ISERME 2021

International Symposium on Earth Resources Management & Environment

10th December 2021

PROCEEDINGS

Organized by
Department of Earth Resources Engineering
University of Moratuwa
Sri Lanka

ISERME 2021

International Symposium on

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10th December 2021, Colombo, Sri Lanka

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Department of Earth Resources Engineering
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Message from Dr. G.V.I. Samaradivakara - Symposium Chair

Welcome to the fifth International Symposium on Earth Resources Management and Environment [ISERME 2021], organised by the Department of Earth Resources Engineering, University of Moratuwa, Sri Lanka.

The research culture of the department revolutionary changed sixteen years back, and in year 2005 with the implementation of World Bank funded project for Improving Relevance and Quality of Undergraduate Education [IRQUE]. We are grateful to the "IRQUE" Project for encouraging and funding the department to organise the first annual research conference in year 2006, which eventually evolved up to an International Symposium in year 2017.

Since 2016, the department has been networking with the international Mining Engineers and Earth Scientists in organising this symposium, annually.

In the past year, we had scheduled to hold this symposium in Hokkaido, Japan, based on the collaboration that we have with Hokkaido University, Japan. However, as Hokkaido was affected by the spreading virus at that time, we had to change our plans. We look forward to having our symposium jointly with them and in Japan also, in the coming years, once the circumstances will back to normal.

On behalf of the Department of Earth Resources Engineering, I wish to extend my sincere thanks to Professor N.D. Gunawardena, Vice-Chancellor of the University of Moratuwa and Professor N.K. Wickramarachchi, Dean-Faculty of Engineering of the University of Moratuwa for granting their kind permissions to hold the inauguration of this year's symposium in the hybrid-mode at the University of Moratuwa.

I greatly appreciate Professor Ajith De Alwis, Dean-Faculty of Graduate Studies of the University of Moratuwa for providing Boardroom facilities of the Faculty, for hosting the inauguration of the symposium in hybrid-mode.

The external reviewers of the symposium are commended for their timely given insightful reviews. Thanks are also extended to all the authors for their excellent submissions made to this symposium.

I'm certain that this symposium will continue to network with more researchers in the fields of Mining and Earth Resources Engineering, in the coming years.

I wish you all, a productive and enjoyable Symposium!

Dr. G.V.I. Samaradivakara - Symposium Chair - ISERME 2021
Head - Department of Earth Resources Engineering, University of Moratuwa
02nd December 2021

Message from Dr. (Mrs.) A.B.N. Dassanayake - Symposium Secretary

On behalf of the Organizing Committee and as the coordinator of the symposium, I would like to warmly welcome your participation in the 5th International Symposium on Earth Resources Management and Environment organised by the Department of Earth Resources Engineering, University of Moratuwa, Sri Lanka. As with the most scientific forums, in ISERME 2021 we have to move to a virtual format due to the challenging times with the COVID-19 pandemic situation.

The event with the theme of "Earth Resources Management and Environment" brings together many academics, students, industry leaders, alumni, and well-wishers and is graced by Prof. P.K.S. Mahanama, Deputy Vice-Chancellor, University of Moratuwa as the Chief Guest, and Dr. Manoj Verman, a well-known Tunneling and Rock Engineering Expert and the President of International Commission on "Hard Rock Excavation" as the keynote speaker. This event creates an international forum for academics, researchers, industry leaders, professionals, and alumni to come together sharing the latest findings, and have constructive discussions across a broad range of disciplines related to Mining and Earth Resources Engineering. The event also seeks to network with a large number of organisations and individuals at national and international levels.

I wish to express our special recognition to our sponsors and supporting institutions for their financial support and dedication. I sincerely thank the organising committee and technical program committees for their contribution toward ISERME 2021 and all the academic and non-academic staff members of the Department of Earth Resources Engineering for effectively fulfilling the individual tasks undertaken to make the event a success.

We would like to express our deepest appreciation to the authors whose technical contributions are presented in these proceedings. We have been able to prepare this proceeding, because of their excellent contribution made and hard work.

Dr. (Mrs.) A.B.N. Dassanayake
Senior Lecturer – Symposium Secretary
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Table of Contents

Keynote Address

Challenges of Tunnelling

Dr. Manoj Verman
Tunnelling & Rock Engineering Expert
President, International Commission on "Hard Rock Excavation"
President, Indian National Group of ISRM

These are exciting times for infrastructure growth. With the focus on development in several developing countries, we in the infrastructure sector are hoping for a great future, and tunnelling is one of the most promising areas.

Tunnelling has become a vast subject, encompassing a wide variety of topics within its realm. It is a surprise that, despite its vastness and its significant relevance to civil engineering, this subject is hardly taught at the graduate and post-graduate levels. The world is looking at a substantial shortage of engineers skilled at various aspects of tunnelling. This presentation is an attempt to create a spark in the minds of the youngsters to take up tunnelling as a career while, at the same time, to introduce the uninitiated into the exciting world of tunnelling - both in the difficult terrain of the mountains, and in the tricky conditions of an urban setting.

While keeping the above objectives in mind, the presentation will meander through a range of topics associated with tunnelling – from the basic philosophy to planning to design to construction and so on. Some interesting yet fiddly aspects of tunnelling will be particularly highlighted, and a few case histories will be included to drive the point home. Besides the technical aspects, some non-technical aspects of tunnelling would also be covered.

While it is simply not possible, in a short time, to dwell upon all the challenges that this exhilarating subject throws, an attempt will be made to give the participants a strong flavour of the exciting world of tunnelling.

Session I

Mining and Geomechanics

An Exploratory Factor Analysis on Issues and Constraints in Sri Lankan Aggregate Quarry Industry

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Abstract
Mining activities are influenced by stakeholders such as regulators, operators, service providers, customers, residents, and other interest groups. The governing bodies seem to fail to adopt a stakeholder inclusive approach to gain the balance between the interest of the stakeholders and the organisational plans and revenues. Such issues are known at the surface level, but no solid quantitative approach has been used to scientifically prove their existence and associations. Thus, this study aims to find the issues with statistical evidence. Convenient sampling was used due to time restrictions and new work norms to pick a substantial number of respondents. A numerical assignment and logical ordering to qualitative data were done to perform the quantitative analysis. Factor analysis was used to find the principal components and the variables which mainly loaded the components showed significant interpretable correlations. The most significant factors associated with issues in quarry operations and management are found in terms of quality, reliability, and usage of monitoring, application of safety measures, quarry type based hazardous conditions, planning and maintaining quarry activities, combatting drilling issues via site planning, use of initiation technique by quarry type, the impact of environmental and social issues, handling public complaints and strategies to improve quarry operations.

Keywords: Correlations, Principal components

1. Introduction

Quarrying is a branch of surface mining method that is used to extract minerals. By the end of 2019, Geological Survey and Mines Bureau (GSMB) had issued 3918 valid mining licenses, out of which 1812 were representing aggregate quarries. Western and Northwestern provinces have marked 25.88% and 20.31% out of the total quarries licensed in Sri Lanka. Mining and quarrying activities account for Rs. 358,287 million from Gross Domestic Product at Current Market Prices in 2018, showing a 2.1% growth compared to 2017 [1]. Aforesaid statistics reveal that there is a significant contribution by mining and quarrying to the economy of the country.

Quarries are categorised considering the type of their mining license and the minerals that they mine by the GSMB. However, quarries can be characterised by the development concept, category of the mining licenses, and the absolute land right of the mine.

The mining industry has a wide range of stakeholders, including regulators, operators, service providers, customers, residents, and other interest groups. However, in the execution of its governance role and responsibilities, the governing body should adopt a stakeholder-inclusive approach that balances the concerns, interests, and expectations of material stakeholders in the best interests of the organisation over time.

Procedural complications will accompany the operational complications. 20-30% of energy dissipated by explosives is directed to rock fragmentation, and the rest is wasted as ground vibration (GV), air blast overpressure (ABOP), noise, and fly rocks [2]. Guidelines have been formulated to determine the safe levels of blasting vibrations to minimise the damage by GV, ABOP, and annoyance.

Generally, issues will emerge when the provided guidelines are not managed and when those are not feasible for every quarry due to site-specific conditions and more. Apart from that major technical, procedural, environmental, and sociological aspects will generate issues and constraints for the operation of a quarry in many ways. Issues and constraints in the aggregate quarrying industry are known, but the extent of existence and the distribution of them are not analysed properly, and no solid quantitative approaches have been taken to scientifically prove the existence of the issues with their interrelations [3]. This research paper will explore the extent of existence, interrelations of the known issues, and the underlying traits which have not been exposed yet.

2. Methodology

2.1 Data collection

A questionnaire was developed to collect the opinions of the stakeholders. Most of the questions in the questionnaire were made available with multiple choices with comprehensive answering options, which made the questionnaire more familiar to the respondents.

Intending to collect responses from the mining operatives (mining engineers/ managers/ owners), the questions were reordered rationally into two clusters, namely technical aspects and general opinion. The questionnaire consists of 38 questions, out of which the first 8 questions were informative, and the rest of 30 questions were classified orderly into aforesaid clusters.

The google form-based questionnaire was intended to reach the mining engineers/ managers and owners to collect opinions of them regarding the issues and constraints in the aggregate quarrying industry.

Data collection was carried out not only through online services (Google Form) but also by interviews through mobile phone calls (most of the quarry owners). The respondents were asked the same questions in brief, and the required data was collected contemporarily.

Altogether from the online surveys as well as from interviews over the phone, it was able to collect 67 responses within the time frame.

2.2 Sampling plan

The sampling plan was developed considering the quarry categorisation criteria in Figure 1. Quarries were classified by considering the license type, absolute ownership of the land, and development type of the quarry.

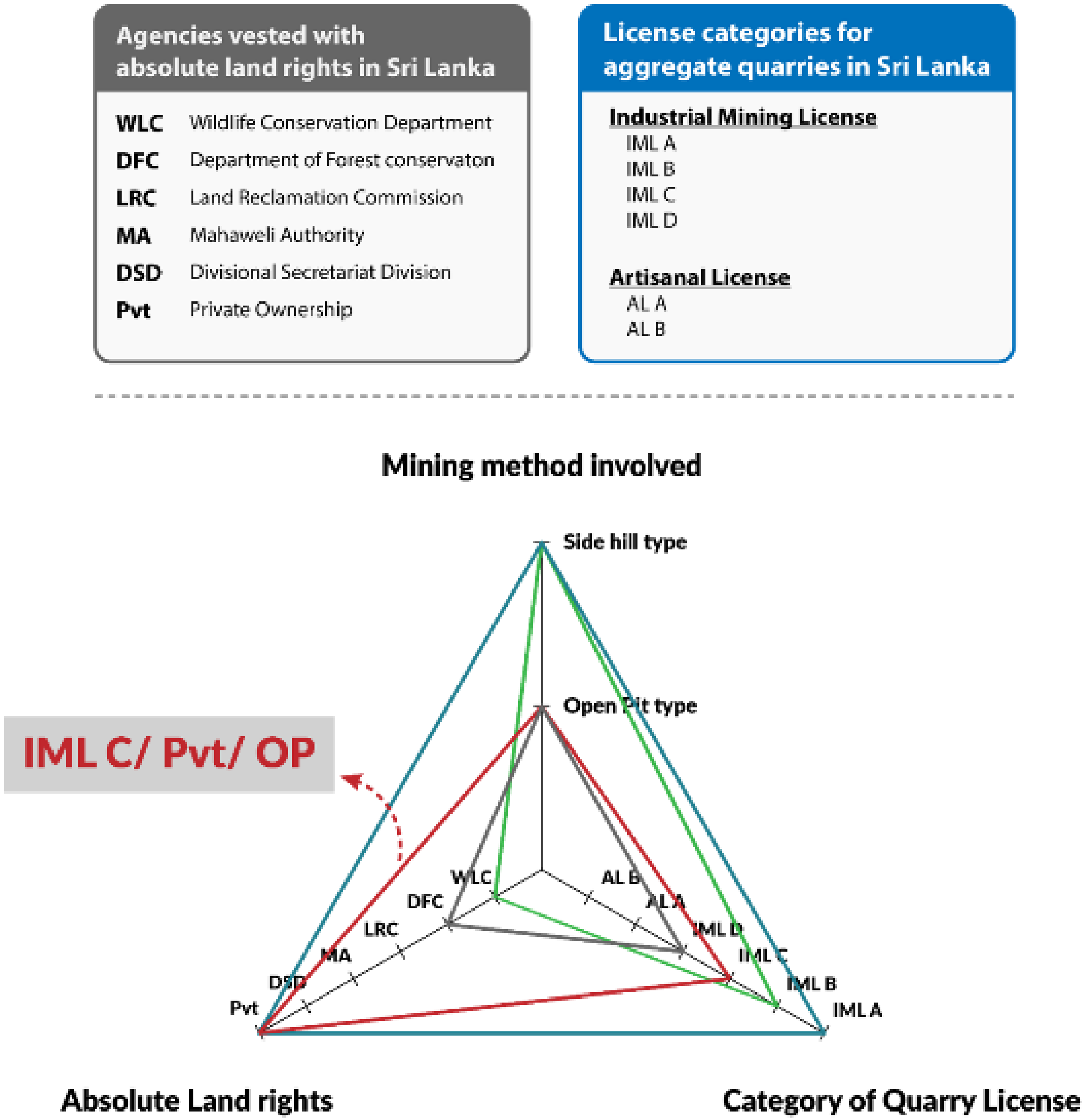

Figure 1: Quarry categorisation criteria.

The total sample size was determined using the "Table for Determining Sample Size for a Finite Population" [4]. It was planned to apply a multistage stratified random sampling technique to gather data from stakeholders. This sampling plan was intended to launch in two stages to select the quarries and the stakeholders.

Stage 1: Sampling by quarry type

The population is classified into 72 substrata according to quarry categorisation criteria in Figure 1; the sample size from each substratum is calculated, proportioning the stratum to the population. Then the sample is selected from each substratum utilising simple random sampling technique.

Stage 2: Sampling by the stakeholder

A selected quarry from sampling stage 1 will undergo steps in sampling stage 2.

The following factors were considered in stage 2.

i. Operators are limited in number, so that no sampling was done.

ii. Environmental activists in the region were considered.

iii. Regulators are regional.

iv. The residents around active zones need to be determined by site investigation. A residential population for each quarry must be identified within a radius of 0.3km from the quarry site. Adhering to that sample frame, a random sample of residents can be determined.

NB: Instead of the described two-stage stratified random sampling, convenient sampling had to be used, obeying the work norms and time restrictions due to the pandemic situation.

2.3 Quantifying responses

Questions were constructed in a strategic way whilst their answering options were ordered with logical reasoning. Then the responses received were treated as ordinal, and the principal component analysis was successfully performed. As the first step of quantifying data, a numerical assignment was done to the responses, as shown in Table 1.

Table 1: Quantifying Question 1

Q1. How frequently are the production blasts conducted at your site?	
Responses	Ordinal Value
Daily basis	1
Several times a week	2
Once a week	3
Once in a few weeks	4

2.4 Data Analysis

The factor analysis was performed using ordinal data and extracted the required principal components. Their factor loadings indicated the contribution to each principal component by the questions. Thus, the correlations were examined only between the questions identified with the highest factor loadings within each principal component. Correlation analysis was done at a 5% significance level.

Factor analysis was carried out for 10 classes of questions that correspond to the concerns under analysis, as shown in Table 2.

Table 2: Variables of Class 1

Class 1	Frequency of blasting / GV and ABOP monitoring (Significant overlapping check)
Var. No	Variable
1	Frequency of production blasts conducted at a site
7	Frequency of monitoring Ground Vibration (GV) due to quarry operations
8	Frequency of monitoring Air Blast Over Pressure (ABOP) due to quarry operations
10	Vibration monitoring mechanism in the site
11	Type of monitoring activities
12	Quality and management of the vibration and air blast overpressure measurements

3. Results and discussion

3.1 Factor Analysis

Quantitative statistical analysis was performed to identify the factors underlying the variables. The correlations between the components/variables which contributed highly towards composing factors under each class of variables are used for interpretations. The following data shows the results of the factor analysis. The results of factor analysis of variables under class 1 only are listed and explained below in detail.

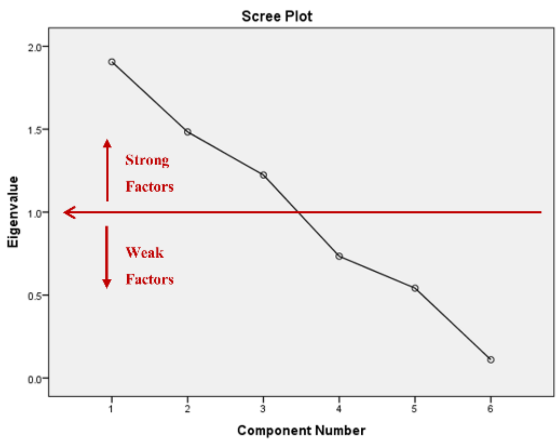

Figure 2: Scree plot for the components (Class 1)

Class 1

Extraction Method: Principal Component Analysis

Principal component analysis extracts strong factors with high loadings which are expected to represent real underlying factors. In Table 3, components with eigenvalues greater than 1 are shown with a name for the latent factor. The low eigenvalues are not assumed to represent real factors underlying the traits in the quarrying industry.

Table 3: Class 1 Total Variance explained.

Component	Extraction Sums of Squared Loadings		
	Eigen Value	% of Variance	Cumulative %
1-Monitoring Quality	1.907	31.776	31.776
2-Reliability of Monitoring	1.483	24.724	56.500
3-Usage of Monitoring	1.224	20.403	76.903

76% of the total variability of the data gathered under the 6 variables is explained by the three principal components extracted, as shown in Table 4.

Following Figure 2 clearly shows the difference between the eigenvalues associated with components.

Table 5: Component matrix of class 1.

	Component 1	Component 2	Component 2
V1	0.141	0.311	**-0.762**
V7	**0.958**	-0.164	-0.019
V8	**0.923**	-0.289	0.104
V10	0.252	**0.795**	-0.148
V11	0.021	0.288	**0.749**
V12	0.231	**0.750**	0.222

After examining the three factors, according to the factor loadings, the most important variables were recognised and followed to look at their relationships through a correlation analysis. The following matrix shows the correlations, which were significant at the 0.05 level.

The highest contributions were made by variables 7 and 8 on composing component 1: Monitoring Frequency. The significant correlation between V7 and V8 (0.860) indicates that when the ground vibration monitoring frequency increases, the frequency of air blast overpressure monitoring has also been increased. Variables 10 and 12 have made the highest loading on the second component: Quality of Vibration Monitoring Management. It reveals that by using a regulated monitoring mechanism, the sites tend to obtain reliable data. Consequently, their decision-making has been effectively supported by these data. Although V1 and V11 make the major contribution to defining the third component: Usage of Vibration Monitoring, they do not depict any significant relationship between the two variables.

Table 4: Class 1 Correlations Matrix.
**Correlation is significant at the 0.05 level.*

		V1	V7	V8	V10	V11	V12
V1	Pearson Correlation	1	0.166	-0.018	0.258	-0.237	0.034
	Sig. (2-tailed)		0.175	0.885	0.055	0.055	0.784
	N	68	68	67	56	66	67
V7	Pearson Correlation	0.166	1	0.860*	0.152	0.000	0.121
	Sig. (2-tailed)	0.175		0.000	0.263	0.998	0.331
	N	68	68	67	56	66	67
V 10	Pearson Correlation	0.258	0.152	-0.004	1	0.087	0.419*
	Sig. (2-tailed)	0.055	0.263	0.977		0.522	0.001
	N	56	56	55	56	56	56

Although V1 and V11 make the major contribution to defining the third component: Usage of Vibration Monitoring, they do not depict any significant relationship between the two variables.

In general, the above-mentioned monitoring exists in IML A quarries where a mining engineer is essentially occupied. Thus at these sites, they will monitor ABOP and GV and adjust their specific charge suiting to the local conditions using the given limits as reference under a mining engineer's observation.

3.2 Summary of Findings

i. The existing regulatory guidelines and procedures seem not to be conducive for quarry operations even though they are highly influential and controlling.

ii. The stability of rock slopes is more important than maintaining design parameters for the quarries at which mining activities are continuing based on site conditions and production requirements.

iii. Lack of induction and training for employees regarding their work responsibilities and practices at quarry sites leads to malpractices.

iv. Stable quarry slopes and regularly updated site-specific blasting parameters are common in the sites where mining engineers are present.

v. The fact that the public complaints on vibration and fly rocks become high when the blasting frequency increases is contradicted to a considerable extent by the responses received regarding public complaints by the quarry owners.

vi. Public complaints are found to be false and are made to receive compensation mostly at the sites where the blasting frequency is comparatively low.

vii. IML B and C categories rely more on annual monitoring compared to the A category, which conducts frequent monitoring. This reflects that they do not tend to analyse the effects of ABOP and GV on the surroundings unless any issues occur since they focus mainly on production.

viii. Environmental and social issues can be highly influential to the quarry operations as environmental issues amplify the social issues.

ix. Although all the mining entities are complying with the permitted initiation mechanisms provided by the license conditions, they still face practical issues while adopting and maintaining such practices.

x. Misfires and hazardous conditions in IML C quarries are mostly due to the malfunctioning of fuse caps.

5. Conclusion

The most significant factors associated with identifying traits in the mining industry are,

- Monitoring Quality.
- Reliability of Monitoring.
- Usage of Monitoring.
- Application of safety measures.
- Quarry type-based hazardous conditions.
- Planning and maintaining quarry activities.
- Combatting rock drilling issues via site planning.
- Use of initiation technique by quarry type.
- Impact of environmental and social issues.
- Handling public complaints.
- Strategies to improve quarry operations.

6. Recommendations

Any researcher can take the listed factors, in conclusion, to investigate more on the traits faced by the mining industry in detail along the prioritised dimensions with reference to any selected group of quarries.

By regulations, there are restrictions on the amount of production per month that a licensed quarry should comply with [5]. If any effort is taken to exceed the prescribed amount, that will be considered as a violation of the law.

i. Engineers should monitor and assess the harm caused by altering the design parameters.
ii. Production management plans should be formulated.
iii. Try not to enter contracts that request production levels that the quarry cannot cater for.
iv. Change the design parameters by keeping the guidelines by GSMB as the reference.
v. In-house training programs should be initiated to train the employees for specific jobs.
vi. People should be hired not in relation but with suitable skills.
vii. New recruitments should be done checking whether the candidate possesses a training certificate related to mining or heavy machinery.

Acknowledgement

We express our heartfelt gratitude to the Geological Survey and Mines Bureau staff members who helped us with the necessary information for the study. We would like to thank the Department of Earth Resources Engineering, University of Moratuwa, for facilitating this research.

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Numerical Analysis of Effects of Clay on a Cut Rock Slope Deformation at an Open-pit Limestone Mine, Japan

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Abstract
Rock slope instability is one of the major challenges of rock engineering projects, including open-pit mining. In this regard, rock slope deformation due to excavation, change in temperature, and influence of rainfall and snowfall have been previously investigated to understand characteristics and causes of slope deformation observed at an open-pit limestone quarry in Japan. The results only revealed characteristics of the deformation as forward and downward displacement of the cut rock slope, but its causes were not clarified. To deduce the causes of the rock slope deformation, we employed the 2-dimensional finite element method (2-D FEM) to investigate the deterioration effect of clay found at the footwall of the rock slope in terms of reduction in Young's modulus of the clay based on experimental results. Firstly, change in distances was analysed from displacement data measured by the automated polar system (APS) over five years, which decreases gradually with time. Secondly, the simulation results were discussed and then compared with the measured displacement data, which shows similar tendencies at the middle and top of the rock slope revealing maximum displacement at the middle of the rock slope. Conclusively, deterioration of clay at the footwall of the rock slope is one of the possible causes of the deformation in the quarry.

Keywords: 2-D FEM, Deterioration effects, Rock slope deformation, Stability assessment, Young's modulus

1. Introduction

Regarding the increased demand for deep resources and minerals exploitation and environmental problems, rock engineers have become much more interested in the study of rock slope instability in order to provide the basis for deformation, stability and safety of rock engineering projects, including open-pit mining, which largely depends on the strength and deformability of rock masses [1]. These have led to numerous researches intending to cover all aspects of rock mechanics from theories to engineering practices, emphasizing on the future direction of rock engineering technologies [2]. In mining engineering, open-pits account for the major portion of the world's mineral production. However, the instability of rock slopes has been a concerning issue that commonly occurs in open-pit mines around the world. Therefore, it is essential to ensure a degree of stability for the slopes in the mines to minimize the risks related to the safety of operation and economic risks to the reserves, mainly in the complex vicinity of ore bodies where exact geological and mechanical properties of cut rock slope are uncertain [3]. These complicate the prediction of the stability of rock slopes.

In Japan, there are large reserves of limestone that have been quarried for cement productions, construction aggregates, and consumption in iron and steel industries [4]. Therefore, large-scale limestone quarries, mostly open-pit limestone quarries, are still under operation. However, most of the limestone deposits are located at steep mountainous terrain [5-6], excavation mostly progresses from either the top or side of the mountain to its foot. Sometimes, these left a huge rock slope in the quarry during operation and/or after being mined out [7]. In this regard, extensive studies have been done on mining-induced deformation of rock slopes in pit-type mines, Japan [8-11].

Previously, the elastic deformation of the cut rock slope due to excavation and backfilling, effects of change in temperature, the influence of rainfall and snowfall on the slope stability have been investigated to understand the characteristics and causes of the slope deformation observed at an open-pit limestone quarry in Japan, but these only reveal the characteristics of deformation modes as forward and downward displacement of the cut rock slope while causes of the rock slope deformation were not clarified. In that regard, the Finite Element Modelling (2D-FEM) method was employed to investigate the deterioration effect of clay found at the footwall of the rock slope in terms of reduction in Young's modulus of the clay quantified by experimental results. Finally, the deformation mechanism was discussed based on the comparison of the measurement and simulated results of the rock slope displacement.

2. Overview and displacement characteristics of the rock slope at of the studied limestone quarry

2.1. Overview of the studied limestone quarry

The studied quarry is a small-scale open-pit limestone quarry, which has been under operation for more than 100 years with an annual production of 200,000 tons [12]. The rock mass of the quarry consists of complex geological rock types of main limestone, schalstein and slate rocks. However, the schalstein and slate rocks have been weathered intensively, resulting in the formation of clay of about 70 m thick (elevations of 440-370 m) at the footwall of the rock slope, as illustrated in Fig. 1.

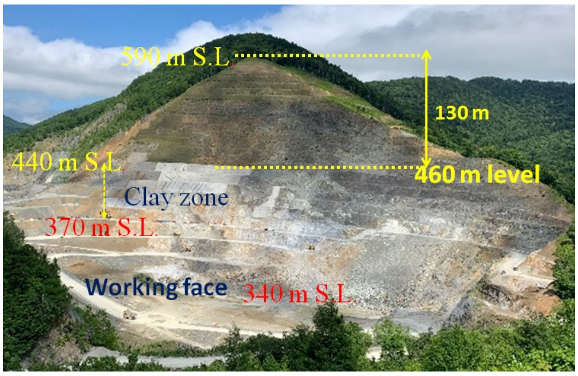

Figure 1: Plane view of the rock slope layout.

At present, mining has been undertaken at 340 m above sea level, whereas backfilling has been done on the northern side of the quarry. Initially, the quarry was designed with a bench height of 10 m at a slope angle of about 70°, as illustrated in Fig. 2. Currently, as the operation continues, the height of the rock slope has increased to about 130 m, as seen in Fig. 1. The final slope angle has been reduced to 55°.

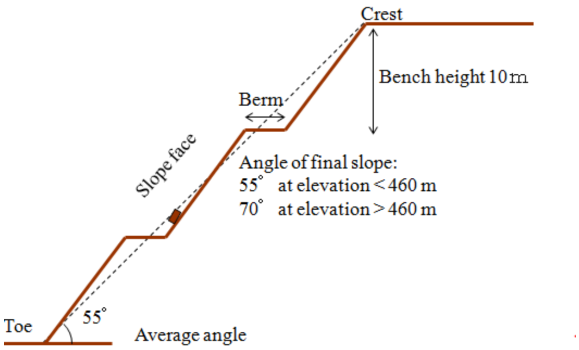

Figure 2: The bench design of the quarry.

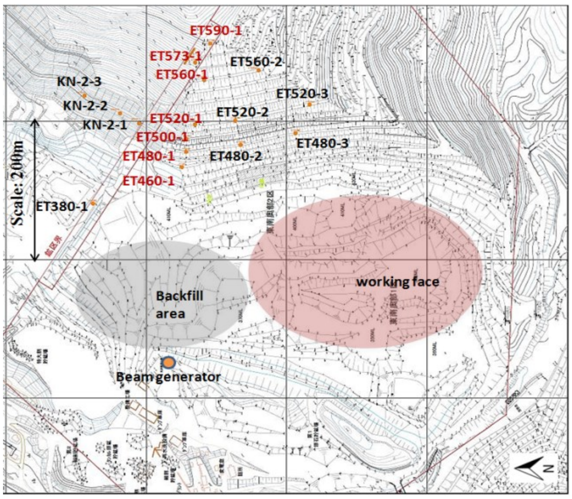

Figure 3: Map showing the APS layout. The ET represents the mirror point locations of automated polar system (APS) set along the rock slope. Each mirror point is represented with a number that indicates its level of elevation.

2.2 Rock slope displacement measurements

In order to ascertain the characteristics of rock slope deformation observed at the quarry, displacement has been measured for more than five years using an automated polar system (APS). In APS, the travelling times of laser beam from a beam generator to mirrors located at various points along the slope (Fig. 3) were measured. Thereafter, changes in distance between each of the mirror points and the beam generator point were calculated from the change in travelling time and velocity of the laser beam. The calculated change in distance from 11 APS data measured from January 2014 to April 2019 are shown in Fig. 4. The results show that change in the distance at all elevations decreases gradually with time, although its decreasing rate depends on mirror points. Total changes in distance are approximately between 20 mm and 100 mm. Fig. 5 shows the relationships between change in distance and elevations of the quarry, which revealed that maximum displacements occurred at the middle (elevation 520 m) of the rock slope. This implied that rock slope displacements depend on the elevations. The result was also used to validate the simulation results.

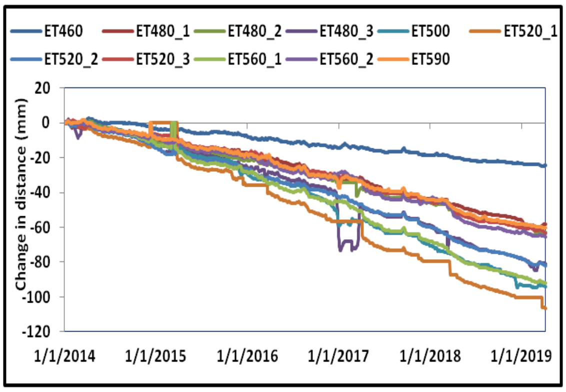

Figure 4: Change in the distance from 2014 to 2019.

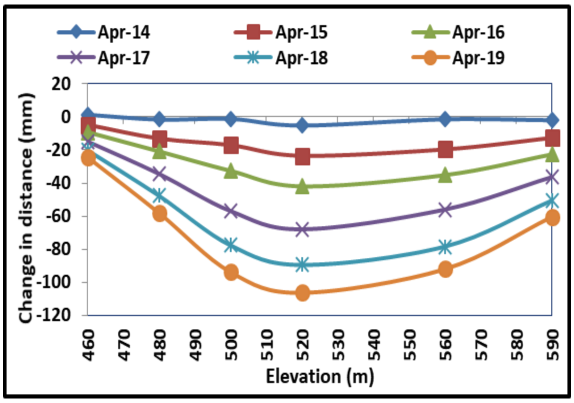

Figure 5: Relationship between change in distance from 2014 to 2019 and elevations.

3. Numerical analysis of rock slope displacement induced by deterioration effect of clay

In this section, the open-pit quarry consisting of limestone, schalstein and slate rocks as main rock types were modeled as homogeneous limestone, except within the clay zone in the vicinity of the footwall of the rock slope by using 2-D FEM in terms of reduction in Young's modulus of the clay-based on the quantitative experimentally results. This is to understand the deterioration effect of the clay on the cut rock slope deformation.

3.1 Simulation conditions

2-D FEM was undertaken using MIDAS GTS/NX 2014 (V2.1) [13] finite element code to simulate deformation induced by a reduction in Young's modulus of clay. Finite element meshes on cross-sectional areas of the quarry, as shown in Fig. 6, were generated using six-node triangular elements based on the elevations read from the regional contour map. Fig. 7 shows the entire analytical model with the dimension of 830 m and 1489 m.

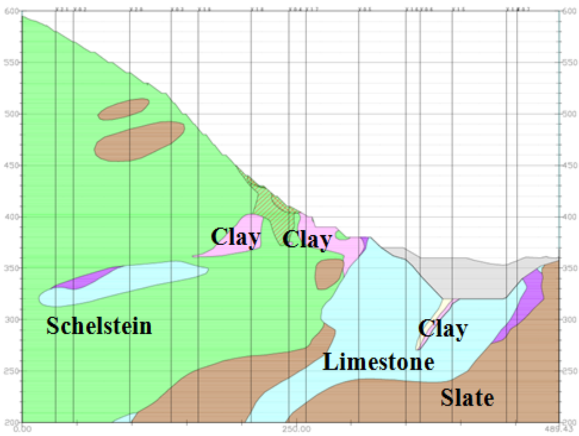

Figure 6: The cross-sectional area of the quarry.

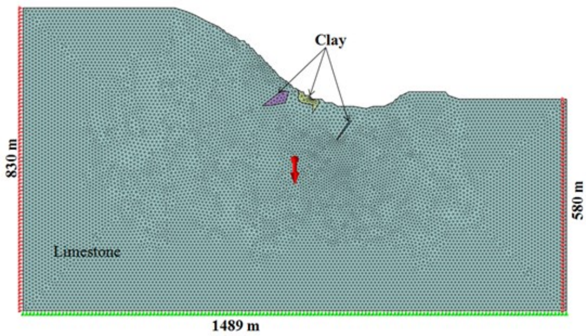

Figure 7: The entire model generated based on the elevations read from the cross-sectional area of the quarry.

The rock mass was assumed to be homogeneous limestone, except within the clay zone in the vicinity of the footwall of the rock slope. The mechanical properties of the limestone and clay zone for this simulation are presented in Tables 1 and 2. In order to clarify the deterioration effects of the clay on the slope deformation, two basic analyses were made based on experimental results, as shown in Tables 1 and 2. Firstly, Young's modulus of clay was set as 50 MPa and 20 MPa at initial stages, then it was assumed to have deteriorated to 20 MPa, and 3 MPa, defined as Case 1 and Case 2 as Young's modulus of limestone was set as 1 GPa, respectively (Table 1). Secondly, Young's modulus of limestone was increased to 5 GPa, whereas Young's modulus of clay was the same, then defined as Case 3 and Case 4, respectively, as shown in Table 1. The analyses were carried out under a plane-strain condition. The nodal displacement perpendicular to the right-left and the bottom surface of the model were fixed at zero. Nodal forces due to gravity were applied to the entire model in the vertically downward direction to generate the initial stress field. Afterward, the relative displacements induced by a reduction in Young's modulus of clay were calculated for each of the models by subtracting the displacement at initial Young's modulus from that of after deterioration.

3.2 Simulation results and discussions

The relative displacements induced by a reduction in Young's modulus of clay (50MPa → 20 MPa) and (20 MPa → 3 MPa) were shown in Fig. 8-11, where horizontal displacement (X-direction) and vertical displacement (Y-direction) are shown on the right-hand-side and left-hand-side of the figures, respectively. The positive values (in X-direction) and the negative values (in Y-direction) indicate forward and downward displacement of the rock slope, respectively. The results revealed that forward surface displacement occurred mostly at the top of the cut rock slope, whereas high downward displacement occurred below the clay zone near the footwall of the rock slope. It also shows similar tendencies of displacement but different magnitude, which increases with reduction in Young's modulus of clay, revealing that displacement depends on the deterioration of the clay. Fig. 12 shows the result of the change in distance calculated from the simulated relative displacement, revealing similar tendencies but different magnitude depending on Young's modulus of clay and limestone. To validate the simulation results, the simulated change in distances was compared with the result of the measured displacements (Fig. 5); both show similar tendencies at the middle and upper part of the cut rock slope. This suggested deterioration of clay found at the footwall of the rock slope as one of the possible causes of the rock slope deformation observed at the quarry.

4. Conclusion

In this paper, displacement measured over five years by APS was analyzed in order to understand the characteristics of rock slope deformation observed at the quarry. The measured results revealed that the distance between the beam generator and mirrors at all elevations on the rock slope gradually decreases with time. Although, the magnitude of change in distance differs at all elevations, and the maximum displacement occurs at the middle of the rock slope. Subsequently, the causes of deformation observed in the quarry were investigated numerically by considering the deterioration effect of clay deposited at the footwall of the rock slope. The results revealed that deterioration of the thick clay could be the possible cause of the slope deformation in the quarry.

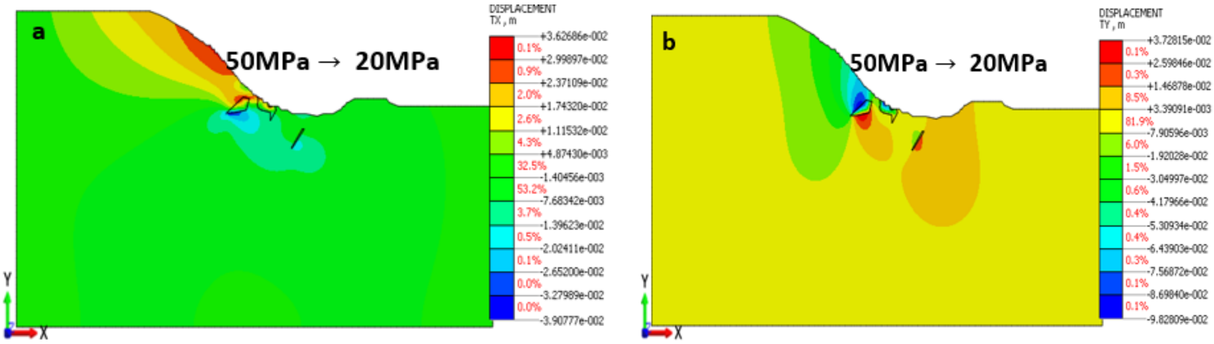

Figure 8: Relative displacement distribution in X-direction (a) and Y-direction (b) (Case 1).

Table 1: Young's modulus of limestone and clay rock.

Case	Clay (GPa)		Limestone (GPa)
	Initial	After deterioration	
Case 1	0.05	0.020	1.0
Case 2	0.02	0.003	1.0
Case 3	0.05	0.020	5.0
Case 4	0.02	0.003	5.0

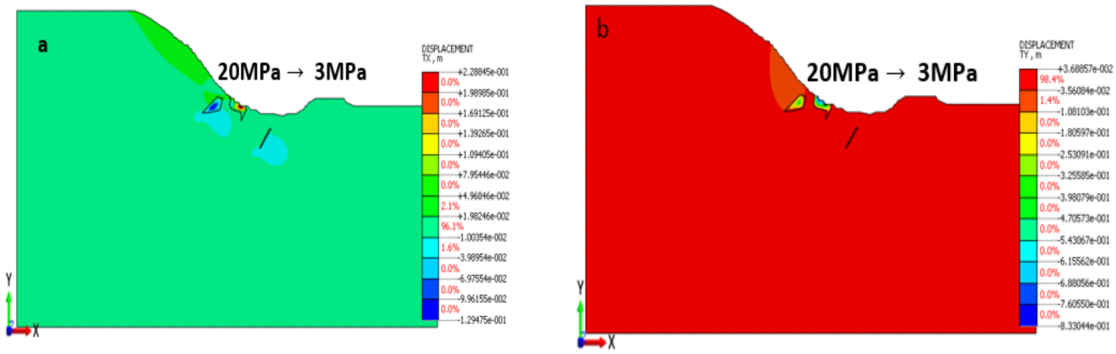

Figure 9: Relative displacement distribution in X-direction (a) and Y-direction (b) (Case 2).

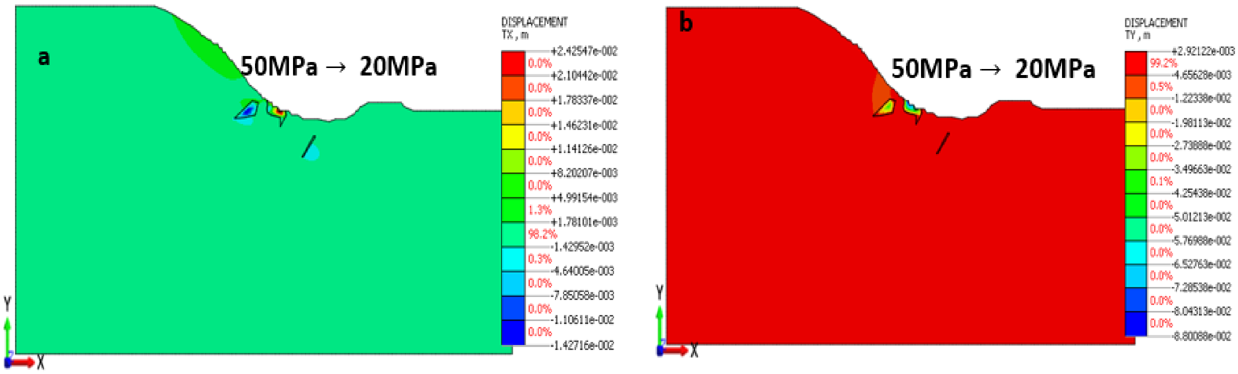

Figure 10: Relative displacement distribution in X-direction (a) and Y-direction (b) (Case 3).

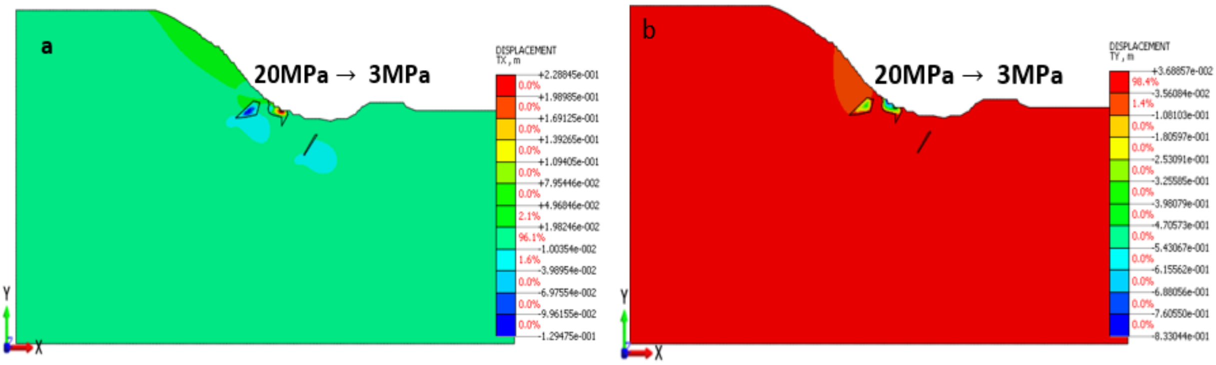

Figure 11: Relative displacement distribution in X-direction (a) and Y-direction (b) (Case 4).

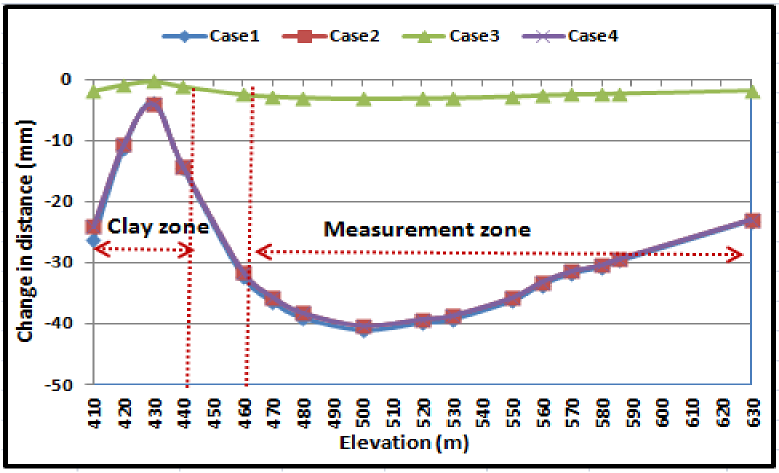

Figure 12: Change in the distance calculated from simulated relative displacements.

Acknowledgement

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A Role of Information Technology in Geotechnical and Mining Engineering for the Past, Present, and Future

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Abstract

In this paper, the development of information technology is divided into three parts: 1975-2000 (past), 2000-2025 (present), and beyond 2025 (future). During the last century, mainframes were considered topmost in terms of processing power. However, with the evolution of personal computers and laptops, computational power has increased in an exponential manner, and at the same time cost of hardware has dropped substantially. Various tech giants are famous for their specific products, such as Microsoft for their MS Office, Oracle for databases, routers from CISCO, and SAP for ERP. In the mining industry, information technology has always been used as a supportive role in various functions such as exploration, drilling and blasting, production and quality monitoring, financial accounting, inventory management, and dispatch of the minerals. The computer science department or information technology department has become crucial to support all functions in geotechnical and mining engineering. Various exploratory and mine planning software such as Surpac and Data Mine are commonly used for exploration and short to long-term mine planning. With the increase in processing power, even the mobile phone has become a powerful and essential device for communication. With enhanced power of the camera and increased data storage, internet connectivity, several useful applications are developed on mobile phones. Various OEMs have their own software for monitoring mining equipment which has benefited mining companies. In the future, artificial intelligence and machine learning models shall be applied in a geotechnical mining operation to improve productivity, safety, and sustained mining operation.

Keywords: Artificial intelligence, Drilling and blasting, Information technology, Machine learning, Mine planning software

1. Introduction

Information technology is well known since the last century for the creation, processing, storing, retrieving and exchanging of every type of electronic data. Mainframe computers slowly decreased from the 1950s to the 21st century with the introduction of intel chips and personal computers [1]. In 1965, Golden E Moore made an observation that the number of transistors on a microchip would double about every two years, though the cost of computers would be halved [2]. His prediction, even after 50 years, appears to be correct. In the mining industry, personal computers were introduced in 1990 [3]. Mining, whether opencast or underground, is a complicated process having many dimensions. Initially, linear computerised models were developed to solve operation research-related problems. Long-term scheduling was always challenging in order to solve how any deposit could be economically mined with a long time span. Computers and quantitative techniques played an important role in solving scheduling problems in mine planning and comparing the progress of mine faces in operation by taking into consideration geology of the deposit, overburden or inter burden [4]. Transport cost optimization of dump trucks and loading was the priority. Handling of overburden and storage of the same with minimum cost was challenging. Lack of computer models at mine site was the main big disadvantage during early computerisation. Experts were available at the central corporate office. Much of the time was spent on understanding the problem at the mine site and discussing with the computer experts and specialists at the mining organization's computer centre, grasping the problem by them and making them interested with mine experts or mining operation personnel. Several times the process was repeated to understand the solution, developing belief by mines management and permitting to test solution to achieve the results. This created frustration and misunderstandings among team members leading to futile efforts. Further experiments were blocked without knowing real or anticipated difficulties. Also, there was a difficulty in training mining and geotechnical engineers who could act as computer experts. As a general rule, all models for the mining industry were expected to be menu-driven with self-explanatory and with help support. However, there was a change in the scenario as cost economical Personal Computers (PC), and later laptops were evolved. Microsoft came up with its operating system, which became most popular among all users. Thus computing capacity was locally available for geotechnical and mining engineers at the site or the office and making efficient operation. There was also improvement in the internet networks-Wide Area Network (WAN) and Local Area Network (LAN). Various mine planning software was evolved, such as Surpac, Data Mine. Explosive companies developed various software for the prediction of fragmentation, ground vibration etc. Many software programmes were developed for the maintenance of mining equipment. During the 1990s, large mining organizations started implementation SAP ERP software. Oracle is one of the large database software companies having an 80% market share. CISCO company has a majority of routers that are used in mining and other industries.

Figure 1 shows how the demand for various electronic items and gadgets have been increased. The first million radio sets were sold in 65 years. It took eight years to sell the first million TV sets. The first million PC's were sold in 3 years. On the other hand, the first million laptops were sold in 1 year. The first million mobile were sold in less than one year. Presently, the new models (1 million sets) of any mobile premium company are sold in a couple of minutes.

This paper further discusses the present (2000-2025) era of information technology and the future (2025-2050) expected development in Information Technology which will benefit Mining and Geotechnical Engineers.

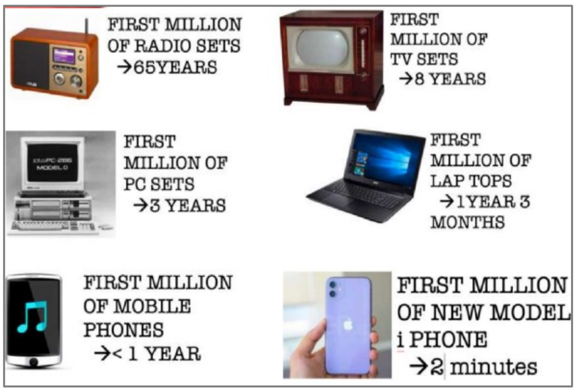

Figure 1: Technological revolution where the demand for various electronic gadgets has increased exponentially.

2. Present Era of Information Technology

Various artificial intelligence/machine learning techniques are applied in the present era for the prediction of performance, optimization or minimization of any mining operation or maintenance issues of mining equipment. Following are some of the algorithms discussed.

2.1 Artificial Neural Network

ANN is a part of Artificial Intelligence, along with Case-Based Reasoning, Expert Systems and Genetic Algorithms. An information processing system is similar to the human brain in structure and functions. ANN is capable of 'learning' to 'recognise' a complex input pattern and predict the output pattern thereof. The network is then able to recognise similarities in new input patterns and can predict the output. This property of a neural network makes it very useful for noisy (inexact) data to be interpolated and outputs predicted in terms of patterns that are already 'known' to it. ANN has three fundamental components [19] Transfer Function, Network Architecture and Learning Law.

2.2 Network Training

The BPNN consists of three layers: Input Layer, Hidden Layer and Output Layer. Daisy Chain – layer using neurons. Neurons under changes in Hidden Layer and Changes to Neurons are determined via transfer function. The transfer function acts as a filter. The transfer function is designed to map the output received from a set of neurons or layer of neurons to the pre-recorded actual output and establish a pattern.

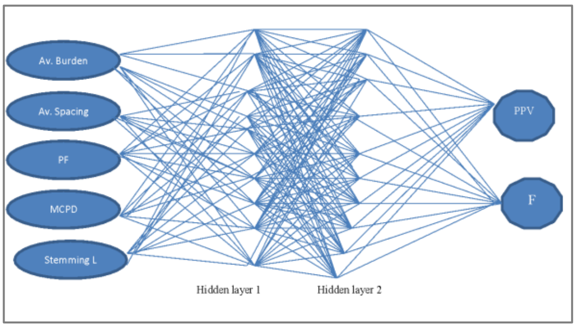

Figure 2: Example of prediction of ground vibration due to blasting using artificial neural network [5].

Three waves of machine learning are shown in Figure 3. During the 1950-1980s, Chinese people had a vision of computers as "Electronic Brains". During 1980-2010, powerful and efficient algorithms were developed for research purposes instead of industry. The present trend is data-driven machine learning with powerful and efficient algorithms.

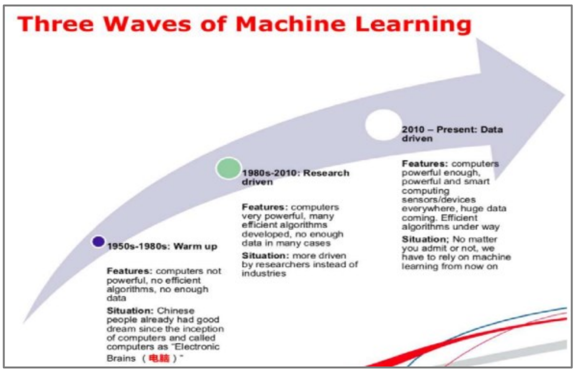

Figure 3: Three waves of machine learning [6,7].

4. Future of Information Technology

Information Technology will be driven by the data of customers. The data from mining operations shall be captured at every stage and stored in cloud storage. For obtaining data, knowledge of digital engineering will be necessary. Resilient decisions can be taken with ample data/ Information technology will be playing a crucial role.

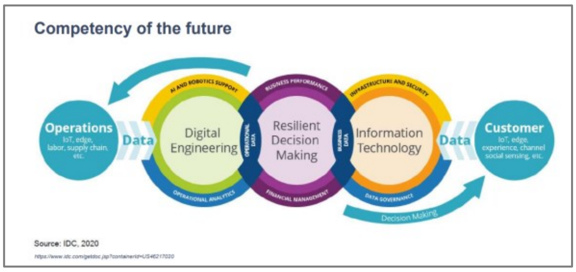

Figure 4: Competency of the future

5. Conclusions

During the last 50 years, information technology is evolving as computer processing power has increased exponentially. On the other hand, the cost of hardware has dropped tremendously. With the application of the Internet of things, improvement in computer proficiency among professionals and ease in availability of hardware, the mining industry is benefited.

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Applicability of Tunnel Muck as An Alternative for Fine Aggregates in Cement Concrete

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Abstract

One of the recent applications of Tunnel boring Machines (TBM) in Sri Lanka is Uma Oya Multipurpose Development Project (UOMDP). During rainy periods, the washed tunnel muck particles sediment in low agricultural lands forming infertile soil. It mainly affects the agricultural sector in the respective area of the country. Therefore, it appears that there is a need to study the reuse of the tunnel muck as an alternative for sand which is a very expensive construction material these days, or any other field as a useful material in an effective manner. The objective of this study is to determine the applicability of tunnel muck as an alternative for sand in concrete as fine aggregate. The material properties of tunnel muck were analysed. Water absorption and workability were higher in the concrete mixed with tunnel muck because the tunnel muck particles are finer than normal sand particles. It was observed that there was a slight decrease in the compressive strength of the concrete casted with tunnel muck. However, the compressive strength could be increased with higher cement content in the concrete. Furthermore, the mix designs with adjusted values were proposed for the concrete mixed with tunnel muck as fine aggregates.

Keywords: Mix designs, River sand, Tunnel boring machine

1. Introduction

It can be identified two main tunnel excavation methods in the tunnel construction industry, as drilling and blasting (D&B) and Tunnel Boring Machine (TBM). Uma Oya Multipurpose Development Project (UOMDP) is one of the recent applications of TBM. It can be observed that a large amount of tunnel muck has been placed in UOMDP, and currently, it has caused severe environmental and agricultural issues [1]. The washed tunnel muck particles are transported with the runoff water during rainy periods and deposits in low land areas causing infertile soils for agricultural activities [1].

As a solution to this, the removed tunnel muck can be used as an alternative in the construction industry or any other industry [2]. Therefore, it appears that the research studies have to be carried out to determine the applicability of tunnel muck as an alternative for construction material in the construction industry.

River sand is normally used as the fine aggregates of cement concrete, and it is one of the major ingredients of cement concrete [3]. However, the demand for the river sand is very high, and the cost for the cement concrete would be reduced if an alternative for the river sand is introduced. That kind of alternative would support the protection of the riverbeds where the river sand is mined.

Here in this study, the applicability of tunnel muck as an alternative for the river sand is determined, and it is analyzed using basic parameters used in cement concrete.

2. Objectives

The objectives of this research are:

- To determine the applicability of Tunnel Muck at Uma Oya Tunnel as a fine aggregate for concrete.
- To determine the properties of concrete made with the tunnel muck removed as a waste of UOMDP.
- To compare the properties of concrete with different water-cement ratio values.

This study consists of a comparison of the test results that have been carried out in laboratories on concrete samples to determine the applicability of tunnel muck as an alternative to river sand and the optimum mix ratio of tunnel muck to achieve the maximum characteristics of concrete.

3. Methodology

Tunnel muck samples are collected from the UOMDP site, and firstly they are tested for sieve analysis. The results are checked with the standard limit range for the particle size distribution curve as per BS 882:1973. Furthermore, the concrete samples made with river sand and the samples made with tunnel muck were tested for the basic tests for concrete, and the results are analyzed.

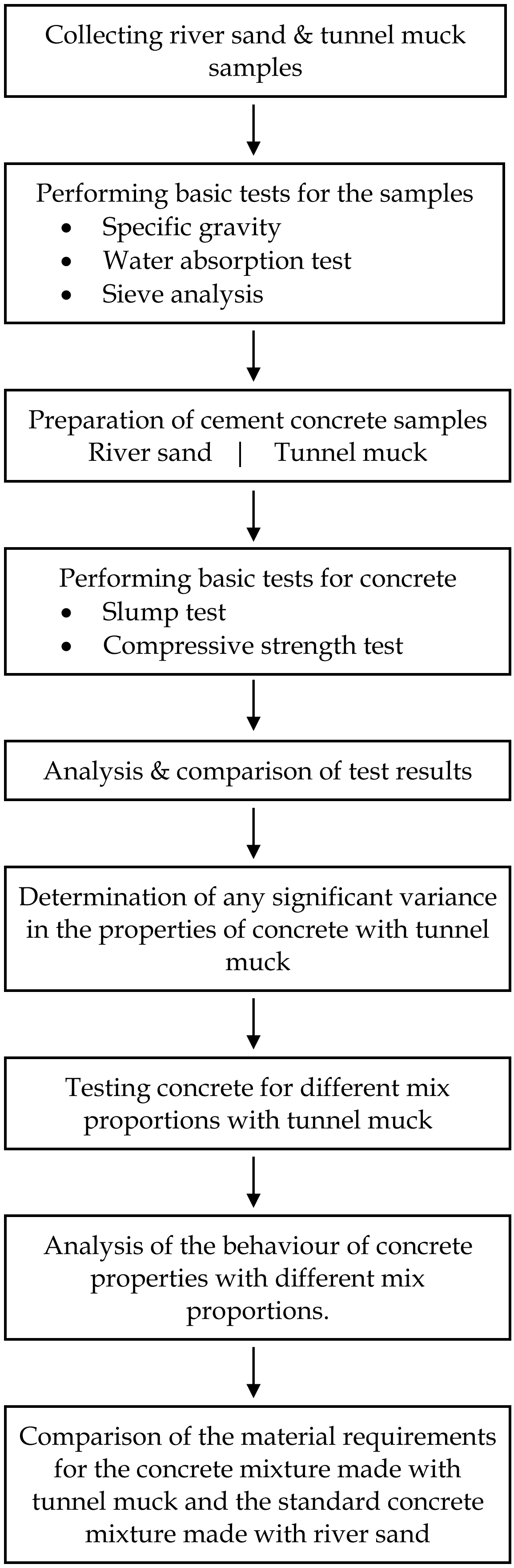

Figure 1: Methodology.

The results are compared with the values available in the literature to determine whether the addition of tunnel muck makes a considerable deviation. The behavior of the concrete made with tunnel muck was observed by changing the mix proportions.

4. Results and Analysis

4.1 Sieve analysis test

Sieve analysis tests were carried out for both tunnel muck and river sand samples to check the applicability in accordance with BS 882:1993 (Grading of fine aggregates). A representative sample of 0.6 kg from tunnel muck and 1 kg of river sand were used for the analysis. Figure 2 shows the results of the sieve analysis.

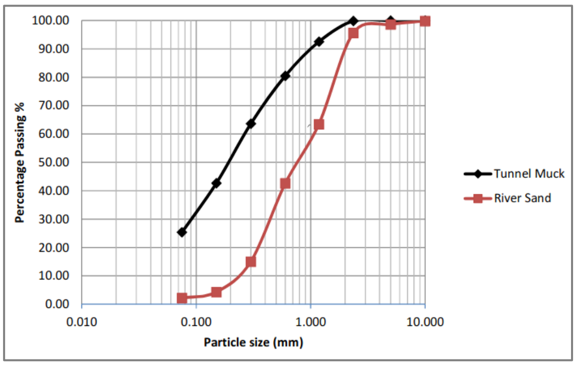

Figure 2: PSD curve for tunnel muck and river sand.

It was observed that the particle size distribution (PSD) of selected river sand samples was within the acceptable limit range (in zone 2) mentioned in BS 882:1973. However, the PSD of the selected tunnel muck sample was not within the acceptable range.

4.2 Specific gravity and Water absorption test

Specific gravity tests and water absorption tests were carried out for both tunnel muck and river sand samples. The results are as follows,

Tunnel muck:

Specific gravity = 2.64

Water absorption = 2.49%

River sand:

Specific gravity = 2.65

Water absorption = 0.60%

It appears that the specific gravities of both materials were almost the same. However, the water absorption of tunnel muck is considerably higher than river sand.

Further, the specific gravity and water absorption values of coarse aggregates (nominal size = 20 mm) were determined for the use in mix designs.

4.3 Slump test

Mix designs were carried out for each grade of concrete with river sand and tunnel muck. The results of slump tests for each grade of concrete are tabulated in Table 1.

Table 1: Slump test results.

Grade	With River sand (mm)	With Tunnel muck (mm)
M10	63	81
M15	61	78
M20	54	90

It was observed that the slump values of concrete made with tunnel muck are relatively higher compared to the river sand for the considered grades of concrete. Further, it was noticed that the concrete made with river sand has a low to medium workability and the concrete made with tunnel muck has a medium to high workability (BS 882:1973).

4.4 Compressive strength test

Compressive strength was tested for the concrete made with river sand and tunnel muck for 7 days and 28 days. The average values are compared in Figures 3 and 4.

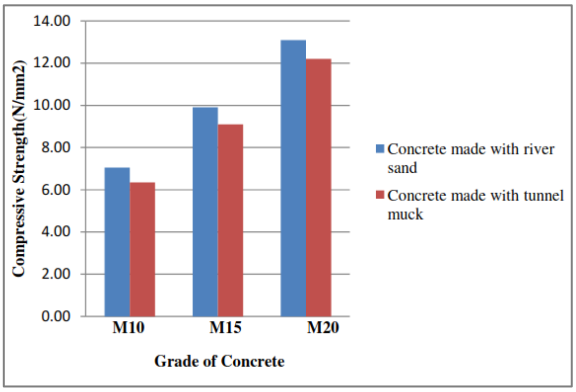

Figure 3: Results of compressive strength test (7 days).

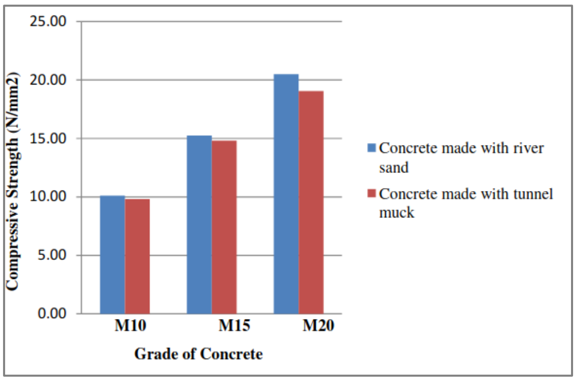

Figure 4: Results of compressive strength test (28 days).

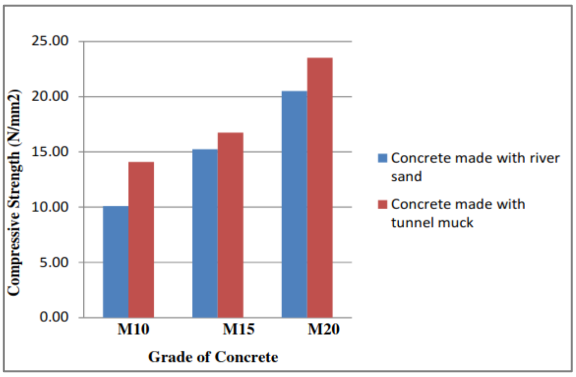

Figure 5: Results of compressive strength test (28 days) for adjusted mix proportions.

4.5 Results for adjusted mix design

It was observed that the compressive strengths of all considered grades of concrete are lower in the concrete made with tunnel muck. Therefore, the mix designs were adjusted by decreasing the water-cement (w/c) ratio. The adjusted w/c ratio values are tabulated in Table 2 with the initial w/c ratio values.

Table 2: Adjusted w/c ratio values.

Grade	Initial w/c ratio	Adjusted w/c ratio
M10	0.85	0.5
M15	0.80	0.5
M20	0.75	0.5

The compressive strength was tested for the concrete made with the adjusted mix proportions by decreasing the w/c ratio. The results are compared with the compressive strength values of concrete made with river sand and depicted in Figures 4 and 5.

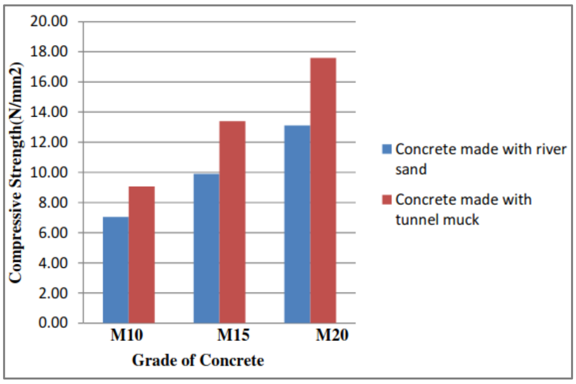

Figure 4: Results of compressive strength test (7 days) for adjusted mix proportions.

It was noticed that the compressive strength was increased by 45% on average in 7 days and 25% on average in 28 days of concrete made with tunnel muck using adjusted w/c values.

5. Discussion

This research was carried out as an experimental study to investigate the applicability of tunnel muck as an alternative for fine aggregate in concrete.

As per the results of the sieve analysis test, it was noticed that tunnel muck is finer and poorly graded relative to the river sand. Further, the particle size distribution of tunnel muck is not compatible with any zone mentioned in BS 882:1973. However, the availability of finer particles may cause higher workability of the tunnel-muck concrete.

In this study, a higher w/c ratio was initially selected to study the behavior of the strength of concrete at a higher w/c ratio with tunnel muck. It was observed lower compressive strength for higher w/c ratio and higher compressive strength in low w/c ratio. Therefore, the targeted compressive strength can be obtained by using tunnel muck by optimizing the w/c ratio further.

The sample collected site was infected by an environmental issue that the low land areas are becoming infertile due to the transportation of tunnel muck. Therefore, the use of tunnel muck as an alternative to river sand would be a good solution. However, the chemical composition of the concrete with tunnel muck and the effects will have to be investigated.

6. Conclusions

Based on the experimental results, the following conclusions could be drawn as the outcome of this study.

- The workability is higher in the concrete made with tunnel muck.
- The target strength cannot be achieved by the tunnel-muck concrete with a higher w/c ratio. However, the target strength can be achieved by a low w/c ratio (i.e., high cement content).

The Following can be recommended as further developments for this topic.

- Determination of an optimum mixture of tunnel muck and river sand for optimum properties of concrete.
- Determination of the optimum proportion of tunnel muck for reinforced concrete and precast concrete.
- Study about the chemical composition of the tunnel muck and the behavior of the concrete based on chemical composition.
- Determination of strength parameters of concrete made with tunnel muck by adding admixtures and for grades higher than M20.

Acknowledgement

The authors are thankful to the staff of UOMDP for their cooperation during sample collection & to the staff of the Engineering materials division, Irrigation Department, Sri Lanka, for providing laboratory facilities.

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Correlations between Durability, Mineralogy and Strength Properties of Limestone

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Abstract

Durability is used to depict the obstruction of rock to weathering and the pace of events of such changes. Wetting and drying cycles cause evaluation of physical changes and slaking behaviour of rocks, and that can be appraised by the slake durability test. Correlations between durability, mineralogy, and other physical-mechanical properties are different with the rock. Physical and mechanical characteristics of High-grade limestone samples and Low-grade limestone samples from the Aruwakkalu mine were determined according to ASTM standard methods. X-ray diffractometer (XRD) analysis and scanning electron microscope (SEM) analysis were used to determine the mineralogical contents of the studied samples. Regression analyses were performed between the slake durability index and the physical-mechanical properties of limestones to identify the best wetting and drying cycle to assess the relationships. Differences between the physical, mechanical properties, and mineralogical contents of High-grade limestone and Low-grade limestone were also identified from the laboratory test results. Finally, this study will help to understand any rock engineering problem relating to durability, mineralogy, and other physical-mechanical properties of areas with limestone.

Keywords: Limestone, Regression analysis, SEM analysis, Slake durability, XRD analysis

1. Introduction

Physical properties, strength, and durability are essential properties that help to assess the rocks for various applications. Determining the suitability of rocks for use under various environmental and stress conditions is mostly rely on these physical properties. The durability and strength of rocks generally help to categorize different types of rocks. The ability of a material to resist deformation induced by an external force is known as Strength [1]. Durability is an indicator of the ability of rocks to withstand and sustain their distinctive strength and resistance characteristics.

Rocks are subjected to alternate dry and wet conditions and undergo repeated water absorption and dehydration steps. This cycle is called the Wetting and Drying cycle of rocks [2]. Many physical and mechanical properties of the rocks are changing with these cycles. The load of past durability characterizations depends on the subsequent cycles' durability index [1]. The most immediate control on durability is provided by the mineralogical composition of the network in rocks. The existence of expansive clay minerals has a strong connection with the effect of wetting and drying cycles on durability [3].

In this study, the relationship between the slake durability index of high-grade and low-grade limestone and the physical-mechanical properties of limestone has been investigated. Mineralogical contents of the limestone samples have also been investigated by X-ray diffraction (XRD) analysis and scanning electron microscope (SEM) analysis.

2. Methodology

2.1 Sample Preparation

High-grade limestone and Low-grade representative samples were extracted as boulders (Approximately 30kg each) from the Aruwakkalu limestone mine (Figure 1). Polythene covers were used to wrap the samples to avoid the loss or gain of moisture during transport and storage.

For the Point load test, core samples were extracted from the boulders using a rock coring machine. For the Los Angeles Abrasion test and Aggregate Impact Value test, limestone boulders were first crushed using a hammer and then from the jaw crusher. Then the limestone samples were sieved into the required sizes for the tests. For SEM analysis and XRD, limestone samples were first crushed using a jaw crusher and then crushed using the laboratory Tema mill to reduce average grain size. Powdered samples were sieved using a 63 µm sieve.

Ten representative limestone specimens were prepared, each weighing 40 g to 60 g by breaking limestone fragments using a hammer for slake durability tests. Sharp corners were broken off and dust was removed from those specimens before performing the Slake durability test. All samples were oven-dried for more than 24 hours at a temperature of 105° C to avoid the effect of pore water on the results.

Figure 1: Sample locations.

2.2 Testing

All the tests were performed according to the ASTM standard testing procedures. Strength properties of high-grade and low-grade limestone were obtained by performing a point load test. The aggregate degradation properties of both types of limestones were obtained by the Los Angeles Abrasion Value test (LAAV) and Aggregate Impact Value test (AIV). Prepared limestone powder samples were analyzed using SEM and XRD to identify the mineralogical content difference between high-grade limestone and low-grade limestone. Durability indexes of all limestone samples were obtained using a slake durability test using tap water as the slaking fluid. Slake durability test was performed up to 2 cycles for each limestone sample.

2.3 Regression Analysis

In this research, slake durability test was done for up to 2 cycles. Linear regression analyses were undertaken for the two cycles of the Slake durability test and the relationship between the Slake durability index and Uniaxial Compressive Strength (UCS). By analyzing in MATLAB via linear regression analysis with a 95% confidence level, the best correlations between parameters were obtained.

3. Results

3.1 Results of strength and degradation tests

Point Load test, LAAV test, AIV test, and Slake durability test were performed on high-grade limestone and low-grade limestone to obtain material properties. The average values of the results are shown in Table 1.

Table 1: The results of laboratory tests.

Laboratory Tests	High-grade limestone	Low-grade limestone
Point load index (kPa)	746.6	328.6
Calculated UCS (MPa)	16.6	7.3
Aggregate Impact Value	23.4	41.2
Los Angeles Abrasion Value	59.7	75.8
Slake Durability Index 1 [Id(1)]	95.5	93.4
Slake Durability Index 2 [Id(2)]	93.9	90.5

3.3 XRD Analysis

Mineralogical contents of both High-grade limestone and Low-grade limestone were determined by the results of XRD. The difference between the mineralogical content of two limestones was identified with these results, as shown in Table 2.

From these results, a slight difference between mineralogical contents of high-grade limestone and low-grade limestone was identified. The percentage of Ettringite was much higher in low-grade limestone.

Table 2: The results of limestone analysis by XRD

Compound	High-grade Limestone %	Low-grade Limestone %
Calcite	95	94.2
Portlandite	1.3	1.2
Ettringite	0.7	3.2
Mullite	1.3	1.4
Quartz	0.9	0.1

3.4 SEM-EDX Analysis

Scanning electron microscopy-energy dispersive X-ray analysis (SEM-EDX) provides a quick nondestructive determination of the elemental composition of the samples. Both high-grade limestones and low-grade limestones were analyzed by SEM-EDX analysis. According to the weight percentage differences of high-grade limestone and low-grade limestone of SEM analysis, C K, O K, AlK and CaK were higher weight percentages in high-grade limestone than low-grade limestone. According to the atomic percentage, C K and O K have higher percentages in high-grade limestones, and SiK and CaK have a higher percentage in low-grade limestones. FeK was only identified in the high-grade limestones as per the SEM-EDX analysis.

4. Discussion

4.1 Physical-mechanical properties and mineralogical content

According to the results of strength and degradation tests, high-grade limestones are tougher, more impact resistant and have more strength than low-grade limestone. But both high grade and low grade have LAAV and AIV values which are higher than the specified maximum values for road construction aggregates. As per the results of the Slake durability test, the % reduction of durability with drying and wetting cycle is higher in low-grade limestone than high-grade limestone, which implies low-grade limestone subjected to weathering effect or durability degradation. According to the Slake durability Index classification [4], Low-grade limestones have high durability (76-90), and High-grade limestones have very high durability (91-95).

When it comes to XRD analysis, there was a slight difference between mineralogical contents of high-grade limestone and low-grade limestone. Basically, the Calcite and Quartz percentage of high-grade limestone is quite higher than low-grade limestone. The percentage of Ettringite was much higher in low-grade limestone. According to the SEM-EDX analysis, C K, O K, AlK and CaK have higher weight percentages in high-grade limestone than low-grade limestone. Although FeK was only identified in the high-grade limestones as stated by the SEM analysis.

4.2 Regression Analysis

Regression analyses were performed between the lake durability index and physical, mechanical properties. The correlation coefficient (r^2) was obtained using MATLAB via regression analysis with a 95% confidence level.

For the two cycles of the Slake durability test and the relationship between the Slake durability index and UCS, linear regression analyses were performed.

The value of the Correlation Coefficient (r^2) of the second cycle of durability for high-grade limestone (Figure 3) is much closer to 1 than the first cycle of durability (Figure 2).

The value of the Correlation Coefficient (r^2) of the second cycle of durability for low-grade limestone (Figure 5) is much closer to 1 than the first cycle of durability (Figure 4).

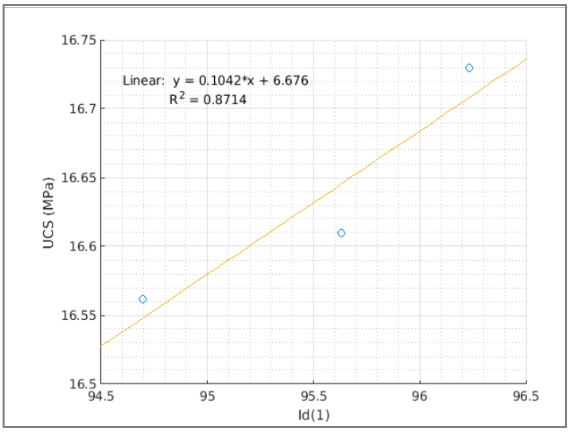

Figure 2: Durability index 1 (Id(1)) and Strength regression analysis of high-grade limestone.

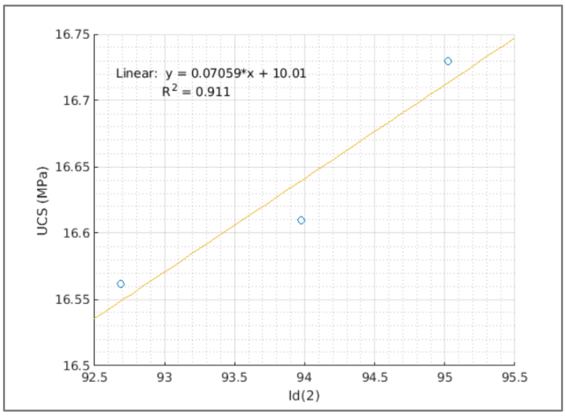

Figure 3: Durability index 2 (Id(2)) and Strength regression analysis of high-grade limestone.

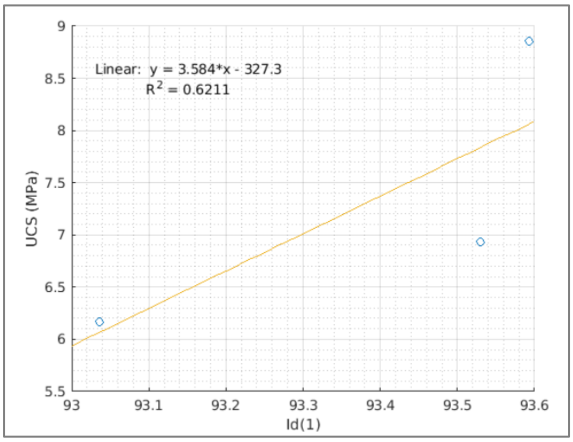

Figure 4: Durability index 1 (Id(1)) and Strength regression analysis of low-grade limestone.

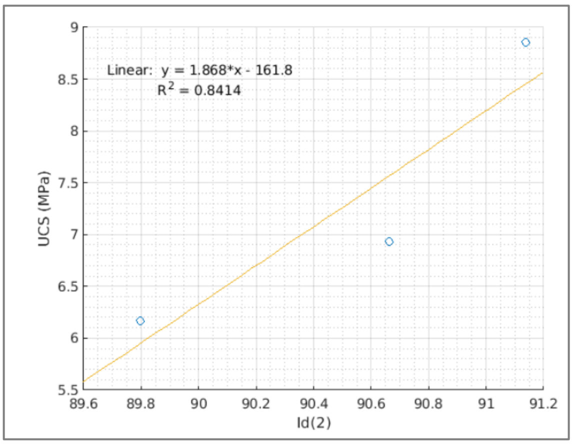

Figure 5: Durability index 2 (Id(2)) and Strength regression analysis of low-grade limestone.

According to the Correlation Coefficient (r^2), UCS values for the second cycle of durability indicated a stronger relationship between the variables than the first cycle.

5. Conclusions

Durability and weathering characteristics of rocks are the key factors in determining the suitability and usefulness of them as different engineering materials, and they also control the stability of surficial and underground excavations.

Low-grade limestone is subjected to weathering effect or durability degradation than high-grade limestone.

Based on LAAV and AIV, this limestone is not suitable for the construction of roads as a base course material. Also, it is insufficient for use as highway surface material or railroad ballast.

Acknowledgement

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Critical Evaluation of Industrial Mineral Mining Methods in Sri Lanka

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Abstract
Sri Lanka is rich in industrial minerals, including graphite, apatite, ilmenite, rutile, quartz, feldspar, kaolin, apatite, garnet sand, mica, calcite, and dolomite. Even though lots of mining methods are used to extract such minerals, only limited studies have been carried out in Sri Lanka to evaluate mining methods by means of main mining principles, such as safety, economy, and efficiency. This study focuses on the Bogala graphite mine, a narrow vein medium depth underground mine, and Aruwakkalu limestone open pit mine. To evaluate the overhand cut and fill mining method in Bogala mine, Equivalent Linear Overbreak or Slough (ELOS), which is a useful measurement for quantifying the unplanned dilution in narrow vein mining, is used. The favourability of the underground mining method is evaluated for a particular mine site using Key Deposit Indicators (KDIs), and it provides the best suited underground mine based on characteristics of the ore body. The fracture system presence in the Bogala mine requires a good support system to ensure the safety of underground workers. The efficiency of the loading and hauling equipment in the Aruwakkalu mine site is evaluated by Match Factor (MF).

Keywords: ELOS, KDI, Limestone, Match factor, Narrow vein, RMR

1. Introduction

Industrial minerals include all those materials that man takes out of the earth's crust except for fuels, metallic ores, water, and gemstones [1]. Consideration of basic mining principles (safety, efficiency, and economy) is essential in the evaluation of mining methods.

1.1 Sri Lankan Mining Industry

About 2.3% of the Gross Domestic Product (GDP) by industrial origin is due to the mining and quarrying industry as per the Central Bank Annual report 2019 [2]. Mainly graphite, ilmenite, rutile, zircon, kaolin, ball clay, feldspar, vein quartz, silica, apatite, calcite, dolomite, mica and are mined as industrial minerals using modernized or traditional mining methods in Sri Lanka.

1.2 Mining Method in Bogala Graphite Mine

In Bogala Mine, the overhand cut and fill mining method is practised with the deepest operation level of 503 m. Prior to the mining, 1.1 m * 60 m * 60 m (W*L*H) block is defined, including two main levels and two winzes. Then 1.1 m * 2.0 m (W*H) area is excavated along the vein.

1.3 Mining Method in Aruwakkalu Limestone Mine

Aruwakkalu Limestone Mine extends its land coverage up to an area of 4454 acres, and more than 450 acres has been mined out already, which is about 10-15% of the demarcated land area for the mining. The open-pit mining method is practised at this mine. The stripping ratio of the mine is 1.6. Several nearly vertical benches are developed on the red soil layer, with approximately 2.1 m bench height and 4 m bench width. The specific charge value stipulated by the Geological Survey and Mines Bureau (GSMB) is 0.15 kg/mt for Aruwakkalu Mine [3].

1.4 Evaluation of Bogala Mine

The average vein thickness of graphite in Bogala Mine is in the range of 20 to 40 cm. It is called a narrow vein graphite mineralization. However, currently, Sri Lanka is the only country mining vein-type graphite for the commercial purpose [4]. During narrow vein mining, dilution can extremely be increased. ELOS index measures the unplanned dilution in underground mining. The main advantage of the ELOS index is that it doesn't depend on mining width. ELOS is a useful measurement for quantifying the dilution in narrow vein mining because narrow vein dilution is highly sensitive to the stoping width.

Bieniawski (1973) published an RMR classification system to classify the rock mass and to decide the suitable support system for underground excavations. Due to the fracture system present in the Bogala host rock, a heavy support system is used. This consists of steel as well as timber. Therefore, RMR value for host rock and consumption of support material per block of Bogala Mine was estimated in this study. Generally, there are three kinds of supporting systems practised in Bogala Mine as cap support, set support and rock bolting.

Open stope mining method was replaced by overhand cut and mining method in Bogala Mine after 165 m level [5]. To adopt most favourable mining method based on spatial characteristics of ore body, Key Deposit Indicators (KDIs) are used. Ore strength, host rock strength, deposit shape, deposit dip, deposit size, ore grade, ore uniformity and deposit depth are the main parameters in the determination of KDIs.

1.5 Evaluation of Aruwakkalu Mine

The largest material handling process in the Sri Lankan mining industry is practised at Aruwakkalu Limestone Mine. A Caterpillar 374F L type excavator and three HD 465-7R type dump trucks are used in the Aruwakkalu quarry for material handling with 6 to 7 tons and 55 tons of capacity, respectively.

Evaluation of cycle time of loading and hauling process is measured as per the optimum requirement of machinery to increase the process efficiency with minimum cost. Match Factor (MF) is measured suitable fleet composition number between loading and hauling equipment. MF=1 indicates 100% efficiency of operation between loading and hauling equipment.

2. Methodology

2.1 Data Collection

Mining engineers were interviewed during the field visit to obtain necessary data related to mining operations on sites. In Bogala Graphite Mine, a survey was conducted using Total Station (Sokkia iM-52) to determine the unplanned dilution at 275 FM level in Bogala Mine. A field survey was conducted using a stopwatch to calculate the cycle time of dumpers and the excavator at Aruwakkalu Limestone Mine.

2.2 Analysis of Data Evaluation of Bogala support system

A detailed fracture survey has been conducted on the main crosscut at 503 m level in "Kumbuk Vein" to identify the weathering condition, continuity and spacing of discontinuities, roughness of the surface, mineral intrusion, and groundwater condition within fractures. Occasionally, Quartz, Pyrite and Mica can be observed within fractures as mineral intrusions. According to the survey observations, fractures are continuous, and most of them have rough surfaces. On average, 1,000,000 m^3 of water is pumped out from Bogala Mine every day.

According to the observation rate for relevant parameters in the RMR classification system are included in Table 1. Consumption of support material used in Bogala Mine for cap supports and rock bolts and cost of them are included in Tables 2, 3 and 4, respectively.

Table 1: Rate for parameters in RMR for Bogala Mine.

Parameter	Observation	Rate
UCS	137.9 –206.8 MPa	12
RQD	99%	20
Spacing of discontinuities	2.4 m	20
Condition of discontinuities	Slightly rough surface, Separation < 1 mm, Highly weathered wall rock	20
Groundwater	Infiltering moderate pressure	4
Orientation of discontinuities	Favourable	-2

Material consumption for a 60 m * 60 m block

Space between two successive anchors (Y) = 4 ft

Overlap length of two successive timber planks (X) = 2 ft

Overall Length of one support cycle (X+Y) = 6 ft

No. of support cycle using timber caps per block = 1450

No. of support cycle using SH rails per block = 50

Table 2: Consumption of material for cap support per 60 m * 60 m block.

(D-Diameter, L-Length, T-Thickness, W-Width)

Support Type	Dim. of Unit Item	No of Items per cycle	Total Qua. Per cycle	Total Quantity per block
Timber cap	D=25 mm L=1.1 m	1	1.1 m	1595 m
Timber plank	T=2" W=6" L=8'	7	56'	84000'
SH rail	L=1.1 m	1	1.1 m	55 m
Anchor	D=30 mm L=4′	2	8′	12000′
Anchor support	D=30 mm L=1′	2	2′	3 000′

Table 3: Consumption of material for rock bolts per 60 m * 60 m block.

Support Type	Dim. of Unit Item	No of Items per block	Total Quantity per block
Stem of Rock bolt (3 ft)	D=25 mm L=3′	30	90 ft
Stem of Rock bolt (5 ft)	D=25 mm L=5'	10	50 ft
Nut	D=25 mm	40	40
Wedge	L=2″ W=1″ T=3 mm	40	40 * (2″X1)=80 $inch^2$
Plate washer	L=5″ W=5″ T=3 mm	40	40 * (5″X5″) =1000 $inch^2$

Labour cost estimation for a 60 m * 60 m block

Duration of a working shift	5 hrs
Total hours required to complete a support cycle	2 shifts
Total number of hours required to fix a support	10 hrs
Number of workers required to fix the support system	3
Average monthly salary per labour	Rs. 80,000
Average labour salary per hour (80 000/200)	Rs. 400
Total labour cost per block (400*10*3*1500)	**Rs. 18,000,000.00**

Cost of material for support system per 60 m * 60 m block

Table 4: Cost evaluation of material for cap supporting and rock bolting per block.

Type of support	Req. quantity	Unit Cost (Rs)	Total Cost (Rs)
Logs	1595 m	31.63/ dc^3	2,475,196.00
Timber plank	84,000 ft	350/ft	29,400,000.00
SH rails	183.15 ft	1 045/ft	191,392.00
Anchor	12,000 ft	280/ft	3,360,000.00
Anchor support	3,000 ft	280/ft	840,000.00
Stem of rock bolts (3 ft)	90 ft	208/ft	18,720.00
Stem of rock bolt (5 ft)	50 ft	208/ft	10,400.00
Nut of rock bolt	40	80	3,200.00
Wedge of rock bolt	80 $inch^2$	2.50/ $inch^2$	200.00
Plate washer of rock bolt	1000 $inch^2$	2.50/ $inch^2$	2,500.00
Total (Rs)		**36,301,608.00**	

Evaluation of unplanned dilution in Bogala Mine based on ELOS index

Based on data collected from the survey, an isometric 3D view was prepared using AutoCAD software to determine excavated rock volume.

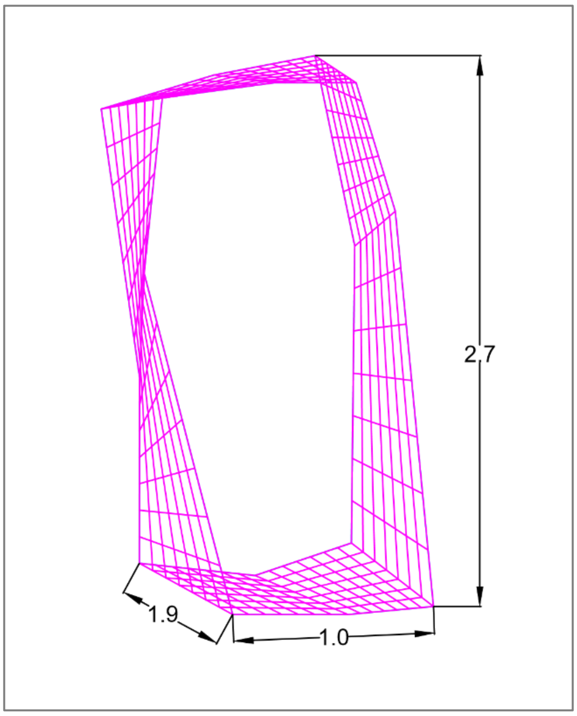

Figure 1: Isometric view at location no:1 at 275 FM level (not to scale).

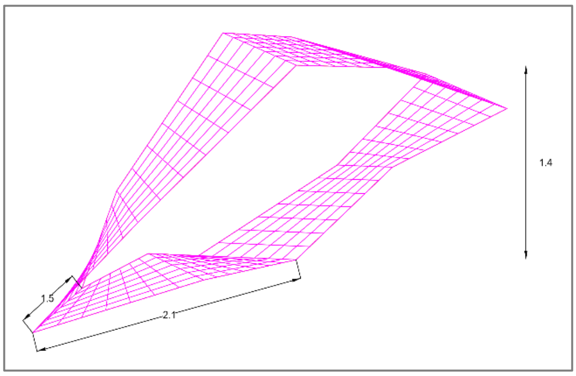

Figure 2: Isometric view at location no: 2 at 275 FM level (not to scale).

Table 5: Calculation of ELOS index for 275 FM level of Bogala Mine.

Location	Total In-situ Blasted Volume[m^3]	Expected Volume[m^3] (H*W*D)	Overbreak Volume [m^3]	Expected Stop Height[m]	Expected Strike Length[m]	ELOS Index [m]
1	3.845	2*1.1*1.2=2.64	1.205	2.0	1.2	0.5021
2	2.935	2*1.1*1.2=2.64	0.295	2.0	1.2	0.1229

The solid rock volume expected to be excavated in the blasting was estimated based on dimensions, and ultimately differences between the actual and expected rock volume (overbreak) were calculated. Then ELOS was determined using equation (1).

ELOS=Volume of overbreak or slough/ stope height*wall strike length (1)

Evaluation of performance in material handling at Aruwakkalu Limestone Mine

Cycle time of dumper is the summation of loading time, hauling time, dumping time, returning time, spotting time and waiting time. Spotting time is defined as the positioning time of the hauler at the loading or dumping point. Excavator cycle is given by summing loading time, drawing time, swing time with load, swing time without load and waiting time.

Table 6: Performance of dumpers.

Machine	Avg. cycle time (min:sec)	Avg. load per cycle(t)	Production Capacity (t/h)
DT 551	14:34	51.55	212.33
DT 552	15:43	53.54	204.43
Dt 553	16:50	53.89	192.08
Average	**15:42**	**52.99**	**202.48**

Avg. cycle time of excavator = 49.24 sec

MF is measured as per equation (2).

$$MF = \frac{NF*CTL*NH}{CTH*NL} \quad (2)$$

Where:
MF = Match Factor
NF = no. of filling for each hauling equipment
CTL = cycle time of loader (sec)
NH = no. hauling equipment
CTH = cycle time of hauler (sec)
NL = no. of loading equipment

MF for Aruwakkalu Limestone Mine = 1.0977

The hauling process at the Aruwakkalu quarry takes 9.5 hours per day in the week. However, material hauling activities are completely stopped in November and December month due to heavy rainfall. Therefore, the contribution of the material handling process to achieve the annual production is targeted as follows:

Total no of working days per year	300 days
No of working hrs/day	9.5 hrs
No of dumpers	3
Avg Cycle Time of dumper	942 sec
Avg load per cycle	52.99 t
No of cycles per day by a single dump truck (9.5*3600/942)	36
Achievable production target per day (36*3*52.99)	5722.92 t
Achievable production target per year (5722.92*300)	1,716,876 t

3. Results

RMR value is calculated by summing all RMR parameters as follows:

RMR for Bogala Mine = 12+20+20+20+4-2
= 74

The total cost for support material (cap support and rock bolting) and labour per

60 m * 60 m block are LKR 36,301,608.00 and LKR 18,000,000.00, respectively.

The average ELOS for Bogala Mine is 0.31 m.

Based on field survey data and available literature, KDI is determined as follow:

Table 7: Resulted KDIs for Bogala Mine.

Key Deposit Indicator	KDI Value for Bogala Mine
Host Rock Strength	3
Deposit shape	4
Deposit dip	4
Deposit Size	1
Ore Grade	4
Ore Uniformity	2
Deposit Depth	3

At Aruwakkalu Limestone Mine, limestone loading and hauling equipment are resulted in Match Factor as 1.098. Based on existing efficiency and recorded cycle times, Aruwakkalu quarry can achieve 1.72 Mn of tons production target per year in term of loading and hauling process.

4. Discussion

Total ratings for rock mass classes published by Bieniawski (1973) was used to evaluate overall condition of host rock in Bogala Mine as follow:

Table 8: Rock classes based on RMR.

Ratings	100–91	90–71	70–51	50–25	<25
Class No.	I	II	III	IV	V
Description	Very good	Good	Fair	Poor	Very poor

According to the result obtained from RMR calculation, rock mass at Bogala Graphite Mine is classified as good rock (class no: 2).

ELOS index less than 0.5 m is considered blast damage only, ELOS index between 0.5 and 1.0 m is minor sloughing, 1.0 to 2.0 m is moderate sloughing and greater than 2.0 m is severe sloughing or possible wall collapse [6]. Average ELOS index for Bogala Mine is 0.31 m. It belongs to blast damage only range which means very low level of unplanned dilution is occurred during the blasting in Bogala Graphite Mine.

According to Table 9, most of the indicators for Bogala Graphite Mine are similar to the favourable KDI of the cut and fill mining method.

Table 9: Favourable KDI for overhand cut and fill mining method.

Key Deposit Indicator	Favourable Condition	KDI Value
Ore strength	Moderate to strong	2,3
Host rock strength	Weak to fairly weak	1, 2
Deposit shape	Any shape	1, 2, 3, 4
Deposit dip	5^0 to 45^0	2, 3
Deposit size/thickness	Small to medium	1, 2
Ore grade	Fairly high	3, 4
Uniformity	Variable or moderate	1, 2
Deposit depth	Moderate to deep	2, 3, 4

Slight differences of Bogala Mine with respect to the favourable condition of cut and fill mining method are host rock strength and deposit dip. The actual strength of the host rock is higher than the expected value. Table 9 indicates that cut and fill mining method is best for weak to fairly weak host rocks. According to the values of Bogala Mine, its host rock strength is high. However, fractures existing within the rock mass decreases the overall strength of the host rock. Although KDI of deposit dip is considered as 4, favourable condition for cut and fill mining method is 2 or 3. It means the dip of the deposit should be between 5° and 45°. However, all graphite veins in Bogala Mine are approximately within 65° to 90° range.

If MF = 1, it indicates 100% efficiency operation between loading and hauling equipment. While, If MF < 1, this indicates hauling equipment works at 100% effective and loading equipment allocates more time for waiting. If MF > 1, this indicates that loading equipment works effectively but waiting time is generated by hauling equipment [7]. MF for Aruwakkalu Mine was 1.098, it is nearly equal to one. So, loading and hauling equipment at Aruwakkalu Limestone Mine works nearly 100% efficiently and at optimum performance under the current conditions.

Further, the annual production target of the Aruwakkalu Limestone Mine is approximately 1.3 million metric tons [8]. According to the current material handling condition, it can achieve a 1.72 million metric tons production target per annum.

5. Conclusions

In Bogala graphite mine, significantly higher cost is incurred for supporting to meet the safety requirement of underground workers. However, economically it is more prefer replacing of existing support materials with a material that ensures the safety requirements with a minimum cost.

According to resulted ELOS index for Bogala Mine, unplanned dilution is maintained at a lower level so that there is no any significant overbreak in the blasting. According to the KDIs overhand Cut & Fill Mining Method is the most favourable mining method for Bogala Mine. According to MF, a highly efficient material handling process exists at Aruwakkalu Mine. Under the current efficiency in the material handling process, Aruwakkalu Mine can achieve a higher production capacity than the expected value.

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Session II

Mining Industry and Mine Safety

Mine Safety Issues in Quarry Industry: Case Studies of Recent Fatal Accidents in Sri Lanka

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Abstract

Sri Lankan metal quarries supply almost all grades of aggregates throughout the country for the construction industry. But the future of these metal quarries may look bleak owing to the rise in the number of mine accidents being reported. The Geological Survey and Mines Bureau (GSMB) is the main regulatory body that regulates the mineral industry, issues licenses to almost all quarries in Sri Lanka. When considering the industrial mining licenses, there are four types of mining licenses, i.e., type-A, type-B, type-C, and type-D, issued by the GSMB. Of the four types of categories, the multi borehole blasting method can be adopted for type-A and type-B category mining licenses, while the single borehole blasting method is adopted for the type C and type D categories. Of the two types of initiation methods majority of cases with regard to accidents being reported from the single borehole blasting method. But they were not properly documented earlier. However, with the dawn of the Mining Safety Unit, these cases were documented. Of the cases being documented, five cases, i.e., fatalities, occurred due to fly rock, falling to a lower level, collapsing rock overhang on to the people who work underneath it, burning of explosives, and unintentional blasting due to lightning were described in this paper for risk assessment. When these cases are analysed, it is observed that among the contributory factors, excessive face height and human negligence are the most predominant factors for these incidents. At the end, it is mentioned that how this method is unsafe and proposed a multi-hole blasting method under the guidance of a suitable person.

Keywords: Aggregate quarries, Mine accidents, Mine safety

1. Introduction

As a government regulatory body for the mineral industry, the GSMB issues mining licenses for conducting mining activities in quarries in Sri Lanka. For conducting mining activities, the GSMB issues two types of mining licenses, namely Industrial and artisanal mining licenses for all categories of minerals. Of the two types of licenses, the number of artisanal licenses gradually decreases and currently, they are limited to a few due to difficulty in executing blasting activities. In this paper, risk assessment is done for quarries owing to lots of fatalities being reported in quarries. Though lots of mine accidents have occurred in Sri Lankan Metal quarries, they were not properly documented. However, with the inception of the mine safety unit in the GSMB, these incidents are documented in such a way that it could be helpful for authorities to take appropriate actions to improve the mine safety of these quarries. In this connection, this paper is presented to highlight the number of fatal accidents that occurred in Sri Lankan quarries and what factors contributed to these incidents occurring and who is at risk and what remedial actions should be taken to prevent these incidents. Of all documented cases, five incidents are taken for assessment.

1.1 Case #1

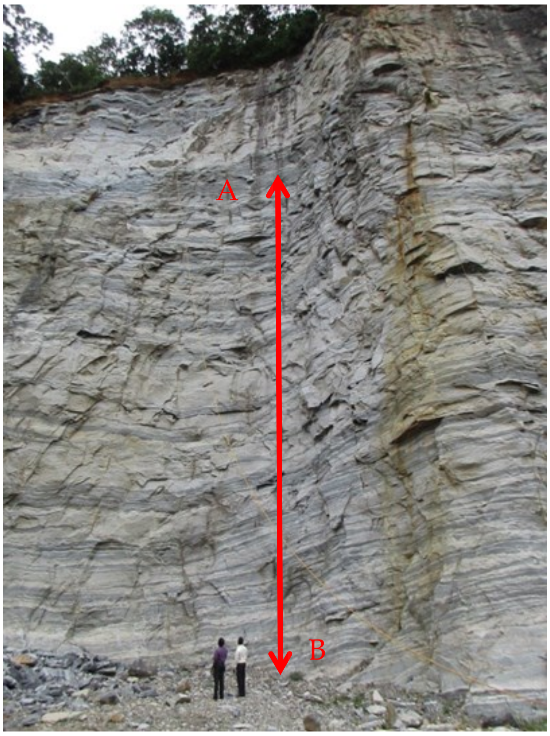

Figure 1: Place of the Incident

A- The place where the Deceased people were engaging in drilling
B- The place where the Deceased people have fallen
AB-70m-80m

Two fatalities were reported in a quarry on 09-07-2021, where the license-IML/C/COL/N/025/R/01 was issued. One of the victims was 42 years old resident of Kaduwela, while the other one was 51 years old resident of Yatiyanthota. Two victims were casual labourers of the quarry and had been working in the quarry for more than ten years. The licensee has been operating the quarry for more than twelve years. It has only one face with a bench height of more than 200 feet. Workers normally reach the top of the face by the ropes since there is no any other access path to reach the blasting location. The place where the incident occurred is shown in Figure 1.

1.2 Case # 2

The quarry where the incident occurred is located Diyagampola GS division of Divulapitiya AGA division of Gampaha District. The GSMB has issued an IML/C/HO/N/9030/LR/5 to this quarry. This incident took place while two employees and the deceased were engaging in removing loosen boulder; suddenly, the loosen bolder started to move down slowly. At that time other two had managed to hold the rope and narrowly escaped with injuries. But the deceased, who was sitting idling at the time, lost his balance and fell under the rubble and was killed. The cause of the death was cited as multiple injuries to the internal organ of the body of the deceased.

1.3 Case # 3

This incident belongs to two quarries located at Paragoda GS division of Divulapitiya AGA division of Gampaha District. The GSMB has issued an IML/C/GM/N/13 & IML/C /GM /N/14 for both quarries, and both licenses were valid when this incident took place. This incident took place when three workers were engaged in welding a metal door inside the explosive storage cabin. While the trio were engaging in welding, sparks came out, and a large fire was broken out inside the cabin. The trio fell victims to the fire.

According to the investigations, it has been revealed that some amount of Black Powder and pieces of safety fuse were laid on the floor of the compartment while they were engaging in welding. Due to sparks, black powder started to burn almost instantaneously. Black Power is a mixture of potassium nitrate, charcoal and sulfur, which is very sensitive to flams and sparks. Therefore it is undoubtedly said that black powder was ignited by sparks. It is therefore undoubtedly said that considering the severity of the incident, some amount of black powder was stacked on the floor of the compartment at that time.

1.4 Case # 4

The quarry where the incident occurred is located at 399/A Pananwela GS division of Dompe AGA division of Gampaha District. The GSMB has issued an IML/C/GM/N/46 to this quarry. This incident took place when the deceased was removing loosen boulders and suddenly lost his balance and was fallen onto the floor of the quarry and was killed. At the time of the incident, the deceased was not tightened to the rope, and he was fallen at the height of about 15 feet onto the floor. The cause of the death was cited as multiple injuries to internal organs due to falling from a height.

1.5 Case # 5

The quarry where the incident occurred is located at Kalahagala in Thamankaduwa AGA division of Polonnaruwa District. The GSMB has issued an IML/A/HO /1535/LR/8 to this quarry. According to the statements given by workers, this incident took place when they were charging the blast holes thunder shower started at once. At that moment, lightning was being experienced in the area where the quarry is located, and they all except the deceased fled away to the shed in fear. In a few moments, they heard the large explosion at the place where they charged the blast holes, and they only saw the mutilated body of the deceased lying under rubbles. They suspected that blast was triggered by lightning

2. Methodology

In the proceeding chapters, what leading factors contributed to these incidents are highlighted. The objective of this risk analysis is to identify risk factors that are dominant in metal quarries and to find ways to remove these risks. In this connection, risks are analyzed qualitatively and quantitatively.

Table 1: Reported fatalities against each reason and its probability of occurrences.

Reason	Fatalities occurred due to each reason during 2020 and 2021, respectively and Probability of Occurrence
Falling rock overhang	1 2 17.6%
Improper explosive management	3 0 5.8%
Accidental Falling	2 5 41.1%
Embankment failure	1 1 11.6%
Fly Rocks incidents	0 1 5.8%
Unintentional detonation due to lightning	1 0 5.8%
Falling to water-filled unprotected pits	1 0 5.8%
Vehicle Toppling incidents	1 0 5.8%

To analyze the risks number of cases within a year and its frequency and probability of occurrences are taken. All reported fatalities shown in Table 1 happened due to the following eight reasons. Tables 2 and 3 describe the probability of occurrence against each category of license and the probability of occurrence against the mode of blasting, respectively.

- Falling rock overhang on to employees
- Improper explosive management
- Falling from a height
- Embankment failure
- Projectiles (fly rocks)
- Unintentional detonation triggered due to lightening
- Falling to water-filled unprotected pits
- Toppling of vehicles

***Table 2: Number of reported fatalities (and its probability of occurrence) against each category of licenses*.**

Licence category	Number of Fatalities (Probability of occurrence)
IML(A)	2 (10.64%)
IML(B)	1 (5.3%)
IML(C)	16 (84.2%)
IML(D)	0 (0%)

Table 3: Reported fatalities against each mode of blasting (and its probability of occurrence).

Mode of Blasting	Number of Fatalities
Single-shot Bore blasting	17 (89.4%)
Multi borehole Blasting	02(10.6%)

3. Risk Ratings

During the calculation, the eight reasons leading to fatalities are rated. Rating is taken based on the number of occurrences of cases reported in 2020 and 2021 and the number of fatalities reported in each case. In this table, Accidental Falling, Falling rock overhangs, Embankment failure and Improper Explosive Management is considered as most severe incidents in quarries, and hence the value of rating is taken as 3 while the value of other cases is taken as 1 due to insignificance and are reported occasionally, and the valve of rating against each case is shown in Table 4.

Table 4: Risk Ratings.

Rating	Case
3	Accidental Falling
3	Falling rock overhang
3	Embankment failure
3	Improper Explosive Management
1	All other cases

3.1 Calculation of Risk

The risk of the above eight reasons leading to fatalities in any quarry is basically a function of the probability of occurrences and the rating. Therefore the risk is calculated according to:

Risk=Probability of occurrences of each reason * Rating

and these calculated risk values are shown in Table 5.

Table 5: Value of the risk of each reason.

Risk Category	Risk
Accidental Falling	1.233
Falling rock overhang	0.528
Embankment Failure	0.348
Improper Explosive Management	0.116
All Other Cases	0.058

According to these values, Accidental falling, failing rock overhang on to employees and Embankment failure are the most dominant factors leading to most fatalities to happen and all are reported from the quarries where the single borehole blasting method is adopted.

4. Risk Factors that could lead to these incidents

According to the above statistics of the reported cases, 89% of cases are from IML/C and IML/B type metal quarries. Blasting activities of these types of metal quarries have been adopted by the single-shot borehole blasting method, while blasts are initiated by a plain detonator, safety fuse arrangement. According to the regulations (GSMB), this method allows license holders to fire only one charged borehole at a time. Therefore this blasting method does facilitate limited opportunity to excavate in the manner in which create benches. Therefore excessive bench height (more than 15 feet) could be seen in these types of quarries. On the other hand, people working in these types of quarries choose to engage in blasting activities that would create excessive face height aiming for higher yield as well. As a result of excessive face heights, it is almost impossible to remove rock overhangs created from the previous blasting as the excavator cannot reach the desired location, whereas removing rock overhangs manually is too risky and impossible due to inaccessibility, heaviness etc. Poor weather conditions also contribute to escalating the situation. During the rainy season, runoff water seeps through fracture plains and builds up pressure on the cracking plains to loosen the boulder from the face of the quarry. Therefore as long as they are working in these types of quarries have to bear the risk of accidental falling over rock overhangs onto them.

On the other hand, these types of metal quarries have one or two working face/s, and people have to be on the face during drilling and charging times. The face angle of these quarries normally has 70°-80° to the horizontal. During this time, people are vulnerable to falling over the face accidentally. Due to excessive face heights and no ramps to reach to blasting location, people used to reach the desired location by the use of ropes without using PPE's. Therefore as long as they are working in these types of quarries have to bear the risk of accidental falling to a lower level.

This situation is further aggravated by the lethargic attitudes and knowledge of the people. Most of the people engaging in this industry are unskilled laborers who have little or no knowledge regarding mine safety and always undermine the safety while carrying out their duties. Even if they are provided safety gear, they do not use it. Therefore not wearing PPE's is also contributed to these fatalities.

Of the reported cases mishandling of explosives and poor blasting practice have led to catastrophic incidents. Some sort of knowledge and experience is required to handle matters related to explosives and blasting activities. But people who are dealing with explosives have little or no knowledge about explosives & blasting activities, and they deal with them without taking adequate safety measures ultimately have to sacrifice their own life as well as surrounding people, and it led to property damages as well.

5. Methods of reducing the above mentioned risk

Following methods can be adopted to reduce the risks.

- Blasting activities should be carried out in a manner in which create a face height of at least 10 feet in such a way that the machine could remove rock overhangs.
- Removed overburden must be stored and stabilized at least 5m away from the edge of the face to avoid falling
- Fence should be erected along the periphery of the quarry to avoid accidental falling by outsiders and animals.
- Workers should always wear a harness with a helmet and boots during working time.
- Blasting activities should be carried out by a competent person. Therefore a blasting foreman should be recruited for each and every quarry to avoid recurrence of these incidents.
- Only the optimum quantity of explosives should be used for each and every blasting.
- Blasts initiation by unrecognized means should be avoided.
- An officer should be appointed to deal with mine safety on a full-time basis by the license holder.
- Meetings should be periodically arranged so that workers become aware of mine safety.

- Workers must be given adequate training by sending them to training courses.
- Notice boards should be put up at strategic locations in the quarry so that workers become aware of mine safety.
- Use of Good quality safety gear by all workers.
- Every accident that occurred in a quarry should be recorded, updated and reviewed.

6. Role of the GSMB

The fatalities reported in the quarries where the single-shot blasting method is adopted shows a sad state that needs to address quickly by the authorities. As a regulatory, the GSMB should revise its strategies to regulate mining activities of these quarries back on track. In this connection following steps can be taken by the GSMB.

- The GSMB should increase the frequency of monitoring of activities of these quarries.
- Stern actions like cancelling of license should be taken against license condition violators.
- Awareness programs/ workshops/ training programs in respect of mine safety must be arranged for workers
- Before issuing licenses, the technical feasibility of license holders should be assessed.
- To encourage license holders to shift to multi borehole blasting method wherever possible.

7. Conclusion

As the majority of fatalities are reported from the quarries where the single-shot blasting method is adopted, as a regulatory body, it is high time to take appropriate actions to reduce the risks as people engaging in this industry are more vulnerable to accidents.

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A Study of Underground and Surface Mining Methods in Sri Lanka and its Suitability Assessment

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Abstract
The demand for minerals is rapidly increasing with industrialization, urbanization, and population growth. Sri Lanka has the potential to penetrate the world market with its endowed rich base of minerals despite its small land area. However, selecting a suitable mining method and its optimum utilization have significant roles in the growth of the local mineral mining industry. The lack of systematic approaches in mining methods is suboptimal for the economic growth of Sri Lanka, which demands immediate investigation and reassessment of contemporary mining practices. Thus, this study aims to critique the major mining practices and evaluate their suitability through characteristics and numerical analysis. This investigation was performed for underground mining methods: cut-and-fill stoping and open-stoping; surface mining methods: open-pit, and open-cast placer mining with reference to Bogala, Kahatagaha, Aruwakkalu, and Pulmoddai, respectively. A detailed study complying with the principles of the grounded theory was conducted through a multimethod approach followed by a thematic and content analysis along with the deductive approach. Moreover, the suitability of these methods was evaluated employing the University of British Columbia method through a numerical approach. This study reveals that the selections of mining methods are appropriate for all four mines, and mining practices are modernized and traditional, complying with site-specific requirements.

Keywords: Cut and fill stoping, Mining methods, Open pit, Open stoping, Placer mining

1. Introduction

Despite its small area, Sri Lanka is bestowed with abundant mineral resources and has a long mining history over several centuries [1], [2]. Mineral resources of Sri Lanka such as gems, graphite, and titanium-bearing sand have a greater demand in the world market due to their higher grade. Subsequently, mining and quarrying have contributed over 2.1 per cent to the Gross Domestic Product (GDP) in the past five years [3]. However, the contribution of mining to the country's economic growth has been suboptimal due to the lack of systematic approaches. Even though modernised and traditional mineral mines exist in Sri Lanka and practice a vast range of mining methods, notably, some deposits have not been optimally engineered and exploited fully to their existing potential.

The mining method is selected based on the respective geology of the deposit, hardness of the ore and host rock, depth, thickness, and other geological aspects [4]. Further, the extraction of narrow, steeply dipping vein deposits and deposits at great depth is significantly challenging and requires specific mining techniques. Thus, the mining methods adopted can be categorised into underground and surface mining methods.

Appropriate mining method selection and its optimum utilisation play a significant role in the success of the local mineral mining industry. Besides, incompetent mining practices constrict productivity and limit access to future reserves while damaging the environment. However, in the Sri Lankan context, there is a lack of comprehensive studies on existing mining practices. Thus, this study converges the characteristics and numerical approaches in analysing the suitability and practices of selected mining methods.

2. Methodology

2.1 Site Selection

This study focuses on the significant large-scale operational mines in Sri Lanka to represent underground mines: Bogala and Kahatagaha and surface mines: Aruwakkalu and Pulmoddai.

2.1.1 Bogala Mines

Bogala is a narrow vein graphite mine in operation since the mid-19th century, which extends up to 476 m depth. Moreover, it produces world-class graphite of 99 per cent purity. This underground mine is owned by Bogala Graphite Lanka PLC and is situated in Arugammana, Kegalle.

2.1.2 Kahatagaha Mine

Kahatagaha graphite mine is the deepest among underground mines in Sri Lanka, reaching a depth of 670 m. Further, this state-owned mine operates under Kahatagaha Graphite Lanka Limited and is situated at Maduragoda-Dodangaslanda, Kurunegala.

2.1.3 Aruwakkalu Limestone Quarry

Aruwakkalu Limestone Quarry is the fully mechanised and largest open-pit mine in Sri Lanka managed by Siam City Cement Lanka Limited. Further, it is located in Eluwankulam, Puttalam.

2.1.4 Pulmoddai Placer Deposit

Pulmoddai placer deposit is a source of predominant heavy minerals sand, which extends for 7 km along the Eastern coastal belt of Sri Lanka. Moreover, it is managed by state-owned Lanka Minerals Sands Limited.

2.2 Data Collection and Analysis

Existing underground and surface mining methods practiced in the selected mines were subjected to a detailed study complying with the principles of the grounded theory through a multimethod approach, comprising semi-structured interviews, participant and non-participant observations, and document analysis. The semi-structured interview method was followed with the objective of formal data acquisition. Moreover, questions were prepared intentionally standardised open-ended for more depth and clarified answers. Further, direct structured participant field investigations at Bogala and Aruwakkalu and direct structured non-participant investigations at Kahatagaha and Pulmodai were conducted to acquire information on the mining practices. The gathered data from these acquisition methods were subjected to a thematic and content analysis along with the deductive approach.

2.3 Suitability Assessment

Outcomes from the data analysis were critically assessed, and the suitability of the mining methods practices at each site was evaluated. Further, mining method selection criteria: University of British Columbia (UBC) method [5] was utilised to evaluate the mining methods at Bogala, Kahatagaha, and Aruwakkalu. The inputs were given for geometry and grade distribution, such as general shape, ore thickness, ore plunge, grade distribution, and depth in accordance with the gathered data. Further, the suitability rankings of mining methods were estimated through the generated numerical rating.

3. Results and Discussion

3.1 Evaluation through UBC Method

Table 1: Numerical ratings from UBC method evaluation.

Mining Method	UBC Rating		
	BO	KH	AR
Block Caving	-38	-38	9
Cut and Fill Stoping	18	18	14
Longwall Mining	-37	-37	11
Open Pit	-42	-42	15
Room and Pillar	-37	-37	14
Shrinkage Stoping	16	16	-40
Square Set Stoping	10	10	7
Sublevel Caving	-37	-37	10
Sublevel Stoping	4	4	16
Top Slicing	5	5	9

Where BO = Bogala, KH = Kahatagaha, and AR = Aruwakkalu

3.2 Cut and Fill Stoping

Bogala adapted open stoping at its earlier stage and has been replaced with the overhand cut-and-fill stoping method beyond the depth of 165 m below the pit head level [6]. Presently, it is the only underground mine in Sri Lanka that practices the cut-and-fill mining method.

Table 2: Characteristics of Bogala mines (after Hettiwatte, 2014) [7].

General Shape	Steeply dipping narrow vein
Depth	> 500 m
Thickness of Ore	20 - 40 cm
Dip/Dip Direction	Na vein: 65/330
	Kumbuk vein: 75-80/210
Mineralogy	99.9% carbon graphite
Host Rock	Metamorphic Garnet-biotite gneiss
Rock Strength	Compressive strength - 143.75 MPa
Planes of Weaknesses	Number of fractures propagating in different directions
Groundwater conditions	Underground seepages - Wet mine
Economic Factors	High-grade graphite with good grade distribution
Labour	Easily available
Environmental Concerns	Dust control by wet drilling
	Water pollution is low
	Ground subsidence is minimal/totally absent

Bogala is a wet mine scoring 4 for groundwater conditions in RMR rating as it has the possibility of water ingress through the fractured planes. Moreover, the RMR of this mine is 71 due to the lower ratings for the condition and orientation of joints and groundwater [7]. Further, Table 2 endorse that the most suitable mining method to extract steeply dipping narrow vein graphite from Bogala underground mine is cut-and-fill stoping as the country rock is highly fractured and incompetent. Furthermore, the numerical rating of the UBC method validates its suitability with the rating of 18, as shown in Table 1.

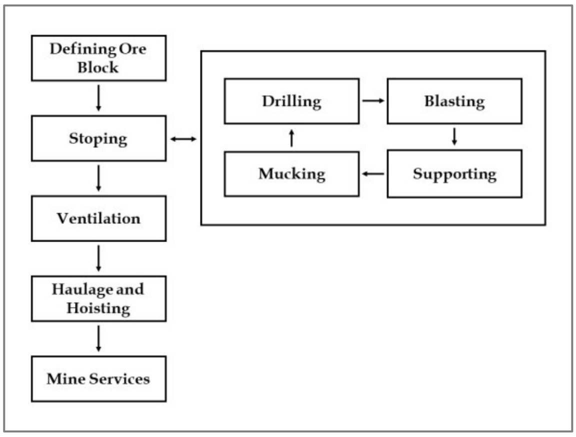

Figure 1: Mining flowsheet at Bogala.

In addition, Bogala practices this overhand cut-and-fill stoping through a cyclic process with the aid of other supportive services, as shown in Fig. 1.

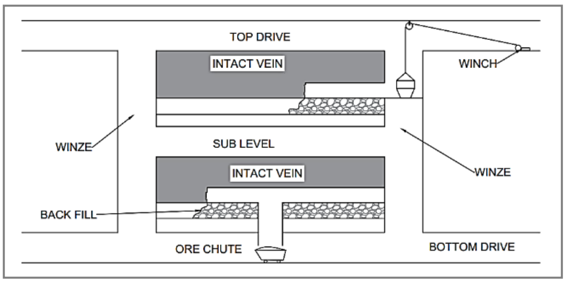

Figure 2: Conceptual diagram of ore block at Bogala.

An ore block is bounded by two drives and winzes comprising an area of 60 m x 60 m, as illustrated in Fig. 2. It permits simultaneous stoping, maximum ore recovery, efficient haulage and hoisting, adequate ventilation, and reasonable ground control. Subsequently, stoping initiates from the lower main level of the block with the dimension of 1.8 m x 1.3 m and progresses upwards. This optimal stope dimension opens the way for man passage and haulage of materials in the narrow vein environment.

Notably, stope advances through a mining cycle: drilling, blasting, mucking, and supporting, as exhibited in Fig. 1. Usually, a pneumatic-powered jackleg drill having 3 feet drilling rod with a chisel bit or button bit of 34 mm diameter is used to drill graphite ore and intact rock. Moreover, water-gel and ANFO (ammonium nitrate with fuel oil) are used as primary and secondary explosives, respectively, along with the electric detonators (ED) to initiate the blast. Immediately after, scaling is performed using a steel bar or high-pressure water jet to ensure the workplace's safety from the loosening of rocks. In addition, roof support is established using timber logs or S-H rails and wooden planks before approaching the blasted face. Mucking is carried out manually using hands to maximise the ore recovery.

Furthermore, the ventilation network of the Bogala mines is supported by an exhaust ventilation system. However, workplaces are not exposed to the main ventilation network; thus, compressed air is used to provide comfort for workers [8]. Moreover, winzes and ore chutes are used for ore haulage, and the materials are transferred to the surface through the Alfred and Gabriel shafts. In addition, a dedicated team upholds uninterrupted mine services.

3.3 Open Stoping

Kahatagaha has a vast range of parallel graphite veins in the East-West direction with southerly dip and practices the conventional open-stoping method to extract the higher-grade graphite ore.

Table 3: Characteristics of Kahatagaha mine.

General Shape	Steeply dipping narrow vein
Depth	> 600 m
Thickness of Ore	10 – 25 cm
Dip	60° - 70°
Mineralogy	99.9% carbon graphite
Host Rock	Garnet granulitic gneiss
Rock Strength	Compressive strength - 123.75 MPa
Planes of Weaknesses	Competent rock
Groundwater conditions	Dry mine
Economic Factors	High-grade graphite with good grade distribution
Labour	Easily available
Environmental Concerns	Dust control by wet drilling
	Water pollution is low
	Ground subsidence is minimal/totally absent

Kahatagaha is a dry mine as the groundwater conditions are categorised under the damp condition with the rating of 10 in the RMR classification system. Moreover, the host rock of Kahatagaha is classified as a very good rock under the Class I category, and RMR is determined to be 97 as it possesses a higher rating for condition and orientation of joints and groundwater [7]. In addition, Table 3 depicts that the characteristics of Kahatagaha are favourable for open stoping, even though UBC ratings recommend cut-and-fill stoping with the value of 18, as shown in Table 1. It is evident that open stoping is more cost-effective than cut and fill stoping for the mines with good ground control.

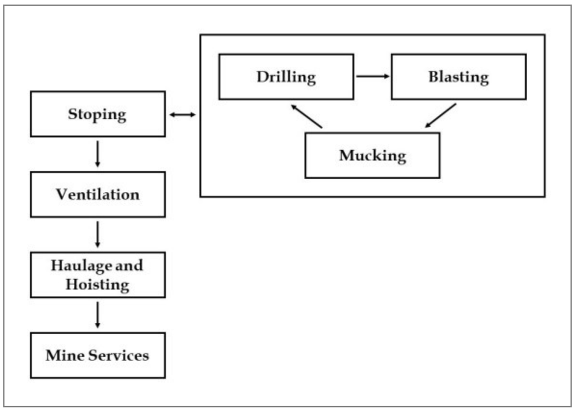

Figure 3: Mining flowsheet at Kahatagaha.

Further, Kahatagaha follows the mining practices, as shown in Fig. 3. Stoping progresses upwards from the main drift using the overhand method while the stoping width varies from 20 cm to 50 cm.

Conventional drilling and blasting is practiced for the extraction, and on most occasions, only the graphite veins are extracted without creating space for man passage. Albeit it reduces the cost for barren rock disposal, it halts the maximum ore recovery. A steel rod 'Kuththu' is used to drill the graphite ore manually, and a pneumatic-powered jackleg drill is used for mechanised drilling of intact rock. Although manual drilling consumes a longer time, it contributes to the maximum recovery of the narrow vein with a lower ore dilution.

No benching system exists on stopes, and the men work by standing on anchors fixed on the footwall, which may reduce the production efficiency and safety. Occasionally, wooden planks are placed on the anchors across the stope to provide a working platform. A blast hole of 36 mm diameter is charged with water-gel and ANFO as primary and secondary explosives, respectively, and the blast is initiated by a combination of plain detonator and safety fuse.

Figure 4: Unloading from 'Winch Box' at Kahatagaha mine.

The blasted ore is piled up in the drift and mucked manually by the hands. After that, it is packed in gunny bags and hoisted to the surface. Level-to-level hoisting is done using 2 feet x 2 feet x 2 feet dimensioned wooden boxes called 'Winch Box,' as shown in Fig. 4. However, the existing infrastructure developments are not optimised to meet the production demand as it consumes much time for haulage and hoisting.

Further, the mine is ventilated by an exhaust ventilation system, comprising a main downcast shaft: Kahatagaha, upcast shaft: Kolangaha, and five booster fans at the return airways. However, open-stoping is a disadvantage in ventilating deeper levels, as air leakage and short-circuits are dominant through mined-out areas. Thus, the temperature below 2000 feet level rises and may reduce workers' efficiency.

3.3 Open Pit Mining

Aruwakkalu limestone deposits are found beneath the thick overburden layer of red soil, and the limestone is extracted by the open-pit mining method.

The limestone deposit is laterally extended in a shallower depth less than 70 m with an overburden thickness of modestly 25 m to 30 m, as given in Table 4. Thus, the open-pit mining method is economically viable as the ore is easily accessible by removing the overburden. In addition, results from the UBC method recommend the open-pit mining method with a score of 15, as depicted in Table 1, which validates the suitability of the existing mining method. Although the UBC method rates the underground method: sublevel stoping at the top, it is not feasible as it demands competent ground control.

Table 4: Characteristics of Aruwakkalu mine.

General Shape	Horizontally bedded deposit
Depth	< 70 m
Thickness of Ore	9 – 12 m
Mineralogy	Miocene Limestone
Overburden	Red soil (25 - 30 m)
Inter-burden	Kartsified limestone (12 - 15 m)
Groundwater conditions	Mining above mean sea level
Economic Factors	Raw material for cement production
Labour	Easily available
Environmental Concerns	Animal rescue
	Restoration of flora

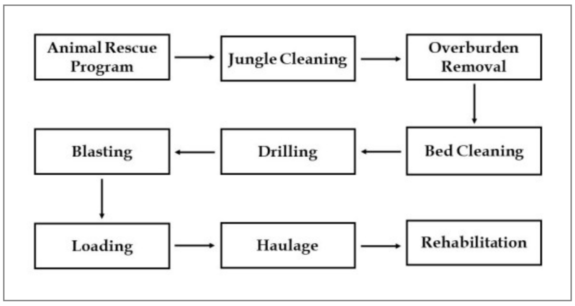

Figure 5: Mining flowsheet at Aruwakkalu.

Mining practices at Aruwakkalu limestone quarry are proceeded, as indicated in Fig. 5. The quarry management carries out a detailed study at the beginning on the area's ecology, flora, and fauna. Further, jungle cleaning is followed by removing the overburden consist of red soil and low-grade limestone layers. Moreover, mining progresses by advancing the benching system to form the open pit, as shown in Fig. 6, until it reaches the limestone bed with an ideal lime saturation factor (LSF) as the cut-off grade.

Figure 6: Aerial view of Aruwakkalu mine.

The pit is developed with a 3.5 - 3.8 stripping ratio by multi-benching (5 - 7 benches) on the red soil layer, with 2.1 m bench height, 4 m bench width, and 90° slope angle. The 40 per cent of the intermediate layer is removed by ripping and dozing, and the rest is extracted by blasting. Ripping and dozing is highly preferred in the soft parts of the Karstified layer as it is comparatively cheaper than drilling and blasting. Further, the high-grade limestone is extracted by drilling and blasting as the hardness of the layers increases with depth. Furthermore, the drilling is performed using hydraulic drill machines to 10.3 m depth with a drill hole diameter of 76 mm. Moreover, water-gel and ANFO are charged along with electric detonators in a staggered pattern with spacing: 2.8 m and burden: 2.5 m.

The blasted material is loaded using a hydraulic shovel of 4.8 m^3 bucket capacity and hauled to the dumpsite located 2.5 km away from the pit using dump trucks of 55 tonnes payload each. Moreover, rehabilitation progresses simultaneously along with the mining by refilling the pits and reforesting the preserved species to ensure the safe locomotion of the native wild animals.

3.4 Open Cast Placer Mining

Pulmoddai placer deposits are among the richest occurrences of the heavy mineral sand along the beach from Nilaweli to Mullaitivu in Eastern Sri Lanka. The pre-mining resource was estimated as 12.5 million tonnes averaging 90 per cent heavy minerals, and the average assemblage comprises 65 per cent ilmenite and leucoxene, 10 per cent zircon, and 10 per cent rutile [9]. The remaining non-valuable heavy minerals are predominantly monazite, garnet, and sillimanite. These heavy minerals are extracted using the mechanised open cast mining method.

Table 5: Characteristics of Pulmoddai deposit.

Deposit type	Replenishable beach placer
Length	7 km
Width	100 m (only 30 m is minable)
Major Heavy Mineral Content	Ilmenite, Rutile, Garnet
Climatic conditions	Heavy rain during North-east monsoon
Economic Factors	Raw material for titanium products
Labour	Easily available
Environmental Concerns	Beach erosion
	Affect flora along the shoreline

The deposit covers an area of 18 hectares (0.18 km^2) which extends along the beach for 7.2 km with an average width of 50 m, while the maximum width is 250 m [9], [10]. Pulmoddai practices a mechanised open cast mining method as the deposit is easily scrapable using machines with a higher mechanical and economic efficiency, and the site characteristics are tabulated in Table 5. Further, there is high uncertainty in the deposition of heavy minerals as it is naturally replenishable and time-dependent. Thus, an increment in the mining rate may deplete the grade of mineral sand. It is evident that the mechanised open cast mining method is the most suitable method to extract mineral sand.

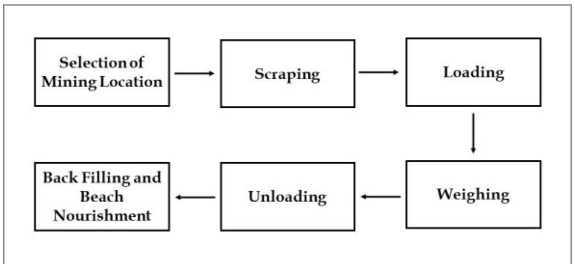

Figure 7: Mining flowsheet at Pulmoddai.

The mining sequence at the Pulmoddai placer deposit is indicated in Fig. 7. The processing plant at Pulmoddai is designed to operate with a minimum grade of 40 per cent. Thus, viable mining locations are selected from the laboratory testing of auger samples collected along the shore to satisfy the above requirement. Meanwhile, it is required to maintain face angle of beach slope between 10° to 12.5° for ilmenite entrainment (70-79 wt%) and between 12° to 14.5° for garnet deposition (20-29 wt%) [11]. However, it is hard to establish this face angle along with the dynamic wave actions.

Figure 8: Mechanical scraping operation at Pulmoddai using wheel loader.

A thickness of 0.5 – 2 feet (0.15 - 0.61 m) beach placer along the shore in predetermined locations are scraped using the mechanical loaders as shown in Fig. 8 with a bucket capacity of 0.75 cubes (2 m^3). Further, mined raw mineral sand is hauled in wet condition using private tractors with a trailer capacity of 1.5 cubes (4 m^3). After that, it is unloaded to the pile near the processing plant, preceded by weighing at the measuring point. However, during the Northeast monsoon period, limited mining activities occurred due to heavy rain in the Pulmoddai zone as the waves get rough and cause an unsuitable environment for mining.

4. Conclusion

This study underlined the mining methods: cut-and-fill stoping, open-stoping, open-pit, and open-cast placer mining, practises at Bogala, Kahatagaha, Aruwakkalu, and Pulmoddai, respectively, are suitable according to the site characteristics and numerical evaluation results of the UBC method. Further, all these four mines adapt a cyclic process for mining, aligned with the predefined flowsheets. Moreover, it is highlighted that hybrid modernised-traditional approaches in mining methods at these mines comply with the site-specific requirements.

5. Recommendations

It is recommended to carry out advanced numerical analysis to validate the suitability of practising mining methods meticulously.

Acknowledgement

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Development of Mathematical Model to Decide the Optimal Graphite Product Mix to Enhance the Profit

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Abstract

Sri Lanka is the only country in the world known to extract and produce commercially viable quantities of natural crystalline vein graphite. Currently, Bogala graphite mines and Kahatagaha graphite mine are famous underground mines and the largest natural graphite producers, which supply natural vein graphite in the form of various product categories to the international graphite market. Out of that two major mines, Bogala graphite mines extracts approximately 350 metric tons of natural graphite per month. When it comes to extracted graphite from underground (run of mine), the raw graphite consists of various carbon contents in the form of lumps, chips, and powder. Such graphite is subjected to hand sorting or mechanical separation before it is used for further processing to produce various product categories as requested by local or international customers. During the graphite processing stage, it is a real dilemma to decide that with available graphite in raw form in which carbon content varying from 80% to 99%, to produce saleable product to which customer orders with a view to getting maximum profit out of various pricing for various saleable graphite products that Bogala graphite mine produces. So, it is worth finding out which product mix gives the highest monthly revenue utilising its limited monthly underground mine production and limited machine capacities. The problem addressed here is to determine the product mix (combination of sales package) to be adopted by the company for selling her graphite products at which the optimal profit level would be attained.

Keywords: Carbon, Crystalline vein graphite, Processing, Run of mine, Underground

1. Introduction

Crystalline vein graphite is believed to be a naturally occurring pyrolyte graphite condensed from gas or liquid phase under high temperature and pressure well below in earth crust. Deposits consist of veins of variable thickness and various carbon content ranging from 80% to 99% carbon contents. In Sri Lanka, currently, Bogala Graphite mines and Kahatagaha Graphite mines are famous underground mines and the largest natural graphite producers, which supply natural vein graphite in the form of various product categories to the international graphite market. Out of two major mines, Bogala graphite mines extracts approximately 350 metric tons of natural graphite per month. When it comes to mined out graphite from underground (run of mine), the raw graphite consists of various carbon contents in the form of lumps, chips and powder. Such graphite is subjected to hand sorting or mechanical separation before it is used for further processing to produce various product categories as requested by local or international customers.

Below diagram is useful to identify the production flow and the machineries used for preparing each graphite grade. The inter-relation of the machinery can also be clearly understood with that schematic diagram. Final graphite product grades which are manufactured using different machines, are shown in the diagram as well.

Schematic diagram of Production Flow of Graphite Processing in Bogala Graphite Mines

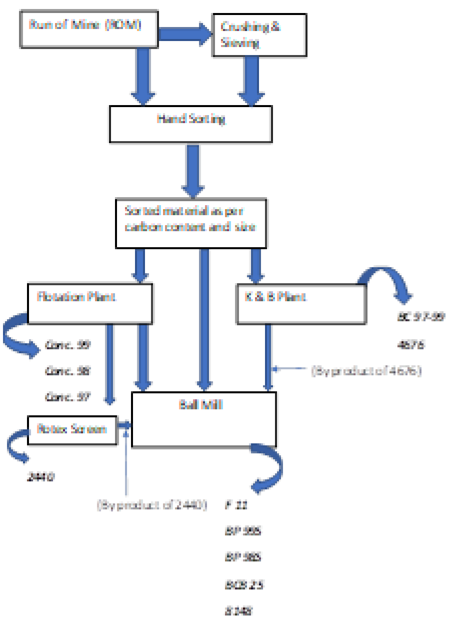

Fig. 1: Major steps in Graphite Processing

During the graphite processing stage, it is a real dilemma to decide that with available graphite in raw form in which carbon content varying from 80% to 99%, to allocate raw material to which customer orders with a view to getting maximum profit out of various pricing for various graphite products that Bogala graphite mine produces. So it is worth finding out which product mix gives the highest monthly revenue utilizing its limited monthly underground mine production, which is 350 mt. The problem addressed here is to determine the product mix (combination of sales package) to be adopted by the company for selling her graphite products at which the optimal profit level would be attained.

1.1 Literature Review

It is generally agreed that the use of mathematical algorithms came into existence as a discipline during World War II when there was a critical need to manage scarce resources. However, a particular algorithm and technique can be traced back as in world war-I, when Thomas Edison (1914-1915) made an effort to use a tactical game board for a solution to minimize shipping losses from enemy submarines instead of risking ships in actual conditions [2]. So under this heading, we shall review the existing recent literature which are related to the topic furthermore.

According to Mille in 2007, linear programming is a generalization of linear algebra used in modelling so many real-life problems ranging from scheduling airline routes to shipping oil from refineries to cities for the purpose of finding inexpensive diets capable of meeting daily requirements. Miller argued that the reason for the great versatility of linear programming is due to the ease at which constraints can be incorporated into the linear programming model [4]. Linear programming techniques have been used in many industrial applications with a view to getting optimal solutions in different requirements [1]. The beverage industry, Oil refinery, Agriculture, manufacturing of different product categories using same raw material, energy sector, Facility location, vehicle routing and scheduling, personnel, machine and job scheduling, product mixes, and inventory management are a few fields where LP models applied to get optimal solutions for achieving highest profit, maximum use of scarce raw material and efficient output.

So the summary of Linear Programming model formulation steps for a product mix company can be given as;

1. Define the decision variables,
2. Define the objective function to maximize profit, and
3. Define the constraints.

With that understanding, it is possible to go for good decision-making to maximize the profit of a company engaged in manufacturing different product mixes.

When it comes to the application of operation research or mathematical modelling in mine planning and product mix optimization for maximizing profits using mathematical modeling, it dates back to the 1960s [3]. Since that time, optimization and simulation, in particular, have been applied to both surface and underground mine planning problems, including mine design, long and short-term production planning, equipment selection and dispatching etc.

2. Methodology

Bogala Graphite Lanka Plc. is chosen for this research study for two main reasons. First, it uses natural graphite mined as run-of-mine from its own underground mine and is used for producing various graphite products by varying its carbon content and particle size and some other physical properties. Secondly, Bogala Graphite mainly exports many different graphite products to many international customers with different price categories, which decide the profits to the company based on profit margins of each product category.

Primary/ Secondary data that is going to be used are as follows, and all that was taken from the available records in Bogala Graphite Lanka PLC.

- Carbon grade-wise quantities sorted from monthly production.
- Product categories processed as per orders from buyers and prices.
- Possible product grades to be processed using available raw material or sorted material.
- By-product information when preparing each product category.

Considering the constraints of raw material available for a month and underground production, which is always limited, a mathematical technique such as linear programming would be much useful to solve and obtain optimum solution for product mix. Linear Programming is a mathematical technique for generating and selecting the optimal or the best solution for a given objective function. It may be defined as a method of optimizing (i.e., maximizing or minimizing) a linear function for a number of constraints stated in the form of linear equations. Generally, the objective function may be a maximization of profit (which is the focus of this research work) or minimization of costs or labor hours. Moreover, the model also consists of certain structural constraints, which are a set of conditions that the optimal solution should justify. Examples of structural constraints include raw material constraints, production time constraints, and skilled labour constraints, to mention a few.

The general linear programming problem (or model) with *n* decision variables and *m* constraints can be stated in the following form.

(Max or min) $z = c_1x_1 + c_2x_2 + \ldots\ldots + c_nx_n$
(Objective function) -----------------> (1)

Subject to linear constraints of the form

$a_{11}x_1 + a_{12}x_2 + \ldots\ldots a_nx_n \ (\leq, =, \geq)\ b_1$

$a_{12}x_1 + a_{22}x_2 + \ldots\ldots a_{2n}x_n \ (\leq, =, \geq)\ b_2$

....

....$a_{m1}x_1 + a_{m2}x_2 + \ldots\ldots a_mx_n \ (\leq, =, \geq)$

$x_1 + x_2 + \ldots\ldots + x_n \quad \geq 0$

Model Assumptions made for the analysis are as follows.

a. The production quantity is the same as the sales. As it is unrealistic to produce what is not sold.
b. There is a linear relationship among the variables used in the model.

2.1 Data Collection and Analysis

The data for this research study are in the below tables which were extracted from the production and sales records of Bogala Graphite Lanka Plc. In Bogala graphite mine, the average monthly underground mine production (run of mine or ROM) is 400 metric tons, and for this research, that figure has been assumed to be fixed for all calculations. Once sorted by manual curing, the composition of ROM according to carbon % is as follows.

Table 1: Composition of run of mine.

Carbon %	Composition	Form
99+	1%	Lumps
97-99	21%	Lumps
90-97	16%	Lumps/ chips
70-90	48%	Tub dust
Below 70	11%	Tub dust
Pure rock	3%	

Product grade requested by buyers and its Carbon %, profit margins (assumed values) and demand in metric tons (assumed values and varies monthly) and available machine hours of each plant per month are given in below two tables.

Table 2: Product grade with C%, profit margin and demand.

Product Grade	Carbon Content	Profit (Euro)/ mt	Demand Qty. (mt)
2440	99%	1329	40
BP99S	99%	1241	25
Conc. 99	99%	717	60
BC 97 99	97-99%	597	40
4676	97-99%	559	60
8148	97-99%	517	20
BP98S	97-99%	510	25
BCB 25	97-99%	414	30
F 11	97-99%	339	40
Conc. 98	97-99%	267	60

Table 3: Machine hours availability for each plant

No	Product grade	Plant used	Machine hrs reqd. / mt
1	F 11	Ball mill	2 ½
2	2440	Rotex Screen	1 ½
3	BC 9799	K & B Plant	¾
4	BP 99 S	Ball mill	3
5	BP 98 S	Ball mill	2 ½
6	BCB 25	Ball mill	2 ½
7	8148	Ball mill	2 ½
8	4676	K & B Plant	1 ½
9	Conc. 99	Flotation	1 ¾
10	Conc. 98	Flotation	1 ½
11	Conc. 97	Flotation	1 ½

Formulation of Objective function:

Maximize profit = (1329) X1 + (1241) X2 + (717) X3 + (597) X4 + (559) X5 + (517) X6 + (510) X7 + (414) X8 + (339) X9 + (267) X10

Where X1, X2,…. X10 will give the possible production quantity of each graphite grade in making an optimum profit with respect to the graphite grades 1 to 10 in the above table.

Decision variables and Constraints were identified for available raw material and available machine hours in a particular month. That information was used for MS Excel Solver with a view to analyse the best solution for the situation.

The aim of the analysis is to use the above data in a suitable linear programming model so that to find the product mix that gives the highest profit margin.

3. Results

By using MS Excel Solver, the product mix that gives the highest profit margin against the sales demand can be found. So the company can go for the highest profit with that mix, and the changes of raw material grades in the run of mine can change the mix in each month. In such a situation, this model can be run with input changes so that each month different product mix values can be taken to achieve the highest profit margin.

Answer report given by Excel Solver for this particular analysis shows the required mix of product grades which gives the highest profit margin against the product demand of that particular month.

Microsoft Excel 16.0 Answer Report
Worksheet: [LP model.xlsx]Sheet1
Report Created: 7/06/2021 10:16:00 AM
Result: Solver found a solution. All Constraints and optimality conditions are satisfied.
Solver Engine
Engine: Simplex LP
Solution Time: 0.031 Seconds.
Iterations: 9 Subproblems: 0
Solver Options
Max Time Unlimited, Iterations Unlimited, Precision 0.000001, Use Automatic Scaling
Max Subproblems Unlimited, Max Integer Sols Unlimited, Integer Tolerance 1%, Assume NonNegative

Objective Cell (Max)

Cell	Name	Original Value	Final Value
I17	Z = Lumps/chips	0	82994.20922

Variable Cells

Cell	Name	Original Value	Final Value	Integer
E18	X1 Demand	0	60	Contin
E19	X2 Demand	0	50	Contin
E20	X3 Demand	0	60	Contin
E21	X4 Demand	0	40	Contin
E22	X5 Demand	0	30	Contin
E23	X6 Demand	0	50	Contin
E24	X7 Demand	0	30	Contin
E25	X8 Demand	0	40	Contin
E26	X9 Demand	0	20	Contin
E27	X10 Demand	0	20	Contin

Figure 2: Answer report given by MS Excel Solver

4. Discussion

An algorithm of Linear Programing with the support of MS Excel Solver was used to solve the product mix problem. This algorithm solved the linear program problem and gave the optimal product mix against the sales demand of customers so that the company can decide how to promote and sell to produce the optimal product mix that gives the highest profit in a particular month.

Optimization problems in many fields can be modeled and solved using Excel Solver as done in this particular scenario. It does not require knowledge of complex mathematical concepts behind the solution algorithms.

5. Conclusion

The objective of this study was to apply a suitable mathematical algorithm or technique to optimize profit margin against sales demands for various graphite product grades in a particular month. For solving this problem, a linear programming model using MS Excel Solver has been developed. The Linear programming technique determined optimum profit values for each graphite grade demanded by customers and gave the product mix, which gives the highest profit.

Based on the findings of the research, the following conclusions can be made:

1. Optimal product mix giving highest profit in selling graphite products can be achieved with the application of this linear programming model. The system of equations can be expanded or reduced to accommodate any variety of system combinations.
2. The research is significant in the sense that it will assist the company in making corrective decisions well in time using the methods of linear programming. This will determine the future production patterns and outlook resulting in the establishment of new production grades while planning for sales profit maximization of the company.
3. Although this research deals with maximizing profit, it can be applied to other process industries with similar production processes.

So manufacturing companies can develop linear programming models to help decide how many units or quantities of different products they should produce to maximize their profit (or minimize their cost), given scarce resources such as raw material, machine hours and available labour.

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Evaluation of Ventilation Network through Hybrid Analytical-Numerical Approach in Underground Working Block

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Abstract
The mine environment is complex and highly dynamic due to the developments over time and surrounding climatic changes. Heedlessness to supply adequate quantity and quality of air will catalyse short and long-term ailments to the workers. Therefore, this study emerges as the new research frontier in incorporating software-assisted numerical simulation with analytical computations. This investigation assesses the existing ventilation parameters at the Bogala underground graphite mine for the propriety of the working environment. The uttermost bottom block between 240 and 275 fathoms (FM) levels was examined. The parameters were obtained through the in-field ventilation survey. Measured air quantity, psychometry, and air quality values were analysed and fed to the computer-simulated model. Moreover, the re-entry time for a development drive at 275 FM level was estimated using the throwback method. Adequacy assessment unveils that all the parameters besides air quality are inadequate at most stations for optimal mine conditions to attain maximum efficiency. Furthermore, the re-entry time after the development blast at the selected drive is meager and necessitates re-calculation for each blast. Moreover, stale air mixing and air recirculation are extant at 240 FM and 275 FM levels, respectively. Thus, mine ventilation at Bogala needs to be optimised, admitting workers' health, safety and comfort, and productivity of the mine.

Keywords: Adequacy assessment, Mine safety, Mine ventilation, Re-entry time, Throwback method, Ventsim

1. Introduction

Mine ventilation, also described as the lifeblood of a mine, is the provision of sufficient quantity and quality of air to all the working places and travel ways to dilute the hazardous and unhealthy atmospheric contaminants to an acceptable level [1],[2]. Air quantity, velocity, humidity, effective temperature, dust, and contaminants concentration are the primary ventilation parameters. The threshold limits of these parameters are hinged on tolerance to provide a safe and comfortable working environment, and the standards have been raised substantially in recent years [3]. Different countries, territories, and regions adopt various standards according to their environment and human characteristics to regulate their mines. However, achieving a safe, effective, and efficient ventilation network is cost-effective and humanitarian.

Sufficient airflow is essential to satisfy the human respiratory needs and evacuate the strata gas, diesel exhaust fumes, dust, toxic fumes of explosives, and heat [3]. Nevertheless, these requirements depend on mine operating conditions and can be assessed individually or in combination. Psychometric properties have a significant role in providing reasonable comfort conditions in the work environment, which adversely affect the performance of workers and the productivity of the mine. Moreover, excessive temperature and humidity lead to loss of interest in the tasks, coordination, and dexterity, and in the worst-case fainting, exhaustion, cramps, rash, and stroke may result [1].

The ventilation network should be optimised to deplete the contaminant gases to yield a safer environment with good air quality. Means of contaminant addition to the working environment are personnel expiration, explosive blast fumes, and combustion engine emissions [3]. The primary gases of immediate concern to human lives are oxides of Nitrogen (NO_x, NO_2), Carbon monoxide (CO), and Ammonia (NH_3); exposure to these gases above the threshold limit would even cause fatalities [5]. Thus, re-entry to the workplaces after explosions should also be considered in ventilation planning to evacuate the toxic fumes.

This study mainly focuses on mine ventilation at Bogala graphite mine, one of the deepest narrow vein mines in Sri Lanka extending up to 500 meters, which adopts the cut and fill mining method. The exhaust ventilation system, which contains one upcast ventilation shaft with a vertically fixed fan, provides fresh air to this mine. Alfred shaft is the downcast primary shaft that extends up to 72 fathom (FM) level, and the blind shaft Gabriel carries the fresh air further to the deeper levels, as shown in Fig. 1. Besides, other downcast surface openings: Karadawatha shaft and Ravanamidulla adit contribute to the primary ventilation by a significant intake and act as maintenance and emergency routes. In addition, the installation of a booster fan at 275 FM level and three auxiliary fans at 142 FM, 240 FM, and 275 FM supports the ventilation system. However, a dynamic mine climate with an average temperature of 26° C and high humidity of 70 to 80 per cent [4] urge the necessity to evaluate the existing ventilation network at Bogala mines.

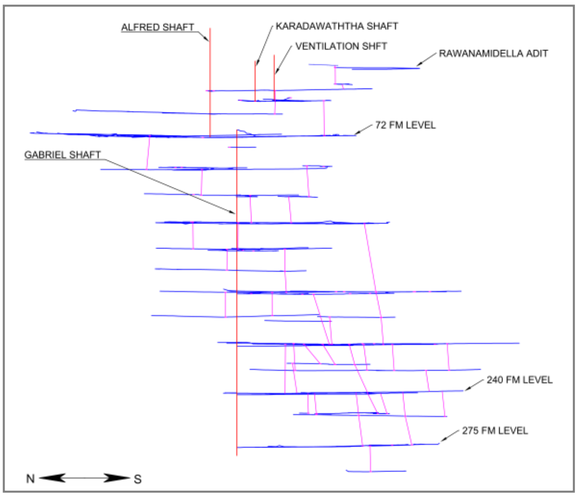

Figure 1: Ventilation Network of Bogala Mines.

2. Methodology

This assessment was performed to evaluate the adequacy of the ventilation network of a selected block between two primary levels of Bogala mines: 240 FM and 275 FM. These levels contain a higher number of workplaces and considerable air recirculation than other working levels. This study was performed through air quantity and quality survey, software-assisted numerical analysis, and analytical assessment of parameters.

2.1 Ventilation Survey

The required ventilation parameters: air quantity, psychometry, pressure, and air quality were measured through the surveys adhering to the standard practices in the selected ventilation stations during January 2019.

Cross-sectional areas of the airways were taken using the offset method. Meanwhile, the air velocity was measured using a vane anemometer (Testo 410-2) by a highly reliable fixed-point method and then validated through the traverse method. Moreover, minor airflows were measured using a low-range vane anemometer. The whirling hygrometer was utilised to obtain dry and wet bulb temperature. Further, barometric pressure at each station was recorded using a digital pressure meter (Testo 511). In addition, the concentrations of O_2, CO_2, and NH_3 were acquired using a multi-gas detector.

2.2 Air Flow Quantity Calculation

The existing airflows at the ventilation stations were calculated utilising the measurements from the survey using,

$$Q = A \times v \tag{1}$$

Where: Q = Volume flow rate (m^3/s), A = Cross-section area of the airway (m^2), v = Velocity of the air (m/s)

2.3 Numerical Simulation

The centerline model drawn using AutoCAD was imported as airways into the Ventsim software and simplified using the filter tool. The generated block model is illustrated in Fig. 2. The shape of the drives and crosscuts were set as irregular, and shaft and winzes as rectangular. Then the area measurements were fed with the calculated values and the perimeters with the standard values. Crosscuts and drives were deemed to have a rough blasted surface and shaft as average blasted surface. The Ventsim itself assigned frictional factors based on the characteristics of these surfaces.

This software model was simulated by allocating two-thirds of calculated airflow quantities as training samples. The model was executed with four iterations using the Hardy cross method [3] and validated by the rest of the calculated values. Finally, this model was utilised to obtain inaccessible and un-measurable airflow quantities for the adequacy assessment.

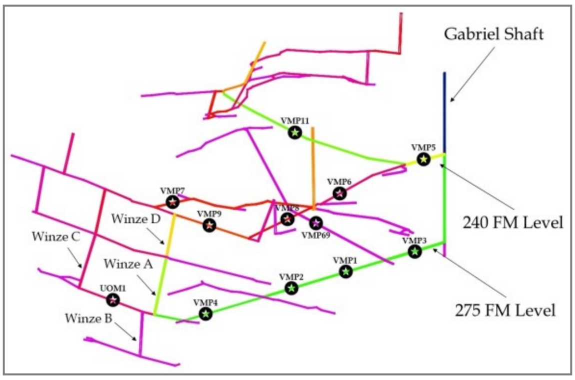

Figure 2: Ventsim model of the block between 240 FM and 275 FM.

2.4 Assessment of Parameters

Relative humidity was obtained from the psychrometric chart of 105 kPa, and the effective temperature was acquired from the effective temperature chart using wet and dry bulb temperature and air velocity. A reverse calculation was performed considering the maximum allowable effective temperature as 27° C to estimate the required air velocity. Furthermore, the required airflows for the workplaces and airways were calculated to maintain an adequate supply for persons considering the standard effective temperature.

2.5 Re-entry Time Calculation

Re-entry time after the blast in a development drive at 275 FM of 'Na' vein was ascertained by the throwback method amidst the following assumptions [1],[5],[6]:

1. Gases from explosives are evenly mixed within the throwback volume.
2. Gases are uniformly diluted over time with fresh air supply by evenly mixing in contaminated volume.
3. All the nitric oxides produced by the explosive are converted rapidly to nitrogen dioxide in the presence of air and water vapour.

Throwback distance of fume is estimated by:

$$L = \frac{KM}{FaD\sqrt{A}} \qquad (2)$$

Where: L = Length of fume throw back (m), K = Constant (usually 25), M = Mass of explosives used (kg), Fa = Face advance (m), D = Density of rock (t/m^3), A = Area of face (m^2)

The time taken for the dispersion of gas to a threshold level is defined by:

$$t = \frac{V}{Q} \ln\left(\frac{G_c}{G_t}\right) \qquad (3)$$

Where: t = Time to achieve target concentration (s), V = Volume of gas filled space (m^3), Q = Flow rate of fresh air (m^3/s), G_c = Initial gas concentration (ppm), G_t = Gas concentration at time t (ppm)

The explosive statistics of December 2018 were utilised to obtain the number of workplaces, explosive consumptions, and production quantities. Besides ANFO factor and water-gel factor were determined and tabulated in Table 1.

Table 1: Blast statistics of Bogala in December 2018.

Description	Value
ANFO factor (kg/m^3)	0.1707
Water-gel factor (kg/m^3)	0.4506

In addition, the required airflows for the production stopes were calculated from the reverse calculation of the throwback method.

2.6 Adequacy Assessment

Adequacy assessment was performed by correlating the measured, calculated, and simulated parameters with the standard provisions.

Table 2: Threshold limit values (TLV).

Description	TLV
NO_2 concentration (ppm)	3
CO concentration (ppm)	50

3. Results and Discussion

3.1 Ventsim Simulation

Gabriel shaft is the only intake airway to the considered block. The stale air exits to the upper levels through five winzes from 240 FM level. The model was generated considering the conservation of mass and simulated as compressible airflow, which resulted in the deviation between intake and exhaust airflow quantities. The summary of the simulated ventilation network is given in Table 3.

Table 3: Ventilation characteristics summary of simulated Ventsim model.

Description	Value
Total airflow intake (m^3/s)	15.4
Total airflow exhaust (m^3/s)	15.5
Total mass flow (kg/s)	24.3
Equivalent resistance for study area (Ns^2/m^8)	0.21886

3.2 Air Quantity

The adequacy of ventilation parameters was evaluated for each survey station located in the primary airways where the passage of men and haulage of materials occur. According to the standards emphasised in [3], the ventilation requirement for the sustenance of human life is a minimum of 0.01 m^3/s per person. It usually exceeds 0.1 m^3/s per person based on the entire mine network's functions and occasionally surpasses 1.0 m^3/s per person.

The required airflows in the working places are determined to satisfy the respiratory need of workers and maintain the maximum effective temperature, which is 0.1 m^3/s per person and 27° C, respectively. Besides, this study does not consider combustion engine emissions as mechanised equipment operates on compressed air at Bogala.

In light of this, it is found that only three stations: VMP11 at 240 FM level and VMP1 and VMP3 at 275 FM level meet the required quantity, and the rest have insufficient air supplies, as shown in Fig. 3. However, the primary ventilation system does not ventilate the production and development workplaces, resulting in no airflow; instead, compressed air is utilised.

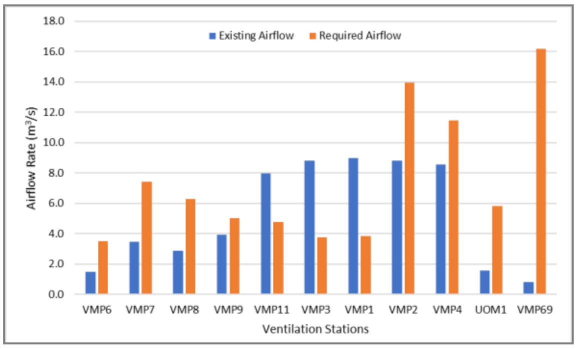

Figure 3: Existing and required airflows at ventilation stations.

Further, the required airflow for a working stope is calculated by considering the air quantity required to dilute the explosive fumes to a threshold value within 75 minutes, as shown in Table 4. In parallel, the sufficient airflow for three workers at a stope is estimated as 0.3 m^3/s, higher than the dilution requirements. Nevertheless, the effective temperature must be taken into account in further studies.

Table 4: Existing and required airflows at production stopes

Description	Value
Existing airflow (m^3)	0.0
Re-entry time between two shifts (min)	75.0
Time taken to fill the stope by the contaminants (min)	10.4
Time to dilute to limit (min)	64.6
Required airflow based on explosives consumption (m^3/s)	0.21
Required airflow based on number of workers (m^3/s)	0.30
Required airflow for a stope (m^3/s)	0.30

Utilising numerical simulation results of Ventsim software to obtain the existing airflow in inaccessible and unmeasurable locations is advantageous to this study. It reveals that airflows through winzes A, C, and D are adequate to supply fresh air to the adjacent stopes, as shown in Fig. 4. However, there is no proper mechanism in practice to make use of the airflows from the winzes.

The simulation unveils two potential threats of mixing fresh and stale air in the 240 FM and 275 FM levels. Stale air from the 275 FM level of the Kumbuk vein is directed to 240 FM level through winzes towards the 'Na' vein and mixes with the fresh air at the 'Na-Kumbuk' crosscut junction.

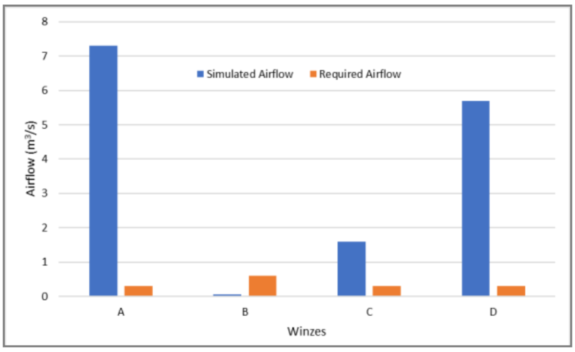

Figure 4: Simulated and required airflows at winzes.

Moreover, the axillary fan at 275 FM level of the 'Na' vein is installed incorrectly and causes recirculation in the same airway. As a result, this airway recorded the mine's highest wet and dry bulb temperature of 32° C and 100 per cent humidity, as shown in Fig. 5 and Fig. 8, respectively.

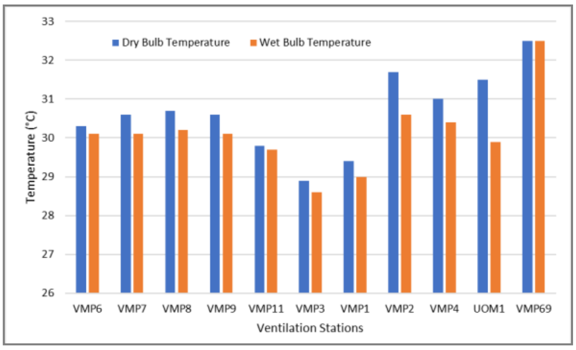

Figure 5: Dry and wet bulb temperatures at ventilation stations.

3.3 Air Velocity

The standards specified in [1] recommend maintaining the air velocity less than 4.0 m/s to avoid dust generation and provide comfort. In addition, the rule of thumb suggests maintaining the air velocity in the drift at 1.0 - 3.0 m/s and the stopes at

2.0 m/s unless it is affected by excessive cold or heat [3].

The air velocity assessment results show the stations: VMP11 at 240 FM level and VMP1 and VMP3 at 275 FM level have the adequate velocity to maintain the effective temperature within the boundary, and all other measured stations failed to satisfy the necessities as illustrated in Fig. 6. In addition, the air velocities at production stopes are nil. The required air velocities at all the stations were determined utilising analytical methods, and this revealed that none of them can re-entrain settled dust as it remains within the maximum limit.

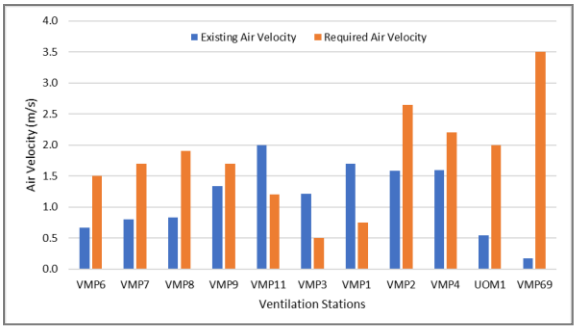

Figure 6: Existing and required air velocities at ventilation stations.

3.4 Effective Temperature

From the analytical calculations for the effective temperature, it is observed that VMP2, VMP4, VMP6, VMP8, VMP69, and UOM1 exceed the prescribed standard of 28° C, as shown in Fig. 7, which may adversely affect the duration of continuous work. Moreover, this study utilises 27° C as the reasonable threshold limit for the effective temperature.

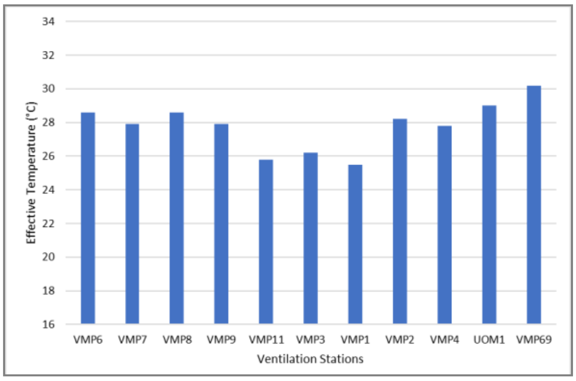

Figure 7: Effective temperatures at ventilation stations.

However, the stations VMP4, VMP7, and VMP9 surpass this limit. Moreover, according to [1], the effective temperature should be maintained lesser than 28° C for continuous work, and the work should be terminated as the temperature rises above 32° C. However, it is possible to lower the effective temperature by increasing the air velocity.

3.5 Humidity

Humidity around all the stations except UOM1 is witnessed higher than 90 per cent, as shown in Fig. 8, and it can cause fog even in the higher effective temperatures. High humidity can affect human comfort and may produce falls of roof and spalling of the sides of airways through physical and chemical reactions between the airborne water and hygroscopic minerals within the strata [1]. Moreover, high humid conditions may result in decay and corrosion as Bogala utilises timber and steel for primary supports. The natural wet condition of the mine and wet drilling practices exacerbate dust problems, though it is the main reason for the high humidity. However, a comprehensive study on dust must be carried out to ensure the quality of air.

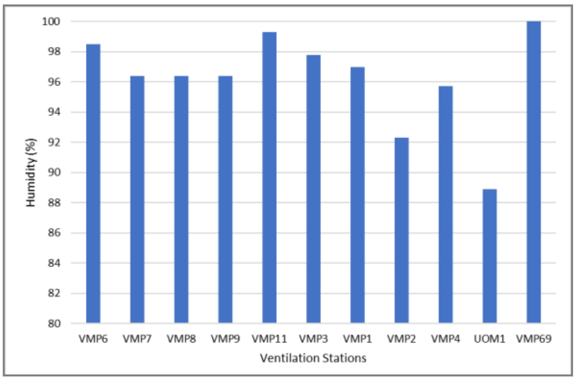

Figure 8: Humidity levels at ventilation stations.

3.6 Air Quality

Gas concentration measurements tabulated in Table 5 imply that the occupational level of O_2, CO_2, and NH_3 are within the acceptable limits in the primary airways.

Table 5: Presence of O_2, CO_2, and NH_3 levels at ventilation stations.

Ventilation Stations	O_2 (%)	CO_2 (ppm)	NH_3 (ppm)
VMP6	20.9	250	0
VMP7	20.9	2050	0
VMP8	20.9	250	0
VMP9	20.9	2050	0
VMP11	20.9	2050	0
VMP3	21.2	250	0
VMP1	20.9	150	0
VMP2	20.9	150	0
VMP4	20.9	150	0
UOM1	20.9	150	0
VMP69	20.9	1350	0

3.7 Re-entry Time

The usage of ANFO and Water-gel as explosives liberates toxic gases into the airways. Therefore, the re-entry time is evaluated using the throwback method at a development drive in a dead-end, and the results are tabulated in Table 6. Accordingly, the available time duration of 75 minutes between two shifts is inadequate to dilute the contaminants to their threshold limits in that drive. Habitually, CO and NO_2 gases are treated separately in calculating the threshold limit of the gas mixture [1]. In light of this, 116 minutes is required to provide a safer environment utilising the existing auxiliary ventilation.

Table 6: Concentration of contaminants after blast and re-entry time.

Description	NO_2	CO
Concentration of contaminant after blast (ppm)	4012.0	3884.9
Concentration of contaminant when filled the airway (ppm)	89.4	86.6
Time to fill the airway (min)	21.3	21.3
Time to dilute to limit (min)	73.8	94.6
Re-entry time (min)	95.1	115.9

5. Conclusion

From the assessment through the analytical and numerical approaches, it can be concluded that the parameters including air quantity, velocity, wet and dry bulb temperature, effective temperature, and humidity in the block between 240 FM and 275 FM levels of Bogala mines are inadequate for optimal mine conditions in most occasions. It may lead to loss of productivity of the workplaces as it has an adverse effect on the health, safety, and comfort of workers and the physical conditions of the mine. Nevertheless, the gas concentrations of O_2, CO_2, and NH_3 in the primary airways meet the regulatory requirements. It has been calculated that the airflow needed for three workers in the stopes is 0.3 m/s. Although winzes carry adequate air quantity, the appropriate mechanism is absent in practice to deliver it into the workplaces. Re-entry time after the blast was calculated as 116 minutes in a development drive at a dead end, and it is higher than the existing time gap between two shifts of 75 minutes. Exposure to these toxic fumes may cause severe illnesses, which is a major concern regarding workers' health. Moreover, a simulated Ventsim model reveals the stale air mixing at the 240 FM level. In addition, it exhibits an air recirculation at 275 FM level as the installed auxiliary ventilation in the dead-end is not exposed to the fresh air intake.

6. Recommendations

Optimising the ventilation network by installing auxiliary fans and dehumidifiers is recommended to provide a comfortable working environment in production stopes and development areas. In addition, automating the mine ventilation through real-time monitoring sensors and three-dimensional (3D) simulations would optimise the dynamically changing mine environment with developments by reducing cost and energy consumption. Moreover, re-entry time after explosions should be accounted for each ventilation design occasion to provide workers with a safe atmosphere.

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Determination of Depleted Rock Volume in Open Cast Mines Using Photogrammetric Techniques for the Purpose of Royalty Calculation

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Abstract

In Sri Lanka, Geological Survey and Mines Bureau (GSMB) imposed royalty levy for aggregate production volume using an equation (indirect method) which utilises the quantity of explosives for the calculation. Since the equation resulted in higher deviations and the previous studies emphasise the advantages of photogrammetric 3-dimensional (3D) modelling (direct method) when determining production volume of bench blast, the present study focus on investigating its applicability to irregular faced dynamic quarry with uneven overburden. Pre and post 3D Digital Surface Models (DSMs) of the quarry were generated using the structure from motion (SFM) algorithm with Real-Time Kinematic (RTK) positioning system and Pix4D mapper software. Golden Software Surfer 16 was used to determine depleted rock volume as the difference between pre and post 3D DSMs. Results indicate a 5.50% deviation of the proposed method from true depleted rock volume determined by truck measurements due to uncleaned quarry face during the pre-Drone survey and unaccounted soil overburden removal. Presence of overburden while generating DSMs can be overcome by pile volume estimation of overburden and decreasing it from depleted rock volume when calculating production volume. GSMB equation calculated production deviate -32% from true production due to the unaccounted explosive amounts which contributed to the production and confirm the suitability of the proposed direct method (5.5% deviation) for determining the depleted rock volume in open-cast mines.

Keywords: Digital surface model, Drones, Pix4D, Real-Time Kinematic positioning

1. Introduction

The mineral wealth of Sri Lanka is regulated by the Geological Survey and Mines Bureau (GSMB). GSMB uses a hybrid system composed of unit-based and value-based systems for royalty calculation of aggregate products [1]. Equation 1 was presented by the Defence Ministry of Sri Lanka and utilised by the GSMB when calculating aggregate yield volume in cubes until 2019 [1].

$$\text{Yield} = 2 \times (a + b) + c \qquad (1)$$

Where,

Yield volume in cubes

a Amount of water gel used (kg)

b Amount of ANFO used (kg)

c Amount of black powder used (kg)

Equation 1 depends on the amount of explosives used for the blast, and its origin have not been proved scientifically [1,2]. Further, Equation 1 does not provide a clear understanding about the impact from the grade of the quarry and bulking of materials. Later in 2019, a new equation (Equation 2) was introduced based on powder factor, bulking factor and explosive amount.

$$\text{Yield} = ((a \times RWS + b)/PF) \times BF \qquad (2)$$

Where,

Yield volume in m^3

a Amount of water gel used (kg)

b Amount of ANFO used (kg)

RWS Relative Weight Strength of water gel

PF Powder factor (kg/m^3)

BF Bulking factor (1.6)

The relative weight strength of watergel was taken as 1.2 in Equation 2. Moreover, factors like the Industrial Mining Licence (IML) category of the quarry, drilling depth, borehole diameter, production and explosive amount were considered when allocating powder factors as stated in GSMB Circular no. 189/01/2021. However, bulking factor was taken as a universal constant which equals to 1.6, regardless of fragmentation, particle size and blasting parameters.

Jayawardana et al. [1] and Perera et al. [2] recommended deploying photogrammetric 3D modelling to calculate the depleted (production) volume of quarries instead of explosive based equations since the results obtained by Drone surveys were deviating -5% from true depleted volume. However, they [1], [2] conducted Drone surveys for bench blasting, and the applicability of the method for the entire quarry was not examined. Besides, the applicability of photogrammetric 3D modelling for small scale quarries with extremely irregular quarry face is unknown. Therefore, the applicability, pros and cons and solutions for encountered complications when employing photogrammetric 3D modelling to a dynamic quarry having irregular quarry face with a weathered rock and soil overburden have been investigated in this study.

2. Methodology

True depleted rock volume was determined using truck measurements compared with depleted rock volume determined by the Drone survey to validate the applicability of photogrammetric 3D modelling in determining aggregate yield volume for the purpose of royalty calculation.

2.1 Site Selection

The study was carried out by selecting an IML C category quarry which is located in Bathalawatta, Dambugolla, Ambepussa in Sri Lanka. Irregular faces, unavailability of benches, topsoil overburden with weathered rock layer are the main features identified. Explosive usage and production were monitored for a time period of four months.

2.2 Determination of depleted rock volume using truck measurements

In-situ production volume and in-situ waste rock volume were calculated using truck weights and truck volumes, respectively. Depleted rock volume was calculated by adding in-situ production volume and in-situ waste rock volume. Depleted rock volume determined by this method was considered as the true value.

2.2.1 In-situ production volume

Truck weights of blasted rocks (tons) were measured to determine the in-situ production volume. The specific gravity of the rock samples obtained from the quarry site were tested in the laboratory to estimate the average density of the rock (kg/m^3), and in-situ production volume (m^3) was calculated using Equation 3.

$$\text{Volume} = \frac{\text{Tonnage hauled x 1000}}{\text{Density of rock}} \qquad (3)$$

2.2.2 In-situ waste rock volume

Truck volumes of bulk waste rocks (cubes) were recorded and converted to in-situ waste rock volume (m^3), assuming the bulking factor as 1.6.

2.3 Determination of depleted rock volume using aerial photogrammetry

Aerial images can be used in generating the 3D model of an object and obtain accurate measurements such as distance, area and volume. This method can be used to obtain a 3D DSM of the quarry.

2.3.1 Drone surveying for 3D modelling

In this study, pre and post Drone surveys were conducted for the purpose of generating 3D DSMs.

2.3.2 Data acquisition

DJI Phantom 4 Pro Drone was used to obtain the aerial images. Specifications of the Drone are indicated in Table 1.

Prior to conducting the Drone flight, Ground Control Points (GCPs) were marked on permanent structures or places that would not be removed until the next survey was conducted. Coordinates of each GCPs were recorded using a highly accurate RTK positioning system during the pre-Drone survey.

A drone survey was conducted after selecting a suitable height, ground sampling distance and grid pattern to cover the project area.

2.3.3 Data processing

Data processing was done using two software viz.

1. Pix4D mapper
2. Golden Software Surfer 16

Aerial images of pre and post surveys were processed using the Pix4D, and georeferenced DSMs were generated (Spatial resolution 2.84cm per pixel). Pre and post-survey DSMs were developed in Surfer software in colour relief format, and the area of which the volume should be calculated was demarcated. Depleted rock volume was determined as the difference in volumes of the same area in pre and post-survey DSMs.

Table 1: Specifications of the Drone.

Model	DJI Phantom 4 pro
Weight	1388 g
Max speed	P-mode: 31 mph (50 kph)
Max flight time	Approx. 30 minutes
Satellite positioning systems	GPS/GLONASS
Max wind speed resistance	10 m/s
Operating temperature range	32° to 104° F (0° to 40° C)
Camera sensor	1′′ CMOS Effective pixels: 20M
Lens	FOV 84° 8.8 mm/24 mm (35 mm format equivalent) f/2.8 - f/11 auto focus at 1 m - ∞
Gimbal stabilization	3-axis (pitch, roll, yaw)
Battery capacity	5870 mAh
Battery type	LiPo 4S
Voltage	15.2 V
Remote controller operating frequency	2.400 - 2.483 GHz and 5.725 - 5.825 GHz
Max transmission distance	7 km

2.4 Determination of yield volume using equations utilised by the GSMB

Explosive quantities consumed during the study period were recorded and used in Equations 1 and 2 to determine the yield volumes.

3. Results

3.1 Calculation of depleted rock volume using truck measurements

3.1.1 Density of the rock

The specific gravity of rock samples (Table 2) was obtained by performing laboratory tests based on Archimedes' principle, and the average density of the rock was derived, assuming the density of distilled water as 1000 kg/m^3.

Table 2: Specific gravity and average density of rock.

Sample	Specific gravity	Average density (kg/m^3)
1	2.67857	2671.11
2	2.66154	
3	2.68148	
4	2.65957	
5	2.67442	

3.1.2 Calculation of in-situ production volume

Production values (tons) were obtained by measuring truck weights of the blasted rocks with the use of a weighbridge (Table 3).

Table 3: Tonnage of blasted rock.

Period	Production (tons)
11th March – 31st March	3155
April	2850
May	4855
June	1800
1st July - 17th July	2150
Total Production (tons)	**14810**

The average density of the rock (Table 2) and total production tonnage (Table 3) were substituted in Equation 3 to compute the in-situ production volume as 5544.51 m^3.

3.1.2 Calculation of in-situ waste rock volume

Waste rocks were not weighted, but instead, broken waste rock volume was measured using truck volumes (cubes) and converted to in-situ waste rock volume. The bulking factor was assumed as 1.6 (as declared by the GSMB) (Table 4).

Table 4: Waste rock volumes.

Period	Waste rock volume	
	Bulk (Cubes)	In-situ (m^3)
11th March – 31st March	51	90.24
April	27	47.77
May	48	84.93
June	21	37.16
1st July - 17th July	18	31.85
Total	**165**	**291.95**

In-situ production volume and in-situ waste rock volume were added to determine the depleted rock volume by truck measurements (Table 5).

Table 5: Volumes calculated by truck measurements.

In-situ production volume (m^3)	In-situ waste rock volume (m^3)	Depleted rock volume (m^3)
5544.51	291.95	5836.46

3.2 Calculation of depleted rock volume using Drone surveys

Figure 1 illustrates the generated DSM of the pre-Drone survey. The demarcated boundary of pre and post-survey models in colour relief format generated using Golden Software Surfer 16 (given in Figure 2 and Figure 3, respectively.)

Volumes calculated from the surfer software are presented in Table 6. Volume of the post survey was subtracted from the pre survey to determine the depleted rock volume (in-situ) by Drone surveys.

Figure 1: DSM of the pre-Drone survey.

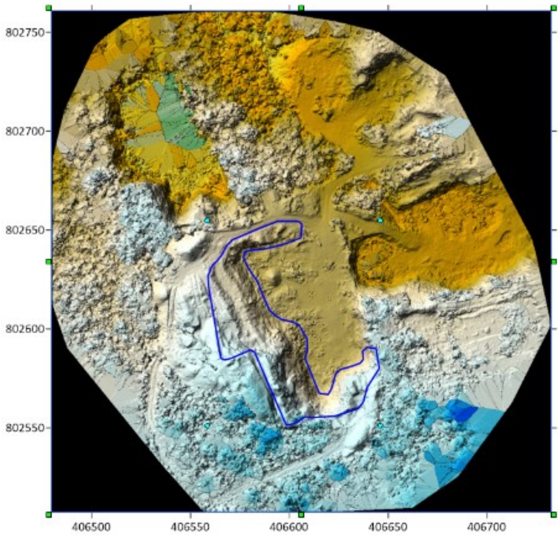

Figure 2: Pre-Drone survey DSM colour relief format with demarcated boundary.

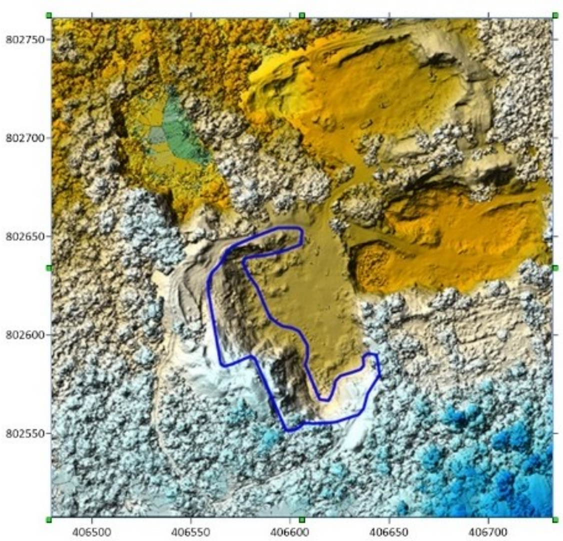

Figure 3: Post-Drone survey DSM colour relief format with demarcated boundary.

Table 6: Volumes calculated by surfer software.

Pre survey volume (m^3)	Post survey volume (m^3)	Depleted rock volume (m^3)
56403.70	50246.16	6157.54

3.3 Calculation of production volume using explosive based equation

Utilised explosive quantities (Table 7) recorded during the study period were used in Eq1 to calculate bulk production volume by explosive based equation. The bulking factor was assumed as 1.6 when converting bulk volume to in-situ volume (Table 8).

Table 7: Explosive utilisation during the study period.

Explosive	Amount (kg)
Water gel	92
ANFO	865

Table 8: Production volume calculated by explosive based equation.

Bulk volume (cubes)	In-situ volume (m^3)
1914	3386.58

3.4 Calculation of production volume using powder factor based equation

Similarly, utilised explosive quantities (Table 7) during the study period were used in Equation 2 to calculate bulk production volume by powder factor based equation. According to the selected quarry type, PF and BF in Eq2 were taken as 0.259 kg/m^3 (based on GSMB Circular no. 189/01/2021) and 1.6, respectively. Estimated bulk production volume was divided by BF to determine in-situ production volume calculated by powder factor-based equation.

Table 9: Production volume calculated by powder factor based equation.

Bulk volume [m^3]	In-situ volume [m^3]
6025.63	3766.02

4. Discussion

Depleted rock volumes determined by truck measurements (M1) and Drone surveys (M2) are tabulated in Table 10. M2 shows 5.50% deviation from M1 (true value) (Table 10). Unaccounted removal of weathered rock overburden during study period resulted in volume reduction of M1 as well as the presence of blasted rock on quarry face during pre-Drone survey resulted in volume exaggeration of M2. Uncleaned blasted rock (bulk volume) on quarry face was identified as the in-situ volume when determining M2. Deviation of M2 can be minimised by clearing the quarry face before Drone surveys and, in addition, by improving the resolution.

Table 10: Comparison of depleted rock volume calculated by each method

Method	Depleted rock volume (m^3)	Absolute deviation (m^3)	Relative deviation (%)
M1	5836.46	-	-
M2	6157.54	321.08	5.50%

M1: Depleted rock volume determined by truck measurements (true value)

M2: Depleted rock volume determined by Drone surveys

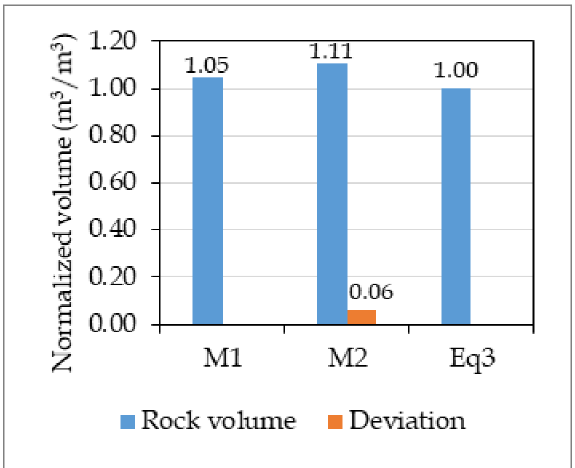

Figure 4: Depleted rock volume comparison (refer Table 10 for M1 and M2; Table 11 for Equation 3)

Figure 4 illustrates the depleted rock volumes obtained by truck measurements (M1) and Drone surveys (M2) normalised by production volume (5544.51 m^3) calculated using tonnage (Equation 3). Figure 4 indicates that 1.05 m^3 of rock should be excavated to produce 1 m^3 of aggregate due to the weathered rock layer existing in the studied quarry. Moreover, M2 shows 0.06 m^3/m^3 deviation (discussed 5.50% relative deviation, Table 10) with reference to M1 as a result of preventable reasons (Figure 4).

Table 11: Comparison of production volume calculated by each equation.

Equation	Depleted rock volume (m^3)	Absolute deviation (m^3)	Relative deviation (%)
Eq1	3386.58	-2157.93	-39
Eq2	3766.02	-1778.49	-32
Eq3	5544.51	-	-

Eq1: Production volume calculated by explosive based equation

Eq2: Production volume calculated by powder factor based equation

Eq3: Production volume calculated by tonnage (true value)

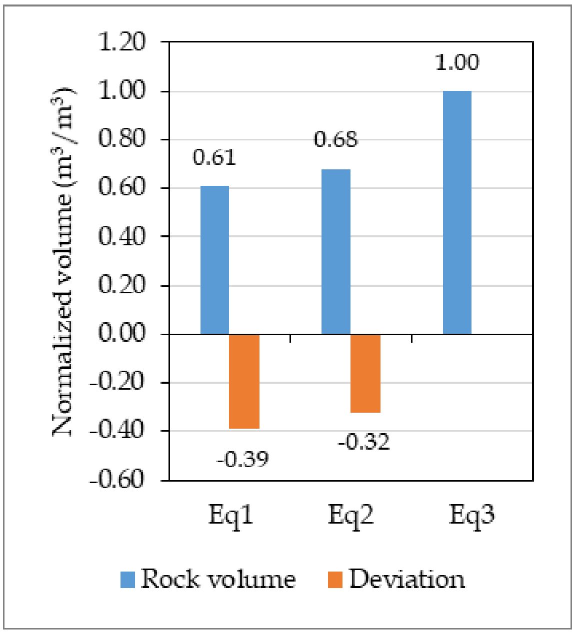

Figure 5: Production volume comparison (refer Table 11 for Eq1, Eq2 and Eq3)

Table 11 includes the production volumes calculated by explosive based equation (Equation 1), powder factor based equation (Equation 2) and directly estimated tonnage (Equation 3). Equations 1 and 2 show -39% and -32% relative deviations from Equation 3 (true value) respectively (Table 11). The relative deviation of Equation 1 from Equation 3 was -18% and -8% - +34% when computing the results obtained by Jayawardana et al. [1] and Perera et al. [2], respectively. Relative deviations obtained by the present study indicate intense negative value compared to previous studies [1, 2]. The reason for such intense deviation was due to the presence of uncleaned blasted rock on the quarry face, which resulted from blasts conducted prior to the pre-Drone survey. The weight of said uncleaned blasted rock was counted to tonnage (Equation 3) since it was removed from the quarry as production during the study period. However, the used explosive amount for the said uncleaned blasted rock was not counted to explosive utilisation during the study period (Equations 1 and 2) since those blasts were conducted prior to the pre-Drone survey.

Figure 5 shows the production volumes obtained by explosive based equation (Equation 1) and powder factor based equation (Equation 2) normalised by production volume (5544.51 m^3) calculated by tonnage (Equation 3). Equation 2 exhibit a 0.07 m^3/m^3 increase than Equation 1. However, production volume calculated by Equations 1 and 2 deviates from the true value (Equation 3).

Equations 1 and 2 depends on the utilised explosive amount. Although these equations calibrate to calculate quarry production, they may not derive quarry production accurately during a specified period, possibly due to unaccounted explosive amounts which contributed to the production, similar to the situation encountered during the present study. Conversely, depleted rock volume determined by Drone surveys indicates an acceptable correlation with the depleted rock volume determined by truck measurements (true value) (Figure 4), validating the applicability of the proposed Drone-based method to calculate the depleted rock volume in quarries. In addition, a Drone survey consists of less data acquiring time and straightforward data processing. Further, volume calculation using DSM is an accurate and efficient method that can deploy to determine depleted rock volume in quarries.

Inability to calculate the depleted overburden volume directly by generating pre and post-survey DSMs counted as the disadvantage of this method. However, collected and piled overburden volume can be estimated using a Drone survey [1]. Thus, the disadvantage of the method can be eliminated by utilising the pile volume determination method, which needs training and expertise on drone surveys.

5. Conclusion

Results obtained by the study confirm the suitability of the suggested Drone-based methodology for calculating the depleted rock volume in open-cast mines for the purpose of Royalty calculation.

Furthermore, the present study confirmed the inapplicability of equation-based methodology, which cause significant production volume deviations.

Problems encountered during the implementation of the proposed methodology can be eradicated by following the solutions recommended in the study. Finally, the current loss of revenue to the national economy due to erroneous estimation of aggregate production can be reduced significantly by implementing the suggested method.

6. Recommendations

If the proposed method is selected to determine the aggregate production volume by the GSMB, the following precautions should be adopted as standards to acquire true in-situ rock DSM.

- Quarry face should be free from loosen rock prior to Drone survey.
- Quarry floor should be cleared at least 3 m distance from the toe.

Moreover, frequent Drone surveys are recommended for accurate results.

The applicability of the proposed methodology to other types of open cast mines (such as limestone, quartz and feldspar) should be examined further.

Acknowledgement

The authors wish to extend their sincere gratitude to all academic and non-academic staff of the Department of Earth Resources Engineering in the University of Moratuwa for their assistance throughout this study. Special thanks are also due to Mr. Nalin De Silva, Director (Mineral Survey) and Mr. SB Harankahawa, Geologist of GSMB, who gave immense support for this research providing us with necessary assistance. We would also like to express our sincere gratitude to Eng. HND Padumasiri, the Mining engineer of the quarry and all the staff in the quarry for providing their workplace to do our research.

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Session III

Environmental Engineering

Decision Tree Regression Approach for Detecting Spatiotemporal Changes of Vegetation Cover in Surface Water Bodies

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Surface water bodies in urban areas, such as Bolgoda lake, show complex vegetation dynamics, typically noticeable by the fluctuating vegetation cover throughout the year. Primary factors governing these fluctuations include wastewater discharge, anthropogenic activities (e.g., surface mining), invasive plant growth, and climate change. It is exceptionally challenging to physically measure and monitor these dynamics over the spatial extent of these waterbodies consistently over many years. Recent studies have explored the potentials of employing satellite imagery to quantitatively detect spatiotemporal changes of surface water vegetation cover. Such attempts have utilised vegetation detection indices, such as the normalised vegetation index (NDVI), to classify the vegetation cover with significant statistical accuracy. However, these conventional geospatial analyses require substantial computational power. They are limited to small timescales and spatial extents. This study employs the computational power of the google earth engine to address this limitation. Moreover, it integrates a machine learning classification approach, namely decision tree regression, to monitor the vegetation cover change over coarser and finer temporal resolutions using Landsat 8 hyperspectral imagery. Initially, NDVI classification was performed on 390 Landsat 8 images acquired throughout 2013-2021. Five locations, which represent different vegetation cover characteristics on the lake, were selected to generate the time series of the NDVI classified values. The results show that the vegetation cover varies at two temporal frequencies. The annual variation of the water, vegetation, and non-vegetation classes are undetectable. However, vegetation dynamics fluctuate rapidly at a finer temporal resolution (i.e., on monthly cycles). The statistically significant results claimed in this study will be further explored to support policymakers in optimising environmental resource management strategies and prioritising eco-preservation that can enhance the health and productivity of urban surface water bodies.

Keywords: Google Earth Engine, Decision Tree Regression, Invasive plants, Surface water bodies

Assessment of the Effect of Aruwakkalu Waste Dump on Surrounding Water Resources

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Abstract

In developing countries like Sri Lanka, waste management is a vital necessity. Solid waste dumps play a major role in handling waste in most countries. These solid waste dumps can cause severe environmental pollution via leachate generation and transport of toxic material along with both surface and underground water flows. The Aruwakkalu waste dump, which is the subject of the study, is a sanitary landfill situated in Puttalam district, Sri Lanka. The landfill is planned to receive and store municipal solid waste (MSW) from Colombo metropolitan area. Locations for these waste dumps must be selected strategically to minimise the risk of contamination of the surrounding environment. A distributed hydrological model was used to determine the effect of rainfall, evapotranspiration, and surface runoff on the site. It identified the direction of the water flow through the waste dump. The analyses have illustrated that the area receives lesser rainfall and a higher rate of evapotranspiration. The hydrological analysis illustrates the water flow direction from the waste dump to the outside is towards the West and away from the nearby settlement areas. The results were used to assess the effect of the waste dumpsite on the surrounding water resources.

Keywords: Evapotranspiration, Flow direction, Hydrology, Landfill, Rainfall

1. Introduction

Solid waste is one of the leading undisputable problems in most developing countries. With increasing population and urbanisation, the management of solid waste is getting complicated day by day [1].

Solid waste is considered as any non-flowing/non-fluidic substance which has no use or no immediate economic demand. Municipal solid waste (MSW) can be generated via various human aspects such as domestic, hospital, industrial and agricultural activities [1]. The quality and the quantity of the MSW depends on many factors such as population, lifestyle, food habits, standards of living, the extent of industrial and commercial activities, cultural practices, and climate. The collection, transportation, and disposal of MSW in unscientific ways caused chaotic results in many countries [2].

One of the main means of waste disposal is by sanitary landfills, where waste materials are spread over the land surface and covered and compressed by soil. These landfilling sites can cause surface water as well as groundwater pollution unless proper management and monitoring are done [3].

Solid waste in these landfills causes environmental pollution as well as water pollution both on the surface and underground. Direct precipitation on the site and surface runoff through the site can carry the waste unless the dumpsites are properly designed [4]. Precipitated water can be easily mixed with waste materials in the dumpsite and produce a mixture called leachate. This mixture can access nearby surface water flows and groundwater aquifers unless proper controlling measures are implemented [5].

In Sri Lanka, the Aruwakkalu sanitary landfill located in Puttalam district is one of the recently designed MSW dumpsites, which is at the final stages of completion. The people living in the vicinity of the dumpsite consume well water as their primary water source. The location of the dumpsite and the effect of surface water runoffs are critical factors which determine the degree of impact on waste dump contaminants on these settlement areas.

Runoff modelling and hydrological analyses are commonly used to determine the effect of precipitation, evapotranspiration, and water flow direction in an area. These methods can illustrate the nature of the water flow and the potential changes of the hydrology of a site utilising key factors like precipitation, temperature, elevation, and surface water flow.

This research study is mainly intended to determine the nature of the surface flow along the waste dump area and to determine the possibility of water contamination of the nearby settlements via surface runoff.

Figure 1: Study area location map.

2. Methodology

2.1 Study Area

The Aruwakkalu sanitary landfill is situated in Puttalam District, Northwestern province of Sri Lanka. Upon completion, it is expected to receive 1600 metric tons of waste daily. The waste dump is planned to store more than 4.7 million m^3 solid waste in the coming ten years [6].

As illustrated in Figure 1, the waste dump location is in the proximity of the Puttalam lagoon, and there are several settlement areas like Sewarakkuli, Karaitive, Periyanaga Villu etc. The area receives a total average annual rainfall of 2500 mm, and the temperature ranges from 23° C to 34.4° C [6].

2.2 Project Flow

The flow of the project can be logged into four parts.

1. Data acquisition
2. Model setup
3. Hydrological analysis
4. Analysis of results

2.3 Data Acquisition

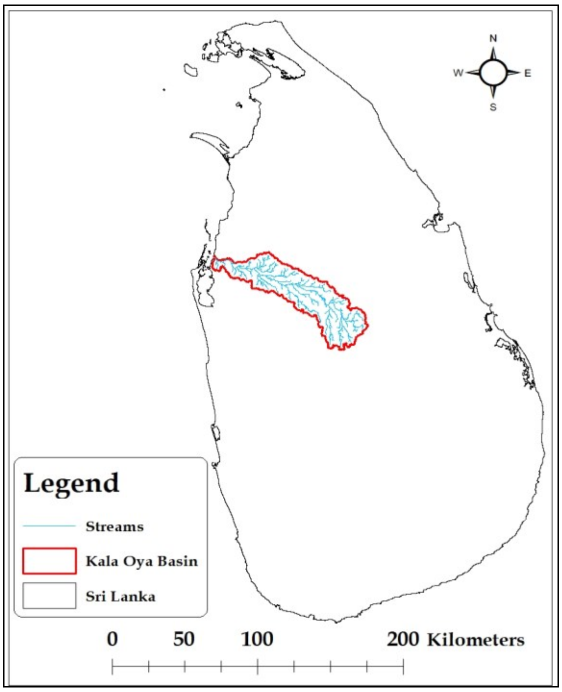

Figure 2: Locations of the Kala Oya Basin in Sri Lanka.

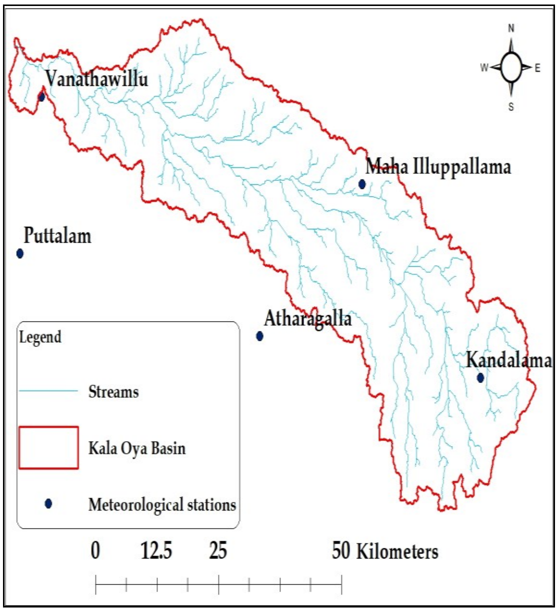

Figure 3: Locations of meteorological stations relative to the Kala Oya basin.

Data for the surface runoff model was collected as per the requirements from the Department of meteorology, Sri Lanka. As illustrated in Figure 3, daily rainfall data and maximum and minimum temperature data of 5 stations for the selected Kala Oya basin were acquired. The location of the Kala Oya basin in Sri Lanka is illustrated in Figure 2, and the selection of the Kala Oya basin was made considering the location of the Aruwakkalu waste dump.

Digital Elevation Model (DEM) files with 12.5 m resolution and 30 m resolution were acquired from the USGS and ASF archives.

2.4 Model Setup

The model setup consists of the preparation of the surface runoff model using rainfall and evapotranspiration maps, and the model inputs are given in Table 1.

The model is designed to determine the variations of precipitation and evapotranspiration of the Kala Oya basin.

The model is employed with a self-written Fortran program which consumes daily rainfall, daily maximum, and daily minimum temperature data as main inputs. Using the Hargreaves equation, the potential evapotranspiration is determined. 12.5 m resolution digital elevation file was obtained from ALOS PULSAR satellite. The DEM file was resampled into 100 m resolution.

Table 1: Input data for the model

Required Data	Source(s)
Daily rainfall (2008-2010)	Department of meteorology
Daily maximum and minimum temperature data (2008-2010)	Department of meteorology
Elevation data (DEM)	USGS SRTM digital elevation model
Catchment boundary and terrain	SRTM and ALOS PULSAR digital elevation models
Soil data	World soil map and literature
Land use map	Sentinel 2 satellite image classification via Arc Map 10.5

Data from 5 meteorological stations from 2008 to 2010 were used to map the rainfall and evapotranspiration of the basin via the inverse distancing method. The location of each station with relative to the basin was given using a latitude grid of the same size (1063 columns and 701 rows).

All these above parameters and procedures were included in the self-written Fortran code. The outputs of precipitation and evapotranspiration were in binary format. They were visualised using Envi Classic 5.0 and ArcMap 10.5, and the variations were studied.

The location of the Aruwakkalu waste dump site is in the vicinity of the selected basin. Using the variations and trends observed within the basin, the impact of precipitation and evapotranspiration on the surface water flow along the waste dump was determined.

2.5 Hydrological Analysis

The hydrological analysis was carried out using the digital elevation models (DEM) via Arc Map hydrological Analysis tool. The DEM files were used to create flow accumulation of the area. Flow direction file was created using the flow accumulation to determine the direction of the water flow through the area.

A buffer zone of 1 km around the Aruwakkalu waste dump was extracted from the original flow direction file. A threshold value of 2000 was used to enhance the water flow paths of the area.

Together with the results of the hydrological analysis and the digitised location map of the surrounding settlements, the degree of water contamination of the nearby villages was determined.

The location of the dumpsite within the basins and the elevation of the dumpsite location relative to the surrounding areas were further analysed to determine the water flow direction and the effect of water flow on the location.

3. Results

3.1 Results of the Model

The model was created by the self-written Fortran code, and the resulted precipitation and evapotranspiration files were in binary format. They can be visualised via Envi classic 5.0 and Arc Map 10.5 software. Monthly variations of years from 2008 to 2010 were taken as outputs for analysis of the effect of surface running water along with the waste dump site.

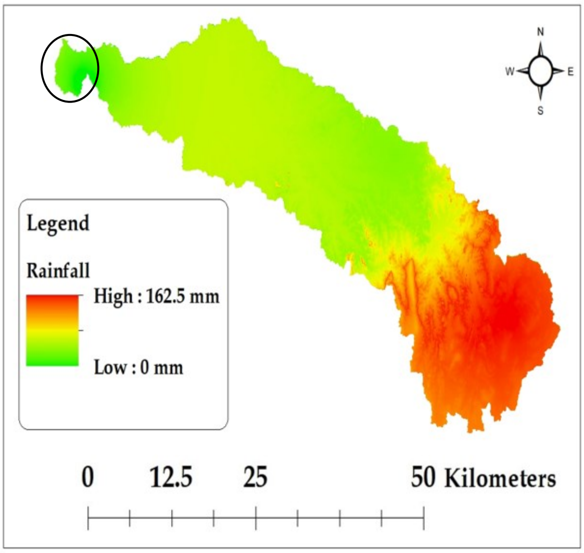

Figure 4: Rainfall map for January 2008.

The circled area of Figures 4 and 5 is the area extent where the waste dump is located within the basin.

Figure 4 and Figure 5 are the resultant rainfall and evapotranspiration maps generated from the Fortran code for the month of January 2008. A total of 72 files for precipitation and evapotranspiration were output by the model, and they can be visualised as GIF files to illustrate the variations.

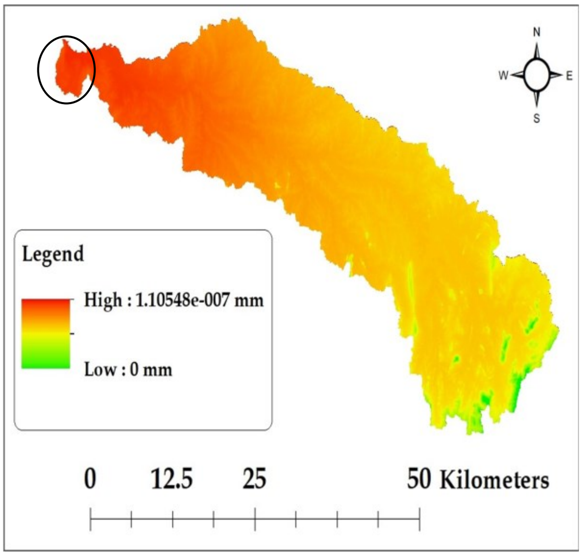

Figure 5: Evapotranspiration map for January 2008.

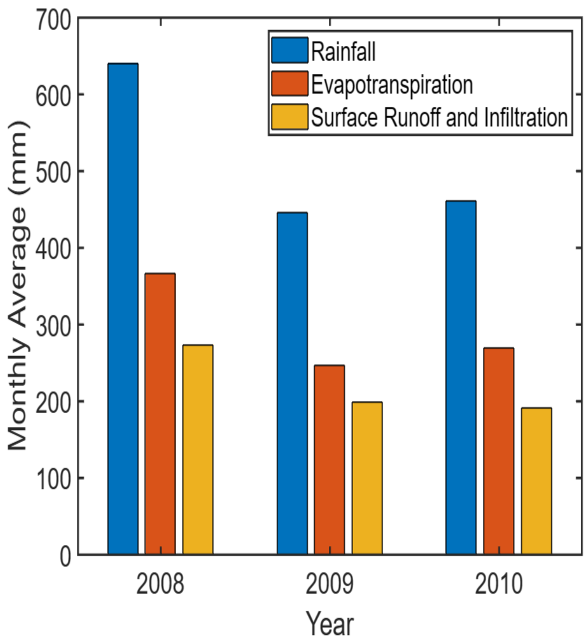

Figure 6: Average monthly rainfall variations, evapotranspiration, surface runoff and infiltration.

The average monthly precipitation, evapotranspiration, surface runoff and infiltration are graphically illustrated in Figure 6.

3.2 Results of Hydrological Analysis

The result of the hydrological analysis is illustrated in Figure 7.

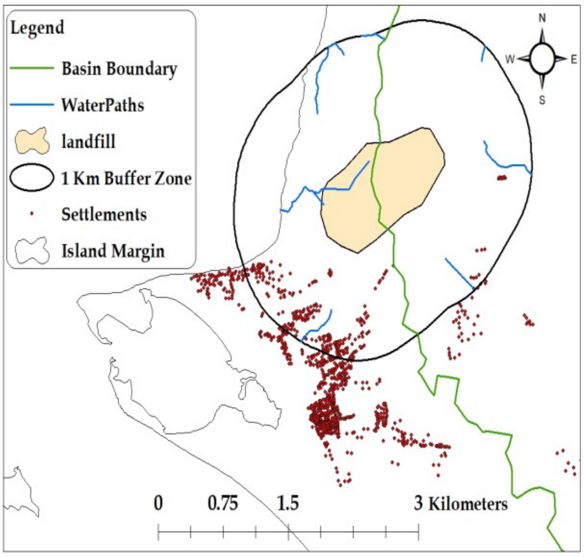

Figure 7: Hydrological map of the Aruwakkalu waste dump area.

4. Discussion

The modelling process was carried out to the Kala Oya basin, and it was observed that the location of the Aruwakkalu waste dump is in the boundary regions of the Kala Oya basin and the basin which is situated in the south of the Kala Oya basin.

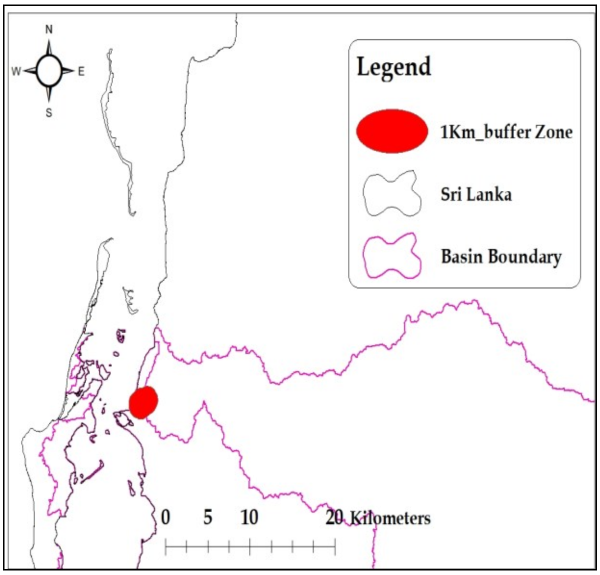

Figure 8: Position of the waste dump and 1 km buffer zone on two basins.

In the hydrological analysis illustrated in Figure 7, water paths through the waste dump area and the buffer zone were determined using a very low threshold value of 2000. This value was selected as there was no major or significant water flow through the area under higher threshold values which are commonly used to generate major water networks in basins.

The elevation of the waste dump area was also considered in the study to further determine the water flow direction and to analyse the position of the waste dump relative to the surrounding areas.

The precipitation and evapotranspiration models, which are created as stacks starting from Figure 4 and Figure 5 clearly indicate that the area that belongs to the Aruwakkalu waste dump (circled area) receives lesser precipitation and is subjected to higher evapotranspiration than the average values of the basin. Figure 6 also illustrates that more than 55% of the average annual rainfall is removed from the area as evapotranspiration. This affects the surface runoff of the area as the generation of runoff is lesser with low-intensity rainfall and high-intensity evapotranspiration.

As illustrated in Figure 8, the location of the Aruwakkalu waste dump is in the boundary areas of the two basins. Also, the location is in low elevation than the surrounding areas. The amount of running water through the area is minimum due to the location at basin boundaries, and as illustrated in Figure 7, the flow of the surface water is towards the lagoon and away from the nearby settlements.

The hydrology analysis clearly illustrates that the water flow gradient extends away from the surrounding settlements within the 1 km buffer zone. The effect of contamination of water sources through surface running water is minimum.

The EIA report [6] reveals that according to the boring tests carried out, the groundwater level fluctuates between –0.05 and -6.85 meters. Thus, the bottom foundation level of the sanitary landfill is higher than that of the groundwater. But in case of a rise in the groundwater table during the rainy season, a bottom liner system is placed to drain the groundwater from the site [6].

The development of the floor of the waste dump is carried out with layers of in situ soil and bentonite. Further, the HDPE sheet layer is used to reinforce the floor by reducing the permeability [6].

According to the EIA report, there is no history of major floods in the area. There were several minor flooding that occurred up to 3 MSL contour levels. Since the waste dump is located above 5 MSL, there will be a lesser impact from flooding in the future [6].

Accumulated drainage flows along minor undulations, and valleys in the area have formed rills causing erosion and fines washout in the sloping ground areas. These flows are finally diverted to the abandoned quarry pits in the North-west and Northside boundaries of the waste dump, and stagnant water is presumed to remain for a short duration after rainfall events before vanishing due to seepage losses and direct and soil evaporation. The average annual reference evapotranspiration and potential soil evaporation are relatively high in this arid region [6].

According to the borehole tests carried out, the groundwater levels were not encountered at any borehole pit, even though the bored pits were driven up to a maximum depth of 33.45 m and 58.00 m, respectively [6].

5. Conclusions

From the results of the model obtained from the hydrological model, it can be cognitively concluded that more than 55% of precipitation in the Kala Oya basin goes out as evaporation and transpiration. The rest is turned to surface runoff and infiltration.

Further, it can be concluded that the lagoon area is more vulnerable than the water sources in the settlement areas in case of a possible contaminant leakage from the waste dumpsite.

The findings of the EIA report of the Aruwakkalu waste dump conclude that the effect of pollution from the waste dump to the surrounding surface water as well as groundwater environment is less.

Following this study, further investigation of the groundwater profile of the area should be conducted to determine the groundwater flow trends of the area. Further, a detailed groundwater quality assessment should be conducted in both upstream and downstream areas of the waste dump, covering both the wet season and dry season of the year.

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Characterisation of Coal Fly Ash for Potential Wastewater Treatment Opportunities

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Abstract

Lakvijaya coal power plant, which is in Norochcholai, Puttalam District, Sri Lanka, is the largest coal power plant in Sri Lanka. It annually produces about 150,000 tonnes of fly ash (FA) as waste, and the management of it is a pressing concern. Various studies have been conducted to valorise FA in a useful manner without simply dumping it into the nearby lands. To this end, we propose the utilisation of FA in wastewater treatments to adsorb heavy metal ions in wastewater to promote environmental sustainability. The direct application of FA for that purpose may be questionable due to the contaminants present in the FA; however, it is reported commercially. Therefore, we attempt to initiate a pre-processed preparation route to ensure the leachate contains fewer contaminants compared to the use of raw FA. Washing is a commonly used preliminary pre-processing step, though it is not studied extensively in the literature with reference to the FA. Herein, we aimed to study the effects of the number of washing cycles and temperature on the characteristics of the FA. With the results, we could conclude that washing is an effective means of pre-processing to alleviate the contaminants of FA, *en route* for wastewater treatment.

Keywords: Preliminary separation, Washing cycles, Particle size distribution

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1. Introduction

Coal is a fossil fuel, and it mainly consists of Carbon. There are four coal types: bituminous, sub-bituminous, lignite and anthracite. It is estimated that there are about 1070 billion tonnes of coal worldwide in 2019. Coal is used in power generation, steel production, cement manufacturing, and heating purposes. Coal plays a significant role as a source of power generation. Globally, 41% of all the sources used in power generation is coal [7].

Coal combustion is the burning of coal in the air to liberate thermal energy. This process is used to generate steam in electric power plants. FA, bottom ash, boiler slag, and flue gases are generated as by-products of coal-burning [6]. In this work, we focus on FA, and it constitutes 64% of the total by-product generation in coal combustion [3]. Though there is a utilising amount of FA, the high FA generating rate has become a huge problem in most countries due to the significant gap between the FA generation and utilising amount. The excess production of FA may cause excessive costs, groundwater contamination, air poisoning, and several health issues, especially in the respiratory system. However, the usage of FA in wastewater treatments to adsorb heavy metal ions is a useful application of FA.

SiO_2 and Al_2O_3 are present in the structure of FA as functional oxidised groups. The surface of silica (SiO_2) has a high affinity towards metal ions. The central ion of silicates (Si^{4+}) has a very strong affinity for electrons; therefore, the oxygen atoms that are bound to the silicon ions have low basicity, making the silica surface act as a weak acid. The oxygen atoms on the silica surface are free to react with water, forming surface silanol (SiOH) groups. The acidity of the silanol (SiOH) groups determines the dependence of the charge of the silica surface on pH. At low pH, a positively charged silica surface results, and at high pH values, a negatively charged surface prevails. Alumina and iron also show the same phenomenon of developing positive or negative charges depending on pH [4].

However, the direct application of FA for the adsorption of heavy metal ions may not give sufficient efficiency. When using FA in the adsorption of heavy metal ions, different pre-processing steps have been reported. FA has been subjected to many types of preliminary treatments such as mechanical activation, chemical modification, impregnation, and thermal or heat treatment (calcination at high temperature) for better performance and to get the required formulation for selected wastewater treatment technique [5].

In this study, we aimed to find the effects of the number of washing cycles and temperature of FA, concerning the particle size distribution (PSD) of residue and pH value and conductivity variations in the effluent.

2. Materials and Methodology

2.1 Sample Preparation

The FA samples used in the study were collected from the Lakvijaya coal power plant in Norochcholai, Puttalam District, Sri Lanka. The coal used in the Lakvijaya plant is imported from Indonesia [2]. The type of that coal is bituminous. The FA type which is generated from the burning of bituminous coal type is class F [1].

Table 1: Composition of FA – Sample A

Mineral ash analysis (Dry Basis)	Unit	ASTM Standard	Results
SiO_2	%	D4326	55.38
Al_2O_3	%	D4326	23.96
Fe_2O_3	%	D4326	8.12
CaO	%	D4326	4.96
MgO	%	D4326	1.26
Na_2O	%	D4326	0.38
K_2O	%	D4326	1.26
P_2O_5	%	D4326	0.33
TiO_2	%	D4326	1.18
Mn_2O_4	%	D4326	0.18
SO_3	%	D1757	2.28
Undetermined	%		0.71
Total	%		100

Table 2: Composition of FA – Sample B

Mineral ash analysis (Dry Basis)	Unit	ASTM Standard	Results
SiO_2	%	D4326	49.88
Al_2O_3	%	D4326	29.22
Fe_2O_3	%	D4326	2.83
CaO	%	D4326	8.55
MgO	%	D4326	1.84
Na_2O	%	D4326	0.18
K_2O	%	D4326	0.68
P_2O_5	%	D4326	1.95
TiO_2	%	D4326	1.52
Mn_2O_4	%	D4326	0.05
SO_3	%	D1757	3.10
Undetermined	%		0.2
Total	%		100

FA sampling was carried out according to the D2234 and D7430 standards in ASTM. Two samples were collected during the entire discharging from the vessel. Both FA samples were analysed using X-ray Fluorescence (XRF) equipment to obtain the composition of the FA of the Lakvijaya coal power plant. The XRF analysis results of two FA samples are presented in Table 1 and Table 2.

Furthermore, twenty 200 g FA samples were measured using an electronic mass balance from OHAUS® Discovery.

2.2 FA Pre-processing

Prepared 200 g FA samples were introduced to the pre-processing stage.

200 g FA sample was put into a 1-litre beaker, and the distilled water from the Banstead EASYpure® II water system at room temperature was poured until the 1-litre mark. To complete a one washing cycle, the sample was stirred for 15 minutes using a magnetic stirrer (AM4) from VELP® and kept settling for 15 minutes (at room temperature). After the settling, the first washing cycle was completed. Then the top water layer was removed, and the bottom FA layer was transferred to another 1-litre beaker, and it was filled with distilled water up to the 1-litre mark and kept at room temperature. Likewise, FA washings were conducted for 1, 3, 5, 7, and 10 number washing cycles at room temperature (RT), 50° C, 70° C, and 90° C.

After finishing the respective washing cycles, residue bottom samples were collected and analysed for the size distribution using FRITSCH ANALYSETTE 22 Laser particle sizer. Finally, the effluents of respective washing cycles were tested for the pH and conductivity values using a SensION-1 pH meter from Hach® and an HQ40D digital conductivity meter from Hach®.

3. Results and Discussion

3.1 Variation of pH Value

The results of the pH value of the solution after a respective number of washing cycles at a respective temperature are presented in Table 3.

The results are graphically represented in Figure 1. The maximum pH value has been reported as 12.10 for the sample with one washing cycle at 50° C. The minimum pH value has been reported as 10.79 for the sample with 10 washing cycles at 90° C. So, it can be concluded that the solution always takes an alkali nature throughout all the washing cycles at different temperatures. When the overall trend of the graph is considered, the pH values of the solutions decrease with the increasing number of washing cycles.

Table 3: pH values of the solutions.

WC	RT	50° C	70° C	90° C
1	11.75	12.10	11.71	11.61
3	11.86	11.73	11.48	11.69
5	11.73	11.29	11.16	11.34
7	10.97	10.89	11.20	11.18
10	10.85	10.89	10.88	10.79

*WC – No. of Washing Cycles

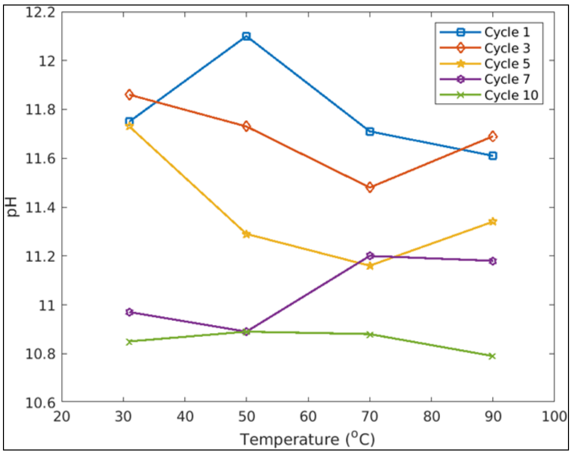

Figure 1: Temperature dependence of the pH of the solution with different washing cycles.

3.2 Variation of Conductivity

The results for the conductivity values of the solution after a respective number of washing cycles at a respective temperature are presented in Table 4.

Table 4: Conductivity values of the solutions.

WC	RT (mS/cm)	50° C (mS/cm)	70° C (mS/cm)	90° C (mS/cm)
1	3.42	6.96	6.40	5.48
3	3.61	4.56	4.34	4.41
5	2.50	2.45	2.30	2.40
7	1.39	1.41	1.89	1.88
10	0.87	0.99	1.83	2.00

The results are graphically represented in Figure 2. The maximum conductivity value has been reported as 6.96 mS/cm for the sample with one washing cycle at 50° C. The minimum conductivity value has been reported as 0.87 mS/cm for the sample with 10 washing cycles at RT. When the overall trend of the graph is considered, the conductivity values of the solutions decrease with the increasing number of washing cycles.

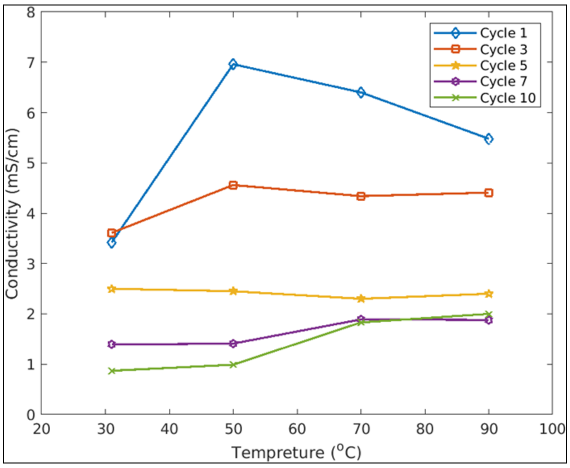

Figure 2: Temperature dependence of the conductivity of the solution with different washing cycles.

3.3 Particle Size Analysis

Table 5: The average mode diameter size of the bottom samples.

Sample name	Average mode diameter of the sample (µm)
Raw FA	31.29
Sample with one washing cycle at RT	24.12
Sample with one washing cycle at 50° C	23.25
Sample with one washing cycle at 70° C	24.47
Sample with one washing cycle at 90° C	20.26
Sample with three washing cycles at RT	25.46
Sample with three washing cycles at 50° C	33.03
Sample with five washing cycles at RT	29.34

A raw FA sample and the bottom samples; the sample with one washing cycle at RT, the sample with one washing cycle at 50° C, the sample with one washing cycle at 70° C, the sample with one washing cycle at 90° C, the sample with three washing cycles at RT, the samples with three washing cycles at 50° C and the sample with five washing cycles at RT were analysed, and the average mode diameter of each sample is tabulated in Table 5.

The result of each sample is graphically presented in Figure 3 to Figure 10. The variation of the average mean diameter of the bottom sample with the number of washing cycles at a similar temperature value is represented in Table 6.

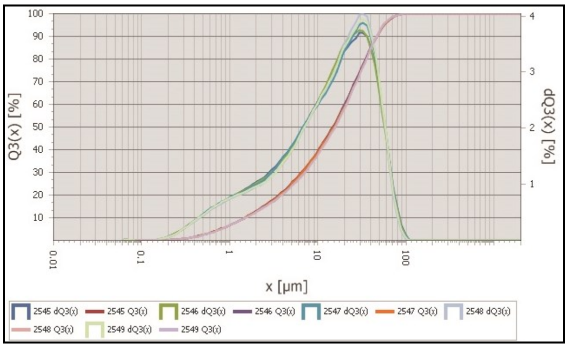

Figure 3: PSD of the raw FA sample.

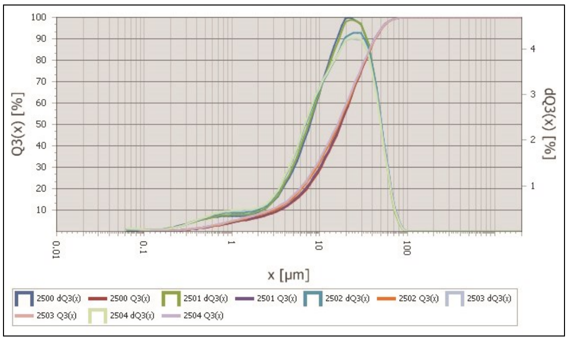

Figure 4: PSD of the bottom sample with one washing cycle at RT.

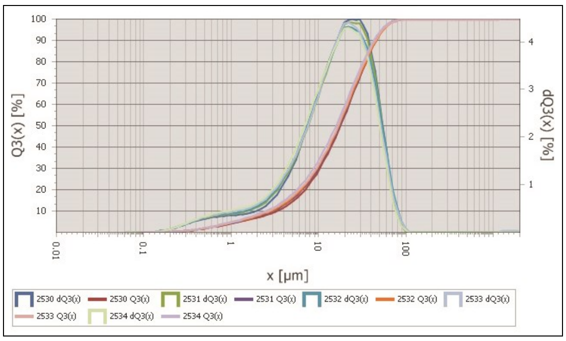

Figure 5: PSD of the bottom sample with one washing cycle at 50° C.

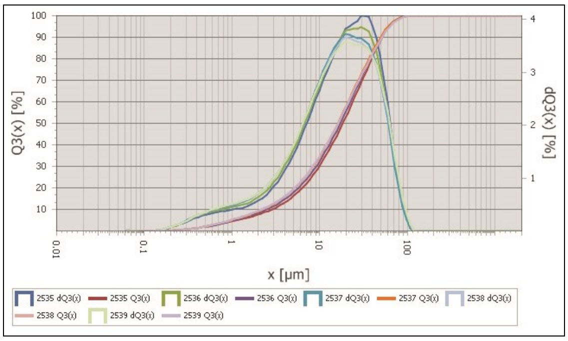

Figure 6: PSD of the bottom sample with one washing cycle at 70° C.

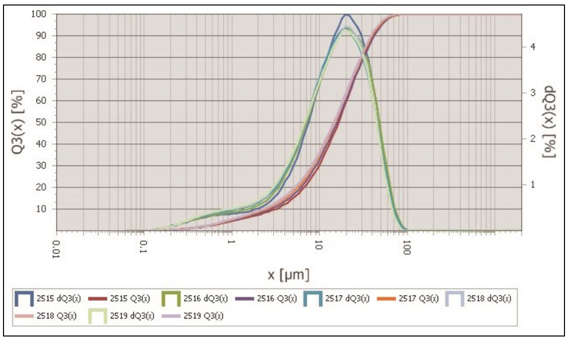

Figure 7: PSD of the bottom sample with one washing cycle at 90° C.

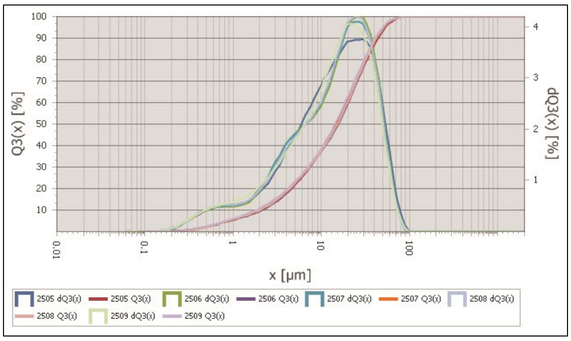

Figure 8: PSD of the bottom sample with three washing cycles at RT.

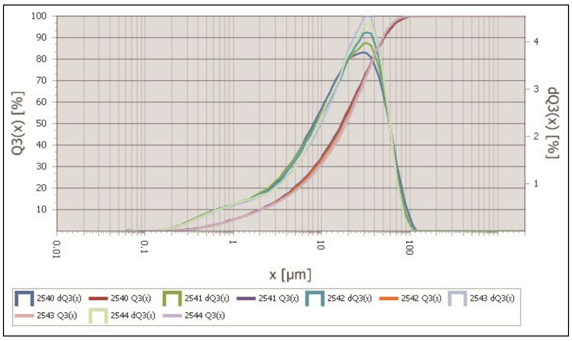

Figure 9: PSD of the bottom sample with three washing cycles at 50° C.

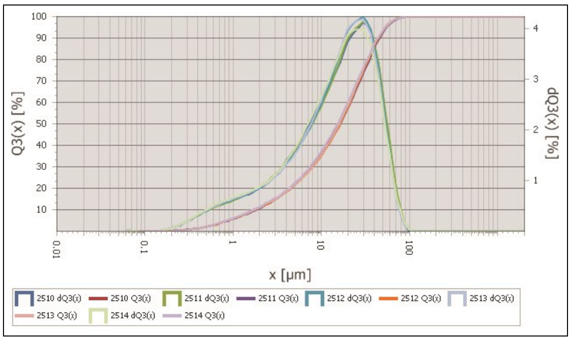

Figure 10: PSD of the bottom sample with five washing cycles at RT

Table 6: Bottom sample particle diameter size variation with the number of washing cycles.

Sample name	**Average mode diameter (µm)**
Sample with one washing cycle at RT	24.12
Sample with three washing cycles at RT	25.46
Sample with five washing cycles at RT	29.34

According to the data of the above table, the mode particle diameter value in the bottom sample is increasing with the increasing number of washing cycles.

The variation of the average mean diameter of the bottom sample with different temperature values at a similar number of washing cycles is presented in Table 7.

Table 7: Bottom sample particle diameter size variation with the temperature

Sample name	**Average mode diameter (µm)**
Sample with one washing cycle at RT	24.12
Sample with one washing cycle at 50° C	23.25
Sample with one washing cycle at 70° C	24.47
Sample with one washing cycle at 90° C	20.26

The highest value for the mode diameter size has been reported to the sample with one washing cycle at 70° C as 24.47 µm. The lowest value has been reported to the sample with one washing cycle at 90° C as 20.26 µm. A visible trend cannot be identified in the available data set for the variation of mode particle diameter in the bottom samples with increasing temperature at a constant number of washing cycles.

4. Conclusions and Recommendations

According to the observed compositions of FA samples, it can be concluded that the raw FA sample shows similar characteristics to class F FA. The pH value of the solution decreases with the increase in the number of washing cycles. But the solution gets an alkali nature throughout all the number of washing cycles. As the ion concentration in the FA sample decreases with the increment of the number of washing cycles, it can be concluded that washing is a good pre-processing method as it can be used to decrease the ion concentration in FA sample, which can be interruptive for the adsorption process of heavy metal ions in wastewater. An AAS analysis can be conducted to identify the types of ions that remain in the solution after the given number of washing cycles. A clear relationship was not observed between the mode particle diameter size in the bottom sample and the temperature value of the water which is used to wash the FA samples. However, further comprehensive experiments are recommended to draw precise conclusions. The mode particle diameter size value in the bottom sample increases when the number of washing cycles is increased.

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Remote Sensing and GIS Approach to Assess Landform Changes in Kaduwela Divisional Secretariat Area and its Impacts to the Environment

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Abstract

Land use/land cover (LULC) change plays one of the major key roles in environmental impacts, and it is common to all nations. Monitoring this LULC change together with quantifications of environmental changes is an important concept in the Sustainable Development process. Therefore, remote sensing and geographic information system technique (RS & GIS) was used to exploit the variation of the LULC pattern, and satellite images of five years between 1997 and 2019 were used in this research. LULC changes in the Kaduwela Divisional Secretariat area were analysed using the Maximum likelihood supervised classification method and found that there was a significant decrease in vegetation cover due to rapid urbanisation. To assess landform changes and their impacts on the environment, normalised difference vegetation index (NDVI), normalised difference built-up index (NDBI), and land surface temperature (LST) were used. Further, relationships in-between them were used to analyse the correlations between NDVI and LST, NDBI and LST, and NDVI and NDBI, and it was noticed that negative, positive, and negative correlations respectively among them. It indicates that healthy vegetation can decrease the land surface temperature, whereas built-up will enhance land surface temperature. More than 70% of overall accuracy for LULC classification was able to achieve in this study.

Keywords: LST, LULC, NDBI, NDVI, Supervised classification

1. Introduction

Land Use/Cover change is one of the major influencing factors in this developing world and to be analysed in a proper way. The rapid growth of population, expansion of urban centres, scarcity of land and development activities are some driving factors. So to do the development activities in a sustainable manner, monitoring these changes in land-use patterns and the related environmental impacts are crucial. Therefore, for landform change analysis, Remote Sensing is a useful technique to achieve the data about ongoing situations and can be used to calculate the change detection, which is proven by many researchers [1]; [2]. The advantages of using remote sensing and geographic information system are large and inaccessible areas can be covered by Remote Sensing, data can be acquired at relatively low cost, long-term time-series analysis can be performed as Remote Sensing has historical digital data available for decades and also the spatial resolutions of remote sensing are suitable for environmental applications [3]. Areas of change in a region can be identified using temporal information on land use/cover [1]. Continuous removal of the vegetation cover may lead to an increase in the surface temperature, and this has the possibility to initiate drought around the area. The expansion of urban centres and the same runoff of vegetation may increase the waterproofing ability of the ground and may lead to problems related to flooding [4]. The resource extraction activities have the chance to cause geological and environmental changes mainly due to ground movements, collision with mining cavities and deformation of aquifers in the nearby areas [5].

In the mining industry also, it is important to assess the environmental impacts due to mining activities such as underground and open cast mining. Mining activities leads to environmental degradation as it is amongst the anthropogenic factor. Continuous observations are required for monitoring these activities, which leads to environmental degradation using automated techniques such as Remote Sensing [6]. There are significant environmental impacts due to the unsystematic surface and subsurface mining activities such as reduction of forest cover, soil erosion, pollution of land, air and water and reduction in biodiversity [7].

In Sri Lanka, the urban centres are clustered in the coastal belts and mainly in the Western and Southern parts of the country. Colombo is the metropolitan region of Sri Lanka and has a high growth rate of urbanisation. Thus the surrounding areas of Colombo, the rate of suburbanisation is very fast, and hence the rural land uses are converted into urban activities. Kaduwela Division is also located in the Colombo suburb area and experiencing rapid urbanisation similar to Colombo city.

2. Methodology

2.1 Study Area

Kaduwela divisional secretariat area is situated in Colombo district in the Western Province of Sri Lanka. The area extends over 87.7 km^2 between Northern latitude of 6°50′0″ N - 6°58′0″ N and Eastern longitude of 79°54′0″ E - 80°4′0″ E (Figure 1).

This division consists of three regions such as Battaramulla, Kaduwela and Athurugiriya. It is located 15 km away from Colombo city and comprises 57 Grama Niladari Division. This is in the fifth urban hierarchy place of Sri Lanka. According to the Census and Statistics data, the total population in 1981 of Colombo district was 1,699,241 and got increased to 2,251,274 in 2001. Also, in Kaduwela divisional secretariat area, the total population in 2001 was 209,741, which is 9% out of the total population [8].

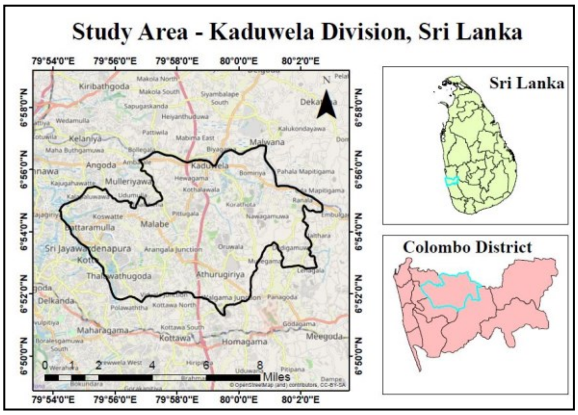

Figure 1: Study Area, Kaduwela Division.

2.2 Data Sources

Almost cloudless 5 Landsat satellite images of 2 different satellite sensors; Landsat 5 Thematic Mapper (TM), Landsat 8 Operational Land Imager and Thermal Infrared Sensor (OLI/TIRS), which has the spatial resolution of 30m each and already georeferenced to UTM coordinate system of Zone 44N taken on 07 February 1997, 02 January 2007, 21 January 2014, 13 January 2017 and 03 January 2019 were used in this study, as shown in Table 1. The specific years were chosen by mainly considering the cloud cover above the study area.

Table 1: Characteristics of Landsat 5TM, Landsat 8 OLI / TIRS.

Acquisition Date	Time (GMT)	Satellite Type	Cloud Cover (%)	Path	Row	Sun Elevation (Degree)
1997/02/07	4:18:24	Landsat 5 TM	8.00	141	55	45.93449564
2007/01/02	4:48:29	Landsat 5 TM	24.00	141	55	49.09226815
2014/01/21	4:54:48	Landsat 8 OLI / TIRS	10.83	141	55	51.05153600
2017/01/13	4:53:49	Landsat 8 OLI / TIRS	2.86	141	55	50.29800365
2019/01/03	4:53:31	Landsat 8 OLI / TIRS	16.94	141	55	50.00682176

2.3 LULC Classification

Land use/land cover classification was done to analyse the change detection over the concerned period, 1997 – 2019. Firstly, the single-band images were combined with multiple-band images using a composite band tool representing the years 1997, 2007, 2014, 2017 and 2019 to produce RGB colour composite images. Then image classification was done using the Maximum likelihood supervised classification technique for the composite images. Finally, the images were classified into four Land-cover classes, namely Water bodies, Vegetation, Barren Land and Built-up areas.

2.4 Normalised Difference Vegetation Index (NDVI)

NDVI is one of the most commonly used vegetation indexes to analyse the change detection of vegetation cover [9]. Depending on the features, absorption and reflection capacity can vary in Near Infrared (NIR) and visible radiation. In the electromagnetic spectrum, green vegetation can strongly reflect the NIR wavelength range (0.64 – 0.67 μm) and strongly absorb by Red portion range (0.85-0.88 μm). The NDVI was calculated using the following Equation (1);

$$NDVI = \frac{NIR - Red}{NIR + Red} \tag{1}$$

Landsat 5; NDVI = (Band 4 – Band 3) / (Band 4 + Band 3)

Landsat 8; NDVI = (Band 5 – Band 4) / (Band 5 + Band 4)

2.5 Normalized Difference Built-up Index (NDBI)

NDBI is one of the most widely used built-up indexes to analyse the change detection of built-up cover. When comparing the other land use/land cover surfaces, Build-up areas have typically higher reflectance in the Short Wave Infrared (SWIR) band range (1.57-1.65 μm) than that of Near-Infrared (NIR) band range (0.85-0.88 μm) [10]. For a proper mapping of urban areas, NDBI is a very effective tool and can be expressed and calculated using the following Equation (2);

$$NDBI = \frac{SWIR - NIR}{SWIR + NIR} \tag{2}$$

Landsat 5; NDBI = (Band 5 – Band 4) / (Band 5 + Band 4)

Landsat 8; NDBI = (Band 6 – Band 5) / (Band 6 + Band 5)

2.6 Land Surface Temperature (LST) retrieval

Here, for the LST analysis radiometrically corrected bands were used. For Landsat 5 TM, Band 3, 4 and Band 6 (Thermal infrared) and for Landsat 8 OLI / TIRS, Band 4, 5 and Band 10 (Thermal infrared) were used. To evaluate the LST, there are some specific steps involved [9].

2.6.1 Convert the satellite Digital Number (DN) into spectral radiance

Initially, from the Digital Number (DN) values, the thermal infrared pixels were converted into spectral radiance. For Landsat 5 TM, Equation (3) and Landsat 8 OLI / TIRS, Equation (4) was used, and the equations are as follows;

$$L\lambda = \left[\frac{LMAX\lambda - LMIN\lambda}{QCALMAX - QCALMIN}\right] \times [QCAL - QCALMIN] + LMIN\lambda \quad (3)$$

$$L\lambda = ML \times QCAL + AL - Oi \quad (4)$$

Where; L_λ is Top of Atmosphere (TOA) spectral radiance (Watts/ (m^2.Srad.µm)), QCAL is quantised calibrated pixel value in DN, LMAX λ and LMIN λ are the maximum and minimum spectral radiance (Watts/ (m^2.Srad.µm)), QCALMAX & QCALMIN are maximum and minimum quantised calibrated pixel value in DN (corresponding to LMAX λ & LMIN λ), ML is Band-specific multiplicative rescaling factor, AL is Band-specific additive rescaling factor and Oi is the correction value for Band 10. All values can be obtained from the metadata.

2.6.2 Convert the spectral radiance into at-sensor brightness temperature

Sensor temperature will be derived [9] from spectral radiance values which represents the black body temperature.

$$BT = \frac{K2}{\ln\left(\frac{K1}{L\lambda} + 1\right)} - 273.5 \quad (5)$$

Where; BT is effective at satellite temperature in Kelvin, and K1 and K2 are Band specific thermal conversion constants that could be obtained from metadata.

2.6.3 Convert the brightness temperature into LST

Equation (6) was used to retrieve the emissivity corrected LST value.

$$Ts = \frac{BT}{1 + \left(\lambda \times \frac{BT}{\rho}\right) \ln(\varepsilon\lambda)} \quad (6)$$

Where; Ts is the LST in Celsius (ºC), λ is the wavelength of emitted radiance (λ=11.5 µm for Band 6 and λ=10.8 µm for Band 10) and ελ is Emissivity and ρ = h(c/σ) = 1.438×10^{-2} mK where; h= Planck's constant (6.626×10^{-34} Js), c = Velocity of light (2.998×10^{8} m/s) and σ = Boltzmann constant (1.38×10^{-23} J/K). For this conversion, emissivity (e or Ɛλ) and proportion of vegetation (Pv) is required and can be calculated from NDVI using the following Equations (7) and (8);

$$e = 0.004\,Pv + 0.986 \quad (7)$$

$$Pv = \left(\frac{NDVI - NDVImin}{NDVImax - NDVImin}\right)^2 \quad (8)$$

2.7 Statistical Analysis of NDVI, NDBI and LST

For the analysis, scatter plots were created first. Then correlation and regression analysis were done to find out the relationship in-between NDVI and LST, NDBI and LST and NDVI and NDBI. To plot the graph, 2000 random spatially unbiased points belonging to all Land Use/Cover classes were selected using the systematic sampling method. The points were extracted from NDVI, NDBI and LST maps using Arc GIS software.

3. Results and Discussion

3.1 LULC Distribution (1997-2019)

The Land Use/Cover classification map of the Kaduwela Division for the years 1997 and 2019 are shown in Figure 2. A similar kind of analysis was done to the years 2007, 2014 and 2017.

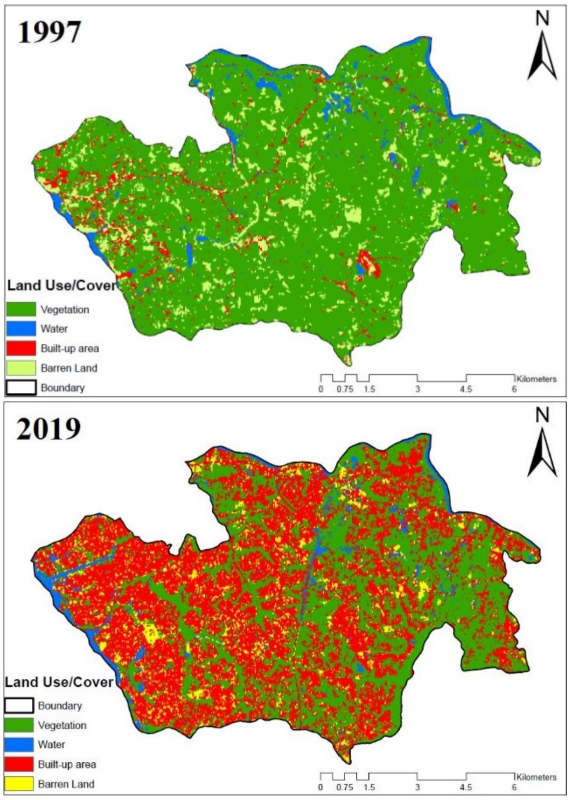

Figure 2: LULC Distribution.

From the statistical data, Figure 3 was generated.

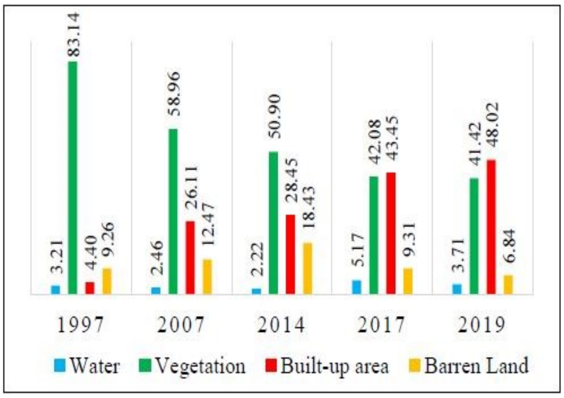

Figure 3: Statistical data of LULC.

According to the results, from 1997 to 2019, there is no significant change in barren land and water cover as they change only 2.42% and 0.5%, but when considering the vegetation cover and built-up area, the changes are 41.72% and 43.62%.

So it can be seen that Kaduwela Division has undergone urbanisation from 1997 to 2019, and especially since 2007, there is rapid urbanisation can be seen. Also, Figure 2 clearly shows that most of the vegetation covers are occupied by the built-up areas, and this may be due to the development activities.

3.2 NDVI Distribution (1997-2019)

NDVI Distribution of Kaduwela Division for the year 2019 is shown in Figure 4. A similar kind of analysis was done to the years 1997, 2007, 2014 and 2017 as well.

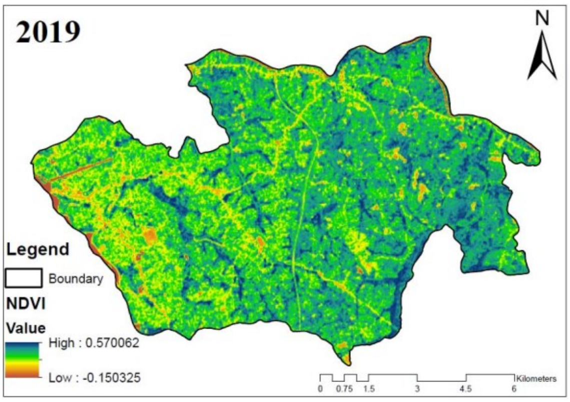

Figure 4: NDVI Distribution, 2019.

According to the analysis, lands occupied by healthy vegetation covers like forest cover and grassland have high value for NDVI, whereas the areas with buildings show low value. For the duration 1997 to 2019, the NDVImax = 0.72 and NDVImin = -0.43.

3.3 NDBI Distribution (1997-2019)

NDBI Distribution of Kaduwela Division for the year 2019 is shown in Figure 5. The same kind of analysis was done to the remaining years too.

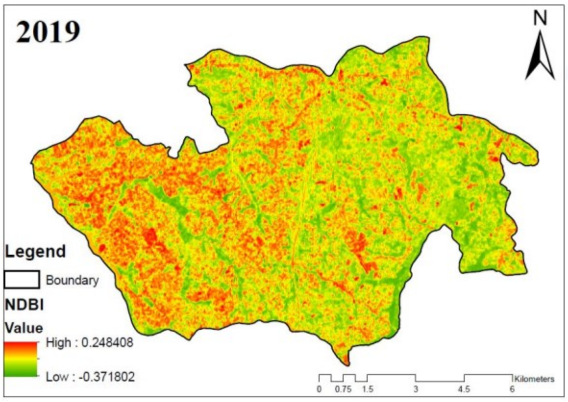

Figure 5: NDBI Distribution, 2019.

Generally, the area occupied by buildings shows a higher NDBI value. For the duration 1997 to 2019, the NDBImax = 0.44 and NDVImin = -0.45.

3.4 LST Distribution (1997-2019)

LST Distribution of the Kaduwela Division for the year 2019 is shown in Figure 6.

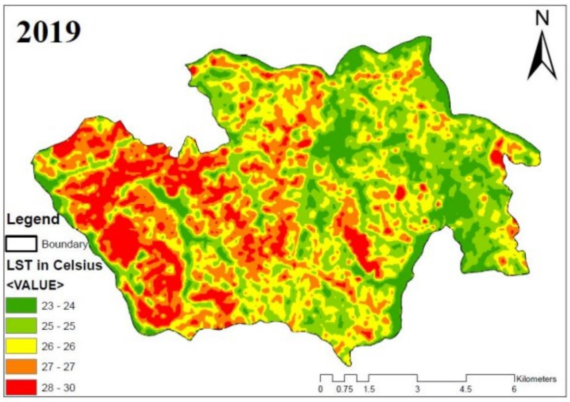

Figure 6: LST Distribution, 2019.

The same kind of analysis was done for the remaining years, and LST distribution maps were developed. The area occupied by buildings shows a higher LST value. For the duration from 1997 to 2019, the LSTmax = 32.46° C and LSTmin = 20.06° C. Due to the rapid development and urbanisation in Colombo suburb, the North-Western and South-Western part of Kaduwela Division shows a higher LST value as a result of the increment of buildings, roads, bridges, industries etc.

3.5 LULC Analysis around Quarry

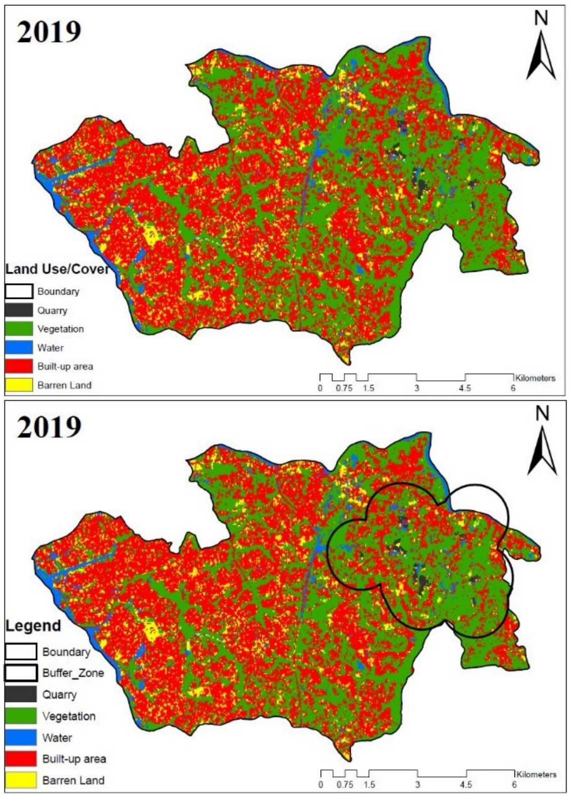

Figure 7: Map of the buffer zone, 2019.

Quarry boundaries were selected to overlay on the LULC classified map to analyse the distribution of the classes around the quarrying area. Even though the quarry site has been presented with minimal background in the study area, the major significance to give concern on this is to analyse whether the quarry development has any considerable effect on the classes concerned. Therefore, a 1 km buffer zone was created around the quarry boundary for this analysis. Figure 7 shows the created buffer zone for the year 2019, and the same kind of analysis was done to the remaining years 2007, 2014 and 2017.

From the statistical data derived from the analysis above, Figure 8 was generated.

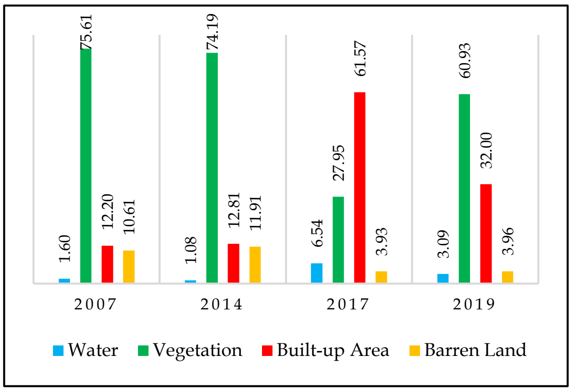

Figure 8: Statistical data of buffer map.

The statistical data shows that quarrying activities also have a certain impact on vegetation cover as they decrease the green cover up to some level. Especially in 2017, the major reason for decreasing the vegetation cover may be the combination of Quarrying activities and urbanisation.

3.6 Relationship between NDVI, NDBI and LST

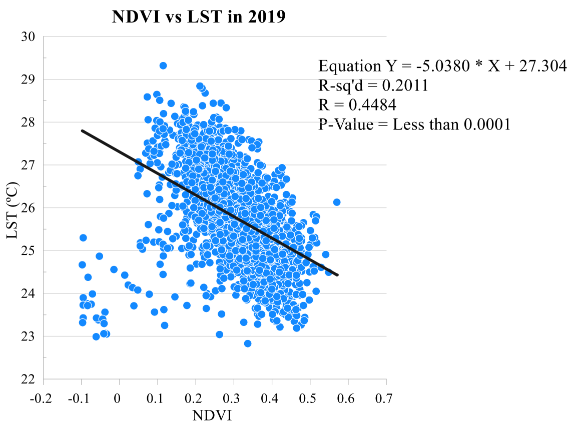

Figure 9: NDVI vs LST in 2019.

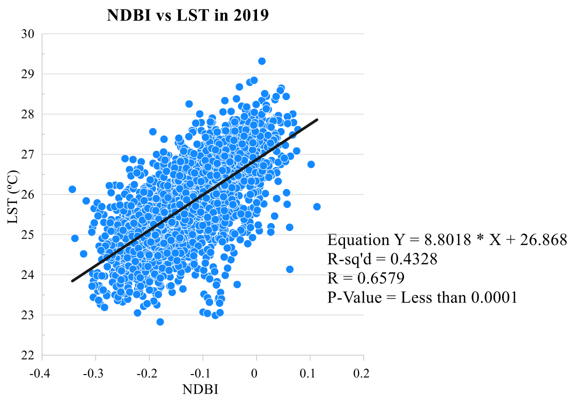

Figure 10: NDBI vs LST in 2019.

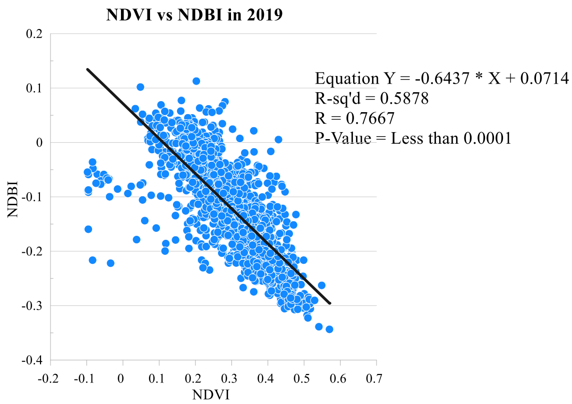

Figure 11: NDBI vs NDVI in 2019.

Scatter plots of NDVI vs LST, NDBI vs LST and NDBI vs NDVI are shown from Figures 9 to 11 for the year 2019 and show an inverse relationship among NDVI vs LST and NDBI vs NDVI and a positive relationship with NDBI vs LST. Similar kinds of graphs were plotted for the remaining years; 1997, 2007, 2014 and 2017. The correlation coefficient (R) of the scatter plots, NDVI vs LST, NDVI vs NDBI and NDBI vs LST, were -0.4484, -0.7667 and 0.6579, respectively. According to the R-Value, NDVI and NDBI and NDBI and LST are highly correlated, whereas NDVI and LST have a moderate correlation. Also, similar kinds of scatter plots were generated for a duration of 22 years (1997-2019), and it was observed that NDVI and LST values have a near-zero covariance, so it is obvious that the relationship is quite nonlinear, and this may be due to the heterogeneity in the land cover.

4. Conclusion

According to the analysis, it was found that there is a significant change in land use/cover pattern during the evaluated years; 1997-2019. Especially in vegetation cover and built-up areas as 41.72% and 43.62% respectively. Here vegetation cover decreased mainly due to urbanisation, quarrying and other industrial activities. Also, more than 70% of overall accuracy was achieved for the Land Use/Cover classification for this study.

As the major purpose of this study is to identify the environmental impacts due to landform changes, the correlation coefficient (R) of NDVI vs LST, NDBI vs LST and NDVI vs NDBI were calculated from the scatter plots and they indicated a negative, positive and negative correlations respectively. When there is an increase in built-up areas together with the decrease in vegetation cover, normally, the surface temperature will increase. However, the healthy vegetation cover will lower the surface temperature. Therefore, it was concluded that vegetation-cover and built-up areas have direct impacts on environmental changes like adverse weather conditions. Therefore, from this study, we found that in the Kaduwela Divisional Secretariat area also, there are temperature variations due to Land Use/Cover changes, especially in the North-Western and South-Western parts of the region due to rapid urbanisation as it locates close to Colombo city. Also, there are environmental issues related to significant climatic changes due to temperature variations, and sometimes it may lead to Urban Heat Island effect as well. Moreover, the decreasing pattern in vegetation cover may lead to not only the surrounding temperature increase but also widespread dust and particulate matter distribution in the study area as rapid urbanisation may cause to increase in the dust emission to the environment. Thus, vegetation cover and rainfall are the key important aspects for dust control in urban areas; in Kaduwela Division, through altering the vegetation cover to urban centres, the reducing regional rainfall pattern may be induced. As a result, deforestation and morphological changes may lead to severe climatic changes in this study area.

Landsat images with 30 m resolution were used in this study as they can be acquired free of charge. So the major limitation we faced was the resolution of the images; we found it difficult to mark the quarry boundaries and differentiate similar spectral classes during the classification due to the low resolution of the images. Landsat images can be acquired free of charge so that, mainly we have focused on Landsat images. Also, it is recommended to consider rainfall data, flood analysis data and socio-economic data for further studies like this one considered only the undesirable weather condition with the use of parameters such as NDVI, NDBI and LST,

Acknowledgement

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Water Hyacinth (*Eichornia crassipes*) as a Phytoremediation Agent for Heavy Metal Removal in Acid Mine Drainages Generated from the Urban Mining of e-Wastes: A Bibliometric Review

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Abstract

Generation of Acid Mine Drainage (AMD) is a problem that is associated with the urban mining of e-waste. Raised acidic conditions resulting from AMD discharge into the surrounding environments and cause toxic heavy metals (HMs) to dissolve, transport, and accumulate in the aquatic environments. Since the elevating concentrations of heavy metals due to AMD discharge exceed the threshold limits, beyond which the health of the living organisms is compromised, remediation of AMD has proven to be taken into consideration. Out of many strategies, passive treatment techniques can be mentioned as the newest approach to remediation AMD. Remediation methods for AMD can be divided into two categories, and they are active systems and passive systems. When comparing these two methods, active systems accrue more financial costs than passive treatments. More than 50 studies have focused on constructed wetland systems under passive technologies since it is self-sustaining once established, and they are cheaper than active treatment systems. The use of several aquatic plants such as water hyacinth, water lettuce, and water cabbage for the remediation process is of significance in constructed wetland systems. This study reviews the experimental findings on HM removal under several conditions using water hyacinth plants in different studies that have been done previously.

Keywords: Acid mine drainage, Heavy metal, Phytoremediation, Urban mining

1. Introduction

Recently the demand for natural resources has increased due to the rapid growth in the population, and there is less supply from conventional mining. Due to those industries are more focusing on the use of recycled materials. Therefore, industries move their focus to the use of recycled materials from e- wastes. As such, industries have explored novel techniques for the recycling process. Urban mining is such a recycling process. Though urban mining is a sustainable method, it has contributed to significant adverse environmental calamities in, and the major environmental impact associated with them is the generation of Acid Mine Drainage (AMD) [1]. AMD is metal-rich water formed from the chemical reaction between water and rocks containing sulfur-bearing minerals. The high acidity conditions in mine drainage resulted in the dissolution of HMs in the adjoining area. Elevating concentrations of heavy metals (HMs) give rise to a considerable threat to the environment, especially when the allowable threshold limits of various HMs which are harmful to living organisms are exceeded [1],[2]. Considering the damages that happen to the environment, several techniques such as pH modification, ion exchange, the addition of chemical and phytoremediation have been introduced for the remediation of these HMs. Among these techniques, phytoremediation plays an important role since it is a green technology.

According to Johnson and Hallberg (2005), oxidation of Iron Pyrite was the main primary cause of AMD generation. pH; temperature; oxygen content of the gas phase; oxygen concentration in the water phase; degree of saturation with water; surface area of exposed metal sulfide; chemical activation energy required to initiate acid generation; and bacterial activity are the primary factors cause for the AMD generation that has been stated by Akcil and Koldas (2006).

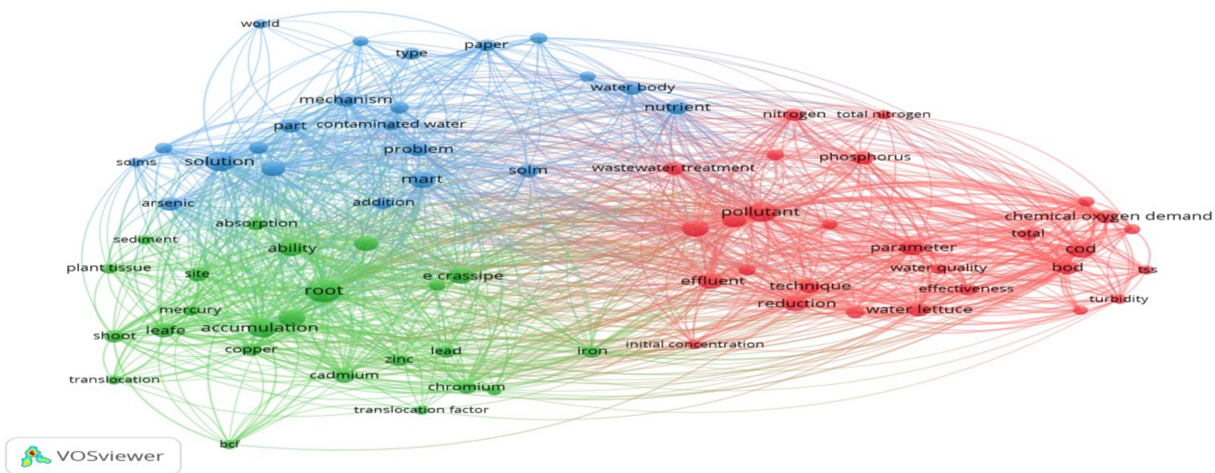

Figure 1: Text data map.

Several authors have also shown that other metal sulfide minerals also produce AMD rather than iron sulfides. Mainly metal contamination associated with this AMD depends on the type and amount of sulfide mineral oxidized, as well as the type of gangue minerals present in the rock. Mainly HMs like Fe, Cu, Cd, Zn and As are present in the mine drainage [1],[3],[4].

A collection of 70 papers from a search in the Scopus database revealed the past and present trends and opportunities. Figure 1 shows the most cited text among these selected research papers related to the plant part considering the heavy metal accumulation. In each network, it shows how the relevant texts are related to the others and from that, the most focused areas can be identified. Through the analysis of this network, plant roots as the remediation method and arsenic as the remediation HM were selected for the studies since they linked mostly with the relevant study. Though water hyacinth and other aquatic plants have the same importance in all studies, water hyacinth was selected due to its availability in Sri Lanka.

Previously many studies have been done to investigate the HM removal capacity by employing Floating Treatment Wetland (FTW) systems. As a phytoremediation media, *Eichornia crassipes* which is a common aquatic weed, is used for the HM remediation process since this plant has the capability of hyperaccumulation [5]. All the tests are carried out under different climatic situations and conditions such as adding of chelating agents. In this study all the results related to the HM removal capacity obtained from the previous studies have been analyzed to obtain knowledge on phytoremediation efficiency as a green technology method.

financial costs due to constant operation and maintenance than in passive systems [7]. Passive systems have an added

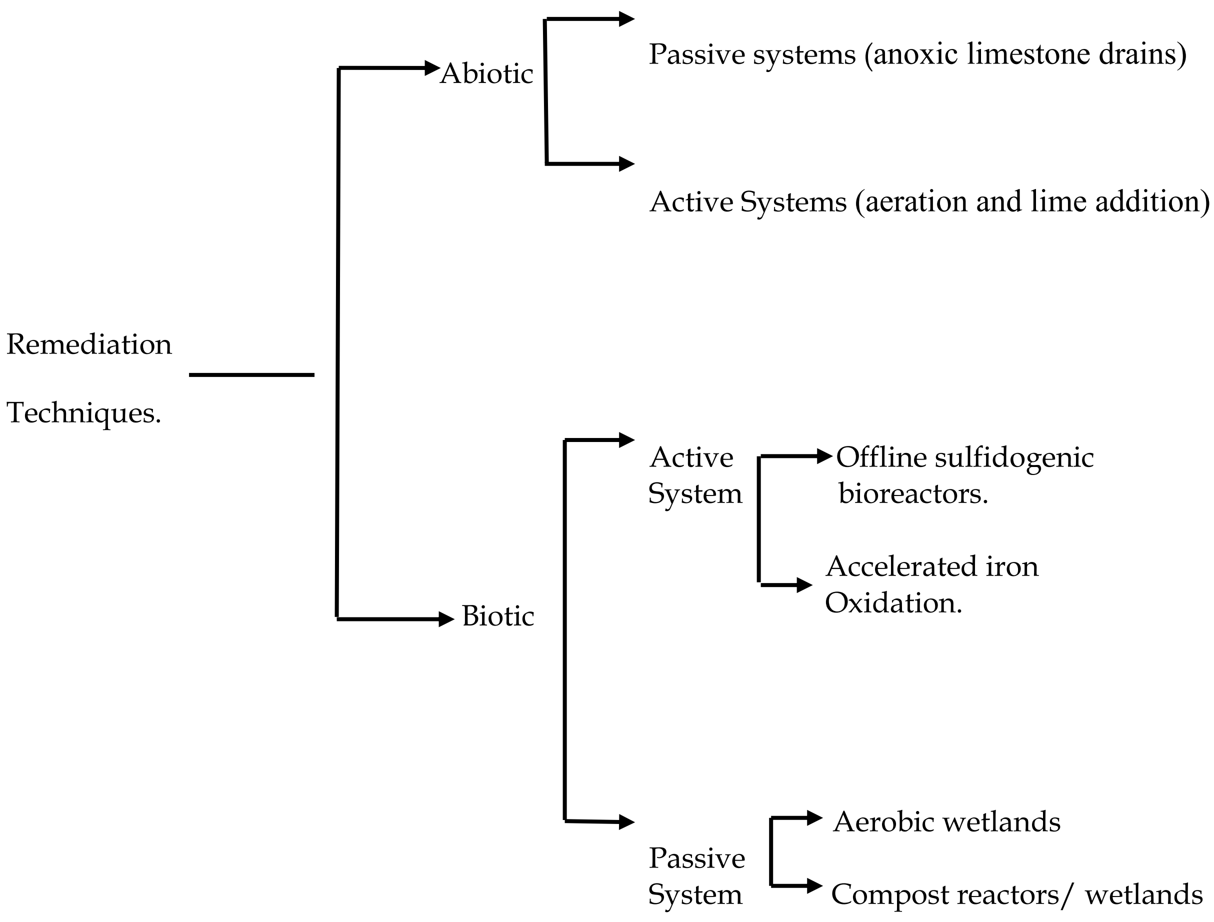

Figure 2: Remediation techniques of AMD.

2. Heavy metal remediation techniques

Removing metals and neutralizing acidity are the main aims of the treatment of contaminated water [6]. Remediation methods for AMD can be divided into two broad categories: active systems and passive systems. Active systems can be employed mainly for continuous operations [6]. Techniques to remediate AMD are shown in Figure 2.

pH modification, ion exchange, biology-based treatments, adsorption, electrochemical treatment, and physical process are the technologies involved in the active remediation system. Active treatment methods generally accrue more

advantage in HM remediation due to the self-sustaining ability of the passive system after its initial setup. Chemical passive treatment by adding chemicals such as limestone, polymers, or others and passive biological treatment represented by primarily constructed wetlands and secondary algal systems or special bioreactors are the treatment methods included in the passive systems [8].

Among passive treatment methods, wetlands have become a favourable option compared with the other treatment methods due to their self-sustaining ability after establishing the remediation process. They are cheaper than active treatment systems [7]. Also, the active treatment technologies require external energy to be supplied in the remediation process. The energy may be supplied physically or chemically to the system. The passive technologies utilize natural sources of energy like sunlight, gravity, chemical and biochemical reactions within the system.

2.1 Remediation method used in wetlands

Phytoremediation is the main remediation method employed in the constructed wetland systems. Phytoremediation has become a recognized pathway for contamination removal from polluted water bodies effectively, and this can mention as a solar-driven technique. Generally, passive techniques are useful for remediating contaminated aquatic environments by AMD over the last two decades [9]. Solidification, soil washing, and permeable barriers are the remediation techniques in constructed wetlands through phytoremediation are more favourable than the others because it is an alternative technique to high cost and high energy conventional method since phytoremediation is cost-effective. This method is considered as "Green insurrection" in the field of innovative clean-up technologies [10].

Due to the applicability even in the remote areas in all climatic zones where plants can have favourable growth and easier of its performance, phytoremediation techniques become more favourable in the remediating process.[11] Various phytoremediation mechanics are available, and they are as follows.

- Phytoextraction – Uptake mechanism of contaminants by plant roots and movement of the contaminants from the roots to aboveground parts of the plant [12]
- Rhizofiltration - adsorption or precipitation of contaminants that are in the solution surrounding the root zone onto plant roots due to biotic or abiotic processes [12]
- Phytostabilization - Immobilization of contaminant through absorption and accumulation by roots, adsorption onto roots or precipitation within the root shoot zone of plants is taken place [11]
- Rhizodegradation- Disintegration of an organic contaminant through a microbial activity that is enhanced by the presence of the root zone [13].
- Phytodegradation - Breakdown of contaminants taken up by plants through metabolic processes within the plant [6].
- Phytovolatization - Uptake and transpiration of a contaminant by a plant with the release of the contaminant or modified form of the contaminant to the atmosphere from the plant [12].

There are many limitations as well as advantages of the phytoremediation process. Time-consuming can be mentioned as one of the main limitations in this process. The rate of remediation under this method depends on the climatic condition, level of contamination, soil chemistry, age of the plant, root depth and contaminant concentration [9].

2.2 Plants for the phytoremediation process

The concept of using plant-based systems and microbiological processes to remediate AMD is used in phytoremediation. Therefore, in constructed wetlands, plants play a major role when remediating the contaminated water bodies. A series of critical physical, biological, and chemical processes within the constructed wetland wastewater treatment system [3]. Wetland plants stabilize the basin substrate, limit the channelized flow, and facilitate the settling of suspended matters by slowing down water velocities. Mainly carbon, nutrients, and trace elements are uptaken by the plant and incorporated into plant tissues. Further, plants transfer gases between the atmosphere and the sediments. Oxygenated microsites within the [14]. The selection of appropriate plant species to employ in constructed wetlands is important in achieving better performance from constructed wetlands. Tolerability of the selected plant to a saturated waterlogged substrate, high concentrations of wastewater loads, and biological traits of the plants such as fast growth rate, large biomass, and well-developed root system are identified as desirable features of plants that can be used in constructed wetlands. Local availability, aesthetic beauty, and the after-use values of the selected plant species are also equally important when selecting plant species used in the constructed wetlands [15]. Water hyacinth and water lettuce are the most common plants among the floating leaf that is being employed for wastewater treatments [16]. Mostly, the accumulation of heavy metals happens through the fibrous root design and the large biomass of these plants.

Figure 3: Water hyacinth.

Figure 4: Water Cabbage.

Figure 5: Water Lettuce.

3. Challenges in the phytoremediation process

The lengthy period for the clean-up, the slow growth rate of the plant and biomass handling after the remediation process can mention as the challenges in the remediation process.

Among these, one of the major problems in the phytoremediation process is to handle the biomass after the accumulation of heavy metals into the plants. If these biomasses are directly introduced into the environment without following proper disposal methods, accumulated heavy metals are again spreading. Van Ginneken et al. (2010) show that combustion of these heavy metal accumulated plants is the preferable method for dispose that can generate energy. Application of the bottom ash non-hazardous solid waste as construction aggregate or in fertilizer formulation in the case of essential trace metals can be mentioned as a revolutionary stage that determines the success of phytoremediation techniques [7].

4. Conclusion

Urban mining-related issues have increased, and the available studies in green remediation techniques have expanded in parallel. The applicability of the water hyacinth plant as a phytoremediation agent is important in the remediation process within the local environment. Nearly 70 studies have revealed the effectiveness of the phytoremediation process compared to the other available techniques. Commonly all the studies reveal that phytoremediation is an effective and cheap method for HM remediation since it promotes green technology. Mostly roots, stems, and leaves of water hyacinth plants can accumulate heavy metals effectively. Indirectly the green technologies such as phytoremediation provide the solution for the problems that cause by aquatic plants such as water hyacinth to the water bodies since these plants show invasive growth.

Acknowledgement

The authors would greatly like to thank all the investigators for the publications on the heavy metal remediation, which was helped inbuilt to this review.

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Assessment of Rehabilitation Options for Environmental Impacts of Abandoned Mines

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Abstract

The quarries which are abandoned ceasing operations without a proper closure plan cause negative impacts on the environment and also risk for the health and safety of society and the economy. This research focused on the environmental aspects that are affected by abandoned quarries and a sustainable methodology to design future land use. The objective of this research is to introduce a quarry rehabilitation method that is effective and has a positive impact on the environment, and applicable for abandoned quarries in Sri Lanka. In this respect, the district of Colombo has been chosen due to its importance in social and economic aspects. The research was conducted by identifying abandoned quarries through remote sensing and geographic information system technique (RS & GIS) analysis of satellite images of the concerned area. Prioritised two quarries among identified quarries for further analysis and suggested possible rehabilitation options. According to the land use analysis around the two quarries, the better rehabilitation option is to transform the lands into a vegetation area.

Keywords: Colombo district, Land use, Quarry rehabilitation, RS & GIS, Sri Lanka

1. Introduction

Mining has become one of the major aspects that contribute to the economy of developing countries. Regardless of the scale of the mine, it causes so many adverse effects to the environment as well as to society during its operations [1]. Some of these effects may impact for decades. It happens due to improper waste management and also due to negligence of rehabilitation measures [2]. Each and every mine has a lifetime for it. With time mines need to shut down their operations on a temporary or permanent basis due to the run out of ore supply, drop in commodity price or the uneconomical nature of operating the mine. After mining, people tend to abandon the mine without any precautions because there is a lack of clearly assigned responsibilities for mine rehabilitation, and it is uneconomical to rehabilitate since there are no fund allocations for it [3].

The main intention of mine rehabilitation is to reduce the long-term effects of environmental contaminations and also the social effects that are caused by abandoned mines [4]. Due to continuous mine waste disposal in mine abandoned areas, it leads to many environmental hazards as the destruction of wildlife and aquatic biodiversity, landscape degradation, dust pollution and water pollution due to siltation, heavy metals and other contaminants. Other than to the environment, abandoned mines cause harmful impacts on society and on the economy also [2]. From the view of society, it brings changes to the lifestyle of the people after abandoning a mine. So, it will lead to the value of nearby lands becoming inferior. The pollution caused by mines will lead to the spread of different diseases by providing grounds for vectors. Other than these, visual pollution may compel psychologically for people. Therefore, these factors are associated with improper mine abandonment.

Since there are a lot of abandoned mines without rehabilitation, prioritisation is important. The methodologies based on GIS, environmental and socio-economic factors, multi-criteria analysis and GIS, hazard maps compilation etc., are used to select the mines that need to be rehabilitated [5].

In the present world, due to the development in the GIS sector, it uses GIS in many circumstances. In mine rehabilitation, also it can be used in every step. For that, it needs satellite images of the particular area. With the use of satellite images, it can identify abandoned mines, their surrounding land uses, visual pollution interpretation etc. [6]. Other than these, it can be used for multi-criteria analysis and select the areas with needed requirements. The selection of practicable rehabilitation options also can be with the use of these. Afterwards, the monitoring and managing of the rehabilitation process can be done by the evaluation of satellite images. So, the usage of satellite images makes the process of rehabilitation effective and efficient.

Under rehabilitation, there are recommended practices for soil management, erosion control, slope stabilisation, species selection, seed collection, nursery establishment and maintenance, seeding and planting strategies and techniques, weed control, fauna attraction and other aspects of rehabilitation techniques.

A long-term sustainable approach should be taken when selecting rehabilitation options. So a mine site can be rehabilitated under the main two categories as a development project or an open space project [7]. When considering about a development project, the mine sites can be converted to industrial plants, residential buildings, cemeteries or infrastructures. As open space projects, particular land can be transferred into agricultural lands, waste disposal sites, leisurely places, and places to attract tourists or can be left for natural vegetation. Among all these options, it can select one of them depending on the nature of the site, requirements and the availability of resources.

To minimise these impacts, it is necessary to follow a proper rehabilitation plan. The rehabilitation plan depends on the nature of the mine as well as on the surrounding nature [7]. Through mine rehabilitation, the productivity of disturbed land can be increased. Thus, by rehabilitating an abandoned mine it can prevent environmental pollution, harmful impacts on the environment and increase the health and safety of the area. Furthermore, it can make abandoned land into a useful, profitable and sustainable asset for the long run [8].

2. Methodology

Colombo district is the main administrative district of Sri Lanka with the highest population density of 20,192 person/km^2. But the land area of Colombo is 699 km^2 [9].

Initially, the abandoned quarries of the concerned area were identified through a classification of satellite images. Through an analysis of land use, rock exposure and topography it identified the possible locations for the availability of abandoned quarries

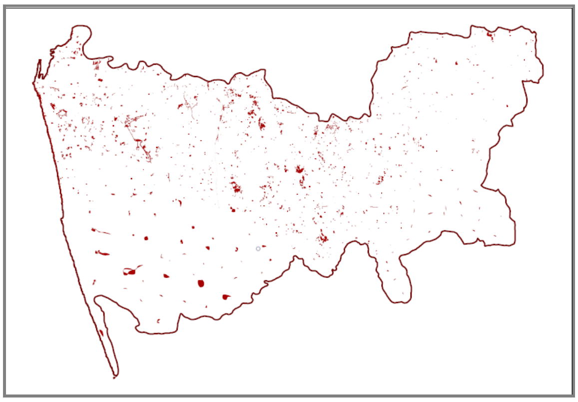

Figure 1:* *Possible locations for the presence of abandoned mines.

These locations were observed using Google Earth on different time scales and separate the quarries that did not operate for the past three years or more.

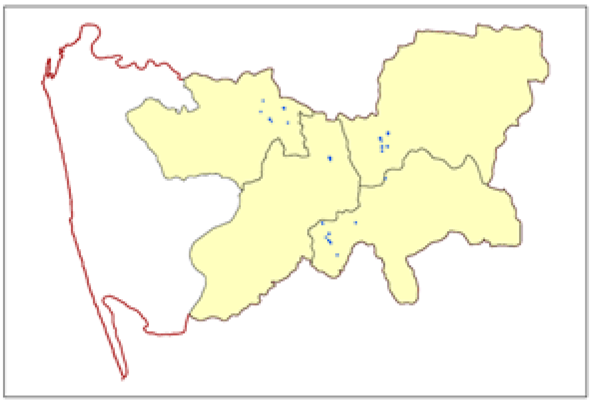

Figure 2:* *Identified abandoned quarry locations in the Colombo district.

Through the analysis, it identified 24 abandoned quarries that were available in the Colombo district. Among these quarries, it prioritised two quarries based on the population density and the presence of water resources around those quarry locations.

3. Results

3.1 Population Density Analysis

The population densities of the identified quarries are in Figure 3. Based on the population density, the quarry in Figure 4 can be identified.

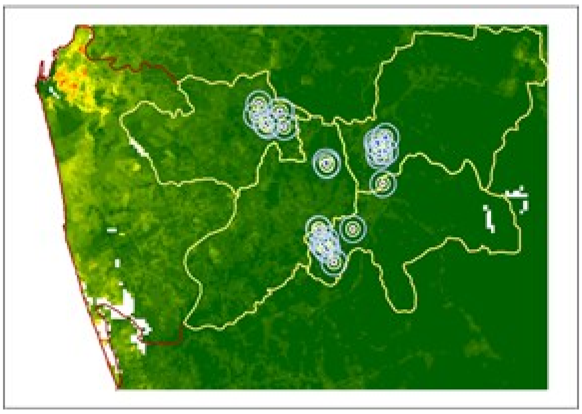

Figure 3:* *Population density of Colombo district.

Figure 4:* *Google Earth image of the quarry with the highest population density.

The operations of the quarry started before 2004 and ceased operations of this part at the end of 2016. This mine has an area of 13,057 m^2. The land use and the topography around the quarry are shown in Figure 5 and Figure 6, respectively. The surrounding area around the quarry was categorised into 100 m, 200 m, 500 m and 1 km buffer zones.

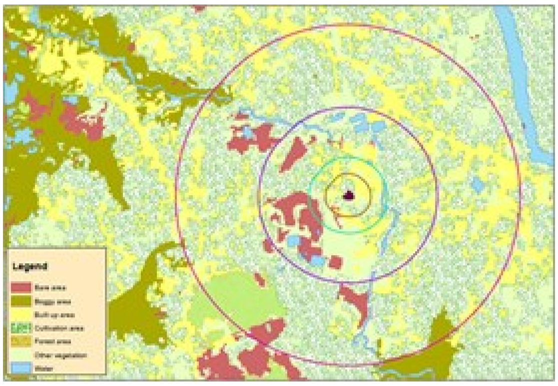

Figure. 5:* *Landuse around the quarry.

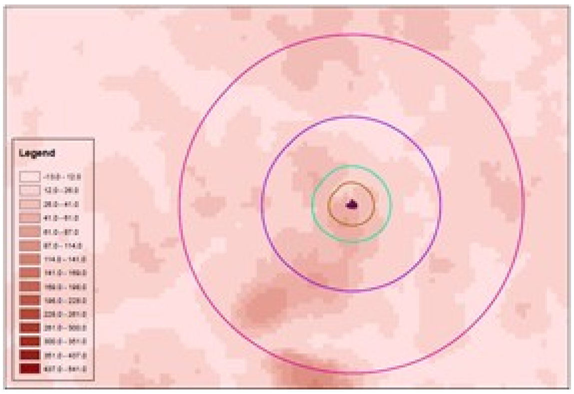

Figure 6: The topography around the quarry.

3.2 Water Resource Availability

The available water resources around the selected quarries are shown in Figure 7.

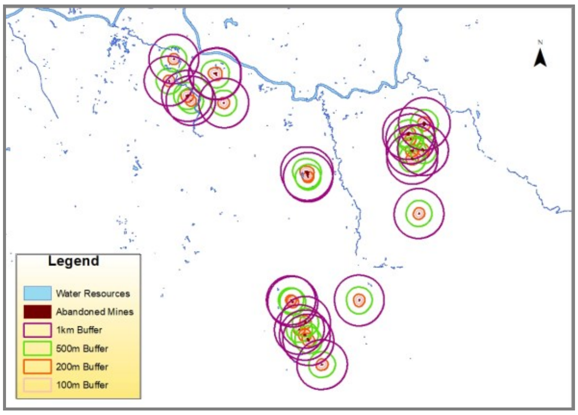

Figure 7: Available water resources around selected quarries.

Based on the presence of water resources closer to the quarry, the following quarry in Figure 8 can be identified.

The Kelani River flows within 1 km of this quarry. Therefore, this quarry mine was selected for the rehabilitation process.

Figure 8: Google Earth image of the quarry with nearby water resources.

This quarry started its operations before 2014 and ceased operations in 2017. It has an area of 17,091 m^2.

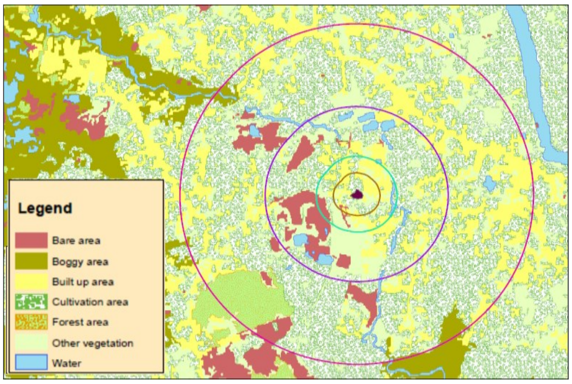

Figure 9: Landuse around the quarry.

The land use around the quarry and the topography of the quarry are demarcated in Figure 9 and Figure 10.

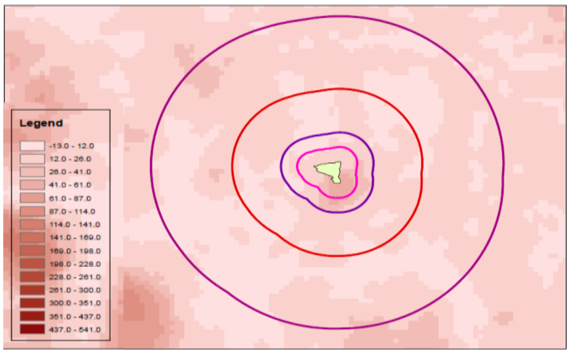

Figure 10: The topography around the quarry.

3.3 Quarry Surrounding Analysis within 200 m Buffer zone

Within the 200 m buffer zone around the abandoned quarry area, the land uses are categorised into four main areas: abandoned quarry area, operating quarry area, residential area and vegetation area.

Figure 11: Google Earth image of quarry 1.

When considering the land use around quarry 1 within 200 m, the major amount is covered by the currently operating quarry (nearly 40% of the area). The rest of the area is for residences and vegetation.

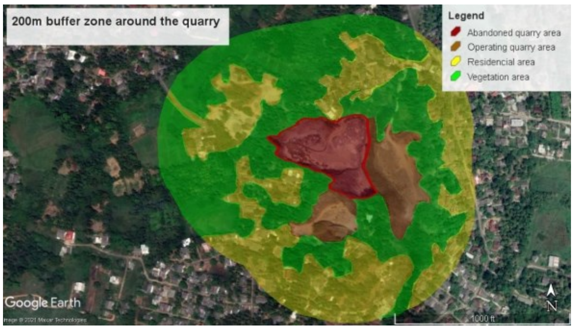

Figure 12: Google Earth image of quarry 2.

Within 200 m around quarry 2, half of the area is covered by vegetation, including farmlands. The operating quarry, which is closer to the abandoned quarry, has approximately similar areas. The human habitats also spread in a considerable area (nearly 36%) within the concerned 200m buffer zone.

4. Discussion

Mining is a crucial source of income for the economy of a developing country like Sri Lanka. As same as the necessity for mining, the impact from it is also higher. Hence, as in the research process, it can identify the abandoned quarry mines in the country and monitor the drawbacks in the mine closure process and make awareness among the relevant stakeholders and authorities.

Identifying abandoned quarries that have been neglected for the past three years or more, without a proper closure plan in Colombo district through a classification of satellite images (Figure 2) and using them to prioritise the quarries according to the selected two criteria (Figure 4 and Figure 8). Based on the topography (Figure 6 and Figure 10) and the land use (Figure 5 and Figure 9) of the surrounding, two quarries are selected and, using the literature review study, sustainable and appropriate rehabilitation methods can be suggested for Sri Lankan quarries.

The method of rehabilitation was selected based on the population and the land use of the surrounding environment. But it can use other factors as soil properties, underlay rock properties, the geography of the area and also the perspective of the community and their standards could also be considered to select the most appropriate rehabilitation method.

Quarry 1 has the highest population density around its location (According to Figure 3). Thus, when rehabilitating this quarry, prioritisation should be given to the community standards, their preferences and their requirements. Therefore, a quarry site in an urban area is better to be rehabilitated into residential areas, infrastructure buildings, park or any other project that support the socio-economic development of the community.

Quarry 2 has the highest impact on water resources (according to Figure 7) than the other selected quarries since it is located closer to the main river of the country, the Kelani River. According to the literature review studies, when there is a water resource closer to the quarry that is to be rehabilitated, it is appropriate to rehabilitate in a manner that manages that water resource to take the optimum use from it. Other than the availability of water resources, the quarry is located in an area with a considerable population. So, it can be recommended to transform this quarry into a farming land or agricultural land with the comfort of accessing water that is required.

Quarry 1 is located on uneven ground when compared with quarry 2, as in Figure 6 and Figure 10.

In further analysis, land uses within 200 m of the quarries are considered as in Figure 11 and Figure 12.

Though there is a high population around quarry 1 than quarry 2, when considering the 200 m buffer zone, the settlements are mostly located around quarry 2 than quarry 1. Furthermore, in the case of considering the size quarry 2 is spread in over a larger area than quarry 1. Therefore, the impact on the community is higher from quarry 2 than from quarry 1. Other than these factors, the common characteristics of these two quarries

(Figure 11 and Figure 12) are that the presence of a currently operating quarry mine closer to these both quarries.

In the case of quarry 1 (Figure 11), the impact from the operating quarry is higher because a major amount of the area within the concerned buffer zone is covered by the operating quarry. And also, the vegetation around that quarry is considerably lesser.

The condition around quarry 2 within 200 m (Figure 12) is different because that area is mainly covered by vegetation and other agricultural lands, and though there is an operating quarry, it is spread over a small area. But it is located near to an abandoned quarry. So, the impact from the operating quarry should also be considered.

Therefore, when introducing a proper rehabilitation method, it needs to take into account mainly the environmental pollution that happens due to these operating quarries. Because, when they are rehabilitated according to the previous recommendations, then those projects may be disturbed by quarries in operation. So, it would be more applicable to transform these two quarry areas into lands with vegetation.

So, according to the study, if the abandoned land is subjected to vegetation, it will act as a green belt and will minimise the environmental pollution from the operating quarries to a certain extent.

Through that, it can achieve three main benefits: minimising the impacts from the abandoned quarry, minimising the impacts from the operating quarry as well as increasing the green cover of the country.

5. Conclusions

By comparing the land use, topography and the population density of the area surrounded by quarries 1 and 2, it can suggest a rehabilitation procedure that is suitable for the abandoned land area. But irrespective of population density, topography and availability of nearby water resources, these two selected quarries have a common feature as both of them are located closer to an operating quarry site. If it uses another rehabilitation method, that may be interrupted by the impacts of the operating quarry.

So, according to the study, it can be recommended to rehabilitate these two abandoned quarry areas by transforming these lands into vegetation areas. Through that, it can allow the vegetation region to act as a green belt that minimises the environmental pollution from the nearby operating quarries while minimising the negative impacts from the abandoned quarry.

Acknowledgements

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Session IV

Mineral Exploration, Subsurface Exploration, and Mineral Processing

Hydrometallurgical Approach to Investigate the Recovery Potential of Gold Available in Waste PCBs

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Abstract

Gold is one of the highly demanded precious metals which have applications in jewellery, investment, electronic and medical industries due to its unique chemical and physical characteristics. Although the demand for gold is continuously increasing, gold producers have failed to meet the existing demand for gold through gold mining. Therefore, seeking out secondary sources of gold is vital. Since gold is one of the major metals used in the electronic industry, e-waste has enormous potential as a secondary source of gold. This study outlines the recovery potential of gold and several other valuable metals quantitatively in Printed Circuit Board (PCB) components of end-of-life computers, namely, microprocessors and Integrated Circuits (ICs), plated connectors in network cards, and plated metallic pins. The research workflow consists of a sample pre-processing and an acid leaching (digestion) process followed by a sample analysis process using an Inductively Coupled Plasma - Mass Spectrometer (ICP-MS). According to the results, the recovery potential of gold is significant in every e-waste component tested under this study.

Keywords: Acid leaching, E-waste, Gold, Metals, Printed circuit boards, Urban mining

1. Introduction

Gold is a transition element situated in group 11 in the periodic table along with copper and silver. The abundance of gold (^{197}Au) in the earth's upper lithosphere is 0.005 ppm [1]. Characteristics such as high malleability, ductility, high thermal and electrical conductivity, corrosion resistivity and rarity have turned gold into the highly desired, precious, and popular metal all over human history [2]. Comparative to the other metals, the demand for gold has an increasing trend. Generally, the jewellery industry accounts for more than 50% of gold consumption [3]. The electronics industry is the third-largest consumer of gold, accounting for nearly 12% of the total gold demand in 2017 [4].

Gold mining is a highly mechanised global industry all over the world. As the three major gold producers, China, Australia, and Russian Federation are account for 1,027.2 tonnes (around 30% of global production in 2019). However, according to the GlobalData, a UK-based data analytics and consulting company, gold production in the year 2020, from the world's eighth-largest producers, fell 6.5%, to 25 million ounces, owing to lower ore grades, asset sales, reduced mill throughput, and lower recoveries [5].

Furthermore, annual demand requires more gold than mined, and the deficit is usually offset by recycled gold (often from the recycling of jewellery). Therefore, recycling from secondary sources is one of the best options to cater for the gold demand in the market. Furthermore, seeking out secondary sources for gold will be a strategic approach for developing countries like Sri Lanka to sustain in the current and upcoming industrial revolutions as gold is essential in the electronic industry. According to the United Nations University (UNU), e-waste is considered as various forms of discarded electronic and electrical equipment that are ceased to be of value [6].

According to the latest global e-waste monitor report, total global e-waste generation and e-waste generation per capita in 2019 is estimated at 53.6 million metric tons (Mt) and 7.3 kg, respectively, and it is a fast-growing waste stream throughout the world with a growing rate of 3-5 % [7, 8].

Moreover, it is predicted that the developing countries in Asia will be producing at least twice as much e-waste as developed countries within the next few years [9, 10].

Almost all electrical and electronic equipment contains metals and plastics. Pure metal and alloy composition is 60 %, while the plastic composition is exceeding 15 % in general e-waste [7]. PCBs are the significant 'electronic' components included in e-waste. Apart from the base metals such as copper, PCBs are a composition of several precious metals, such as gold and platinum [11, 12]. Gold is used as a thin film in contacts, soldered joints and connecting wires [13]. Due to the limitation of natural resources, precious metals such as gold can be recovered through techniques such as acid leaching, cyanide leaching, thiourea leaching, thiosulphate leaching, halide leaching, bioleaching, ion exchange method, borax method, and amalgamation, if the selected e-waste has a considerable concentration of gold [14, 15]. The potential of e-waste as an alternative source for gold will be studied in this study.

2. Materials and Methods

2.1 Sample Collection

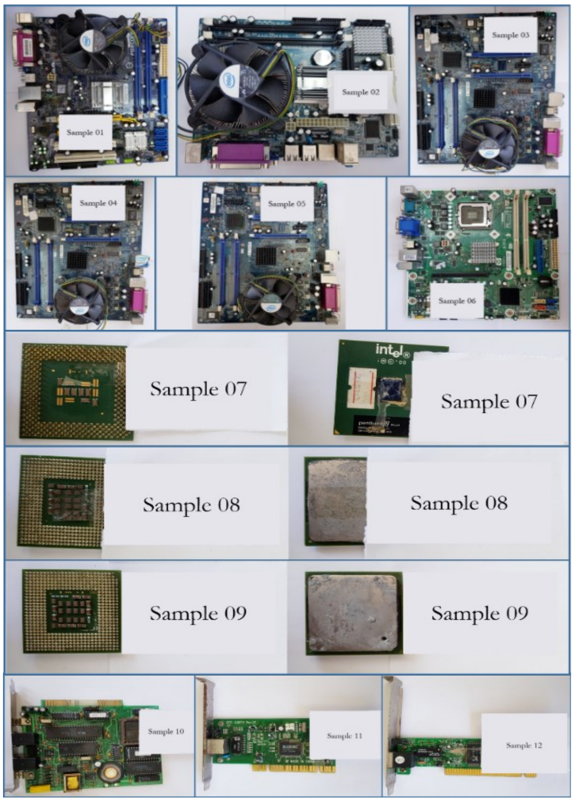

Figure 1: Initial E-waste Samples (01–06: Motherboards, 07–09: Micro-processors, and 10–12: Network Cards).

As shown in Figure 1, 12 e-waste (PCB) samples under 3 categories i.e., motherboards, microprocessors, and network cards discarded by the Department of Computer Science and Engineering, University of Moratuwa were used for this study.

2.2 Sample Pre-processing

Initial PCB samples were dismantled using hand screwdrivers and pliers. Then, 3 types of e-waste components were liberated as shown in Figure 2.

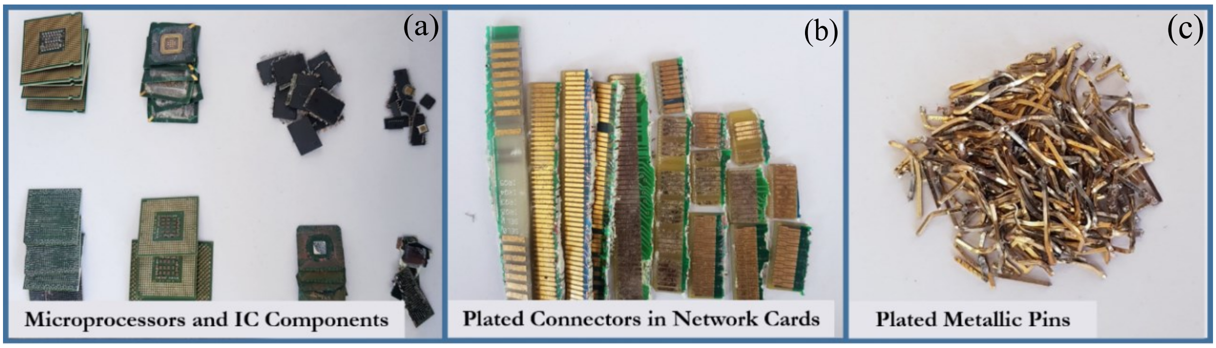

Figure 2: Dismantled E-waste Components ((a) Microprocessors and IC Components; (b) Plated Connectors in Network Cards; (c) Plated Metallic Pins).

An initial manual crushing and a grinding process were used for the size reduction. T.100 type Tema mill (1000 rpm for 5 min.) and a sieve shaker with laboratory test sieves (63 µm BS410/1986) were used for the grinding and sieving process.

2.3 Digestion Process

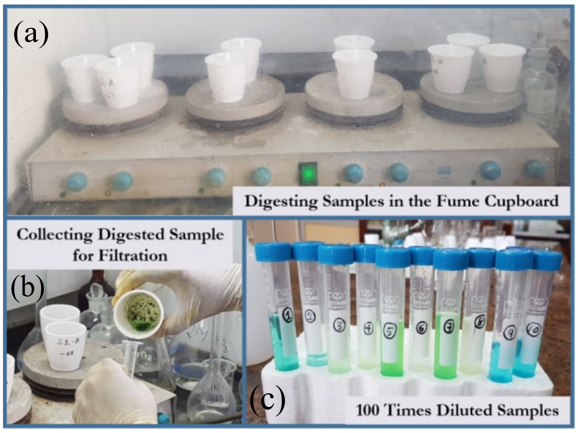

Figure 3: Digestion Process ((a) Digesting samples in the fume cupboard; (b) Collecting digested samples for filtration; (c) 100 times diluted samples).

After the size reduction process, two 0.5 g of each bulk sample were digested separately in Teflon beakers with 4 ml of aqua regia (3 ml of conc. HNO_3 and 1 ml of conc. HCl) at 200 °C for 1 hr.

After the digestion, residue solutions were collected and filtered through 0.45 µm Nylon syringe filters. The filtrates were diluted 100 times with deionised ultrapure water from the Banstead EASYpure® II water system (Figure 3).

Usually, almost all metals, including gold, are oxidised and then solved as ions under aqua regia digestion. Therefore, the actual concentration of selected metals in the sample can be determined using the aqua regia digestion process. Duplications of each sample were digested and analysed using ICP–MS.

2.4 Sample Analysing

The digested samples were analysed through the ICP–MS (ThermoScientific – iCAP RQ) for metal concentrations in the samples. An autosampler unit (TELEDYNE® – ASX-560) was used to feed the sample accurately into the ICP–MS.

Averaged metal concentrations in the digested samples were calculated based on the ICP–MS results.

3. Results and Discussion

Aqua regia dissolves gold, although none of the constituent acids does so alone. Nitric acid is a strong oxidant, which will actually dissolve a virtually undetectable amount of gold, forming gold ions (Au^{3+}).

Hydrochloric acid provides an immediate supply of chloride (Cl^-) ions, which react with gold ions to produce tetrachloroaurate (III) anions, even in solution.

The reaction with hydrochloric acid is an equilibrium reaction that favours the formation of chloroaurate anions ($[AuCl_4]^-$). This results in the removal of gold ions from the solution and allows for further oxidation of the gold. The appropriate equations for aqua regia dissolution are,

$$Au + 3\,HNO_3 + 4\,HCl \rightarrow [AuCl_4]^- + 3\,NO_2 + [H_3O]^+ + 2\,H_2O \qquad (1)$$

Table 1: Metal concentrations in digested samples.

Sample Source	Microprocessors and IC Components (MP/IC)		Plated Connectors (PC)		Plated Metallic Pins (PMP)
Size (μm)	**+ 63**	**- 63**	**+63**	**- 63**	**-5000**
^{52}Cr [ppb]	95.10	46.29	72.85	149.73	58.18
^{58}Ni [ppb]	12,685.93	6,028.53	32,227.37	3,335.66	3,736.24
^{59}Co [ppb]	3,521.25	1,600.38	186.23	71.72	12.48
^{56}Cu [ppb]	264,990.23	124,905.45	296,506.11	19,003.21	480,856.82
^{64}Zn [ppb]	857.16	424.00	1,161.19	295.51	100,846.87
^{75}As [ppb]	782.41	351.90	63.16	18.88	3.29
^{114}Cd [ppb]	1,050.00	562.33	44.78	1195.45	306.09
^{197}Au [ppb]	2,162.86	124.61	1,506.92	2,685.56	4,881.05
^{208}Pb [ppb]	3,230.51	1,860.08	1,294.17	1,705.26	5,750.95

$$Au + HNO_3 + 4\ HCl \rightarrow [AuCl_4]^- + NO + [H_3O]^+ + H_2O \quad (2)$$

Apart from the gold, all other elements shown in Table 1 are reacted with aqua regia, similar to the reaction of copper.

$$Cu + 4\ HNO_3 \rightarrow Cu(NO_3)_2 + 2\ NO_2 + 2\ H_2O \quad (3)$$

According to the results (Table 1), copper and nickel concentrations are higher in every sample. Especially, copper content in the Plated Metallic Pin (PMP) sample is significantly high.

Furthermore, it was evident that the concentration of gold is high in waste PCB components listed in Table 1. The highest gold concentration has resulted in gold plated metallic pin sample, and it was 4,881.05 ppb. When considering gold plated connectors in network cards, gold concentration is higher in the -63 μm sample.

Moreover, toxic heavy metals such as Cd and As concentration is significantly high in MP/IC samples than PC and PMP samples as they are used in manufacturing semi-conductors in MP/ICs. Since Pb is used in both solders and alloy pin manufacturing, the concentration of Pb is high in PMP samples than MP/IC and PC samples.

Moreover, Cr concentration is almost low in every sample. Apart from the MP/IC, Co concentration is lower than 200 ppb. As is also showing a similar variation as Co. Generally, As and Co are used in integrated circuits due to their low diffusion rate and semiconducting properties.

4. Conclusion and Recommendations

The metal composition in the components of waste PCBs and their potential as an alternative source for gold were investigated in this paper.

According to the results, it can be concluded that PCB components such as MP/IC, PC, and PMP have enormous potential for gold recovery; therefore, those waste components will be a good alternative source for gold extraction. Therefore, research on non or less toxic methodologies to extract gold from e-waste are recommended, as most gold leaching techniques are based on cyanide leaching. Furthermore, particle distribution under different size reduction processes should be assessed to identify the optimum size reduction process for gold in e-waste.

Since copper and nickel concentration is higher in MP/ICs, plated connectors, and plated metal pins, further research on the economic viability and the eco-toxicology of the extraction of those elements are encouraged.

Acknowledgement

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Exploration for Potential Sources of Rare Earth Elements in Sri Lanka

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The global demand for rare earth elements (REEs) has been skyrocketing lately due to their excessive usage in numerous high-technologies. Therefore, the current study explores the REE potential in different geological formations of Sri Lanka. Accordingly, REE geochemistry and mineralogy of granites at Massenna (n=10), Arangala (n=6), and Thonigala (n=16); Eppawala phosphate deposit (EPD) (n=20); Ratthota pegmatite (n=6); southwest beach placers (n=18); and Walave alluvial placers (n=20) were analysed by the Inductively-Coupled-Plasma Mass-Spectrometer (ICP-MS) and X-ray Diffractometer (XRD), respectively. Based on the results, only EPD (2676.0-6486.3 mg/kg), Arangala (1634.9-4031.6 mg/kg), and Massenna (65.3-2153.4 mg/kg) showed high total REE (TREE) contents, and they contained REE minerals, such as apatite, monazite, rinkite, mosandrite, and eudialyte. Currently, carbonatites and ion-adsorption clays are the dominant REE sources in the world, and commercial REE extractions are only focused on the minerals: bastnaesite, monazite, and xenotime. In this context, the EPD is the most potential REE source in Sri Lanka (0.46% REO), especially for light rare earth elements considering not only its high TREE content but also the carbonatitic origin and the mineralisation of apatite and monazite. However, the EPD should be further explored for mineralogy, composition, and impurities to assess its viability as a future REE source in Sri Lanka.

Keywords: Eppawala phosphate deposit, Geochemistry, Rare earth exploration, Rare earth mineralogy

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Mine Safety Issues in Quarry Industry: Case Studies of Recent Fatal Accidents in Sri Lanka

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Abstract

Sri Lankan metal quarries supply almost all grades of aggregates throughout the country for the construction industry. But the future of these metal quarries may look bleak owing to the rise in the number of mine accidents being reported. The Geological Survey and Mines Bureau (GSMB) is the main regulatory body that regulates the mineral industry, issues licenses to almost all quarries in Sri Lanka. When considering the industrial mining licenses, there are four types of mining licenses, i.e., type-A, type-B, type-C, and type-D, issued by the GSMB. Of the four types of categories, the multi borehole blasting method can be adopted for type-A and type-B category mining licenses, while the single borehole blasting method is adopted for the type C and type D categories. Of the two types of initiation methods majority of cases with regard to accidents being reported from the single borehole blasting method. But they were not properly documented earlier. However, with the dawn of the Mining Safety Unit, these cases were documented. Of the cases being documented, five cases, i.e., fatalities, occurred due to fly rock, falling to a lower level, collapsing rock overhang on to the people who work underneath it, burning of explosives, and unintentional blasting due to lightning were described in this paper for risk assessment. When these cases are analysed, it is observed that among the contributory factors, excessive face height and human negligence are the most predominant factors for these incidents. At the end, it is mentioned that how this method is unsafe and proposed a multi-hole blasting method under the guidance of a suitable person.

Keywords: Aggregate quarries, Mine accidents, Mine safety

1. Introduction

As a government regulatory body for the mineral industry, the GSMB issues mining licenses for conducting mining activities in quarries in Sri Lanka. For conducting mining activities, the GSMB issues two types of mining licenses, namely Industrial and artisanal mining licenses for all categories of minerals. Of the two types of licenses, the number of artisanal licenses gradually decreases and currently, they are limited to a few due to difficulty in executing blasting activities. In this paper, risk assessment is done for quarries owing to lots of fatalities being reported in quarries. Though lots of mine accidents have occurred in Sri Lankan Metal quarries, they were not properly documented. However, with the inception of the mine safety unit in the GSMB, these incidents are documented in such a way that it could be helpful for authorities to take appropriate actions to improve the mine safety of these quarries. In this connection, this paper is presented to highlight the number of fatal accidents that occurred in Sri Lankan quarries and what factors contributed to these incidents occurring and who is at risk and what remedial actions should be taken to prevent these incidents. Of all documented cases, five incidents are taken for assessment.

1.1 Case #1

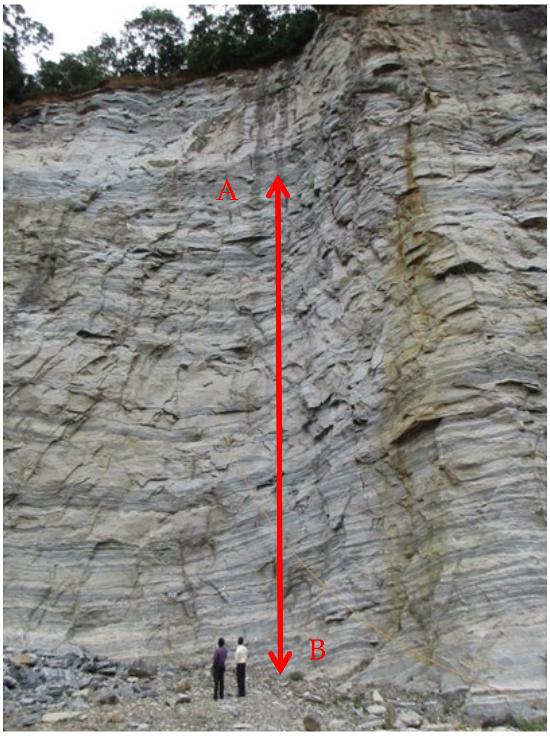

Figure 1: Place of the Incident

- **A-** The place where the Deceased people were engaging in drilling
- **B-** The place where the Deceased people have fallen

AB-70m-80m

Two fatalities were reported in a quarry on 09-07-2021, where the license- IML/C/COL/N/025/R/01 was issued. One of the victims was 42 years old resident of Kaduwela, while the other one was 51 years old resident of Yatiyanthota. Two victims were casual labourers of the quarry and had been working in the quarry for more than ten years. The licensee has been operating the quarry for more than twelve years. It has only one face with a bench height of more than 200 feet. Workers normally reach the top of the face by the ropes since there is no any other access path to reach the blasting location. The place where the incident occurred is shown in Figure 1.

1.2 Case # 2

The quarry where the incident occurred is located Diyagampola GS division of Divulapitiya AGA division of Gampaha District. The GSMB has issued an IML/C/HO/N/9030/LR/5 to this quarry. This incident took place while two employees and the deceased were engaging in removing loosen boulder; suddenly, the loosen bolder started to move down slowly. At that time other two had managed to hold the rope and narrowly escaped with injuries. But the deceased, who was sitting idling at the time, lost his balance and fell under the rubble and was killed. The cause of the death was cited as multiple injuries to the internal organ of the body of the deceased.

1.3 Case # 3

This incident belongs to two quarries located at Paragoda GS division of Divulapitiya AGA division of Gampaha District. The GSMB has issued an IML/C/GM/N/13 & IML/C /GM /N/14 for both quarries, and both licenses were valid when this incident took place. This incident took place when three workers were engaged in welding a metal door inside the explosive storage cabin. While the trio were engaging in welding, sparks came out, and a large fire was broken out inside the cabin. The trio fell victims to the fire.

According to the investigations, it has been revealed that some amount of Black Powder and pieces of safety fuse were laid on the floor of the compartment while they were engaging in welding. Due to sparks, black powder started to burn almost instantaneously. Black Power is a mixture of potassium nitrate, charcoal and sulfur, which is very sensitive to flams and sparks. Therefore it is undoubtedly said that black powder was ignited by sparks. It is therefore undoubtedly said that considering the severity of the incident, some amount of black powder was stacked on the floor of the compartment at that time.

1.4 Case # 4

The quarry where the incident occurred is located at 399/A Pananwela GS division of Dompe AGA division of Gampaha District. The GSMB has issued an IML/C/GM/N/46 to this quarry. This incident took place when the deceased was removing loosen boulders and suddenly lost his balance and was fallen onto the floor of the quarry and was killed. At the time of the incident, the deceased was not tightened to the rope, and he was fallen at the height of about 15 feet onto the floor. The cause of the death was cited as multiple injuries to internal organs due to falling from a height.

1.5 Case # 5

The quarry where the incident occurred is located at Kalahagala in Thamankaduwa AGA division of Polonnaruwa District. The GSMB has issued an IML/A/HO /1535/LR/8 to this quarry. According to the statements given by workers, this incident took place when they were charging the blast holes thunder shower started at once. At that moment, lightning was being experienced in the area where the quarry is located, and they all except the deceased fled away to the shed in fear. In a few moments, they heard the large explosion at the place where they charged the blast holes, and they only saw the mutilated body of the deceased lying under rubbles. They suspected that blast was triggered by lightning

2. Methodology

In the proceeding chapters, what leading factors contributed to these incidents are highlighted. The objective of this risk analysis is to identify risk factors that are dominant in metal quarries and to find ways to remove these risks. In this connection, risks are analyzed qualitatively and quantitatively.

Table 1: Reported fatalities against each reason and its probability of occurrences.

Reason	Fatalities occurred due to each reason during 2020 and 2021, respectively and Probability of Occurrence
Falling rock overhang	1 2 17.6%
Improper explosive management	3 0 5.8%
Accidental Falling	2 5 41.1%
Embankment failure	1 1 11.6%
Fly Rocks incidents	0 1 5.8%
Unintentional detonation due to lightning	1 0 5.8%
Falling to water-filled unprotected pits	1 0 5.8%
Vehicle Toppling incidents	1 0 5.8%

To analyze the risks number of cases within a year and its frequency and probability of occurrences are taken. All reported fatalities shown in Table 1 happened due to the following eight reasons. Tables 2 and 3 describe the probability of occurrence against each category of license and the probability of occurrence against the mode of blasting, respectively.

- Falling rock overhang on to employees
- Improper explosive management
- Falling from a height
- Embankment failure
- Projectiles (fly rocks)
- Unintentional detonation triggered due to lightening
- Falling to water-filled unprotected pits
- Toppling of vehicles

Table 2: Number of reported fatalities (and its probability of occurrence) against each category of licenses.

Licence category	Number of Fatalities (Probability of occurrence)
IML(A)	2 (10.64%)
IML(B)	1 (5.3%)
IML(C)	16 (84.2%)
IML(D)	0 (0%)

Table 3: Reported fatalities against each mode of blasting (and its probability of occurrence).

Mode of Blasting	Number of Fatalities
Single-shot Bore blasting	17 (89.4%)
Multi borehole Blasting	02(10.6%)

3. Risk Ratings

During the calculation, the eight reasons leading to fatalities are rated. Rating is taken based on the number of occurrences of cases reported in 2020 and 2021 and the number of fatalities reported in each case. In this table, Accidental Falling, Falling rock overhangs, Embankment failure and Improper Explosive Management is considered as most severe incidents in quarries, and hence the value of rating is taken as 3 while the value of other cases is taken as 1 due to insignificance and are reported occasionally, and the valve of rating against each case is shown in Table 4.

Table 4: Risk Ratings.

Rating	Case
3	Accidental Falling
3	Falling rock overhang
3	Embankment failure
3	Improper Explosive Management
1	All other cases

3.1 Calculation of Risk

The risk of the above eight reasons leading to fatalities in any quarry is basically a function of the probability of occurrences and the rating. Therefore the risk is calculated according to:

Risk=Probability of occurrences of each reason * Rating

and these calculated risk values are shown in Table 5.

Table 5: Value of the risk of each reason.

Risk Category	Risk
Accidental Falling	1.233
Falling rock overhang	0.528
Embankment Failure	0.348
Improper Explosive Management	0.116
All Other Cases	0.058

According to these values, Accidental falling, failing rock overhang on to employees and Embankment failure are the most dominant factors leading to most fatalities to happen and all are reported from the quarries where the single borehole blasting method is adopted.

4. Risk Factors that could lead to these incidents

According to the above statistics of the reported cases, 89% of cases are from IML/C and IML/B type metal quarries. Blasting activities of these types of metal quarries have been adopted by the single-shot borehole blasting method, while blasts are initiated by a plain detonator, safety fuse arrangement. According to the regulations (GSMB), this method allows license holders to fire only one charged borehole at a time. Therefore this blasting method does facilitate limited opportunity to excavate in the manner in which create benches. Therefore excessive bench height (more than 15 feet) could be seen in these types of quarries. On the other hand, people working in these types of quarries choose to engage in blasting activities that would create excessive face height aiming for higher yield as well. As a result of excessive face heights, it is almost impossible to remove rock overhangs created from the previous blasting as the excavator cannot reach the desired location, whereas removing rock overhangs manually is too risky and impossible due to inaccessibility, heaviness etc. Poor weather conditions also contribute to escalating the situation. During the rainy season, runoff water seeps through fracture plains and builds up pressure on the cracking plains to loosen the boulder from the face of the quarry. Therefore as long as they are working in these types of quarries have to bear the risk of accidental falling over rock overhangs onto them.

On the other hand, these types of metal quarries have one or two working face/s, and people have to be on the face during drilling and charging times. The face angle of these quarries normally has 70°-80° to the horizontal. During this time, people are vulnerable to falling over the face accidentally. Due to excessive face heights and no ramps to reach to blasting location, people used to reach the desired location by the use of ropes without using PPE's. Therefore as long as they are working in these types of quarries have to bear the risk of accidental falling to a lower level.

This situation is further aggravated by the lethargic attitudes and knowledge of the people. Most of the people engaging in this industry are unskilled laborers who have little or no knowledge regarding mine safety and always undermine the safety while carrying out their duties. Even if they are provided safety gear, they do not use it. Therefore not wearing PPE's is also contributed to these fatalities.

Of the reported cases mishandling of explosives and poor blasting practice have led to catastrophic incidents. Some sort of knowledge and experience is required to handle matters related to explosives and blasting activities. But people who are dealing with explosives have little or no knowledge about explosives & blasting activities, and they deal with them without taking adequate safety measures ultimately have to sacrifice their own life as well as surrounding people, and it led to property damages as well.

5. Methods of reducing the above mentioned risk

Following methods can be adopted to reduce the risks.

- Blasting activities should be carried out in a manner in which create a face height of at least 10 feet in such a way that the machine could remove rock overhangs.
- Removed overburden must be stored and stabilized at least 5m away from the edge of the face to avoid falling
- Fence should be erected along the periphery of the quarry to avoid accidental falling by outsiders and animals.
- Workers should always wear a harness with a helmet and boots during working time.
- Blasting activities should be carried out by a competent person. Therefore a blasting foreman should be recruited for each and every quarry to avoid recurrence of these incidents.
- Only the optimum quantity of explosives should be used for each and every blasting.
- Blasts initiation by unrecognized means should be avoided.
- An officer should be appointed to deal with mine safety on a full-time basis by the license holder.
- Meetings should be periodically arranged so that workers become aware of mine safety.

- Workers must be given adequate training by sending them to training courses.
- Notice boards should be put up at strategic locations in the quarry so that workers become aware of mine safety.
- Use of Good quality safety gear by all workers.
- Every accident that occurred in a quarry should be recorded, updated and reviewed.

6. Role of the GSMB

The fatalities reported in the quarries where the single-shot blasting method is adopted shows a sad state that needs to address quickly by the authorities. As a regulatory, the GSMB should revise its strategies to regulate mining activities of these quarries back on track. In this connection following steps can be taken by the GSMB.

- The GSMB should increase the frequency of monitoring of activities of these quarries.
- Stern actions like cancelling of license should be taken against license condition violators.
- Awareness programs/ workshops/ training programs in respect of mine safety must be arranged for workers
- Before issuing licenses, the technical feasibility of license holders should be assessed.
- To encourage license holders to shift to multi borehole blasting method wherever possible.

7. Conclusion

As the majority of fatalities are reported from the quarries where the single-shot blasting method is adopted, as a regulatory body, it is high time to take appropriate actions to reduce the risks as people engaging in this industry are more vulnerable to accidents.

Acknowledgement

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Value Addition of Sri Lankan Dark Spinels through a Heat Treatment Technique

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Ruby, Sapphire, and Emerald are the gem varieties that come under the precious gem category. Therefore, they have a high economic value in the global gem market. Semi-precious gem types are already gaining significant commercial importance in the worldwide gem trade. Gem quality Spinels has a high demand as a semi-precious gem variety. Sri Lanka is very famous for gem-quality Spinels, but apart from that, a high quantity of low gem-quality dark Spinels are found in Sri Lankan gem fields. Currently, the Sri Lankan gem industry is facing the challenge of enhancing the clarity and transparency of dark spinel using heat treatment techniques. Therefore, this study focused on whether the Sri Lankan dark Spinels can be lightened through heat treatment and to develop a heat treatment technique for clarity enhancement of dark Spinels. Spinel samples were analysed to find the content of d block transitional elements which are responsible for the formation of colours in crystals. X-ray fluorescence spectroscopy was used to provide information about the chemical composition and trace element content of Spinel samples. The samples were heat-treated in oxidising and reducing conditions at a temperature ranging from 600° C to 1800° C for 1 hour to 6 hours using the Lakmini gas furnace to find the critical temperature at which Spinels change their current state of transparency. Chemical analysis has proved the dark Spinels primarily consist of high Fe and Zn content. Some rare elements, such as Pt, are also found apart from 3d transitional elements. The critical temperature at which Spinels enhance their clarity was around 1050° C. Prominent clarity enhancement was observed in samples that contained a high percentage of 3d transitional elements such as Mn, Cr, and Cu. The results of this research proved that dark spinel could be value-added by heat treatment.

Keywords: Clarity enhancement, Dark Spinels, Heat treatment, Semi-precious gems, Value addition

Investigation of Rare Earth Element Potential in Granitic Rocks of Sri Lanka Special Reference to Thonigala Granite

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Abstract

Rare earth elements include lanthanide series elements plus Sc and Y. These 17 elements are characterised by the European Union (EU) as critical raw materials with significant supply risk due to their broad of emerging technological applications. Due to this ever-increasing demand for rare earth element (REE) related products, new REE-bearing mineral deposits need to be identified and evaluated for the purpose of filling the supply scarcity in the world. Therefore, this research is focused on investigating REE potential in granitic rocks of Sri Lanka, with special reference to Thonigala granitic intrusions. Thonigala granite is enriched with REEs, relatively with high light REE (LREE) concentration. In the rock samples, the average total rare earth element (TREE), LREE, and heavy REE (HREE) concentrations were 328, 285, and 43.1 mg/kg, respectively. The corresponding sediment values were 619, 472, 147 mg/kg, respectively. The presence of REEs in both insitu rock and sediment samples indicated that significant weathering and erosion occurred in the area. Therefore, this research provides insights into REE potential in granitic rocks of Sri Lanka, focusing on Thonigala granite. More geochemical analysis followed by mineralogical and compositional analysis needs to be carried out for future benefit.

Keywords: Granite, ICP-MS analysis, Rare earth elements, Sri Lanka, Thonigala

1. Introduction

Rare earth elements are a set of 17 chemical elements in the periodic table which consist of 15 lanthanides plus scandium and yttrium. This set of elements is further divided into two categories as light REEs (LREEEs; from La to Eu) and heavy REEs (HREEs; from Gd to Lu and Y) [1], [2].

Currently, the supply of REEs has become a crucial factor due to the expansion of their application in high-tech and green products such as superconductors, mobile phones, LED lightings, metal alloys, catalysts, super magnets, solar panels, etc. [3], [4]. This demand was arisen mainly due to their unique properties such as chemical, physical, luminescent & magnetic properties [1]. As a result, there is a significant gap between the supply and demand of REEs, especially for HREE. This issue was arisen not only due to the demand but also the limited supply of

HREE by the currently identified deposits. In the present world, most of the HREE are produced to the market from ion-adsorption type REE deposits in china [1]. With all the REE productions in China, they control the global REE market with more than 79% of world production and 37% of the world's REE reserves. The ion-adsorption type deposits are commonly formed by the chemical weathering of granitic rocks. The average REE concentration of these types of deposits varies from 300 ppm to 2000 ppm approximately [3].

In Sri Lankan terrain REE containing minerals are mostly found in carbonatites, pegmatites, and beach placer deposits. Among these deposits, Pulmoddai beach sand deposits and Eppawala carbonatite deposits are highly enriched with REEs [3]. Granites are found throughout the Sri Lankan basement, especially in the Thonigala and Ambagaspitiya of the Wanni complex [5]. However, not many prospecting studies were carried out to investigate the REE potential in these granitic rocks of Sri Lanka. Therefore, this study aims to assess the REE potential of Thonigala granitic rocks of Sri Lanka.

2. Methodology

In this study, a total of 20 samples, including 17 rock samples and three sediment samples, were collected considering the geological origin and rock formation of the Thonigala area (Table 1).

All the samples were air-dried, and then sediment samples were further oven-dried at 110° C for 8 hours. All the samples were crushed and powdered using a laboratory jaw crusher and tema mill to reduce average grain size. Powdered samples were sieved using a 63-micron sieve. Sieved samples were then digested using aqua- regia in the presence of conc. Nitric acid, conc. Hydrochloric acid (3:1 volume ratio) and Hydrogen peroxide. After the digestion, samples were analysed using an inductively coupled mass spectrophotometer (ICP-MS) to obtain REE concentrations.

Table 1: Sampling Locations in Thonigala.

Symbol	Northing Coordinate	Easting Coordinate
T- 1	7.90260	79.992444
T- 2	7.902577	79.992435
T- 3	7.902451	79.992507
T- 4	7.90246	79.99248
T- 5	7.902596	79.992815
T- 6	7.902406	79.992634
T- 7	7.902451	79.992789
T- 8	7.90247	79.992915
T- 9	7.902398	79.993015
T- 10	7.90223	79.993043
T- 11	7.901936	79.992962
T- 12	7.901828	79.992989
T- 13	7.901774	79.993053
T- 14	7.902714	79.992815
T- 15	7.902858	79.992579
T- 16	7.901137	79.990977
T- 17	7.901127	79.990751
T-18-S1	7.901246	79.991113
T-19-S2	7.901263	79.99085
T-20-S3	7.901037	79.990869

3. Results

Table 2 represent ICP-MS results for rock samples from T-1 to T-17 and sediment samples from T-18 to T-20.

Accordingly, the ranking of the average REE concentrations in the rock samples followed as Ce > Nd > La > Gd > Pr > Sm > Dy > Er > Tb > Eu > Yb > Ho > Lu > Tm. The average TREE concentration of the rock samples was 328 mg/kg, whereas LREE and HREE concentrations were 285 and 43.1 mg/kg, respectively. The concentrations of LREEs were higher than the HREEs (Table 2), and LREE fraction was about 87%. The average LREE/HREE ratio in the rock samples was 6.6 ranging from 6.0 to 7.7. Ce was the most abundant REE, which accounted for 35.2% of the TREE content. Similarly, Nd, La, and Gd contributed 20.8%, 19.8% and 9.0%, respectively.

The ranking of the average REE concentrations in the sediment samples followed as Ce > Nd > Gd >La > Pr > Sm > Dy > Er > Tb > Yb > Eu > Ho > Lu > Tm.

Table 2: REE concentrations of Thonigala granite rock and sediment samples (in ppm).

Label	TREE	LREE	HREE	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu
T -1	367	318	49.5	52.6	144	25.8	78.7	15.6	1.1	34.3	2.6	6.5	0.75	3.9	0.15	1.2	0.18
T -2	285	250	35.7	51.77	104	20.9	60.5	11.6	0.85	25.0	1.9	4.4	0.48	2.9	0.09	0.78	0.11
T -3	352	307	45.4	65.4	117	28.9	81.3	14.1	0.92	32.0	2.4	5.7	0.57	3.7	0.10	0.85	0.12
T -4	553	479	74.8	98.4	209	37.2	112	20.4	1.7	52.9	3.8	9.2	1.1	6.0	0.22	1.8	0.26
T -5	203	176	26.8	51.5	68.0	11.4	35.6	8.3	1.4	17.2	1.4	4.1	0.58	2.3	0.16	0.94	0.14
T -6	247	219	27.6	73.3	76.9	14.5	45.7	8.3	0.97	18.9	1.5	3.6	0.41	2.2	0.11	0.75	0.11
T -7	580	501	79.6	104	195	42.2	134.5	23.8	1.3	53.7	4.3	11.0	1.3	6.63	0.28	2.1	0.31
T -8	281	244	37.2	76.8	90.6	16.1	48.8	10.9	1.5	25.2	1.9	5.1	0.70	2.9	0.17	1.2	0.18
T -9	267	236	30.8	71.1	89.0	17.4	49.4	9.0	0.84	21.7	1.6	3.9	0.44	2.3	0.09	0.71	0.10
T -10	142	124	17.6	33.1	58.2	6.8	21.3	4.7	0.58	12.6	0.78	2.1	0.30	1.2	0.08	0.50	0.08
T -11	301	261	39.8	49.37	108	22.2	68.2	12.5	0.78	27.0	2.2	5.5	0.61	3.2	0.12	1.1	0.16
T -12	358	309	49.1	60.9	133	25.3	74.3	14.2	0.95	33.6	2.5	6.5	0.81	4.1	0.17	1.3	0.20
T -13	307	263	43.9	48.8	105	23.1	71.4	14.0	1.2	29.2	2.4	6.4	0.77	3.6	0.17	1.3	0.19
T -14	173	151	21.8	40.9	65.2	8.8	27.2	7.5	2.0	14.7	1.14	2.7	0.50	1.6	0.26	0.67	0.23
T -15	614	535	79.1	100	226	45.1	138	23.2	1.2	58.5	4.0	8.5	0.8	5.7	0.18	1.3	0.23
T -16	337	292	45.0	67.6	108	26.2	76.0	13.2	0.9	30.5	2.5	5.9	0.71	3.8	0.16	1.2	0.18
T -17	214	186	28.6	61.9	63.7	12.6	38.9	8.2	0.9	17.4	1.5	4.6	0.67	2.7	0.19	1.3	0.2
Label	TREE	LREE	HREE	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu

T -18	1122	797	324	106	227	113	318	28.1	3.6	228	16.8	35.0	3.7	25.9	0.78	12.6	0.97
T -19	404	350	53.8	30.8	268	8.0	26.1	12.4	5.0	39.0	1.5	5.5	1.0	3.3	0.47	2.6	0.40
T -20	331	267	64.1	71.7	97.6	17.3	60.3	16.0	4.7	30.8	3.1	12.6	2.3	7.8	0.94	5.6	0.93

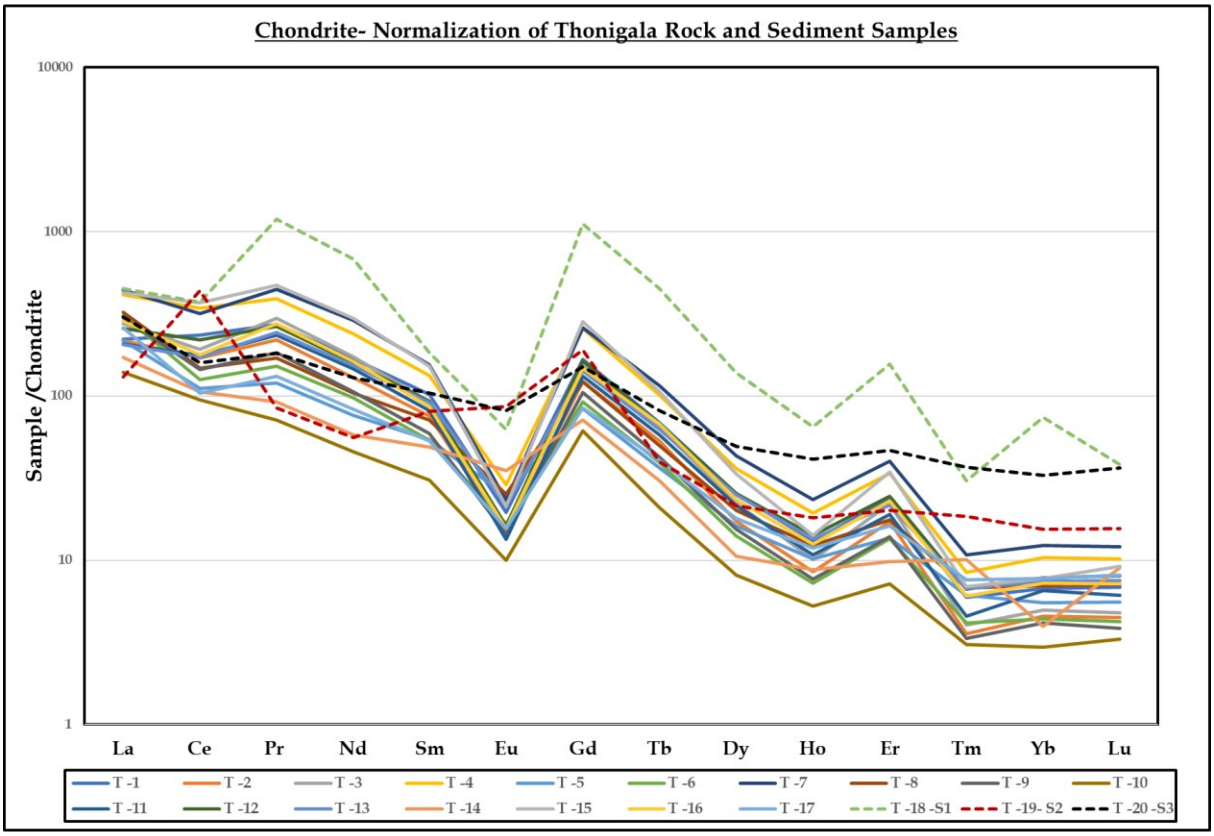

Figure 1: Graph of chondrite – normalisation of Thonigala rock and sediment samples

The TREE, LREE, and HREE concentrations in the sediment samples were 619, 472, 147 mg/kg, respectively. The LREE/HREE ratio ranged from 2.5 to 6.6, with an average value of 3.2. Even in the sediment samples in Thonigala, Ce (31.9%) was the most abundant REE along with high percentages of Nd (21.8%), Gd (16.1%), and La (11.3%). Accordingly, the LREE percentage (76.2%) was higher than the HREE percentage (23.8%) in the sediments. In comparison, all the REEs show high concentrations in the sediments compared to the corresponding rock sample values.

According to Figure 1, most of the samples show a similar chondrite-normalised REE pattern, and they are basically enriched in LREEs than HREEs. All the samples show pronounced Eu anomaly except T-19. Sample T-19 showed a distinct positive Ce anomaly. Most of the samples show Pr, Er and Yb enrichments with a sharp and higher Gd enrichment than most of LREEs. Sediment samples (T-18, T-19, and T-20) show higher REE enrichment compared to rock samples, particularly sample T-18. REE distribution pattern of T-19 is slightly different from other patterns and shows higher Ce, Sm and Eu enrichments. Ce anomaly of the sample T-19-S2 shows a positive anomaly, and also, it does not show Eu anomaly as other samples. Most of the samples show Pr, Er and Yb enrichment with a sharp and higher Gd enrichment than most of LREE.

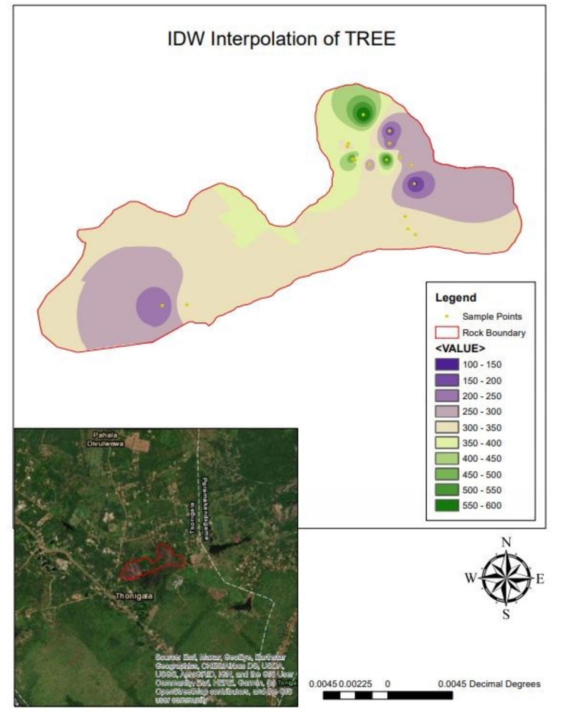

Figure 2: IDW interpolation for TREE distribution for a selected area (values in ppm).

REE distribution for the study area in Thonigala granitic intrusions was estimated using Inverse Distance Weighted Interpolation (IDW) with reference to rock samples (Figure 2). It can be concluded that TREE enrichment can be observed in the Northern part of the map (Figure 2). Moreover, Eastern and S-W regions show significant depletion for the values in the range of 100-300 ppm.

4. Discussion

Results show that Thonigala granite contains an enrichment of LREE over HREE.

All the samples show pronounced Eu anomaly, which is a characteristic feature of granitoid rocks except sample T-19. Sediment sample T-19 showed a distinct positive Ce anomaly, which may be due to the presence of oxidised Ce^{4+} in Cerianite mineral formed by alteration of other REE minerals. Therefore, it can be concluded that there was an oxidising condition during the formation of Thonigala granite.

Most of the samples show Pr, Er and Yb enrichment with a sharp and higher Gd enrichment than most of LREE.

Based on IDW, eastern and S-W regions show significant depletion for the values in the range of 100-300 ppm. This may be due to the considerable weathering and erosion of the parent rock.

As a further approach to the study, the results of Thonigala granite is compared with local and global granitic occurrences. Massenna zircon granite was used for the comparison as a local deposit, while a deposit in Laos was used for the global comparison.

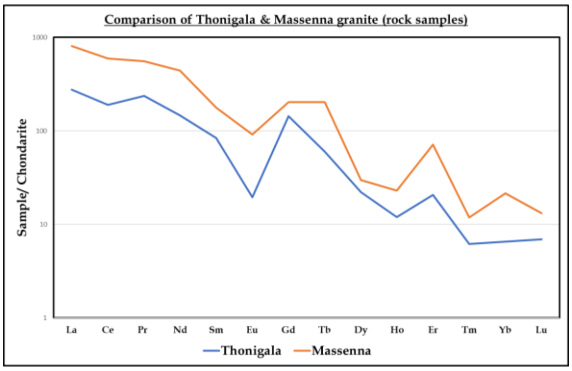

Figure 3: Graph of comparison of Thonigala granite and Massenna granite rock samples

According to Figure 3, it is evident that REEs are enriched in the Massenna granite rock samples compared to Thonigala granite.

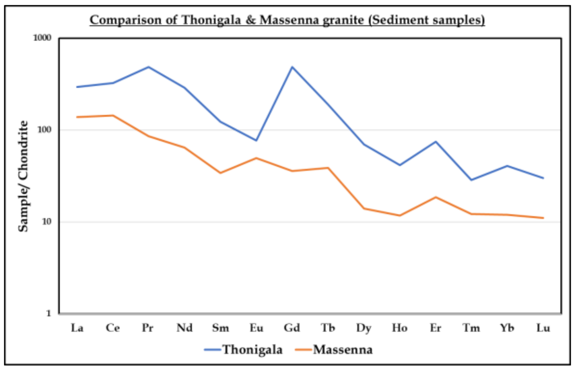

Figure 4: Graph of comparison of Thonigala granite and Massenna granite sediment samples.

Based on Figure 4, REEs are enriched in the Thonigala granite sediment samples than in Massenna sediment samples.

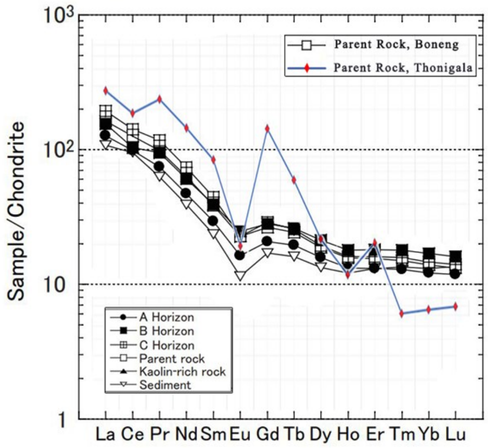

Figure 5: Comparison of Thonigala rock samples with global granite deposit in Boneng, Laos.

When comparing REE potential in Thonigala granite with a global granite deposit, based on Figure 5, Thonigala rock samples were enriched with LREEs than deposit in Laos, while sharp Gd enrichment was indicated by Thonigala granite.

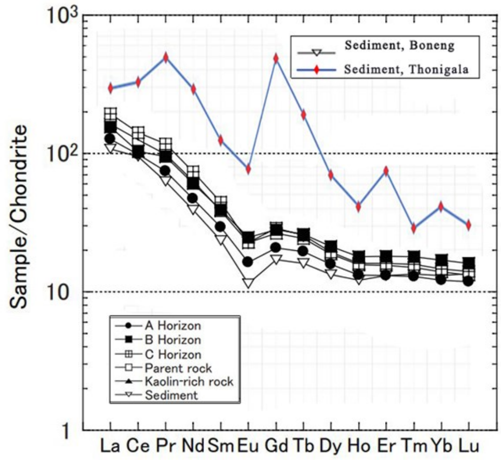

Figure. 6: Comparison of Thonigala sediment samples with global granite deposit in Boneng, Laos.

Based on Figure 6, Thonigala sediments were enriched with all the REEs than deposited in Laos, while sharp anomalies were indicated for Pr, Gd, Er and Yb in Thonigala sediments.

5. Conclusions

REE concentrations in Thonigala granite intrusions were investigated in this work. The average TREE and LREE concentrations in the sediment samples were high compared to the corresponding values in the rock samples.

Thonigala granites are enriched in REEs, particularly LREE, than HREE. Ce was the most abundant REE, which accounted for 35.2% of the TREE content in the rock and 31.9% in sediments. Rock has an average LREE fraction of 87%; thus, it can be concluded that Thonigala granite is a good source rock for LREE.

Since the higher REE enrichment in sediments, it can be concluded that they have been transported and deposited by physical or chemical weathering and erosion processes from the ore body.

Based on IDW, the broad study area is enriched with TREE in the range of 250 - 400 ppm. Also, it indicates significant weathering and erosion in the Eastern and S-W regions of the map.

Thonigala granite rock samples showed relatively low REE enrichment compared to Massenna granite.

Global comparison with the deposit in Boneng, Laos showed relatively higher LREE enrichment and sharp Gd enrichment in Thonigala rock samples, while significant REE enrichment was shown by Thonigala sediments compared to sediments in Laos deposit.

Acknowledgement

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Extraction of Rare Earths from Monazite in Pulmoddai Deposit

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Abstract

Rear earth elements (REEs) have considerable demand in the world and higher market value due to various applications, emerging economies, and continuous depletion of rare earth (RE) metals. Monazite is a perfect source for REEs (61%) and consists of RE and Thorium Phosphate. The Phosphate component was removed by alkaline leaching using NaOH (80 W/V%) at 1800° C for 4 hours at 500 rpm. Thorium was removed by acid leaching using (60 W/V%) HCl at 90° C for 1hour at 500 rpm and 15% pulp density. The solvent extraction method was carried out to separate and purify RE-Oxides from other impurities. 50% Tributyl phosphate (TBP) in kerosene was used as an organic phase with available facilities, and an experiment was done for phase ratio 1 and phase ratio 2, for 3, 6, and 12 minutes of contact time. Inductively coupled plasma mass spectrometry (ICP-MS) analysis was done for elemental analysis. Most abundant REEs in Monazite were Cerium (39%), Lanthanum (20%), Neodymium (12%) and Praseodymium (4%). The main objective of this research was to identify an industrial-level efficient method for separating and purifying RE from Monazite. Economically viable industrial level extraction of REEs was obtained when the phase ratio equals one and 3 minutes contact time.

Keywords: Acid leaching, Alkaline leaching, Dephosphorization, Solvent extraction, Rear earth

1. Introduction

REEs are a group of 17 elements which are 15 Lanthanides series (Atomic number = 57 to 71), Scandium (Atomic number = 23), and Yttrium (Atomic number = 39) in the periodic table. Based on their ionic radius, REEs are mainly classified into two groups as Light REEs (LREEs) and Heavy REEs (HREEs). LREEs have consisted of La to Sm (La, Pr, Nd, Sm), and HREEs have consisted of Eu to Lu (Eu, Gd, Tb, Y, Dy, Lu). These elements are referred to as REEs not only because they are moderately abundant in nature but the concentration of that element is very low.

In the past, REEs were not in high demand due to extraction difficulties. With time, various methods were experimented and utilized to extract REEs efficiently and effectively. The world's energy requirement is steadily rising over time. Thus the requirement of energy is fulfilled by oil, coal, fossil fuels, and natural gas.

Therefore, the amount of CO_2 emitted to the atmosphere increases, and it has various adverse effects. Thus the requirement for a green energy system is increased. Furthermore, economic risk, oil price volatility, and the energy crisis have already created the requirement for a green energy system. Further, demand for REEs is increased due to their applications in many fields such as electronic, medical, military, energy, and manufacturing. Thus, the demand for the REEs market is significantly increased and created a higher request for the global production of REEs.

REEs are found in six ores such as Bastnasite, Monazite, Xenotime, Loparite, Apatite, and ion adsorption clay. Monazite is a perfect source for REEs, specially Cerium (20–30% Ce_2O_3), Lanthanum (10–40% La_2O_3), and Neodymium [1]. Monazite shows radioactive properties due to the presence of Thorium and uranium. In 2020, the amount of global production of REO was 240,500 metric tons. China is the world-leading REEs producer. They produce 140,000 metric tons of Rare Earth Oxides (REO) and have increased their production to 90% of world production. In 2020, Main REEs deposits are located in various countries in the world such as China, Russia, Australia, the USA, Canada, Brazil, Green island, South Africa, India, etc. [1].

Although not very large, Sri Lanka also has some potential to have REEs due to the presence of several RE-sources such as Monazite, Apatite, and gem bearing ores. Although it has some potential to occur Bastaesite and Xenotime deposits with Sri Lankan geology, still those deposits were not found. Thus, it is needed to carry out a proper exploration process to identify and profitably extract these elements.

Pulmoddai beach sand deposit is the major Monazite deposit in Sri Lanka. Although many researchers have developed several methodologies to extract REEs, Sri Lanka has not been able to develop a successful industrial-level methodology to extract REEs yet. Pulmuddai mineral sand contains Ilmenite, Garnet, Rutile, Zircon, Spinal, Sillimanite and Monazite. Ilmenite and Rutile are separated as the main mineral, and Monazite is a by-product. Monazite is consisted of Thorium and REEs (Cerium (28%), Lanthanum (15%), Neodymium (10%), Promethium (3%), Samarium (2%), Gadolinium (2%) and Yttrium (1%)) Phosphates. Monazite concentration in Pulmuddai beach sand is 1.38%, and Monazite has about 61% of REEs [7]. Therefore, these monazites can be utilized effectively by extracting REE.

Several physical separation methods are used to concentrate REEs bearing minerals, such as gravity separation, electrostatic separation, or magnetic separation. Concentrated REEs bearing minerals should be subjected to chemical or physicochemical methods to extract REEs with considerable purity. Main conventional REEs extraction technologies are fractional step method, fractional extraction, ion exchange, etc. With the development of technology, bio-hydrometallurgical method, solvent extraction method, a hydrometallurgical method has a tendency to use to extract REEs. Due to the high chemical and physical stability, the economical extraction and processing of REEs are difficult and costly.

1.1 Physical separation of Monazite

The suitable physical separation method depends on their mineralogical and chemical composition from location to location. Gravitational separation and wet high-intensity magnetic separation (WHIMS) methods are the most commonly used methods for concentrating Monazite. Minerals that have low specific gravity can move easily long distances forward. Minerals that have high specific gravity remain moving a short distance. By applying different intensity magnetic fields, minerals are separated as ferromagnetic, diamagnetic and paramagnetic from each other. The recovery percentage varies with the granule size, the feed rate, the electrode configuration, the high-voltage level and the speed of the roller. All those factors should be optimized to get maximum recovery. For high-grade Monazite mixtures (≥33.3 wt.%), the field intensity investigated were not enough to completely recover. For low grade (Monazite) mixture (≤10 wt.%) WHIMS method is most suitable with higher magnetic field intensity (high-voltage level– 25 kV, roll speed - 60 r/min, feed rate– 1.5 kg/h intensity≥1.08 T).

1.2 Treating with Hydrogen Peroxide

The Monazite can consist of organic matters, and Monazite is treated with Hydrogen Peroxide ($H_2O_{2\ (aq)}$) for several hours to remove organic matters.

1.3 Dephosphorization

RE-Phosphate is a crystalline Phosphate mineral. Due to the high lattice energy of the P-O bond and RE-O bond, $RE(PO_4)_3$ has high thermal and chemical stability. Thus it is unable to decompose even at high temperatures. Thus hydrometallurgical methods or pyrometallurgical methods are used to recover REEs from Monazite. In various researches were done direct leaching of Monazite by using acidic leaching and alkali leaching methods. Sulfuric, Hydrochloric and Nitric acids etc., are used as acid solutions, while Sodium hydroxide, Potassium hydroxide, Sodium peroxide, Potassium chloride, Sodium carbonate, Calcium hydroxide, etc. are used as alkali solutions. Optimum conditions should be used for effective dephosphorization.

Acidic digestion process

$$2REPO_{4(s)} + 6H^+ \longrightarrow 2RE^{3+}_{(s)} + 2H_3PO_4$$

Alkaline digestion process

$$REPO_{4(s)} + 3\ OH^- \longrightarrow RE(OH)_{3(s)} + PO_4^{3-}$$

Experiments were done using different parameters for direct leaching processes of Monazite concentrate by acid or alkali leaching to recover REEs. Table 1 shows the summary of direct leaching processes of Monazite concentrate using acid or alkali to recover REEs.

Table 1: Direct leaching processes of Monazite concentrate using acid or alkali to recover REEs.

Medium	Temperature (^0C)	Time (h)	Reference
(93–96% H_2SO_4)	200–220	3	[2]
50% NaOH (w/v)	170	4	[3]
Na_2CO_3 1:1 (w/w)	900	2	[4]
NaOH 1:1w/w	400	2	[4]
50% NaOH (w/v)	170	4	[5]
80% (w/v) NaOH	150	4	[6]
80% (w/v) NaOH	150	4	[7]

Here are some advantages of using the NaOH digestion process;

- Can recover Tri-Sodium Phosphate as a by-product which is used as fertilizer [4].
- Monazite is transformed into the RE-Hydroxide. Thus RE- Hydroxide will easily dissolve in acid after removal from the supernatant solution of alkaline Phosphates.
- Chemical reactions do not produce any liquid wastes [8].
- No chance to emit acidic gasses to the environment and low maintenance cost [4].
- In the acidic leaching method, separation of Th and RE's is poor, and maintenance cost is high [4].

The disadvantage of the alkaline digestion method is the requirement of a huge amount of NaOH and badly affect the filtration process due to excess NaOH from crystalline and blocking the filtration paths.

1.4 Acid Leaching

Various acids are used for leaching and removing radioactive components such as HNO_3, HCl, H_2SO_4 etc. HCl is the best leachate because it removes the radioactivity component at the beginning and makes it a safer manner to continue the entire process. There are several concentrations, temperature, and time was used in experiments. All the REEs and associated minerals are dissolved in an acidic solution, and Thorium and Uranium remain as residue. The residue is contained by Chlorides form of Thorium and Uranium.

$RE(OH)_{3(s)}+3HCl_{(aq)} \rightarrow RE(Cl)_{3(aq)}+3H_2O_{(aq)}$

$Th(OH)_{4(s)}+4HCl_{(aq)} \rightarrow Th(Cl)_{4(s)}+4H_2O_{(aq)}$

$U(OH)_{4(s)}+4HCl_{(aq)} \rightarrow U(Cl)_{4(s)}+4H_2O_{(aq)}$

1.5 Solvent Extraction

Purity and quality are mainly considering factors for the economic value of the individual REEs. There are some impurities in acid/alkaline solution such as Ca^{2+}, Pb^{2+}, Fe^{3+} and $A1^{3+,}$ which decrease the quality and economic value for REEs. Solvent extraction is the best method for removing those impurities and obtaining REEs with considerable purity. Similar chemical and physical properties make it difficult to separate individual REEs. But solvent extraction method is reported as the best method for selectively extracting individual REEs.

Distribution ratio (D) and separation factor (SF) are used to determine the extraction possibilities and extraction behaviour. These are defined by equations 1 and 2 when C_0 is the initial concentration and C is the final concentration. The distribution ratios of metal ion 1 and metal ion 2 are defined as D1 and D2 [10].

$$D = \frac{(C0 - C)}{C} \tag{1}$$

$$SF = \frac{D1}{D2} \tag{2}$$

Although industrial-level solvent extraction was begun around 1960, it was difficult to isolate a single REEs due to similar physical and chemical properties of individual REEs. The ion exchange method was the only practical method that could extract REEs at the industrial level before 1960.

At present, the solvent extraction method is widely used because it has preferably commercial technology, simple, high capacity, low operating cost, fast kinetics, and high selectivity. The ion exchange method is used only to obtain a small amount of high purity REEs, and it can use for laboratory-scale extractions. Several amines are reported to extract REEs in literature such as TBP, TBA, DOA, D2EHPA, phosphinic acid (Cyanex 272) etc. [11]. When atomic number increase, the most frequently used extractant which used to separate REEs is TBP because extractants are in operation, safety and economically viable. Kerosene is used as an organic diluent. TBP is dissolved in kerosene to prepare an organic phase solution. Two phases of organic and aqueous mixed in a breaker with mechanical stirring. After complete mixing, two phases are separated using a conical separating funnel.

2. Methodology

Monazite sample was directly taken from the Lanka Mineral Sands Limited, which was separated in their processing stage. The initial radioactivity of the sample was measured by the RADALERT100X Giger counter.

2.1 Abundance of REEs in Monazite.

0.5 g of Monazite sample was digested with H_2O_2 (1 ml), HNO_3 (3 ml) and HCl (1 ml) and heated at 200° C for 15 minutes at 1 atm to obtain an abundance of REEs in Monazite.

2.2 Organic matters percentage

14 g of Monazite sample was weighted and proceeded using Hydrogen peroxide ($H_2O_{2\,(aq)}$) to determine organic matter percentage. Hydrogen peroxide was added

to the sample and kept in the fume hood for three hours at 30° C (room temperature) 1 atm (atmospheric pressure) to remove organic matter. The amount of organic matter was obtained by weight difference of the initial weight of the sample and weight of the sample after process with Hydrogen peroxide.

2.3 Monazite Dephosphorization

Three Monazite samples, 30 g for each, were weighted and treated with NaOH (80% W/V) at 180° C for 4 hours. A magnetic stirrer was used at 500 rpm to mix the sample well. After 4 hours sample was washed with hot distilled water several times and filtered using filter paper.

2.4 HCl leaching

Precipitation was leached with HCl (60% w/v) at 90° C for one hour at 500 rpm at 15% pulp density, and samples were filtered using filter paper.

2.5 Solvent extraction

Filtrate solution was put into two beakers of equal volumes. One sample was tested for phase ratio 1, and the other sample was tested for phase ratio 2 to determine the optimum phase ratio. The organic solution was prepared as 50% TBP in kerosene. $RECl_3$ solution was striped with 1M H_2SO_4, and the striped aqueous phase was mixed with organic phase (50% TBP in kerosene) for both phase ratios 1 and 2 in the separatory funnel at 30° C (room temperature) at 1 atm (atmospheric pressure). The stopwatch was started as soon as stopping mixing, and samples were taken from the aqueous phase after 3, 6 and 12 minutes.

3. Result

3.1 Abundance of REEs in Monazite

Initial radioactivity was measured as 40 – 140 CPM, and particles were laid in between 10 – 400 µm. ICP-MS analysis was carried out to determine the ppm level of REEs for the initial Monazite samples. The average ppm level of REEs in Monazite is reported in Table 2.

Table 2: Average ppm level of REEs in Monazite.

REE	Percentage from Total REE (%)	Level (ppm)
La	19.7	4790.0
Ce	38.7	9377.3
Pr	4.0	981.5
Nd	12.4	3002.2
Sm	2.0	492.8
Eu	0.2	56.1
Gd	5.4	1318.8
Tb	0.4	95.4
Dy	1.1	276.3
Ho	0.2	49.6
Er	0.7	167.9
Tm	0.0	10.6
Yb	0.4	86.9
Lu	0.1	13.1
Th	14.2	3453.1
U	0.4	89.1

3.2 Organic matters percentage

Percentage of the organic matters in the sample = 0.86 %.

3.3 Monazite Dephosphorization

ICP-MS analysis was carried out for three samples from $Na_3PO_{4(aq)}$ solution, and Table 3 shows the average ppm level of REEs in the $Na_3PO_{4(aq)}$ solution.

Table 3: Level(ppm) of REEs in $Na_3PO_{4(aq)}$ solution.

REE	Level (ppm)	REE	Level (ppm)
La	0.2678	Dy	0.038
Ce	0.708	Ho	0.004
Pr	0.231	Er	0.032
Nd	0.711	Tm	0.001
Sm	0.038	Yb	0.005
Eu	0.003	Lu	0.012
Gd	0.238	Th	0.134
Tb	0.018	U	0.957

3.4 HCl leaching

ICP-MS analysis was carried out for three samples from $RE(Cl)_3$ solution, and Table 4 shows the average ppm level of Thorium before and after HCL leaching.

Table 4: Level (ppm) of Thorium before and after HCl leaching.

	Before HCl leaching	After HCl leaching
Level(ppm) of Thorium	3,453	54

3.4 Solvent extraction

Figure 1 shows the total REEs extracted from the aqueous phase after solvent extraction in 3, 6 and 12 minutes. Figures 2 and 3 show the ppm level of REEs in the aqueous phase after solvent extraction when the phase ratio is one and two. Figure 4 shows the Individual REEs percentage in the initial and after solvent extraction process when the phase ratio equals one.

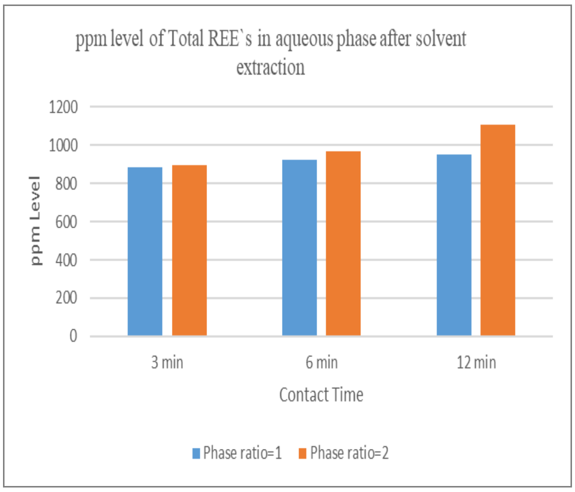

Figure 1: Level (ppm) of Total REEs in the aqueous phase after solvent extraction.

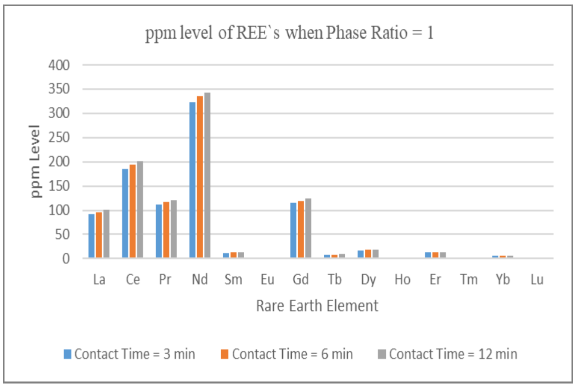

Figure 2: Level (ppm) of REEs in the aqueous phase after solvent extraction when phase ratio is one.

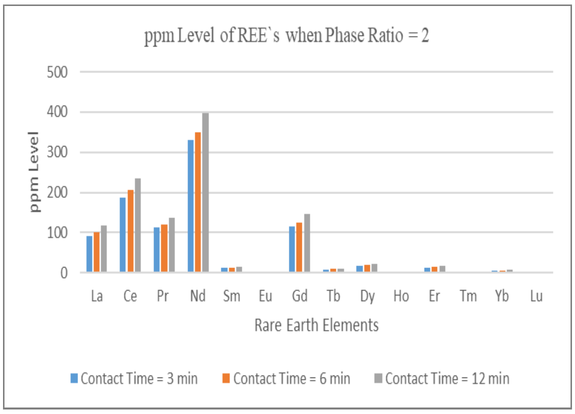

Figure 3: Level(ppm) of REEs in the aqueous phase after solvent extraction when phase ratio is two.

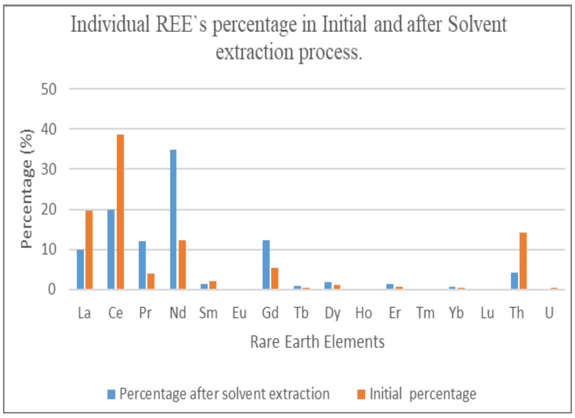

Figure 4: Individual REEs percentage in initial and after the solvent extraction process.

4. Discussion

4.1 Abundance of REEs in Monazite

The most abundant REEs were Cerium (39%), Lanthanum (20%), Neodymium (12%) and Praseodymium (4%). Therefore, Pulmuddai Monazite is a good source for Cerium, lanthanum and Neodymium.

4.2 Determination of organic matters in sample

The presence of organic matter in the Monazite sample was 0.86%. Thus the possibility of having the organic matter in Monazite is negligible.

4.3 Monazite Dephosphorization

Equations Monazite was digested with NaOH and resulted in $RE(OH)_3$, $Th(OH)_3$ precipitation with Na_3PO_4 filtrate and Na_3PO_4 was removed by filtering.

$$REPO_{4(s)} + 3NaOH \xrightarrow[(aq)]{} RE(OH)_{3(s)} + Na_3PO_{4(aq)}$$

$$Th_3(PO_4)_{4(s)} + 12NaOH_{(aq)} \longrightarrow 3Th(OH)_{4(s)} + Na_3PO_{4(aq)}$$

Extra NaOH and Na_3PO_4 were removed from the sample by washing hot distilled water. $RE(OH)_3$ and $Th(OH)_4$ cake remained on filter paper. According to the ppm level of REEs and Th in the Na_3PO_4 solution (Table 3), The amount of REEs passed through the filter paper to Na_3PO_4 solution was small enough to be ignored. Hence, the Phosphate component was successfully separated from REEs and removed from the Monazite.

4.4 HCl Leaching

$RE(OH)_3$ and $Th(OH)_4$ reacted with HCl and formed $RECl_3$ and $ThCl_4$. $ThCl_4$ was precipitated, and $RECl_3$ remained in the filtrate. So radioactive Th components were successfully separated from REEs.

$$RE(OH)_{3(s)} + 3HCl_{(aq)} \longrightarrow RE(Cl)_{3(aq)} + 3H_2O_{(aq)}$$

$$Th(OH)_{4(s)} + 4HCl_{(aq)} \longrightarrow Th(Cl)_{4(s)} + 4H_2O_{(aq)}$$

According to Table 4, the ppm level of Th after HCl leaching is very low. Thus almost all the Thorium ions were precipitated as $Th(Cl)_4$.

4.5 Solvent extraction

According to Figure 1, the ppm level of the aqueous phase for REEs was not significantly different for phase ratios one and two. But ppm level of REEs in the aqueous phase was much higher when the phase ratio was equal to two rather than one. Thus higher number of REEs were extracted by an organic phase when the phase ratio is equal to one. It was economically viable extracting REEs in phase ratio one due to high expensive process of preparing organic phase. In phase ratio two, waste generation is high, and waste disposal expenses also increase. In our research, we consider about extraction of REEs on an industrial scale. Thus it is better to do solvent extraction in phase ratio one.

In phase ratios, one (Figure 2) and two (Figure 3), the minimum ppm level for Lanthanum, Cerium and Neodymium in the aqueous phase was obtained when contact time equals 3 minutes. Thus most of the Lanthanum, Cerium Neodymium ions are extracted by an organic phase when contact time is 3 minutes.

Individual REEs percentage in initial sample and after Solvent extraction in phase ratio one was shown by Figure 4. The abundance of Neodymium and Gadolinium after solvent extraction is higher than the abundance of Neodymium and Gadolinium in the initial sample. Thus this method is most suitable for Neodymium and Gadolinium due to the well extraction of Neodymium and Gadolinium.

5. Conclusions

Most abundant REEs in Monazite, were Cerium (39%), Lanthanum (20%), Neodymium (12%) and Praseodymium (4%). Monazite does not need to process with Hydrogen Peroxide ($H_2O_{2\ (aq)}$) in REEs productions due to the very low possibility of having organic matters. Several steps were done to extract REEs from Monazite in our research methodology. First, Monazite ore (RE-Phosphate) was digested with NaOH (80 % w/v) at 180° C for 4 hours at 500 rpm when the particle size of Monazite was laid in between 10 - 400 µm. Then, RE-Hydroxide (solid) and Thorium-Hydroxide (solid) was precipitated, and $Na_3PO_{4(aq)}$ was removed by filtering. In the second stage, the precipitated cake was leached with HCl (60% w/v) acid at 90° C for one hour at 500 rpm when the pulp density was maintained as 15%. Then RE-Chloride and Thorium-Chloride(solid) was formed. Undissolved Thorium- Chloride was removed from RE-Chloride$_{(aq)}$ by filtering. In the third stage, REEs were extracted by the solvent extraction method. 50% TBP in kerosene were used as the organic phase at 30° C (room temperature) and 1 atm (atmospheric pressure). Phase ratio one was the best and economically viable for industrial level REEs extraction from Monazite. Most of the Lanthanum, Cerium Neodymium ions were extracted by an organic phase when contact time was 3 minutes. This methodology is mostly suitable for Neodymium and Gadolinium due to the well extraction of Neodymium and Gadolinium.

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Characterisation of Sediment Deposition of Bolgoda Lake using Acoustic and Sampling Methods

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Abstract

Bolgoda Lake is one of the main water resources in the western province, which has two major basins explained as Bolgoda south and north lakes. Lake sedimentation is a serious dilemma for water scarcity, productivity, and flooding effect. So, identification of sediment characteristics is most critical. The attempt of this study is to identify the sedimentation pattern, sediment type, and mineral composition of the sediment from the data collected from core sampling, grab sampling, and the bathymetric data obtained from spot depth eco sounder that combines with Magellan 510. Hence, grain size distribution (GSD), scanning electron microscope (SEM), and bathymetric surveys were utilised to recognise sediment type, heavy metal composition, and sedimentation pattern, respectively. These analyses revealed that bottom sediment has poorly sorted, very fine skewed, physical characteristics have statistical distribution and discover the sediment type as sandy silt. Also, bathymetric analysis upholds the sedimentation pattern that accumulates sediment from the left bank to the right bank of the Bolgoda lake via the water column and to identify the sediment distribution along the lake bottom.

Keywords: Bathymetry, Grain size distribution, Kurtosis, Sedimentation, Skewness

1. Introduction

Due to the growth of population and increase of the requirement under the urban development, there is a demand for freshwater. If the water demand is exceeded the available amount, water stress happens. Lakes and reservoirs are some of the main water resources that affect the day-to-day essentials. However, reservoir sedimentation and lake sediment are the major problems that diminish the useful water quantity from the available water resources [1].

Soil erosion due to the effect of the water and wind is one major problem when considering the lake sediment in the lake bottom. As well, sedimentation happens due to the effect from the catchment area and also surface mining, urban construction, etc. [2]. When the water flow energy and velocity are reduced, the sediments tend to settle down along the bottom of the basin of the water resource. Lake sedimentation is a fact that is not to be neglected as it can cause mainly impacts like Biological and physical and Social and economic etc.

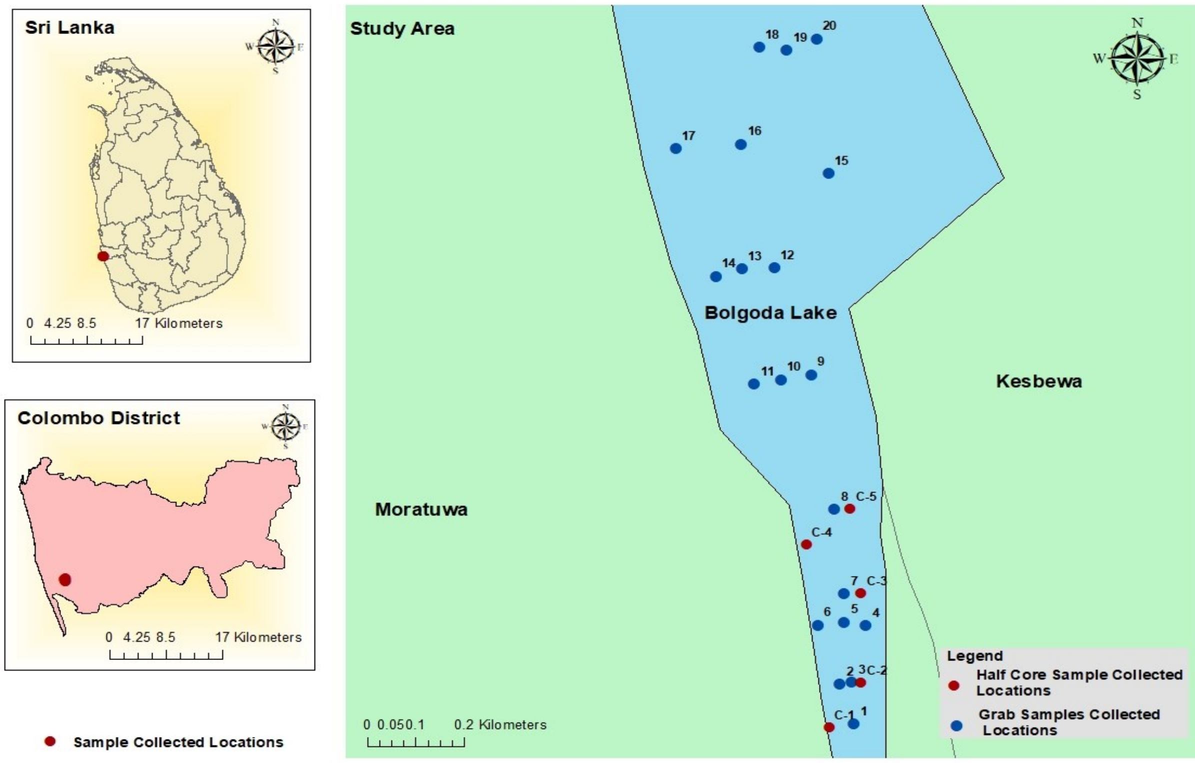

Figure 1: Core sampling and grab sampling locations.

Lake sediments can be categorised as mineral matter's organic matter and inorganic matter. Among these three matters, grain size analysis is done for the mineral maters to identify the sedimentation pattern, to find the origin of the sediment, paleoclimate conditions, etc. [3]. Structure, Texture, and composition are the primary indicators that use to describe the sediment, and they indicate the catchment characteristics, historical conditions, and sediment source, etc. Grain size distribution (GSD) is the main method that is used to identify the critical info about the provenance and nature of the sediment, like transportation and deposition [4]. Hydrodynamic sorting, parent materials, and the way that transport material is indicated by the grain size deposition. As well as GSD is used to identify the mixture of silt, clay, sand in the sediment deposition. The laser diffraction method is the main method used to identify the grain size distribution.

High accuracy results within the nano range can be taken from the GSD. These nanoparticles can be displayed as the 3D image by using the SEM analysis. Also, samples that use GSD analysis can be subjected to the Energy Dispersive X-ray Analysis (EDAX) to identify the elemental distribution of the sediment over the lake [5].

The purpose of this case study is to observe the sediment characteristics and behaviour of the sediment accumulation of the Bolgoda lake by using the core sampling method and grab sampling method. The authors also explore shallow depth environments of the Bolgoda lake by using the acoustic and wave reflection methods.

2. Methodology

2.1 Site Selection

Sri Lanka comprises three main climatic zones called wet, intermediate, and dry zones and also three main morphological regions called coastal lowlands, uplands, and highlands. Bolgoda Lake is one of the main water reservoirs in the western province, which has two major basins explained as Bolgoda south and north lakes. As well as Bolgoda lake is a partially closed low liberation water body [6]. The water flow of the Bolgoda lake finally connects with the Indian Ocean via the large two openings that exist in the western edges.

The study area of our research project is situated near the boatyard of the University of Moratuwa from the 'Karadiyana garbage pit' to the bridge of 'Piliyandala'. According to the seasonal changes, the lake is subjected to floating ferns, and floating plants categorised as genus Eichhorminia, genus Salvinia, Water Hyacinth, etc. [6]. Also, the bathymetry of the water body can be seen in 1-2 m depth in the study area due to heavy sedimentation, but some areas achieved 3m depth.

2.1 Material and Methods

2.1.1 Sampling method, Particle Size Distribution and SEM Analysis

Van Veen grab sampler was used to collect the surface sediment samples; Core samples were also extracted during the boat ride using half corer. When a new sample was taken, the sampler should clean well. Also, samples should name clearly, like Figure 1. There were 42 samples analysed to gain precious and accurate details.

Observed samples were prepared under the ASTM D 6913/D 6913M standards for the particle size analysis with slandered sieving methods. A laser particle analyser (JNGX HMK-CD2) for the granular range of 0.1–1000 μm was used to analyse the prepared 0.35 mg amount of sediment samples. The GDADSTATv.8 program was used to analyse particle size statistics such as mean, classification, curvature, and kurtosis [7].

The remaining prepared samples were subjected to Scanning electron Microscopy. 3D images and energy-dispersive X-ray Analysis (EDAX) were taken to observe the sediment composition.

2.1.2 Bathymetric Surveying

The bathymetric survey was executed to cover the entire study area by using the pre surveying plan that was prepared. Hondex PS 7 that called Spot Depth Echo Sounder, was used to take the Spot Depth measurements with high accuracy. Navigation data were taken by handheld GPS unit with 3 m precision.

Collected data was post-processed using the Surfer 8 and ArcGIS software.

3. Results

3.1 Grain size distribution

Grain size analysis is directly affected by

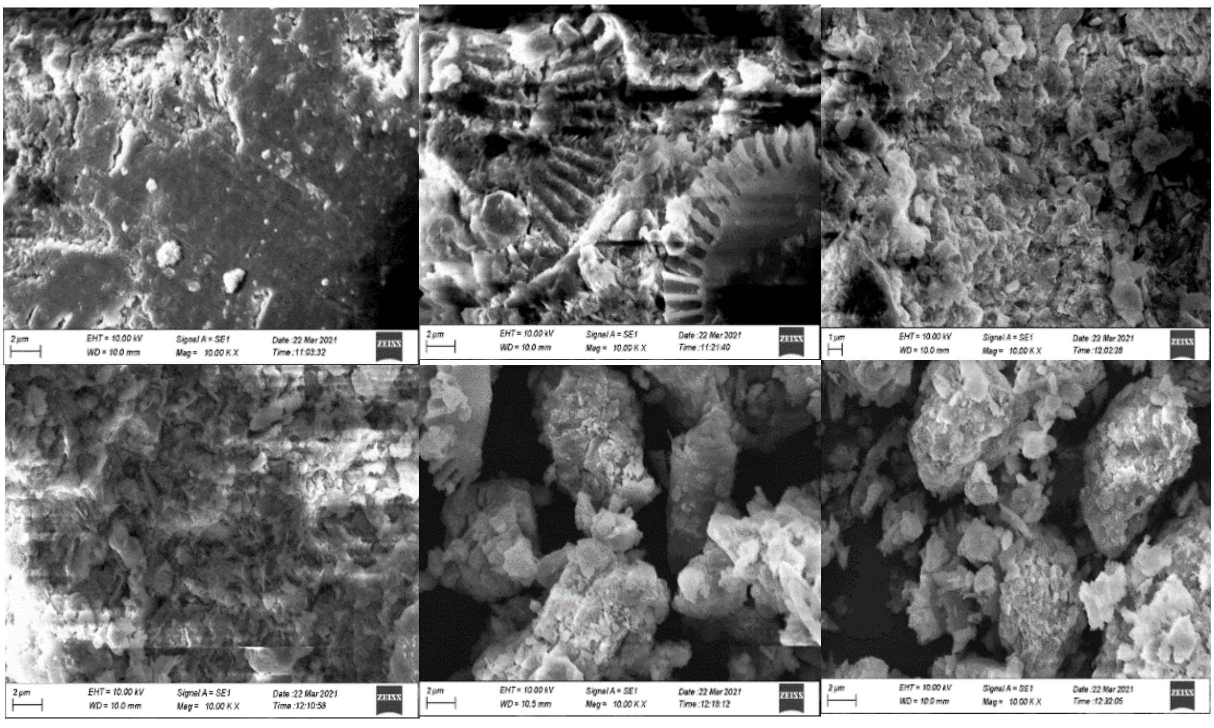

Figure 2: SEM analysis image data.

the lake sediment's geochemical parameters. Also, GSD is given ideas and explanations about the transportation and deposition of the lake sediment. So, the particle size distribution method gives the details about sediment mixture that consists of silt, clay, and sand, etc. The characterisation of the deposit is given by the skewness, mean grain size, kurtosis, and sorting.

The following table shows the grain size distribution taken from the grab sampler and core sampler.

Table 1: Sediment characteristic of the samples (as an average).

Characteristics	Results
Mean	Medium silt
Sorting	Very poorly sorted
Skewness	Very fine skewed
Kurtosis	Platykurtic and very platykurtic

So, the platykurtic nature of the sediment sample observes that two or more particle populations mix together with equal amount or unequal amounts, which explain the polymodal nature of the sediment deposition. It means polymodal sediment mix with silt clay and the sandOnly five samples varied from the other samples because these samples were shown mesokurtic and leptokurtic kurtosis. It means showing a high peak or tail of the grain size distribution curve.

3.2 SEM analysis data

Generally, grain size analysis using nanoscience is one of the important things to gain high accuracy results. We used six samples for the SEM analysis, and 3D images were taken. Samples were taken within the 10*X magnitude to identify characteristics of the sediments. Figure 2 display the high-resolution 3D images that are used to identify the non-circular, 20-100 µm size particles consist in the sediments.

Finally, the composition of the data was obtained by using the EDAX analysis.

According to Table 2, Sample number 7 and sample number 19 are only subjected to the EDAX analysis and observe the weight deference of material in the two locations.

Table 2: eZAF smart quant results.

Element	Weight Percentage%	
	Sample 07	Sample 19
Carbon	15.05	15.75
Oxygen	47.44	49.15
Sodium	0.72	0.50
Magnesium	0.57	0.38
Aluminium	12.63	11.52
Silicon	11.72	14.53
Sulphur	2.33	1.29
Chlorine	0.53	0.43
Potassium	0.40	0.33
Calcium	-	1.17
Titanium	0.63	0.67
Ferrous	7.97	4.27

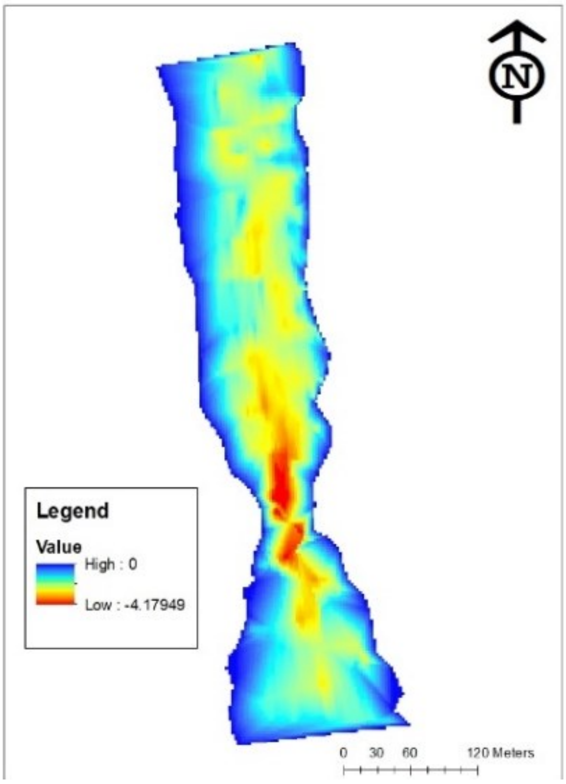

Figure 3: Raster file created using ArcGIS.

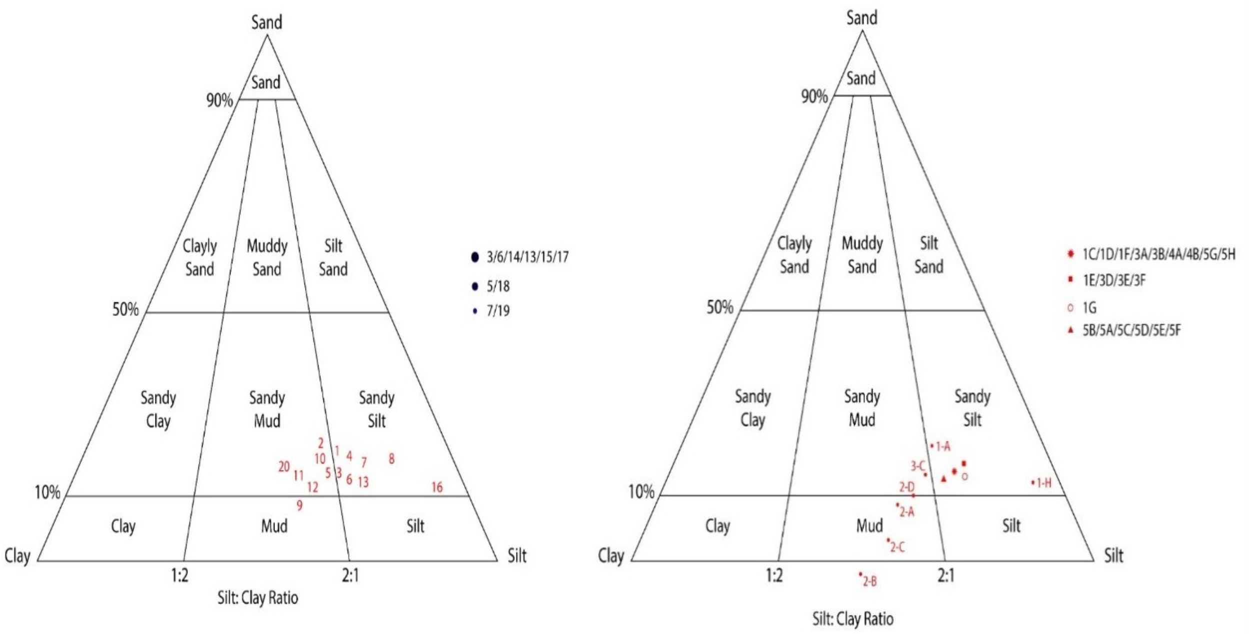

Figure 4: SSC diagram for grab and core sampling.

3.3 Bathymetric Surveying and Map Preparation

Bathymetric Surveying data could not obtain directly. It had to be taken via the post-processing platforms like Surfer 8 and ArcGIS. Using the Surfer 8 3D maps, Sedimentation patterns, shallow and deep areas can be obtained.

Figure 3 shows depth data mapping obtain from the ArcGIS. Also, Figure 4 exhibits the bathymetric analysis that was taken from the Surfer8.

4. Discussion

Folk and Ward's classification can be used to identify the sediment distribution along the lake. Properties of the sediment samples are mentioned in table 1. These results reflect that sediment has not had proper time to sort. Normally lake sediments are well-sorted towards the downstream of the lake, but according to these results, very poorly sorted sediment distribution can be identified towards the downstream. According to the geometric values acquired from the particle size analysis test, a negative skewness value is shown for the data obtained in the lake area. These results also reflect that the lake area is favourable to sediment dispositioning. There is sediment accumulation, but due to the flow rate, accumulated sediment is eroded and distributed downstream. The platykurtic nature of the sediment sample observes that two or more particle populations were mixing with an equal amount or unequal amounts, which explains the polymodal nature of the sediment deposition. It means polymodal sediment mix with silt clay and sand also.

According to [7], sediment types can be classified using the Sand-Silt-Clay(SSC) Diagrams and Gravel Sand Mud Diagram (GSM). It explains the textural group and sediment type of Bolgoda Lake. Figure 5 shows the SSC diagrams for both grab sampling data and core sampling data. According to these data. These results illustrate the Sediment as Sandy Silt and Sandy mud sediment.

SEM analysis results also give an idea about the composition of bottom lake sediment, particle size, shape, and roundness of the sediments. However, these results interpret the same mineral composition around the study area with slightly different weight and atomic percentages. The difference can happen due to the water flow rate and the sediment current of the lake. Table 2 shows the weight percentages near the "Karadiyana garbage pit" and the university area from EDAX. As well, the atomic percentage also displays the same slim difference as the weight percentages. It clarifies that there is a difference in Calcium minerals when comparing the two locations. Calcium may be accumulated due to the effect of the garbage contamination.

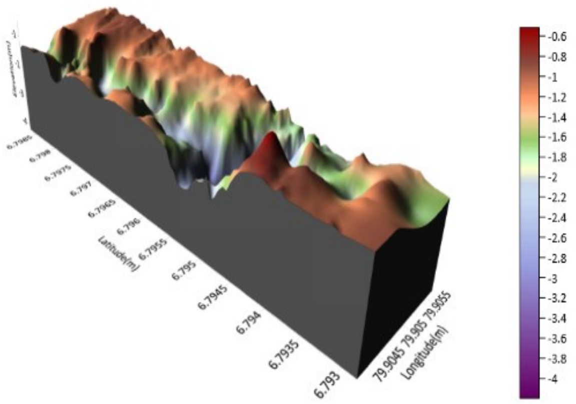

Figure 5: Bathymertic analysis from Surfer8.

It is preferable to do XRD and XRF tests for the sediment sample to extract accurate mineral composition around the lake and to confirm the effect from the Karadiyana garbage pit.

Bathymetric data obtained from Surfer8 is illustrated in Figure 6. The cross-section data obtain from the two places show the sediment accumulation pattern. So, we can ensure that sediments are accumulated in the lakes are like the cross-section of Figure 6. According to bathymetric data, the middle of the lake has a low sedimentation rate, and the side of the lake has higher sedimentation.

5. Conclusions

According to this study, the sediments of the lake mainly come from the upstream area of the Bolgoda lake and due to the erosion of the bank. SSC diagrams show that sandy silt is the sediment that accumulates in the river bottom and geometric values give the idea about sedimentation rate and percentages weights of lake sediments.

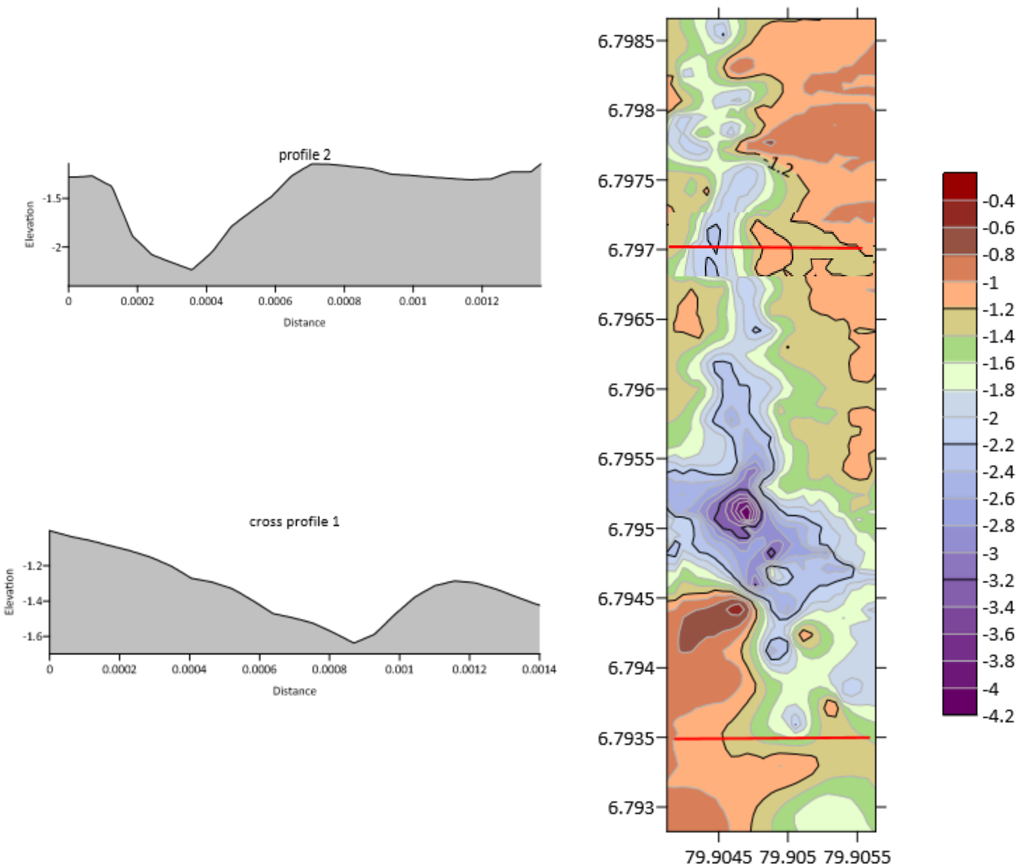

Figure 6: Cross section of the Bolgoda Lake.

It is important to introduce a proper method to minimise the sediment accumulation in the Bolgoda lake to minimise future accumulation.

To reduce the sediment accumulation, mitigation measures have to be done for the upstream area, and a proper urban development plan should be introduced for construction near the lake area.

Future Direction

Bathymetric surveying was done by the Spot depth eco sounder, and research should be done again by using a sub-bottom profiler to identify the sedimentation pattern, sediment thickness, and accumulated capacity.

Identified sediment can be used in the fertiliser industry, construction industry, and ceramic industry, etc. Proper environmental and legal action should be followed before commencing the dredging.

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Investigation of Bolgoda Lake to Establish a Ferry Service for University of Moratuwa

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Abstract

Colombo is one of the major cities which has a high population density with a smaller number of transportation facilities over many years. Even though it has a good canal system, waterway transportation is not implemented for no reason. With the idea of establishing a ferry service from the University of Moratuwa to Panadura along the Bolgoda lake, our research was carried out to determine the main parameters relevant to the aim of the project. Prioritise factors considered are Bolgoda lake bathymetry, identification of restricted areas, highly sensitive areas, material type to be dredged, and possible locations for terminals. Initially, the research area was narrowed down from the boatyard of the University of Moratuwa to the Vijayarama temple. There are several ways to determine the bathymetry of a water body, and echo-sounding techniques were used in our research. Bathymetry of the region was determined with cut volume to be dredged to make the ferry path safe using "Hondex PS 7" echo sounder and Surfer software. "ArcGIS" software was utilised to generate maps, and "Slope W" software was used in analysing dredge slope and its stability. The research was carried out during the dry season, and the water level was indicated as 4.2 m on the gauge at the bridge.

Keywords: Bathymetry, Bolgoda Lake, Echo sounding, Ferry

1. Introduction

Lakes are known as water bodies surrounded by land and water in them is less mobile than in a river. Lakes are more important in storing water in lowlands and urban areas with less hazards. Bolgoda Lake is situated in the South West of Sri Lanka, 19 km from the city of Colombo. It is one of the major water resources in the Colombo district. The lake consists of two major water bodies and covers 374 km^2, partly fresh and brackish water. The largest freshwater lake on the island, the Bolgoda lake, stretches from Anguruwatota to Piliyandala. The northern section of the waters stretches from Colombo to Kalutara while the south side of the lake finds its way to the open seas through the Thalpe canal [1]. It has a great potential of developing tourism with its natural beauty and fisheries in the nearby area [2]. Bolgoda Lake has high biodiversity and has become a home to many animals and fish kinds as well.

Colombo district is a highly populated area with the very limited land facility. Since the high population density, transportation

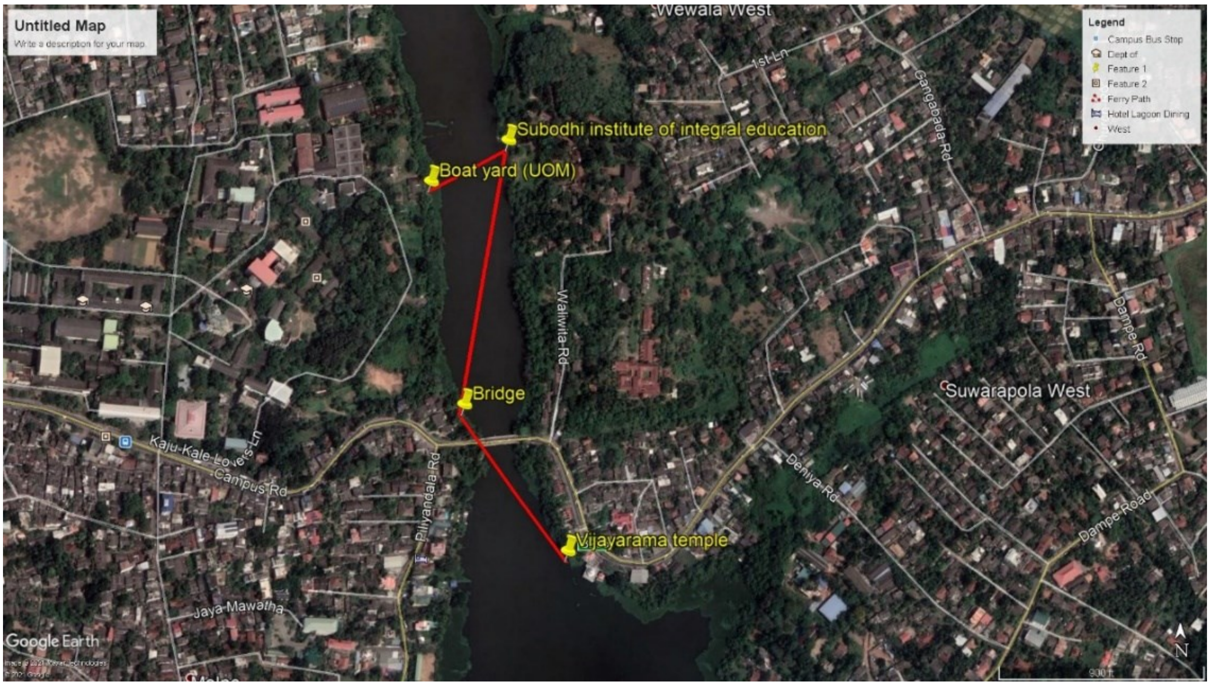

Figure 1: Initial model of ferry service.

facilities should have to be efficient and adequate. The main method of transportation in this area are buses, railways, and private vehicles. Even though trains have the capacity to carry more passengers, located away from the inner cities lead to occur heavy traffic conditions on roads. Ferry transportation will be a great opportunity to have an extra choice for people with less obstacles than roads which leads to saving much time [3].

The research is to determine the potential of Bolgoda lake and its environment to establish a ferry service for the University of Moratuwa. To have a safe and reliable ferry transportation, the path needs to be well suitable. The under-keel clearance, obstacles and restricted sensitive areas should be pre-identified. A detailed bathymetric survey provides such information in a certain path to determine the above-mentioned factors.

2. Methodology

2.1. Study area

This research is to determine the potential for establishing a ferry service from the University of Moratuwa to Panadura. We have identified suitable potential places to establish ferry terminals between University and Panadura through Bolgoda Lake. After that, it is narrowed down to the Vijayarama temple from the University of Moratuwa as the initial stage (Figure 1).

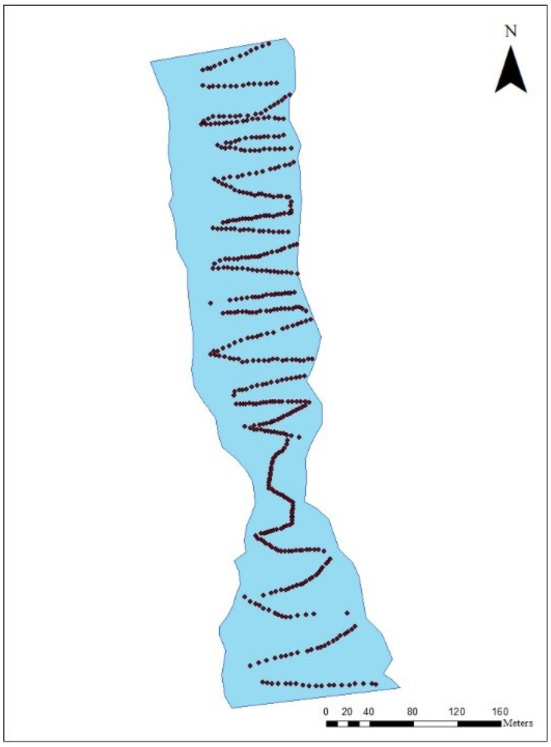

Figure 2: Actual survey path.

2.2. Materials and methodology

2.2.1. Data acquisition

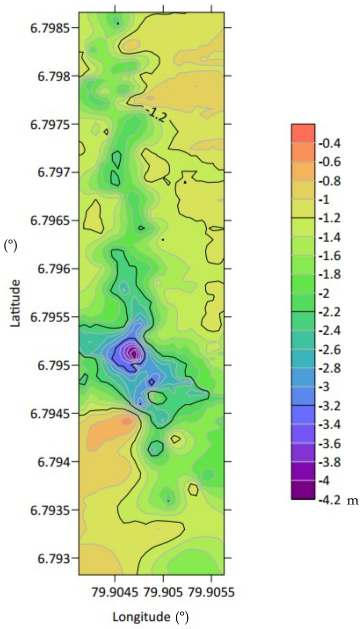

Figure 3: Contour map of the region.

International Hydrographic Organization has declared the types of hydrographic surveys and required features that should be followed to be a standard survey. It provides safety for navigating across the surveyed areas to an expected level. According to the required accuracy and depth limitations, the survey method is varied among Special order, Order 1A, Order 1B and Order 2 [4]. Acoustic waves, either sonic or ultra-sonic, achieve good penetration and propagation through all elastic media once these media can be made to vibrate when exposed to pressure variations. Most of the sensors used for depth determination use acoustic waves. There are several conventional and sophisticated ways of determining the depth values of a water body. Handheld rope and lead method, Echo sounding methods (Simple echosounder, single beam echosounder, Multibeam echosounder, Acoustic Doppler Current Profiler and Sub-bottom profilers) and other methods (lidar and remote sensing) [5]. Bathymetric Surveys were carried to cover the whole area as per Figure 2. Spot depth Echo Sounder (Hondex PS-07) was used for spot depth measurements. Corresponding GPS locations were captured using Magellan eXplorist 510 GPS receiver. The survey was done on March 25th, 2021, when the water level of the gauge at the bridge was indicated as 4.2 m.

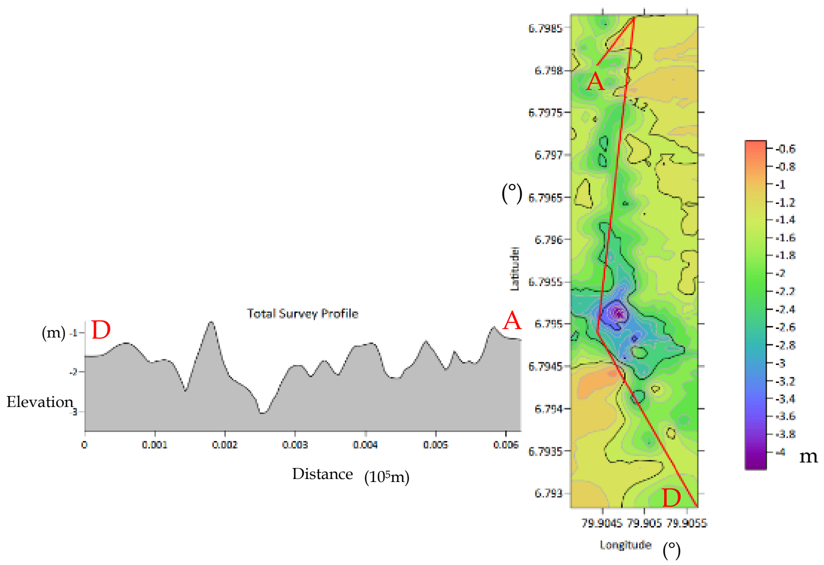

Figure 4: Depth variation along the path.

2.2.2. Data processing

Collected data was arranged using "MS-Excel" software to a CSV (Comma Separated Values) file, and post-processing was done with "Surfer", "ArcGIS" and "Slope W" software platforms to generate contour map (Figure 3), depth variation profiles (Figure 4), 3D model (Figure 6) and determine the Cut volume and slope parameters (Figure 5).

3. Results

It can clearly be identified that the bridge area is deeper than anywhere else in the region.

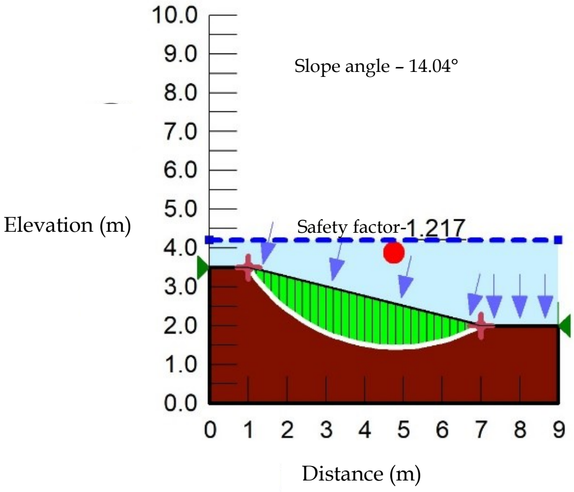

Figure 5: Proposed slope design.

Depth variations between each proposed terminal is given above (Figure 4). All the distance values are in 10^5 m.

It was selected that 2.2 m depth for dredging from the water level for easiness of ferry transportation. The cut volume was observed as 3484.5326601540 m^3 using "Surfer" software according to the Trapezoidal rule.

There should be nearly 10° angle for the slope to have a factor of safety as 1.2 (Figure 6). The material type was taken as sandy clay.

4. Discussion

As the expected objectives of our research, A dredging process should be carried out to remove material at shallow areas, which was mostly along the border of Bolgoda Lake. There were few private and single public land near the proposed terminal locations in the considering zone. The water column has very high turbidity; therefore, it was identified some false reflections during the survey. It was identified that the main problem in this area is the water hyacinth population. It can be packed well like a very thick cover. The actual path was narrowed to the middle region of the lake due to very shallow depths, growth of plants and boat turning radius.

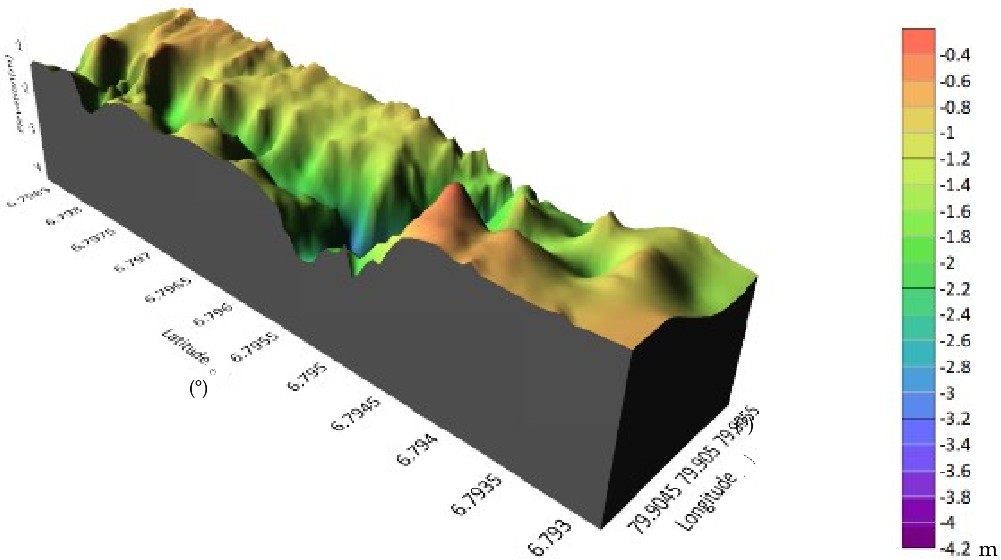

Figure 6: 3D model of the lake bottom in the region.

5. Conclusion

It is possible to establish a ferry service from Panadura to the University of Moratuwa, and the selected path and selected ferry terminal locations are free from disturbances or restrictions. By continuing development of the basic ferry by research work, it should be able to develop either to power by solar + fuel (Hybrid) to cut down the cost and also to minimise the environmental pollution. A stable slope was designed, and there is a much lower angle than we expected. It may be limited to some value due to the width of the lake, and the safety factor would be a low value than this. Therefore, frequent dredging has to be carried out along the path to remove the collapsed material. Otherwise, a proper stabilising process can be implemented. The excavated soil can be utilised as a construction material after carrying out tests and proper beneficiation process. Once the ferry service is established, the next step should be linking this service with other paths of transport facilities such as busses and trains, which will be a nationally important model to establish integrated transport facility accessible even from a website for bookings and travel planning by the individuals.

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