**0**
2	RQD (%)		90-100	75-90	50-75	25-50	<25		
	Rating		**20**	**17**	**13**	**8**	**3**		
3	Spacing of Discontinuities		>2 m	0.6-2 m	0.2-0.6 m	0.06-0.2 m	<0.06 m		
	Rating		**20**	**15**	**10**	**8**	**5**		
4	**Condition of Discontinuities**								
	Persistence		< 1m	1-3 m	3-10 m	10-20 m	>20 m		
	Rating		**6**	**4**	**2**	**1**	**0**		
	Aperture		None	<0.1 mm	0.1-1 mm	1-5 mm	>5 mm		
	Rating		**6**	**5**	**4**	**1**	**0**		
	Roughess		Very rough	Rough	Slightly rough	Smooth	Slickensided		
	Rating		**6**	**5**	**3**	**1**	**0**		
	Infillings (gouge)		None	Hard filling <5 mm	Hard filling >5 mm	Soft filling <5 mm	Soft filling >5 mm		
	Rating		**6**	**4**	**2**	**2**	**1**		
	Weathering		Unweathered	Slightly weathered	Moderately weathered	Highly weathered	Decomposed		
	Rating		**6**	**5**	**3**	**1**	**0**		
5	**Groundwater Condition**								
	General description		Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		**15**	**10**	**7**	**4**	**0**		

Table 2. Rock Class after Total Weight

Rating	**Class**	**Description**
100-81	I	Very good rock
80-61	II	Good rock
60-41	III	Fair rock
40-21	IV	Poor rock
<20	V	Very poor rock

The Geological Strength Index (GSI) [6], developed by Hoek, Kaiser, and Bawden (1995), is used to evaluate the decline in rock mass strength due by various geological circumstances. The geometric shape of the rock blocks that comprise the rock mass, as well as the surface characteristics of the separating planes between the rock blocks, govern it. An angled rock block with a rough surface area has better rock mass strength than a round rock block with a worn surface area (Figure 2).

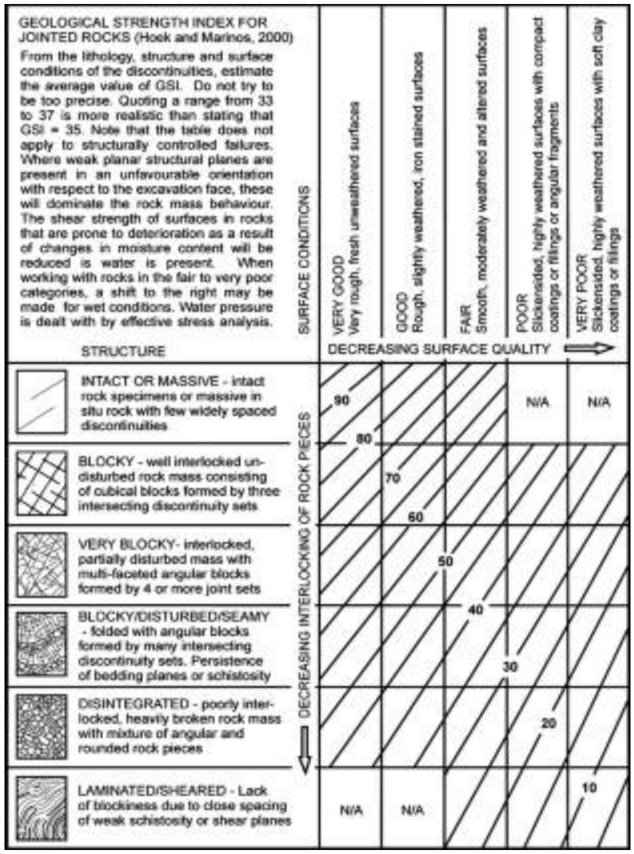

Figure 2. GSI Quantification

The relationship between the Geological Strength Index (GSI) and the Rock Mass Classification RMR) is as follows:

$$\text{For } RMR_{89'} > 23 \quad (1)$$

$$GSI = RMR_{89'} - 5 \quad (2)$$

RESULT DAN DISCUSSION

Rock Mass Rating

Location 1 is a fresh breccia andesite slope, whereas Location 2 is a weathered breccia andesite slope. Tables 3 and 4 offer a summary of the tabulation of RMR values at site 1 and position 2.

Table 3. Results of Rock Mass Classification Location 1

No	RMR Parameter	Hasil	Rating
1	Strengh of intact rock material (UCS)	17.16 MPa (5-25 MPa)	2
2	Rock quality designation (RQD)	99.89 %	20
3	Spacing of Discontinuities	> 2m	20
4	Condition of Discontinuities	15	15
5	Groundwater Condition	Completely dry	15
	RMR total rating		72
	Rock Class		II (Good)

Table 4. Results of Rock Mass Classification Location 2

No	RMR Parameter	Hasil	Rating
1	Strengh of intact rock material (UCS)	6.64 MPa (5-25 MPa)	2
2	Rock quality designation (RQD)	98.59 %	20
3	Spacing of Discontinuities	0.6 - 2m	15
4	Condition of Discontinuities	14	14
5	Groundwater Condition	Damp	7
	RMR total rating		58
	Rock Class		III (Fair)

Geological Srength Index

The results of the RMR are then entered into the equation $GSI = RMR_{89'} - 5$ so that the GSI value for Slope 1 is 67 and is in the Good category (good), while the GSI value for Slope 2 is 53 is in the Fair (medium) category.

Slope Stability and Probability of Failure

The GSI value from the rock mass characterisation is utilized as an input parameter for slope stability analysis, along with other input parameters such as rock constant values (mi) and disturbance factor (D).

Because the stress factor is included in the Finite Element Method approach, it is not only limited to the Safety Factor (SF) that is obtained, but the maximum displacement data when avalanches are also obtained, making it very useful to map the maximum displacement limit of an avalanche slopes as well as useful when reverse analysis of an avalanche [7].

Slope stability analysis using the Finite Element Method approach because the stress factor is included, so it is not only limited to the Safety Factor (SF) that is obtained, but the maximum displacement data when avalanches are also obtained, so it is very useful to map the maximum displacement limit of an avalanche slopes as well as useful when reverse analysis of an avalanche [7].

The appearance of the slopes at locations 1 and 2 is shown in (Figure 3), and the results of the slope stability calculation are shown in (Figure 4).

(a) (b)

Figure 3. (a) Slope of Hargomulyo Hamlet (b) Slope of Ngalang Hamlet

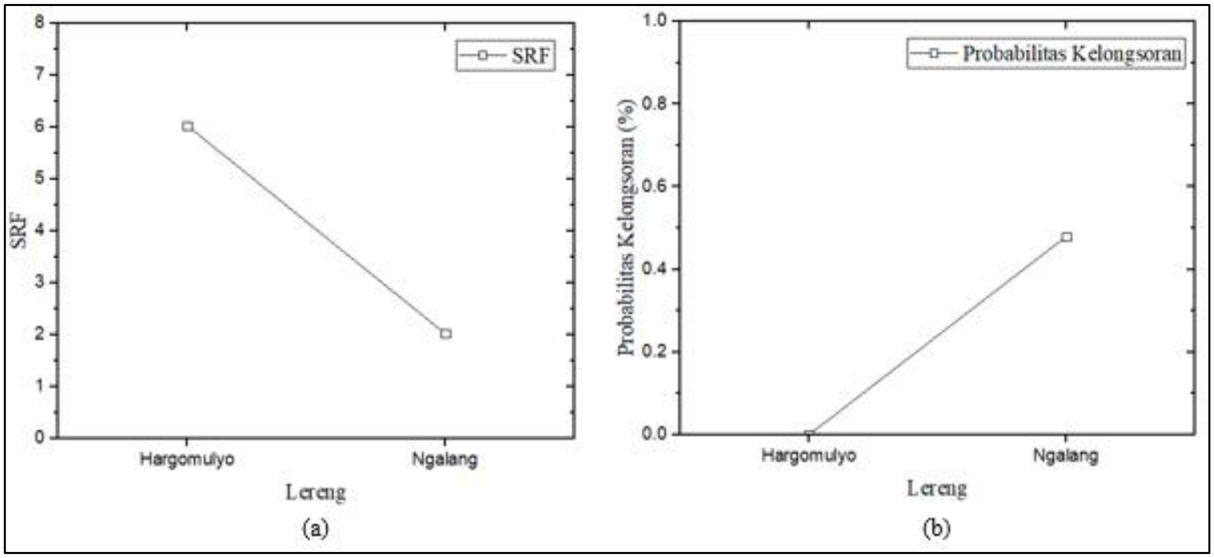

Figure 4. (a) SRF Hargomulyo and Ngalang slopes (b) PoF Hargomulyo and Ngalang slopes

Figure 3 indicates that both slopes are safe, with SRF greater than 1.5. (Slope of Hargomulyo Hamlet with SRF 6.03, PoF 0 percent and Ngalang Hamlet Slope with SRF 2.02, PoF 0.48 percent). The Hargomulyo Hamlet, on the other hand, is in better shape than the Slope of the Ngalang Hamlet. This is proportional to the first estimate of slope stability using the rock mass characterisation technique with RMR and GSI. The slope rock mass of Hargomulyo Hamlet was classed as good by both rock mass categorization methods, whereas the slope of Ngalang Hamlet was classified as fair.

Aside from rock mass classification, another technique was used to determine the source of the discrepancy in SRF values between the two slopes, as shown in (Figure 5).

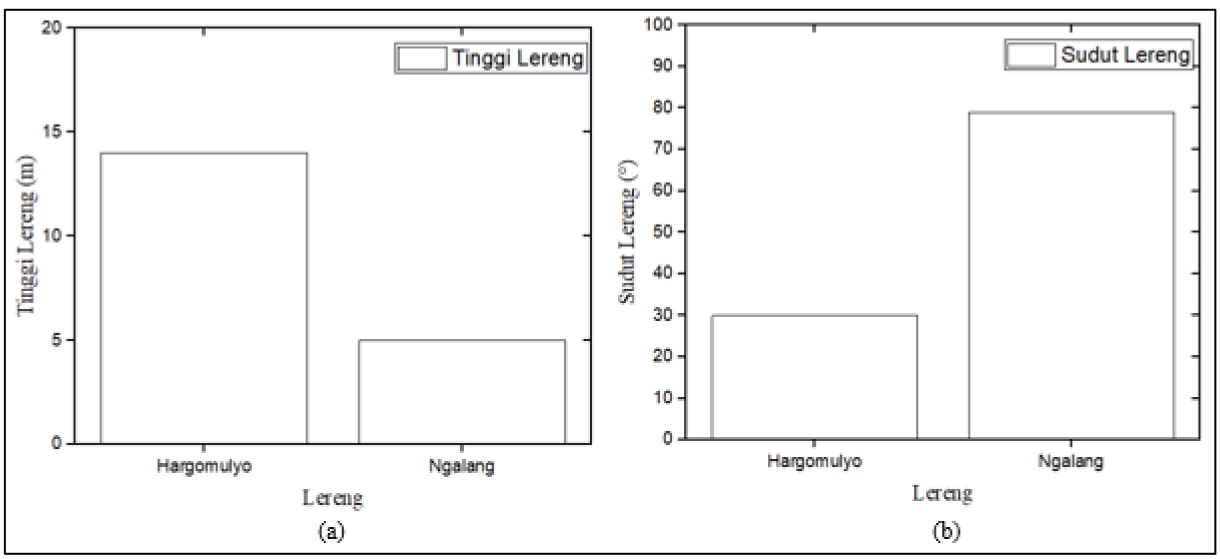

Figure 5. (a) The height of the Hargomulyo and Ngalang slopes (b) The angle of the Hargomulyo and Ngalang slopes

A geometric approach is used to compare the two slopes, and Figure 4 shows that the slopes of Hargomulyo hamlet have a single slope of 14 meters, which is higher than the slopes of Ngalang hamlet, which has a single slope of 5 meters; however, the slopes of Hargomulyo hamlet have a single slope angle that is gentler, which is 30°, and the slope of Ngalang village has a single slope angle of 79°. According to the geometric method, the angle of the slope is an essential aspect that might affect the level of slope stability. Even though the single slope height in Ngalang village is 5 meters, the load received by the slopes is more than the load received by the slopes in Hargomulyo hamlet with a single slope height. 14 meters with a single slope angle of 30 degrees. So lowering the slopes by reducing the angle of the single slope is one technique to strengthen the stability of the slopes in the Ngalang hamlet.

CONCLUSION

Despite the fact that the single slope height in Ngalang village is 5 meters, the load received by the slopes is more than the load received by the slopes in Hargomulyo hamlet with a single slope height. 14 meters with a single slope angle of 30° So, lowering the slopes by reducing the angle of the single slope is one technique to strengthen the stability of the slopes in the Ngalang hamlet.

Despite the fact that the single slope height in Ngalang village is 5 meters, the load received by the slopes is more than that received by the slopes in Hargomulyo hamlet with a single slope height. 14 meters and a single 30° slope angle Sloping the slopes by lowering the angle of the single slope is one technique to strengthen the stability of the slopes in the Ngalang hamlet.

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