INFLUENCE OF MECHANICAL AND AGGREGATE PROPERTIES OF ROCK ON POWDER FACTOR IN ROCK BLASTING

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DECLARATION OF THE CANDIDATES AND SUPERVISORS

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ABSTRACT

Quarry metal is a widely used material in any large scale construction industry. Though demand for quarry metal substantially increased due to recently emerged large infrastructure development projects in Sri Lanka such as highway and port city, profit margins of the quarrying has drastically reduced due to high powder factors recorded in recent rock blasting activities of local quarries. Most possible reason for realizing high powder factors could be the introduction of various new explosive agents such as water-gel in to the local mining industry. Importance of analysing the influence of mechanical and aggregate rock properties on powder factor arises in this context to allow seeking suitable rocks those incur minimum blasting expenditure during the production stage. Outcomes of the project can be used to predict powder factor which could be achieved in blasting operations of a certain rock even before starting the quarry operations to minimize its production cost in the long run.

Eight quarries operates under the close supervision of qualified Mining Engineers were selected for this study to ensure blasting geometry and configurations have being properly managed during the realized powder factor data recorded time period. Random core samples were obtained from each quarry site and they were tested for Density, Uniaxial Compressive Strength (UCS) and Tensile Strength. Similarly, random aggregate samples were taken and performed the Aggregate Impact Value (AIV) test. Rock Mass Rating (RMR) was determined for each quarry using UCS values and other field data obtained at the site. Explosive consumption and drilling records for recent six month were obtained from each quarry for the calculation of powder factor.

Powder factor was plotted against each selected rock property and regression analysis was performed on test results to understand their standalone influence. The only realized best fitting model for the Powder Factor was AIV according to the regression analysis and it is very closely following the quadratic model. Rock property test results and past records of few other quarries were used to validate formulae obtained in this research. Explosive cost and production cost of all the quarries analysed for the same six months period. Results revealed that the production cost is a function of explosive cost since other costs on drilling, machineries and labour are usually incur relatively fixed costs in nature. Hence it can be concluded that the aggregate rock properties, especially Aggregate Impact Value (AIV) influence on powder factor of blasting and furthermore affects economics of the quarry production. More importantly, combined formula derived in this research can be used predicting powder factor of a fresh rock before conducting any blasting activity.

Keywords: Aggregate Impact Value (AIV), Powder Factor in blasting, metal quarrying

DEDICATION

I dedicate this research work to the Sri Lankan mining industry which uplifted my living standard, provided me qualifications plus recognition and continuously energized me to thrive my career thus far.

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Finally yet importantly, I appreciate my wife for lending her devoted time to me during the last few years to read for the MSc and encouraging me to complete this final hurdle. This dissertation stands as a testament of her unconditional love and encouragement.

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LIST OF ABBREVIATIONS

Abbreviation	Description
ANFO	Ammonium Nitrate with Fuel Oil
CEA	Central Environmental Authority
GSMB	Geological Survey and Mines Bureau
UCS	Uniaxial Compressive Strength
AIV	Aggregate Impact Value
RMR	Rock Mass Rating
ACV	Aggregate Crushing Value
RQD	Rock Quality Designation
Pf	Powder factor
DS	Divisional Secretariat
IML	Industrial Mining License
AML	Artisanal Mining License
TNT	Tri Nitro Toluene

LIST OF APPENDICES

1. CHAPTER 01: INTRODUCTION

Importance of sustainability in the Sri Lanka metal quarry industry is discussed in the chapter 01. Further, introduction to the research problem, objective of the research, method of tackling, expected outcomes mentioned and significance of the research is briefly introduced in this chapter.

1.1 Sustainability of Sri Lankan metal quarry industry

Generally rocks are classified under three major types based on their formation process and those are; igneous rocks, sedimentary rocks and metamorphic rocks. Approximately 85% of Sri Lankan terrain is made up of highly crystalline, non-fossiliferous rocks which are fall under metamorphic rock types hence quarry metals produced all over the country are classified under the same category.

Constant and adequate supply of quarry metal or construction aggregates is a vital requirement to continue civil engineering projects without delay in any context. Recently emerged mega projects in Sri Lanka such as port city, highway extension and multistory residences and hotel complex developments have drastically increased the construction aggregate demand prevailed in the market during last few decades. On the other hand, metal quarry industry is challenged by various technical as well as non-technical issues which are directly impact to the economics of the operations. Few such technical issues are; finding rocks which can produce suitable construction aggregates to the civil construction industry, comparatively low metal yield achieve per blast using newly introduced explosive in the local market and rock fragmentation control issues by manipulating explosive loading in the blasting configuration. Protests by environmentalists against environmental impacts caused by the metal quarry industry, complains and objections by the surrounded inhabitants to continuous blasting operations, increasing expenditure on environmental remediation and impact mitigation measures are few of the non-technical issues faced by the local metal quarry industry at the moment. Technical over and above non-technical issues finally boils down on cash flow of the business while affect on profitability of the business will discourage the quarry operators and avert them from quarry metal industry may cause supply shortage as consequence in future. Therefore, sustainability and smooth operation of the metal quarry industry is an essential need to continuation of ongoing and upcoming mega development projects in Sri Lankan.

Allow producing construction aggregates up to the national requirement with minimum environmental as well as social impact while guaranteeing acceptable economics to the quarry operators is one of the greater challenges faced by the Sri Lankan mining industry regulator at the moment. Creating a balance among metal production, surrounded environment and profits of the business are the three main concerns in the quarry metal industry at the moment. Performing research in this context and inventing technological solutions to tackle the issue is a great responsibility of the local Mining Engineering community in this regard.

1.2 Research problems

Similar to any other generic economic model of a business, quarry operation generates income, involves capital and operational expenditure in its cash flow. Capital expenditure is fixed cost incurred and depreciated in the long run such as cost of crusher plant, track drills, excavators, loaders etc and does not vary with the time. Operational costs are variable costs incurred on the daily run and varying with the metal production such as explosive cost, drilling cost, salaries and wages. Profitability of the quarry operation is highly sensitive to operational costs than capital costs.

Although, drilling cost and wages considered as less fixed costs for the accounting purposes of a quarry operation, they are fixed in incurring nature. This scenario can clearly be observed when individually compared total costs versus above two costs of several blasts carried out during a specific period in a particular quarry site. Mining Engineers tend to control blast fragmentation by altering the explosive quantity is the main reason behind this observation. Variation of drilling cost and labour cost are negligible in these cases since number of drilled holes and labour allocation remains unchanged or varies in a minimal way. Secondary blasting cost and rock breaker operating costs are two additional operational costs incurring in a quarry when primary blasting work does not generate required rock fragmentation. Therefore, it can be clearly identified that the effectiveness of a blast i.e. amount of explosives used per blast, metal yield and quality of fragmentation directly affects production cost, consequently profitability of the quarry operation.

Relationship between metal yield per blast and the quantity of explosive used is defined as “Powder Factor” in the Mining Engineering theories. Fragmentation and powder factor shows positive relationship according to the Mining Engineering literature hence it is important to research on factors effecting powder factor of rock blasting. Nevertheless, there are number of empirical equations and proven theories existing on rock blasting powder factor control by manipulating blasting geometry configurations such as spacing, burden, sub drilling, charge length, charge type, stemming height and hole diameter. However, no comprehensive study had been done on influence of mechanical and aggregate rock properties on powder factor thus far.

1.3 Objectives

Objectives of this study are to analyse the influence of mechanical and aggregate properties on powder factor of rock blasting in metal quarry industry and to identify relationships of powder factor to rock density, Uni-axial Compressive Strength (UCS), tensile strength, Aggregate Impact Value (AIV) and Rock Mass Rating.

1.4 Method of tackling

Eight IML A (Industrial Mining Licence A) category quarry sites located in the Colombo district were selected for the research related testing and record collection for the analysis. Since there are existing relationships for rock blasting, powder factor with blasting geometry configurations such as spacing, burden, sub drilling, charge length, charge type, stemming height and hole diameter; there is a need of making constant these parameters to nullify their effect on powder factor variations.

All the above quarries operate under the close supervision of qualified Mining Engineers, hence it was assumed that the blasting geometry and configurations have being properly optimized during the realized powder factor data recorded time period. Therefore, the variations of recorded powder factor, even after blast optimization of a qualified Mining Engineer, were assumed due to geomechanical and aggregate rock property variation inherent to the rock mass. Random core samples were obtained from each quarry site and they were tested for Density, Uniaxial Compressive Strength (UCS) and Tensile Strength. Similarly, random aggregate samples were taken and performed the Aggregate Impact Value test (AIV). Rock Mass Rating (RMR) was determined in each quarry using UCS values and other field data obtained in the site. Explosive consumption and drilling records for recent six months were obtained from each quarry for the calculation of powder factor.

Density was calculated by measuring weight and mass of the intact rock sample. Uniaxial Compressive Strength (UCS) of rocks was directly measured in the laboratory using UCS apparatus. Tensile strength of rock samples was measured using Brazilian test which is most common tensile test is the industry. Aggregate Impact value (AIV) obtained by performing aggregate impact value test.

Mathematical tools such as correlation coefficient and regression analysis were used to analyse the behavior of powder factor against above mentioned mechanical as well as aggregate rock properties. State-of-the-art statistical software “Minitab” was used to perform all the statistical data analysis tasks to confirm the accuracy and confidence level of the research results.

1.5 Significance of the research and expected outcome

Research results will open up an avenue to select a suitable rock mass for a quarry site among several available options based on expected powder factor of rock blasting. The method allows Mining Engineers to select rock which produce quality fragmentation while giving out low powder factor in rock blasting operations and it helps quarry investors to project their future potential operational costs and forecast cash flows as well as profits for the same site. Outcome is an indirect invention to control fragmentation as well as cost of the blasting operations while operational cost controlling methods for the metal quarry industry.

2. CHAPTER 02: LITERATURE REVIEW

An extensive literature survey was carried out on research conducted, thus, far related to mechanical and aggregate properties of the rocks in the mining industry and connected to the powder factor. Important findings of the literature survey are summarized under this chapter. Further, ASTM (American Society for Testing and Materials) standards for each and every rock property test were referred and relevant standard were mentioned under the sub sections where testing procedure is described.

2.1 Powder factor in rock blasting

The use of explosives in mining goes back to the year 1627 when gunpowder was first used in place of mechanical tools in the Hungarian (now Slovakian) town of Banská Štiavnica. The innovation spread quickly throughout Europe and the America. Drilling and blasting saw limited use in pre-industrial times using gunpowder and later realized its potential after inventing safer and powerful explosives such as dynamite and employed powered drills (Buffington 2000). Usage of drilling and blasting techniques evolved with the new inventions and developments in the mining industry. Improvements led conducting blasting operations with minimal cost and damages to the environment while achieving higher production rates of rocks. Amount of explosives used per single blast is a critical and vital factor for all financial, environment and social factors in any mining operation since it differ the level of disturbance to the surrounding inhabitants, severity of environment impact while cost incur to the operator.

Powder factor of rock blasting is defined as requirement of explosive in kilograms to excavate a unit volume of rock in a particular blast and possesses units of kilograms per cubic meter. It is also a measurement of effectiveness of the rock blasting activities. If the powder factor is low, that implies a successful blast where unit volume of rock excavated by spending minimal amount of explosives and vice versa. Usually tunnels and chamber excavations record relatively high powder factors compared to surface blasting activities which are having more than two free faces to throw. (Dick et al 1983)

2.2 Various properties of rocks

Rock properties can be categorized under different criterion. Governing principles are based on chemical, physical, mechanical and mechanical characteristics of rocks. Using either criterion, one should be able to clearly distinguish rocks from each other and capable of selecting them for a specific purpose. Selection of appropriate rock properties are based on the purpose or applicable field but it is useful to understand all these properties in general.

2.2.1 Chemical properties of rocks

Every mineral or rock has its unique chemical composition and can be expressed as an identical chemical formula. In most cases minerals are salts composed of positively charged cations such as K^{+}, Na^{+}, Ca^{+2} or Fe^{+3} and negatively charged anionic groups like CO_3 or PO_4,.other than native elements. Silicates are the largest groups of minerals consist in rocks and about 86% of the earth's volume is made out of silicate.

2.2.2 Physical properties of rocks

Physical characteristics of rocks are often referred as physical properties of rocks. Most of the physical characteristics are measurable according to universal scale hence can be used to distinguish rocks from one another. Most common rock properties applicable in the mining industry are listed below;

a) Density, porosity and saturation
b) Hardness
c) Abrasivity
d) Permeability
e) Streak
f) Colour and Luster
g) Cleavage

2.2.3 Mechanical properties of rock masses

Rock behaviour when subjected to either natural or artificial forces, else geological phenomena is the focus of mechanical property behaviour. Compressive strength, tensile strength and shear strength measurements of a certain rock provides its characteristics when subjected to natural or artificial, compression or tension stresses or strain. Mechanical properties of rock masses usually tested in mining engineering designs are listed below. Rock Mass Rating (RMR) specifically evaluates changes of rock behaviour when subjected to various geological phenomena such as fracture intensity, condition and ground water fluctuations as a whole.

a) Compressive Strength
b) Tensile Strength
c) Shear Strength (Heinio 1999)
d) Rock Mass Rating (Strenght of intact rock, drill core quality, spacing of discontinuties, condition of discontinuties and ground water conditions)

2.2.4 Aggregate rock properties

Aggregate rock characteristics that are considered in the construction industry ranges from grading, durability, particle shape and surface texture, abrasion and skid resistance, unit weights and voids, absorption and surface moisture etc. However, Aggregate Impact Value (AIV) test is the widely applicable aggregate rock property test practiced in the civil construction industry, especially in highway and road engineering disciplines. The aggregate impact value indicates a relative measure of the resistance of the aggregate to a sudden shock or impact which in some aggregates differs from its resistance to a slow compressive load.

2.3 Rock properties focused in the research

During the research, random core samples were obtained from each quarry site and they were tested for Density, Uniaxial Compressive Strength (UCS) and Tensile Strength. Similarly, random aggregate samples were taken from each quarry site and performed the Aggregate Impact Value (AIV) test. Rock Mass Rating (RMR) was determined at each quarry using UCS values realized during the above laboratory tests and other field data such as drill core quality, spacing of discontinuties, condition of discontinuties and ground water conditions obtained at the site.

2.3.1 Density

Density is a common physical property which is influenced by the specific gravity of the minerals constituent and the compaction of the minerals. It is a measure of mass per unit of volume. Density of rock material varies, and often related to the porosity of the rock. It is sometimes defined by unit weight and specific gravity. Granite rocks have density between 2,500 to 2,800 kg/m^3 (Bell 1992). ASTM D 6473 method is used in this research in finding the density value of each rock sample.

2.3.2 Uniaxial Compressive Strength (UCS)

The uniaxial compressive strength of rock is one of the simplest methods to determine the strength of rock sample. It may be regarded as the highest stress that a rock specimen can carry when a unidirectional stress of even load distribution is applied, normally in an axial direction, to the end of the cylindrical specimen. The specimen should be prepared on a requirements of flatness of the end surfaces in order to obtain an even load distribution. In other words the unconfined compressive strength represents the maximum load supported by the specimen during the test per cross sectional area of the specimen. Although its application is limited, the unconfined compressive strength allows comparisons to be made between rocks and affords some indication of rock behavior under more complex stress systems.

The behavior of rock in uniaxial compression is influenced to some extent by the tests conditions. The most important of these is the length diameter ratio or slenderness ratio of the specimen. Dhir and Sangha (1973) mentioned that the most satisfactory slenderness ratio as2.5. At lower ratios, fractures take place in highly restrained specimen ends, while at higher ratios there is an undesirable release of elastic strain energy from the unfractured ends region to the fractured central zone during post failure stressing. In other words such a ratio provides a reasonably good distribution of stress throughout. However, Obert and Duvall (1967) had previously suggested the use of the following empirical expression to relate the uniaxial strength to the length – diameter ratio:

$$\sigma_c = \frac{\sigma_{act}}{0.788 + 0.222 \;{}^{D}/_{L}} \qquad \text{Eq (01)}$$

Where;

σ_c is the Compressive strength of a specimen of the same material having a 1:1 length diameter ratio, D is the diameter of the sample and L is the length of the sample.

σ_{act} is the compressive strength of a specimen for which $1/3 < L/D < 2$.Indeed Obert and Duvall reported that as far as the uniaxial compression of cylindrical specimens is concerned, the size of specimen of cylindrical specimens is concerned, the size of specimen has less effect than the natural variation in the values obtained from testing a given rock type when the specimen length – diameter ratio is kept constant. An approximate relationship between uniaxial compressive strength (σ_c) and specimen diameter (for specimens up to 200mm diameter) is given by;

$$\sigma_c = \sigma_{c\,50} \times \frac{50}{D} \qquad \text{Eq (02)}$$

Where;

$\sigma_{c\,50}$ is the uniaxial compressive strength of a 50 mm diameter specimen and D is the actual diameter of the specimen in millimeters (Bell 1992).

ASTM D 2936 method is used in this research to test the uniaxial compressive strength of rock samples.

2.3.3 Tensile Strength

Rocks have a much lower tensile strength than compressive strength. Brittle failure theory predicts a ratio of compressive strength / tensile strength of about 8:1 but in practice it is generally between 15:1 and 25:1. The direct tensile strength of rock has been obtained by attaching metal end caps with epoxy resins to specimens, which are then pulled in to tension by wires. In direct tensile tests the slenderness ratio of cylindrical specimens should be 2.5 – 3.0 and the diameter preferably should not less than NX core size (54 mm). The ratio of diameter of specimen to the largest grain in the rock should be at least 10:1 (Bieniawski and Hawkes 1978). Unfortunately the determination of the direct tensile strength has proved difficult since a satisfactory method has not been devised to grip the specimen without introducing bending stresses. Accordingly most tensile tests have been carried out by indirect methods.

In the flexural test a cylindrical specimen of rock is loaded between three points at a rate of 1.4 MPa/min until the sample fails. The flexural tensile strength (T_f) is then given by the expression as follows.

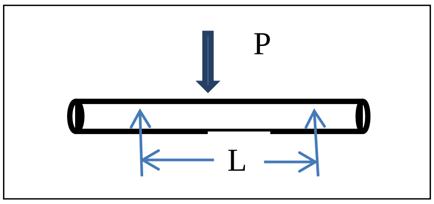

Figure 2.1: Failure load test configuration

$$T_f = \frac{8PL}{\pi D^3} \qquad \text{Eq (03)}$$

Where;

P is the failure load, L is the length between supports and D is the diameter of the specimen which is undergoing testing.,

In the Brazilian test a rock cylinder of length (L) and diameter (D) is loaded (with a load, P) in a diametrical plan along its axis. The sample usually fails by splitting along the line of diametrical loading and the tensile strength (T_b) can be given as follows.

$$T_b = \frac{2P}{\pi LD} \qquad \text{Eq (04)}$$

The use of the Brazilian test as an indirect method of assessing the tensile strength of the rocks is based on the fact that most rocks in biaxial stress fields fail in tension when one principle stress is compressive. Failure, however, may be brought about by localized crushing along the axis of loading and not by diametral tension.

Disc shaped specimens are used in the Brazilian test. Curved jaw loading are sometimes used when disc are tested, in attempt to improve loading conditions. Uncertainties associated with the premature development of failure are sometimes removed by drilling a hole in the center of a disk shape specimen. This has sometimes been referred to as the “ring” test.

The International Society for Rock Mechanics (ISRM) recommends that when a disc shaped specimen is used, it is wrapped around its periphery with one layer of masking tape (Bieniawski and Hawkes 1978). In such cases the ISRM also recommends that the specimen should not be less than NX core size (54 mm in diameter) and that the thickness should be approximately equal to the radius of the specimen. The tensile strength (T_b) of the specimen is obtained as follows.

$$T_b = \frac{0.636P}{DH}\ 0.636 \qquad \text{Eq (05)}$$

Where,

P is the load at failure measured in Newton, D is the diameter of the test specimen measured in millimeters and H is the thickness of the specimen measured at the center measured also in millimeters.

Mellor and Hawkes (1971) stated that the Brazilian test is useful for brittle materials but for other materials the test may give wholly erroneous results. Furthermore, Fairhurst (1964) concluded that the uniaxial tensile strengths of materials with low compression/tension ratios is underestimated by Brazilian tests in which radial loading is applied to disc – shaped specimens (Bell 1992). ASTM D 3967 is used in this research to test the Tensile Strength of rock samples.

2.3.4 Aggregate Impact Value (AIV)

The aggregate impact value indicates a relative measure of the resistance of the aggregate to a sudden shock or impact which in some aggregates differs from its resistance to a slow compressive load

$$AIV = \frac{(\text{Weight of fines passing 2.36mm sieve after test})}{(\text{Weight of the sample})} \text{ x } 100 \quad \text{Eq (06)}$$

Understanding of AIV is very important for construction field and quarry operations. Huge amount of various aggregate varieties are needed for range of development projects and aggregate impact value confirms the suitability of its application for the project. In the road construction field, the main raw material is aggregate or crushed rock. So the suitability of aggregate for the road construction plays a dominant role. Recommended AIV values for the road construction projects are tabulated in the Table 2.1 below.

Table 2.1: Permissible Aggregate Impact values

Aggregate Impact Value	**Classification**
< 20%	Exceptionally Strong
10 – 20%	Strong
20-30%	Satisfactory for road surfacing
> 35%	Weak for road surfacing

Influence of Aggregate Impact value to the powder factor is very important in the quarry industry. If the AIV varies according to the powder factor, it will affect the production cost of the quarry and wear and tear of parts of the crusher plant. This research intends to include and assess the effect of AIV to powder factor variations in rock blasting.

2.3.5 Rock Mass Rating Value (RMR)

The rock mass rating (RMR) system is a rock mass quality classification developed by South African Council for Scientific and Industrial Research (CSIR), closely associated with excavation for the mining industry (Bieniawski 1973). Originally, this geomechanics classification system incorporated eight parameters. The RMR system is used now incorporating five basic parameters as indicated below.

a) Strength of intact rock material: Uniaxial compressive strength is preferred. For rock of moderate to high strength, point load index is acceptable.
b) Rock Quality Designation (RQD) Value.
c) Spacing of joints: Average spacing of all rock discontinuities is used.
d) Condition of joints: Condition includes joint aperture, persistence, roughness, joint surface weathering and alteration, and presence of infilling.
e) Groundwater conditions: It is to account for groundwater inflow in excavation stability.

In the rock blasting, the understanding of RMR value is a very important factor. The RMR value represents the influence of the discontinuities and joints on stability. Powder factor directly depends on the above five basic parameters. The RMR value can be calculated by making use of the graph given below (Hoek 1994). In this study, the ASTM D 5878 – 08 method is used for finding the Rock Mass Rating Value.

Table 2.2: Rock Mass Rating system

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	Parameter		Range of values				
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa 1 - 5 MPa < 1 MPa
	Rating		15	12	7	4	2 1 0
2	Drill core Quality *RQD*		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%
	Rating		20	17	13	8	3
3	Spacing of discontinuities		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm
	Rating		20	15	10	8	5
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1-5 mm Continuous	Soft gouge >5 mm thick or Separation > 5 mm Continuous
	Rating		30	25	20	10	0
5	Ground water	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125
		(Joint water press)/ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing
	Rating		15	10	7	4	0

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)

Strike and dip orientations		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
Ratings	Tunnels & mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

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

D. MEANING OF ROCK CLASSES

Class number	I	II	III	IV	V
Average stand-up time	20 yrs for 15 m span	1 year for 10 m span	1 week for 5 m span	10 hrs for 2.5 m span	30 min for 1 m span
Cohesion of rock mass (kPa)	> 400	300 - 400	200 - 300	100 - 200	< 100
Friction angle of rock mass (deg)	> 45	35 - 45	25 - 35	15 - 25	< 15

E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions

Discontinuity length (persistence) Rating	< 1 m 6	1 - 3 m 4	3 - 10 m 2	10 - 20 m 1	> 20 m 0
Separation (aperture) Rating	None 6	< 0.1 mm 5	0.1 - 1.0 mm 4	1 - 5 mm 1	> 5 mm 0
Roughness Rating	Very rough 6	Rough 5	Slightly rough 3	Smooth 1	Slickensided 0
Infilling (gouge) Rating	None 6	Hard filling < 5 mm 4	Hard filling > 5 mm 2	Soft filling < 5 mm 2	Soft filling > 5 mm 0
Weathering Ratings	Unweathered 6	Slightly weathered 5	Moderately weathered 3	Highly weathered 1	Decomposed 0

F. EFFECT OF DISCONTINUITY STRIKE AND DIP ORIENTATION IN TUNNELLING**

Strike perpendicular to tunnel axis		Strike parallel to tunnel axis	
Drive with dip - Dip 45 - 90°	Drive with dip - Dip 20 - 45°	Dip 45 - 90°	Dip 20 - 45°
Very favourable	Favourable	Very unfavourable	Fair
Drive against dip - Dip 45-90°	Drive against dip - Dip 20-45°	Dip 0-20 - Irrespective of strike°	
Fair	Unfavourable	Fair	

2.3.6 Rock Quality Designation index (RQD)

The Rock Quality Designation index was developed by Deere (Deere et al 1967) to provide a quantitative estimate of rock mass quality from drill core logs. RQD is defined as the percentage of intact core pieces longer than 100 mm (4 inches) in the total length of core. The core should be at least NW size (54.7 mm or 2.15 inches in diameter) and should be drilled with a double-tube core barrel. The correct procedures for measurement of the length of core pieces and the calculation of RQD are summarized in Figure 2.2

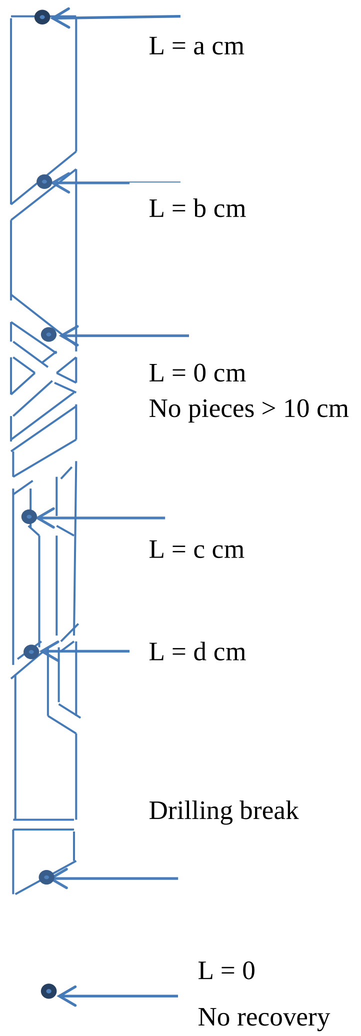

Figure 2.2: Rock Quality Designation evaluation method

$$RQD = \frac{\sum \text{Length of core pieces} > 10\ cm\ length}{\text{Total Length of core run}} \text{ x } 100 \quad \text{Eq (07)}$$

Palmström (1982) suggested that, when no core is available but discontinuity traces are visible in surface exposures or exploration adits, the RQD may be estimated from the number of discontinuities per unit volume. The suggested relationship for clay-free rock masses is:

$$RQD = 115 - 3.3\ J_v \qquad \text{Eq (08)}$$

Where,

J_v is the sum of the number of joints per unit length for all joint (discontinuity) sets known as the volumetric joint count.

RQD is a directionally dependent parameter and its value may change significantly, depending upon the borehole orientation. The use of the volumetric joint count can be quite useful in reducing this directional dependence. RQD is intended to represent the rock mass quality in situ. When using diamond drill core, care must be taken to ensure that fractures, which have been caused by handling or the drilling process, are identified and ignored when determining the value of RQD. When using Palmström's relationship for exposure mapping, blast induced fractures should not be included when estimating Jv.

Deere's RQD has been widely used, particularly in North America, for the past 25years. Cording and Deere (1972), Merritt (1972) and Deere and Deere (1988) have attempted to relate RQD to Terzaghi's rock load factors and to rock bolt requirements in tunnels. In the context of this discussion, the most important use of RQD is as a component of the RMR (Bieniawski, 1989).

2.4 Surface rock excavation using bench blasting

Bench blasting is a technique used for surface rock excavation, as in open pit mines, quarries and on civil engineering construction projects. Blast holes are drilled downwards from a surface to a depth normally not exceeding 20 m. The hole diameter varies from 33mm for the small quarries to 350 mm in the large open – pit mines. The drill rig used for bench drilling is equipped with tracks for mobility, and recognized as a crawler rig. The range of crawler rigs features a choice of rock drills. A simple rigs comes with pneumatic rock drill, where the operator handles the extension rods manually. The sophisticated crawler rig comes with air – conditioned operator's cabin and fully automatic rod handling.

In civil engineering construction projects the object is often to remove a rock mass of irregular shape: for instance, a rock cutting in a mountain side to prepare a base for a road. Bench height and hole depth varies; from zero up to may be 20m. Most of the times crawler rigs or jack hammers are used for these purposes. The oscillating track under carriage enables it to climb steep terrain.

Conditions in quarries are different from those met with in construction projects. The quarry operator wants a steady production of blast holes to feed his crusher, and has an unlimited rock mass at his disposal to penetrate. A quarry is designed with a flat bottom, and/ or benches, on which heavy machines can operate. The bench height is fixed or sometimes varies. Most of the time all blast holes are same. Multi holes are used for a one blast and blasting patterns square pattern and staggered pattern. Most of the time the staggered pattern is used in Sri Lankan quarry industry.

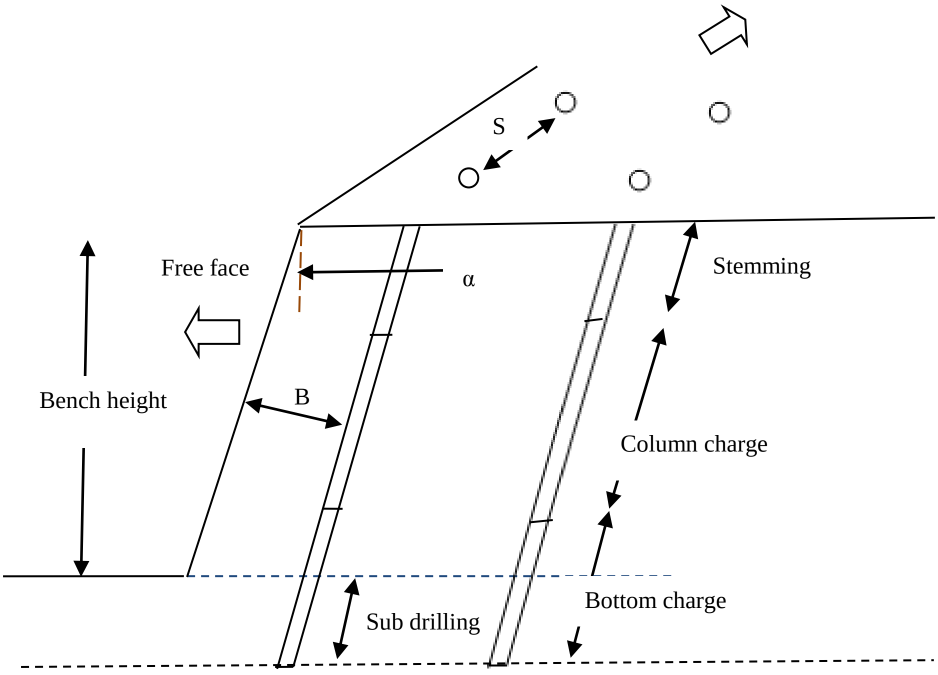

Figure 2.3: Bench blasting parameters (Muhammad 2009)

2.4.1 Free face

This is an exposed rock surface towards which the explosive charge can break out. It resembles a wall. Fragmentation and powder factor directly depends on number of available free face of the blasting bench.

2.4.2 Bench height

This is the vertical distance in meters between floors of the bench and should be at least twice the burden.

2.4.3 Blast hole diameter

Generally, the cost of drilling and blasting decreases as hole diameter increases. The relation between the blast hole diameter and face height is approximately:

$$D = 0.01 \text{ to } 0.02\ H \qquad \text{Eq (09)} \qquad \text{(Konya 1991)}$$

2.4.4 Burden (B)

Burden distance is defined as the shortest distance to relief at the time the hole detonates (Figure 2.1). Relief is normally considered to be either a ledge face or the internal face created by a row of holes that have previously shot on an earliest delay. The section of the proper burden is one of the most important decisions made in any blast design. Of all the design dimensions in blasting, it is the most critical one. If burden are too small, rock is thrown a considerable distance from the face. Air blast levels are high and the fragmentation may be excessively fine. If burdens are too large, severe back break and back shattering results on a back wall. Excessive burdens may also cause blast holes to geyser throwing fly rock considerable distances, vertical cratering and high levels of air blast will occur when blast holes relieve by blowing out. Excessive burdens cause over confinement of the blast holes, which results in significantly higher levels of ground vibrations per kilograms of explosive used. If the burden has some error, the all the other variables in the blast will have error. The following approximate relationship can be given for the burden.

$$Burden\ (B) = 25 \text{ to } 40D \qquad \text{Eq (10)} \qquad \text{(Konya 1991)}$$

If the operator has selected a burden and used it successfully for a drill hole of another size and wants to determine a burden for a drill hole that is either larger or smaller, one can do so quite easily if the only thing that he is changing is the size of the hole and the rock type and explosives are staying the same. To do this, one can use following simple ratio;

$$B_2 = B_2 \frac{D_2}{D_1} \qquad \text{Eq (11)} \qquad \text{(Konya 1991)}$$

Where,

B_1 is the Burden successfully used on previous blast, D_1 is the Diameter of explosive for B_1, B_2 is the new burden and D_2 is the new diameter of explosive for B_2.

2.4.5 Spacing

Spacing is the distance between adjacent blast holes in a row, measured perpendicular to the burden. In row-to-row shooting, spacing is measured between holes in a row; when the shot progresses at an angle to the free face, the spacing is measured at that angle.

Spacing may be somewhat dependent on the timing, but is most often a function of the burden. Close spacing cause crushing and cratering between holes, boulders, and toe problems. Holes spaced too far apart will result in inadequate fragmentation. The assumption of from 1.8 to 2 times the burden is a good starting point for determining the spacing of a blast to be initiated simultaneously in holes in the same row. When shooting sequentially down the row in a box cut or "V" pattern, spacing should be from 1 to 1.2 times the burden (or close to a square pattern).

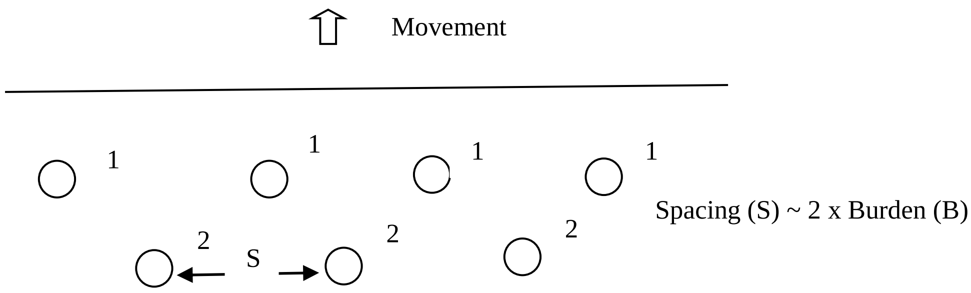

Figure 2.4: Staggered drilling pattern in bench blasting

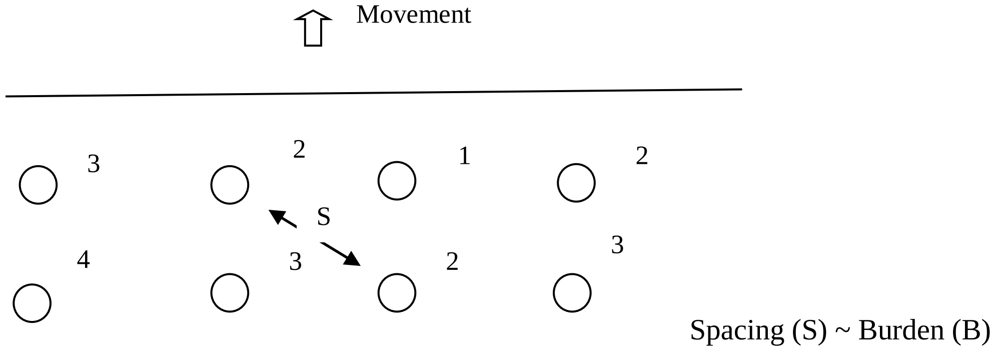

Figure 2.5: Square drilling pattern in bench blasting

2.4.6 Hole angle (α)

If strata conditions permit, inclined blast holes allows better distribution of the explosives. (See Figure 2.6) Inclined blast holes are very effective in eliminating "toe" (which is a hump of solid rock between the free face and the bench floor), and back break. α varies between 0 to 30^0 from the vertical plane. Easer is breaking at the toe and prevents unwanted fracturing of pit floor.

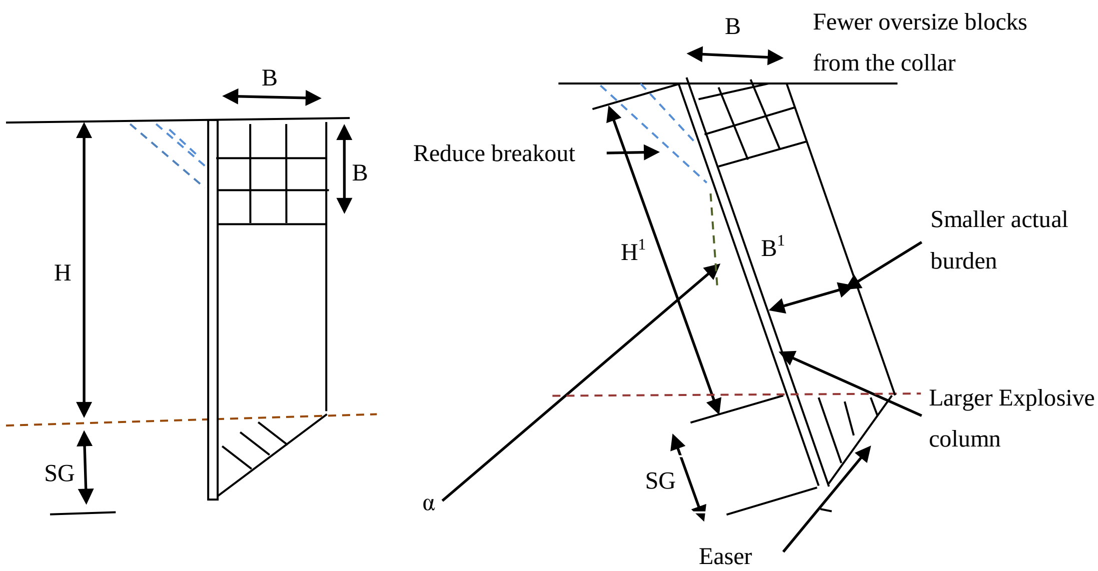

Figure 2.6: Rock breaking mechanisms of vertical and inclined blast holes

2.4.7 Subdrill

Subdrilling is a common term to detonate the depth which a blast hole will be drilled below the proposed grade to ensure that breakage will occur to the grade line. Blast holes normally do not break to full depth. On most construction projects, sub-drilling is used unless, by coincidence, there is either a soft seam or a bedding plane located at the grade line. If this occurs, no sub-drilling would be used. In fact, blast holes may be back filled a distance of 6 to 12 charge diameters to confine the gasses and keep them away from a soft seam (Figure 2.7). On the other hand, if there is a soft seam located a short distance above the grade line and below there exist massive material; it is not uncommon to have to sub-drill considerably deeper in order to break the material below the soft seam. As an example, Figure 2.8 indicates the soft seam one foot above the grade. In this case, a sub-drilling approximately equal to the burden distance was required below the grade to ensure breakage to grade. In most instances, sub-drilling is approximated as follows.

$$J = 0.3 \times B \qquad \text{Eq (12)} \qquad \text{(Konya 1991)}$$

Where,

J is the Subdrilling in feet and B is the Burden in feet

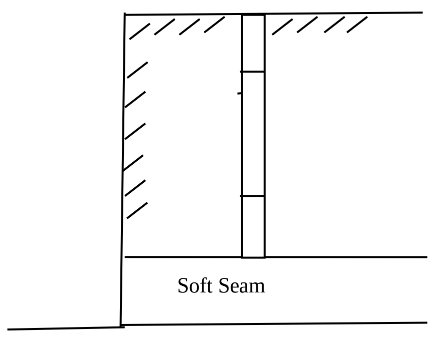

Figure 2.7: Back fill bore hole to soft seam in bench blasting

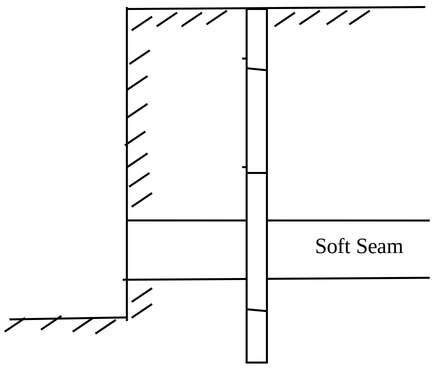

Figure 2.8: Problems of soft seam off bottom in bench blasting

The sub-drilling must not contain drill cuttings, mud or any rock material. If bore hole walls naturally slough and fill in, drilling must be deeper than the sub-drilling previously discussed so that at the time of loading the calculated amount of sub-drilling is open and will contain explosives,

In order to get a flat bottom in an excavation, it makes good economic sense to drill to a depth below grade, which ensures, in spite of random drilling depth errors and sloughing holes, that all hole bottoms will be down to the proper depth at the time of loading. If drilling is done slightly deeper than required and some holes are too deep at the time of loading, the blaster can always place drill cuttings in the bottom of those holes to bring them up to the desired height. The blaster, however, does not have the ability, at the time of loading, to remove excessive cuttings or material which has fallen in to the hole (Konya 1991).

2.5 Powder Factor

The several expressions and formulas have included under the discussion of powder factor.

2.5.1 Definitions and expressions

There are several possible combinations that can express the powder factor. Ashby (1981) developed an empirical relationship to describe the powder factor required for adequate blast based on the fracture frequency representing the density of fracturing and effective friction angle representing the strength of structured rock mass. According to Ashby the powder factor of rock can be determined from the following equation;

$$Powder\ factor\ = \frac{0.56\ \times\ Þ\ \times tan\ (\varphi + i)}{\sqrt[3]{\mathrm{Fracture}/\mathrm{meter}}}\ \mathrm{Kg/Cu.m} \qquad \mathrm{Eq\ (13)}$$

Where,

ϕ is the Basic Friction angle, Þ is the in-situ density of rock formation, i is the Roughness angle, (ϕ+i) is the friction angle and (Fracture/meter) is the represent the fracture frequency (Bhanwar 2013).

2.5.2 Definition of powder factor

The quantity of explosive required fragmenting 1 m^3 or 1 ton of rock is known as powder factor. It can serve a variety of purposes, such as an indicator of hardness of the rock, or the cost of the explosives needed, or even as a guide to planning a shot (Bhanwar 2013).

$$Powder\ factor\ = \frac{Explosive\ usage\ of\ the\ blast\ \ (kg)}{\mathrm{Insitu\ volume\ of\ the\ blast}\ \ \ \mathrm{(m3)}} \qquad \mathrm{Eq\ (14)}$$

2.5.3 Explosives

Explosive is a solid or liquid substance or a mixture of substances which on application of a suitable stimulus is converted in a very short time interval in to other more stable substances, largely or entirely gaseous, with the development of heat and high pressure. There are many types of explosives used in mining activities and quarry operations (Dick 1983).

2.5.3.1 Classification of Explosives

Explosives are categorized in two large groups according to their shock wave velocities. They are; a) rapid and detonating explosives with speed between 2000 to 7000 meters per second and b) slow and deflagrating explosives which are having speeds lower than 2000 meters per second. Detonating explosives are again sub divided in to two as primary and secondary explosives. Primary explosives used to initiate secondary explosives and can be found in blasting caps and cast primers. Few examples for primers are PETN, Pentolite and mercury fulminates. Secondary explosives are those applied to breakage of rocks and they are less sensitive than primary explosives.

2.5.3.2 Types of explosives

The industrial explosives classification commonly used in mining industry is shown in the figure 2.9 below.

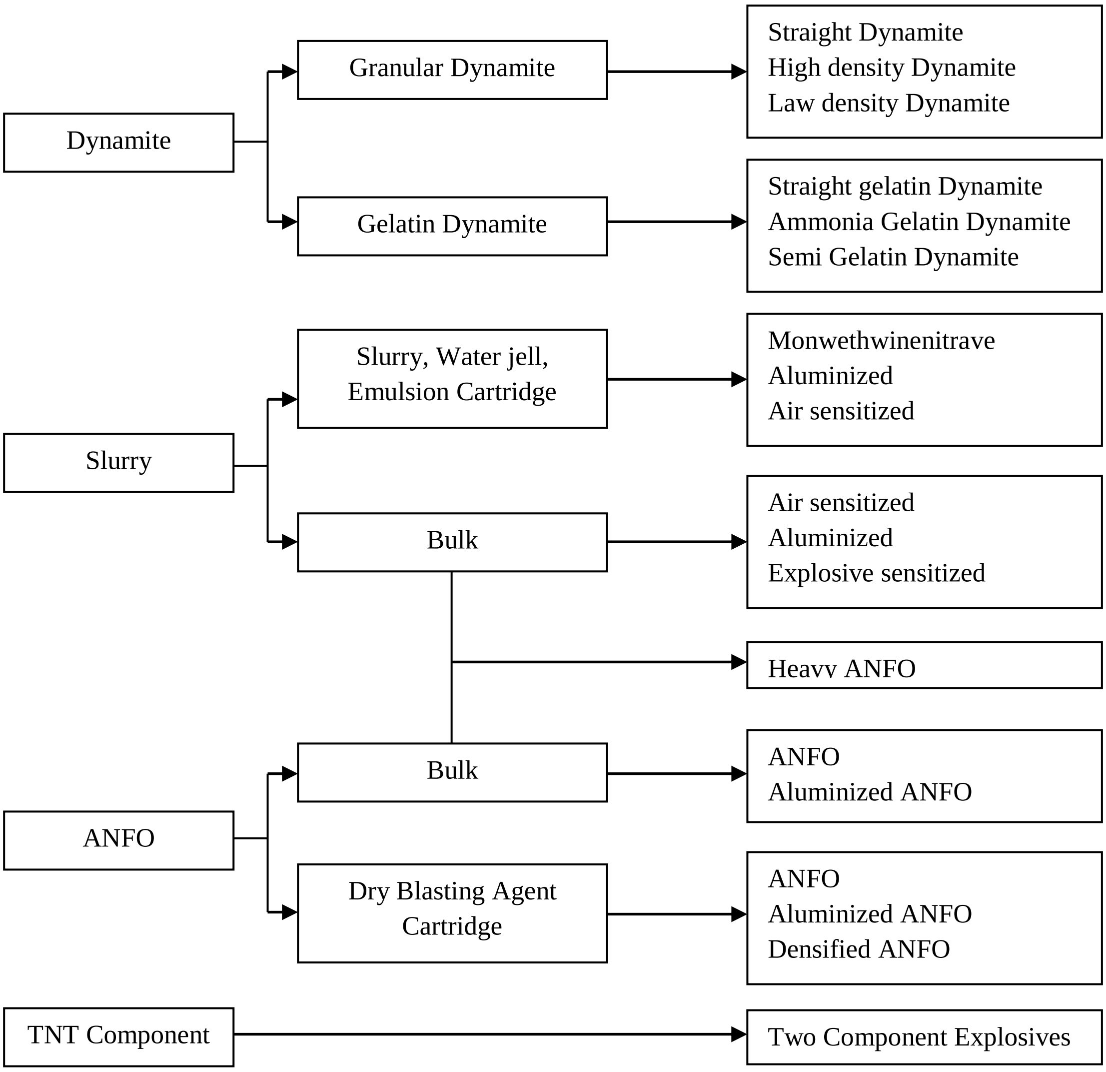

Figure 2.9: Explosive Classification (Dick 1983)

Primary explosives commonly used in the Sir Lankan metal quarry industry to initiate blasting are electric detonators, plane detonators and safety fuses while water-gel, black powder and ANFO use as secondary explosives to rock breakage.

2.5.4 Volume calculation in rock blasting

In situ block size is a key parameter in the mechanical characterization of rock masses. It describes the fracturing of the rock mass and thus is a measure for the degradation of the rock mass strength. Several classification systems use the in situ block size.

Joint Spacing (S) Method
Block Volume (V_b)
Volumetric Joint Count (J_V)

2.5.4.1 Joint Spacing (S) Method

In other cases where an average joint spacing is used and more than one joint set occurs, the following expression may be used:

$$V_b = {S_a}^3 \quad \text{Eq (15)}$$

Here, V_b is the block volume in m^3.

Some rock engineers apply the following expression for the average spacing of the joint sets:

$$S_a = \frac{(S_1 + S_2 + S_3 + \cdots + S_n)}{n} \quad \text{Eq (16)}$$

Where,
S_1, S_2, S_3... etc. are average spacing's for each of the joint sets. But Equation (15) does not correctly characterize the joint spacing.

2.5.4.2 Block Volume (V_b)

For small blocks or fragments having volumes in cubic decimeter size or less, this measurement is often the quickest of the methods, as it is easy to estimate the block size compared to registration of the many joints involved. Where three joint sets occur, the block volume is

$$V_b = \frac{(S_1 \times S_2 \times S_3)}{(\sin\gamma_1 \times \sin\gamma_2 \times \sin\gamma_3)} \quad \text{Eq (17)}$$

Where,

S_1, S_2 and S_3 are the spacing's in the three joint sets, and $\gamma 1$, $\gamma 2$ and $\gamma 3$ are the angles between the joint sets.

2.5.4.3 Volumetric Joint Count (J_V)

The volumetric joint (J_v) count was introduced by Palmstrom in 1982. Earlier, a similar expression for joint density measurements was applied as the number of joints in a blast round. Being a 3-dimensional measurement for the density of joints, J_v applies best where well-defined joint sets occur. J_v is defined as the number of joints intersecting a volume of one m^3. Where the jointing occurs mainly as joint sets

$$J_v = \frac{1}{S_1} + \frac{1}{S_2} + \frac{1}{S_3} + \cdots + + \frac{1}{S_n} \quad \text{Eq (18)}$$

Where

S_1, S_2 and S_3 are the average spacing's for the joint sets

Normally in the rock blasting, expected in-situ rock volume can be calculated by using spacing, burden and bench height as follows.

$$In\ situ\ rock\ volume = \text{Spacing} \times \text{Burden x Hole depth} \qquad \text{Eq (19)}$$

3. CHAPTER 03: METHODOLOGY

Chapter 03 explains the steps followed during the quarry location selection, sample collection, sample preparation and sample testing processes in detail. Further, this chapter elaborates reasons behind selecting locations of the quarry sites, sample collection criteria and each testing procedure according to the American Society for Testing and Materials (ASTM) specifications. Uniaxial Compressive Strength (UCS), Tensile Strength and Aggregate Impact Value (AIV) tests were performed in the rock mechanics laboratory of Earth Resources Engineering Department and Soil Mechanics Laboratory at Civil Engineering Department of university of Moratuwa. Number of samples tested in each quarry site was limited to the minimum required number of samples to be tested (05 samples) defined for the research due to time and resources limitations. Rock Mass Ratings (RMR) for each quarry site was calculated according to the standard procedure by using field data obtained during the field visits.

3.1 Identification of the locations

Eight IML A (Industrial Mining Licence A) category quarry sites located in the Western province were selected to perform research related testing and record collection for the analysis. Detailed map of the above quarry site locations in Western province is enclosed as Appendix A to this report. Details of the quarry sites selected for the research are enclosed to this report as Appendix B and Industrial Mining Licence types and licensing procedures are explained in the Appendix C enclosed to this report.

Bench blasting system employed quarries were specifically used for the research since the capability of accurately calculating powder factors for this excavation method. Since there are existing relationships for rock blasting powder factor with blasting geometry configurations such as spacing, burden, sub drilling, charge length, charge type, stemming height and hole diameter; there is a need of making constant these parameters to nullify their effect on powder factor variations. Hence quarries operate under the close supervision of qualified Mining Engineers were specifically selected for the research purpose. It was assumed that the blasting geometry and configurations have being properly optimized by the responsible Mining Engineers during the realized powder factor data recorded time period. Location details of the selected quarries are tabulated in table 3.1 below.

Table 3.1: Locations of metal quarries selected for the research

Quarry No	DS Division	Name of the Land	Easting (m)	Northing (m)
1	Padukka	Leenawatte	122538	182102
2	Kaduwela	Sampakara mawatha area (Ritigahapitiyawatte)	115793	189513
3	Hanwella	Hanwellawatta Lot 34 & 35	125605	186137
4	Padukka	Meepe	125700	184555
5	Kaduwela	Hokandara, Malabe	110777	187797
6	Homagama	Nawalamulla (Lenagalawaththa)	118928	185762
7	Kaduwela	Korathota Estate	115268	190471
8	Homagama	Habanhena waththa	113912	183936

3.2 Data and sample collection

Explosive usage and drilling geometry data relevant to blasting activities of last six months were obtained. In-situ volumes of the blasts were calculated using drilling data of recent months. Powder factors were calculated using explosive usage and quarry production to estimate powder factor. Area of the quarry span roughly divided to five sections in its plan view and representative rock and aggregate samples were collected from every quarry tabulated in the table 3.1.

3.3 Sample preparations and testing procedures

Core samples were extracted by core drilling through rock samples obtained during the field visits. Samples to test for Uniaxial Compressive Strength and Tensile Strength prepared at the Rock Mechanics laboratory of the Earth Resources Engineering Department of University of Moratuwa. Aggregate samples were collected from same locations of the quarry if available. Else, samples were prepared manually by hammering metal samples obtained from the sampling locations. All the aggregate samples were tested for Aggregate Impact Values at the Soil Mechanics laboratory of the Civil Engineering Department of University of Moratuwa.

3.3.1 Testing for Density of rock

Core samples prepared for UCS were used to calculate the density of the rock. Weight of the core sample measured using electronic balance and height and diameter of the core measured using Vernier caliper.

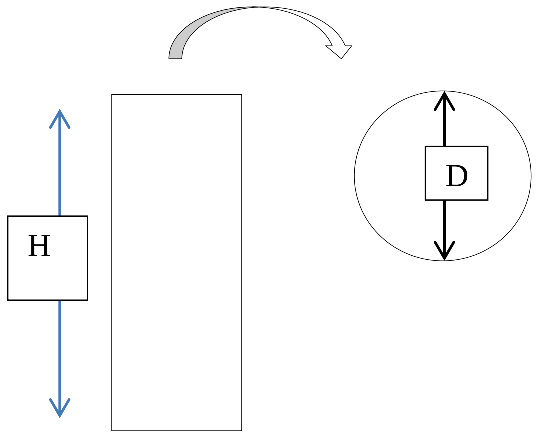

Figure 3.1: Measuring core samples for density

Sample volume was calculated using equation 21 given below;

$$Volume\ of\ the\ sample\ =\ \pi\left(\frac{\mathrm{D}}{2}\right)^{2}\mathrm{H}\qquad \text{Eq (20)}$$

Where,

D is the diameter of core sample and H is the height of the core sample

Density of the sample is calculated using following equation 21;

.

$$Density\ of\ the\ sample\ = \frac{W}{\pi\ \left(\frac{D}{2}\right)^2\ H} \qquad \text{Eq (21)}$$

Where,

W is the measured weight of the core sample, D is the diameter of the core sample and H is the height of the core sample.

This was done for all samples of all the quarries and got the mean density values for all quarries separately.

3.3.2 Testing for Uniaxial Compressive Strength (UCS) of rock

Five representative core samples were prepared from each quarry and Uniaxial Compressive Strength tests were performed on each sample.

3.3.2.1 Sample preparation for UCS

Core sample drilling machine shown in the figure 3.2 and core sample cutting machine shown in figure 3.3 were used to extract five cores per each quarry from collected rock samples. All prepared samples should have its height to diameter ratio around 2.

Figure 3.2: Core drilling machine used for core sample preparation

$$UCS = \frac{4P}{\pi D^2} \quad \text{Eq (22)}$$

Where;

P is the Failure Load and D is the Diameter of the sample

Figure 3.3: Core sample cutting machine

3.3.2.2 UCS testing method

Compressive strength test machine and Vernier caliper were used as apparatus in this laboratory test.

Figure 3.4: Uniaxial Compressive Strength testing apparatus

Specimens from drill cores were prepared by cutting them to the specified length and there after ends were ground and measured. There are high requirements on the flatness of the end surfaces in order to obtain an even load distribution. Ratio of height / diameter of specimens were maintained between 2 and 3.The diameters and heights were measured using the Vernier caliper of each core samples of each quarries separately. Some of the physical properties were identified. One sample was kept in between the two vertical moving plates. After that, the slowly and continuously increasing pressure was applied to the sample and the load at the failure was recorded. This was done for all samples collecting from each quarry. Then Uniaxial Compressive Strengths were calculated in each quarry.

3.3.3 Testing for Tensile Strength of rock

Test samples were prepared using the laboratory instruments in the Rock Mechanics laboratory at the Earth Resources Engineering Department. The core disc samples were prepared for Brazilian disk tests.

3.3.3.1 Sample preparation for Tensile Strength test

Five cylindrical samples were prepared from each quarry as follows

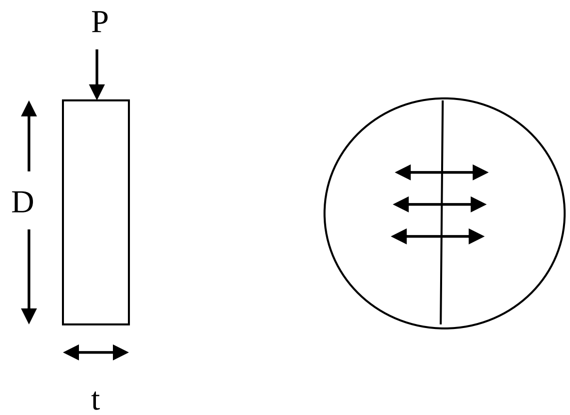

Figure 3.5: Prepared sample for Tensile Strength test

$$Tensile\ Strength = \frac{2P}{\pi Dt} \quad \text{Eq (23)}$$

Where;

P is the Failure Load, D is the diameter of the sample and t is the thickness of the sample (Bell 1992)

3.3.3.2 Tensile Strength testing method

Compressive strength test machine and Venire caliper were used as apparatus during the test. Five cylindrical samples were prepared from each quarry. The diameters and the heights were measured using the Venire caliper of each core samples of each quarries. Some of the physical properties were identified of each core samples. Then the single sample was kept as the outer surface tough the compressive strength machine. After that the pressure was applied as previous uniaxial compressive strength test. This was done for all samples representing individual quarries and mean tensile strength value for each quarry was calculated.

3.3.4 Testing for Aggregate Impact Value

The property of a material to resist impact is known as toughness. Due to movement of vehicles on the road the aggregates are subjected to impact resulting in their breaking down into smaller pieces. The aggregates should therefore have sufficient toughness to resist their disintegration due to impact. The toughness value is very important to breakage of the rock. This is very important for the powder factor. This characteristic is measured by impact value test. The aggregate impact value is a measure of resistance to sudden impact or shock, which may differ from its resistance to gradually applied compressive load. The aggregate samples collected from each quarry were tested for the Aggregate Impact Value during this research.

3.3.4.1 Sample preparation and test method.

Field investigations were carried out at each selected quarry site and five aggregate samples were collected from each. The samples size was prepared in between 10 mm 12.5 mm in each quarry. All Aggregate samples (sample size 10mm to 12.5mm) were washed and dried by heating at 100-110° C for a period of 4 hours and cooled.

3.3.4.2 Testing for Aggregate Impact Value

AIV testing machine has a cylindrical steel cup, a metal hammer and tamping rod. A balance of capacity is not less than 500g and it is readable and accurate up to 0.1.

Figure 3.6: Aggregate Impact Value testing apparatus

Measuring cylinder was weighed and recorded it. After that the aggregates of a sample was filled about just 1/3 depth of measuring cylinder. Then the materials were compacted by giving 25 gentle blows with the rounded end of the tamping rod. Two more layers were added in similar manner, so that cylinder was full and Strike off the surplus aggregates. After that (Aggregate + measuring cylinder) was determined. Then the net weights of the aggregates were measured to the nearest gram (W). Measured (Sample + measuring cylinder) was taken to the impact machine to rest without wedging or packing up on the level plate, block or floor, so that it is rigid and the hammer guide columns are vertical. Fixed the cup firmly in position on the base of machine and placed whole of the test sample in it and compact by giving 25 gentle strokes with tamping rod. Raised the hammer until its lower face was 380 mm above the surface of aggregate sample in the cup and allowed it fell down freely on the aggregate sample. Fifteen such blows were given at an interval of not less than one second between successive falls. Removed the crushed aggregate from the cup and sieved it through 2.36 mm British Standard sieves until no further significant amount passes in one minute. Fraction passing the sieve to an accuracy of 1 gm. was weighted. Also, weighed the fraction retained in the sieve. Aggregate impact value was weighted. This was done for all five samples and got the mean Aggregate Impact Value. Then this was done for all quarries.

$$Aggregate\ Impact\ Value = \left(\frac{W_2}{W_1}\right) x\ 100 \qquad \text{Eq}\ (24)$$

3.3.5 Defining Rock Mass Ratings for quarries

Field observations and Rock Mass Rating assessments for all the eight quarries are enclosed as Appendix 01 to this report.

4. CHAPTER 04: RESULTS AND DISCUSSION

Chapter 04 detailed test results were obtained for Density, Uniaxial Compressive Strength (UCS), Tensile Strength and Aggregate Impact Value test (AIV) during the laboratory testing. Similarly, chapter summarizes Rock Mass Rating (RMR) assessed to each quarry using UCS values and other field data observed in the site. It was assumed that the powder factor variations occurred due to inherent mechanical and aggregate rock property variations of selected eight quarries. Effect of blasting geometry on powder factor is considered as minimal during the data analysis period due to the fact that all these quarries are operated under the close supervision of a qualified Mining Engineer who is responsible to use optimum blasting configurations.

Further, the test results were plotted in graphs, trends were analysed and discussed in this section to correlate powder factor behaviour with selected rock properties.

4.1 Statistical analysis of data

Mathematical tools such as correlation coefficient and regression analysis were used to analyse the behavior of powder factor against above mentioned mechanical as well as aggregate rock properties. State-of-the-art statistical software “Minitab” was used to perform all the statistical data analysis tasks to confirm the accuracy and confidence level of the research results.

4.2 Statistical interpretations

Minitab output for every tested rock property i.e. UCS, Tensile Strength, Density, AIV and RMR were analysed based on below mentioned criteria to define their relationship type.

a) P-value < 0.05; all parameters are significant
b) $R^2 > 80\%$ (Coefficient of determination); How much variability of the observed data has been explained by the fitted model.

c) DW (Durbin Watson statistic); is used to test the randomness of errors. It should be close to 2 for random generation of errors.

d) VIF (Variance Influence Factor); is used to measure the existence of significant correlations among the explanatory variables is termed as “Multicollinearity”. This will pose problems in interpretation. The existence "multicollinearity will violate the assumptions in regression. If VIF < 10 “Multicollinearity” problems won’t be generated.

4.3 Powder Factor versus Specific Gravity relationship

Mean density values obtained from measured samples representing metals at each quarry is tabulated in table 4.1 shown below.

Table 4.1: Calculated density values

Quarry	Mean Density (Kg/m3)
Q1	2805.27
Q2	2653.71
Q3	2756.81
Q4	2721.44
Q5	2718.30
Q6	2701.62
Q7	2663.70
Q8	2696.61

Behaviour of Powder factor of rock blasting against density of the quarry metal was analysed using regression analysis tool in “Minitab” statistical software. Data points were correlated using mainly three models namely linear, quadratic and cubic. Agreements of data for each model were interpreted using all or few of the above mentioned statistical interpretations; P-value, R^2 (Coefficient of determination) values, DW (Durbin Watson statistic) and VIF (Variance Influence Factor).

4.3.1 Linear model

Graph obtained for linear regression model in "Minitab" is shown in the Figure 4.1 below.

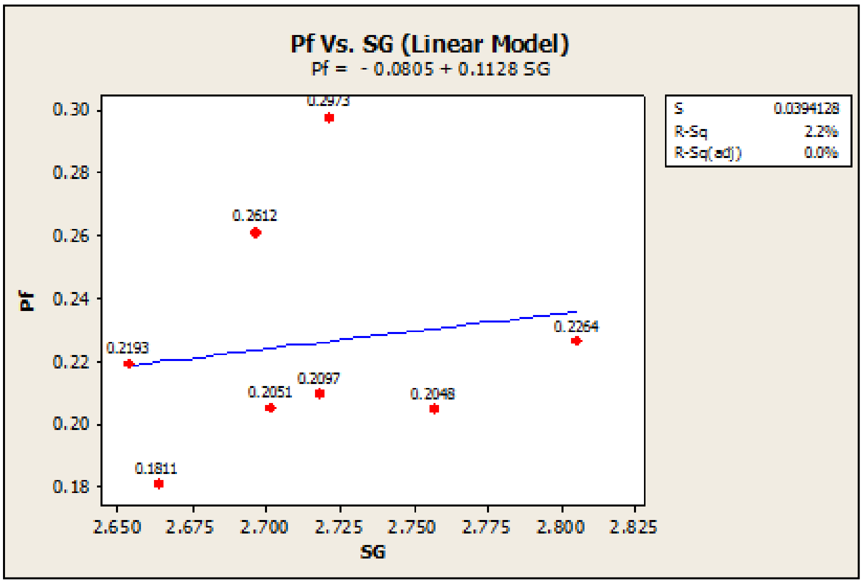

Figure 4.1: Powder factor vs. Specific Gravity (Linear Model)

As per the results of the above linear relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant linear relationship between Powder Factor versus Specific Gravity under 95% confidence level. There could be a linear relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.3.2 Quadratic model

Graph obtained for quadratic regression model in "Minitab" is shown in the Figure 4.2. As per the results of the below quadratic relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant quadratic relationship between Powder Factor versus Specific Gravity under 95% confidence level.

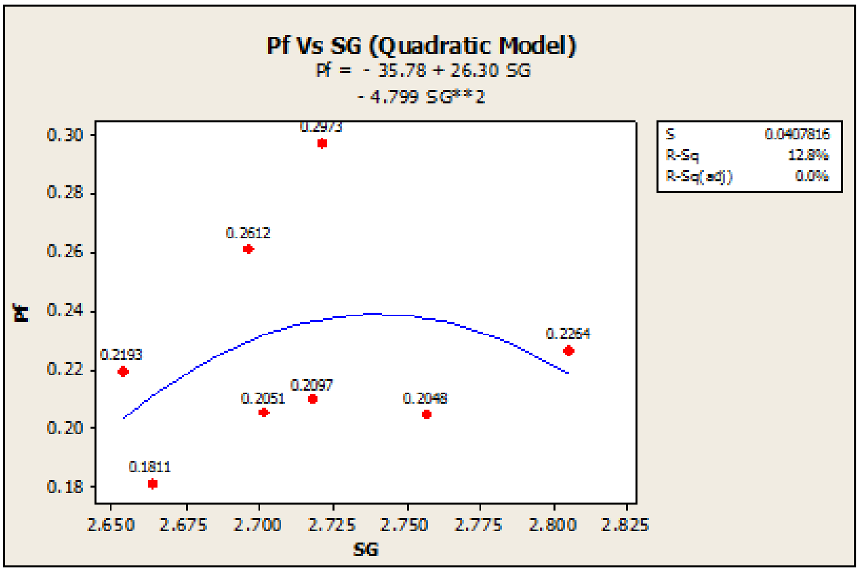

Figure 4.2: Powder factor vs. Specific Gravity (Quadratic Model)

There could be a quadratic relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.3.3 Cubic model

Graph obtained for cubic regression model in "Minitab" is shown in the Figure 4.3 below.

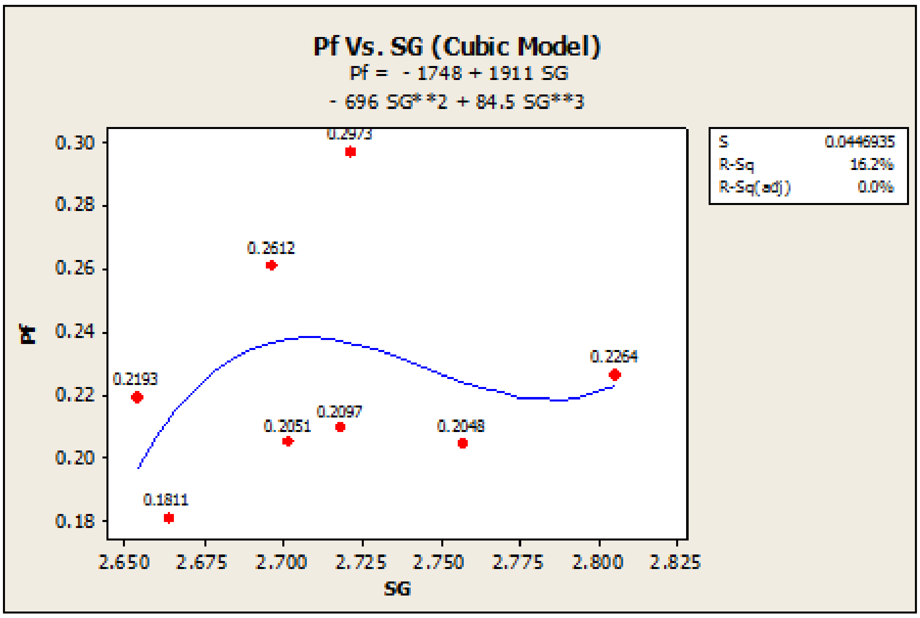

Figure 4.3: Powder factor vs. Specific Gravity (Cubic Model)

As per the results of the cubic relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant cubic relationship between Powder Factor versus Specific Gravity under 95% confidence level.

4.4 Powder Factor versus Uniaxial Compressive Strength relationship

Mean UCS values recorded for cores extracted from rock samples obtained from each quarry is tabulated in table 4.2 shown below.

Table 4.2: Results of Uniaxial Compressive Strength tests

Quarry	UCS Value (Mpa)					Mean UCS Value (Mpa)
	Sample	Sample	Sample	Sample	Sample	
Q1	41	61	0	0	0	51
Q2	78	70	99	55	85	77
Q3	48	83	77	100	75	76
Q4	70	197	104	120	127	124
Q5	260	55	242	37	160	151
Q6	49	91	46	62	60	62
Q7	51	45	49	48	49	48
Q8	109	78	80	55	82	81

Behaviour of Powder factor of rock blasting against UCS of the quarry metal was analysed using regression analysis tool in “Minitab” statistical software. Data points were correlated using mainly three models namely linear, quadratic and cubic. Agreements of data for each model were interpreted using all or few of the above mentioned statistical interpretations; P-value, R^2 (Coefficient of determination) values, DW (Durbin Watson statistic) and VIF (Variance Influence Factor).

4.4.1 Linear model

Graph obtained for linear regression model in “Minitab” is shown in the Figure 4.4 below.

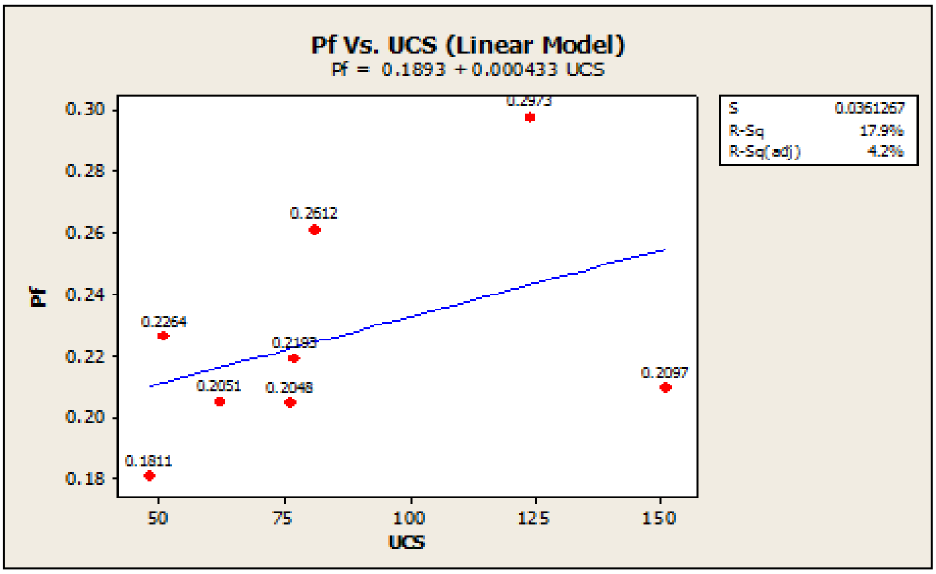

Figure 4.4: Powder factor vs. Uniaxial Compressive Strength (Linear Model)

As per the results of the above linear relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant linear relationship between Powder Factor versus Uniaxial Compressive Strength under 95% confidence level. There could be a linear relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.4.2 Quadratic model

Graph obtained for quadratic regression model in “Minitab” is shown in the Figure 4.5. As per the results of the below quadratic relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected.

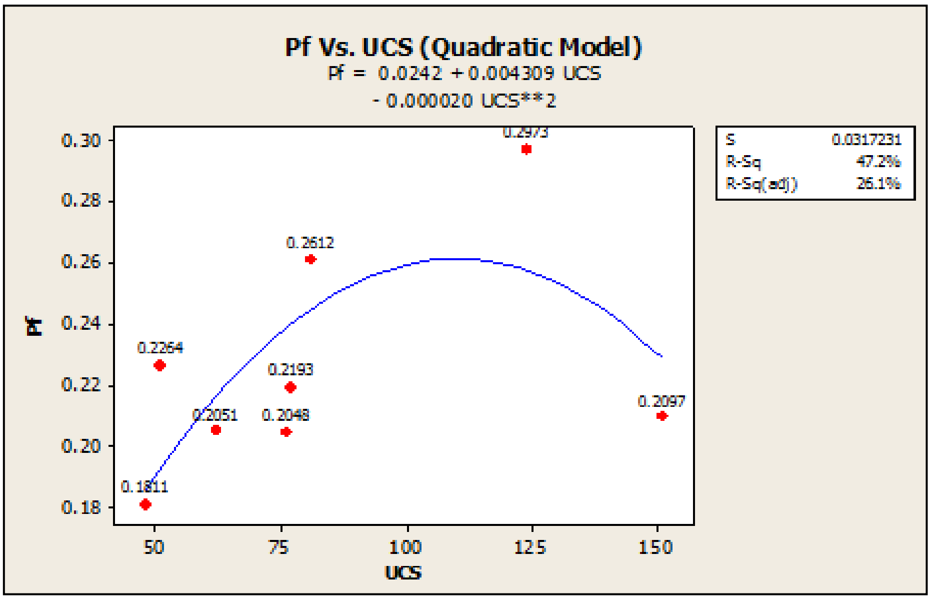

Figure 4.5: Powder factor vs. Uniaxial Compressive Strength (Quadratic Model)

Also, there is no significant quadratic relationship between Powder Factor versus Uniaxial Compressive Strength under 95% confidence level. There could be a quadratic relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.4.3 Cubic model

Graph obtained for cubic regression model in "Minitab" is shown in the Figure 4.6. As per the results of the cubic relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant cubic relationship between Powder Factor versus Uniaxial Compressive Strength under 95% confidence level. There could be a quadratic relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

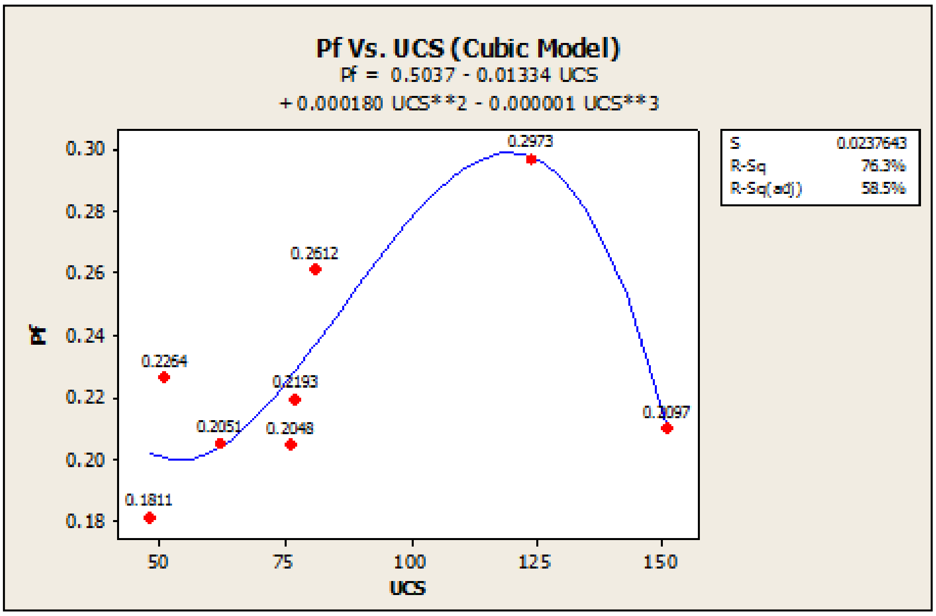

Figure 4.6: Powder factor vs. Uniaxial Compressive Strength (Cubic Model)

4.5 Powder Factor versus Tensile Strength relationship

Mean Tensile Strength values recorded for cores extracted from rock samples obtained from each quarry is tabulated in table 4.3 shown below.

Table 4.3: Applied load for Tensile Strength tests

Quarry	Applied Loads (KN)					Mean Thickness values of the				
	Sample 01	Sample 02	Sample 03	Sample 04	Sample 05	Sample 01	Sample 02	Sample 03	Sample 04	Sample 05
Q1	12.0	7.5	28.5	57.0	39.0	0.02	0.02	0.02	0.02	0.02
Q2	19.5	22.5	88.5	37.5	-	0.02	0.02	0.02	0.02	-
Q3	9.0	9.0	12.0	9.5	43.5	0.02	0.02	0.02	0.01	0.02
Q4	33.0	36.0	31.5	28.5	24.0	0.02	0.02	0.02	0.02	0.02
Q5	13.0	25.5	31.5	16.5	115.0	0.02	0.02	0.02	0.02	0.02
Q6	16.5	36.5	28.5	37.5	28.5	0.02	0.02	0.02	0.02	0.02
Q7	30.0	46.5	22.5	63.0	76.5	0.02	0.02	0.02	0.02	0.02
Q8	36.0	21.0	16.5	28.5	57.0	0.02	0.02	0.02	0.02	0.02

Calculated tensile strengths for above samples are tabulated in the table 4.4 below.

Table 4.4: Calculated tensile strengths

Quarry	**Tensile Strength (MPa)**					**Mean Value (MPa)**
	Sample 01	Sample 02	Sample 03	Sample 04	Sample 05	
Q1	6.00	3.93	15.48	30.61	20.85	22
Q2	10.59	11.45	42.62	19.33		14
Q3	4.91	4.66	5.8519	5.76	24.02	5
Q4	16.33	19.07	16.99	14.96	13.57	16
Q5	7.04	13.26	14.56	8.66	60.10	11
Q6	8.36	17.72	14.50	19.00	14.76	16
Q7	16.37	25.09	12.19	33.37	42.45	34
Q8	18.98	10.68	8.08	14.19	27.28	15

Behaviour of Powder factor of rock blasting against Tensile Strength of the quarry metal was analysed using regression analysis tool in "Minitab" statistical software. Data points were correlated using mainly three models namely linear, quadratic and cubic. Agreements of data for each model were interpreted using all or few of the above mentioned statistical interpretations; P-value, R^2 (Coefficient of determination) values, DW (Durbin Watson statistic) and VIF (Variance Influence Factor).

4.5.1 Linear model

Graph obtained for linear regression model in "Minitab" is shown in the Figure 4.7. As per the results of the above linear relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected.

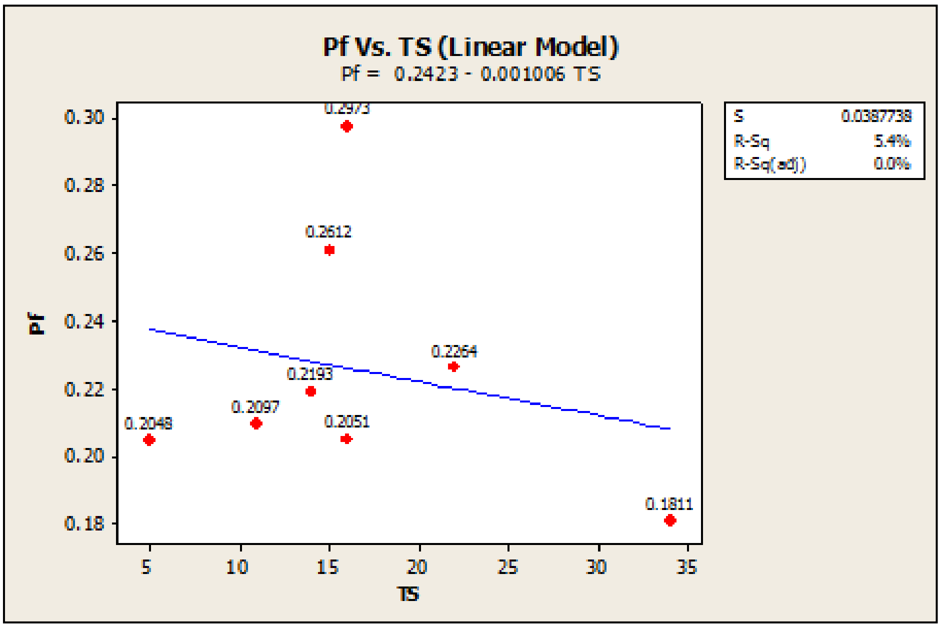

Figure 4.7: Powder factor vs. Tensile Strength (Linear Model)

Also, there is no significant linear relationship between Powder Factor versus Tensile Strength under 95% confidence level. There could be a linear relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.5.2 Quadratic model

Graph obtained for quadratic regression model in "Minitab" is shown in the Figure 4.8 below.

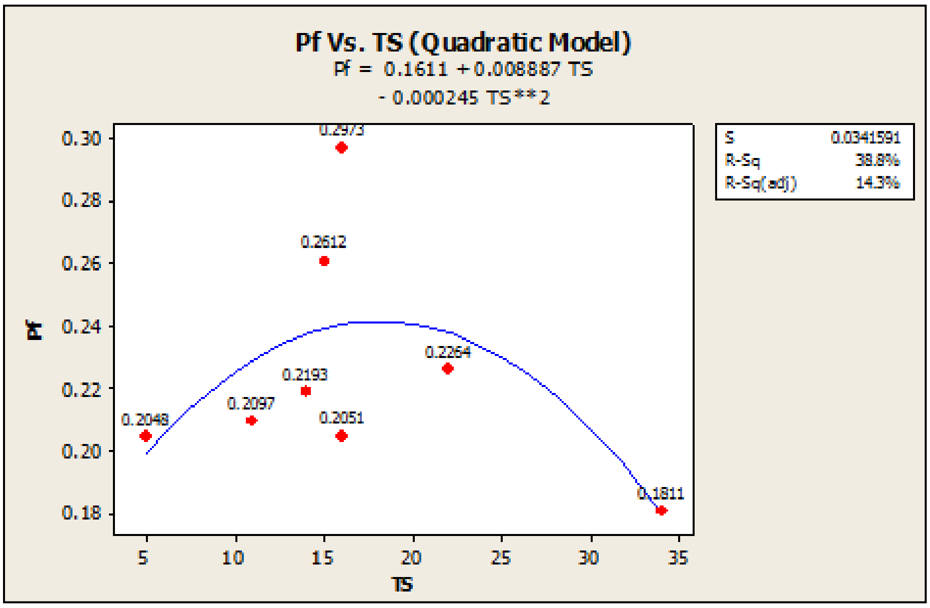

Figure 4.8: Powder factor vs. Tensile Strength (Quadratic Model)

As per the results of the below quadratic relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant quadratic relationship between Powder Factor versus Tensile Strength under 95% confidence level. There could be a quadratic relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.5.3 Cubic model

Graph obtained for cubic regression model in "Minitab" is shown in the Figure 4.9. As per the results of the cubic relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected.

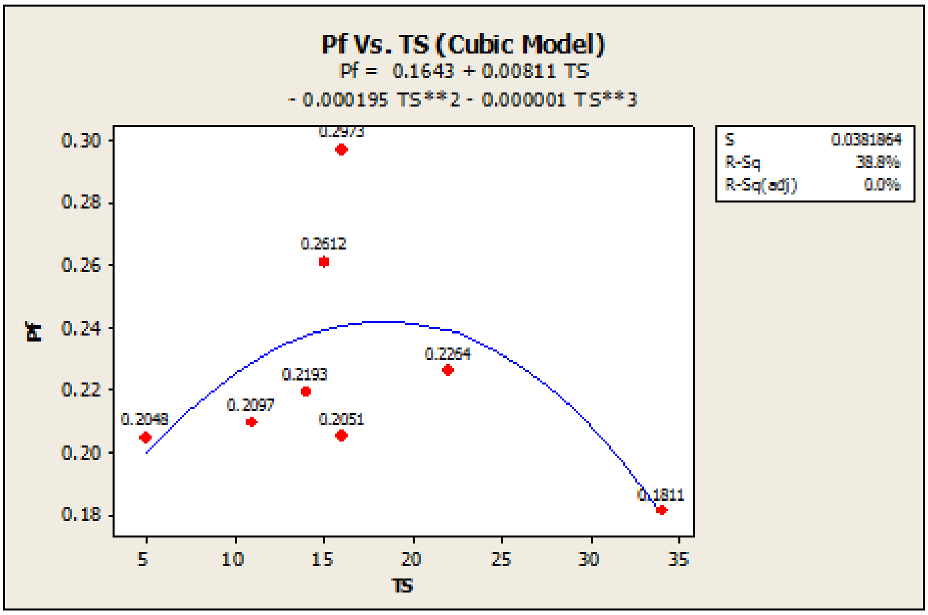

Figure 4.9: Powder factor vs. Tensile Strength (Cubic Model)

Also, there is no significant cubic relationship between Powder Factor versus Tensile Strength under 95% confidence level. There could be a quadratic relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.6 Powder Factor versus Aggregate Impact Value relationship

Weights recorded during the sample testing in order to calculate Aggregate Impact Values for aggregate samples prepared from rock samples obtained from each quarry is tabulated in table 4.5.

Table 4.5: Results of AIV Test

Quarry	Sample weight (Dry) (W1 gm)					Weight of portion passing 2.36 mm sieve				
	Sample 01	Sample 02	Sample 03	Sample 04	Sample 05	Sample 01	Sample 02	Sample 03	Sample 04	Sample
Q1	600	600	600	600	600	122.5	119.	118.0	121.5	123.5
Q2	601	600	601	601	601	138.0	138.	134.5	135.0	138.5
Q3	600	601	600	600	600	131.0	130.	134.0	131.5	135.0
Q4	600	600	600	600	600	101.5	112.	102.5	101.0	103.0
Q5	600	600	600	600	600	141.5	137.	128.0	135.0	136.5
Q6	600	600	600	600	600	162.0	163.	161.0	164.0	162.5
Q7	592	600	600	600	600	139.5	137.	142.0	140.5	136.5
Q8	600	600	600	600	600	106.5	112.	113.0	112.0	110.5

Aggregate Impact Values calculated for above aggregate samples and the mean Aggregate Impact Values are tabulated in the table 4.6 below.

Table 4.6: Aggregate Impact Values

Quarry	**Aggregate Impact Values (AIV) percentage**					**Mean AIV**
	Sample 01	Sample 02	Sample 03	Sample 04	Sample 05	
Q1	20.4	19.9	19.6	20.2	20.5	20.1
Q2	22.9	23.0	22.3	22.4	23.0	22.7
Q3	21.8	21.6	22.3	21.9	22.5	22.0
Q4	16.9	18.7	17.0	16.8	17.1	17.3
Q5	23.5	22.9	21.3	22.5	22.7	22.6
Q6	27.0	27.1	26.8	27.3	27.0	27.0
Q7	23.5	22.8	23.6	23.3	22.7	23.2
Q8	17.7	18.7	18.8	18.6	18.4	18.4

Behaviour of Powder factor of rock blasting against Aggregate Impact Value of the quarry metal was analysed using regression analysis tool in “Minitab” statistical software. Data points were correlated using mainly three models namely linear, quadratic and cubic. Agreements of data for each model were interpreted using all or few of the above mentioned statistical interpretations; P-value, R^2 (Coefficient of determination) values, DW (Durbin Watson statistic) and VIF (Variance Influence Factor).

4.6.1 Linear model

Graph obtained for linear regression model in "Minitab" is shown in the Figure 4.10 below.

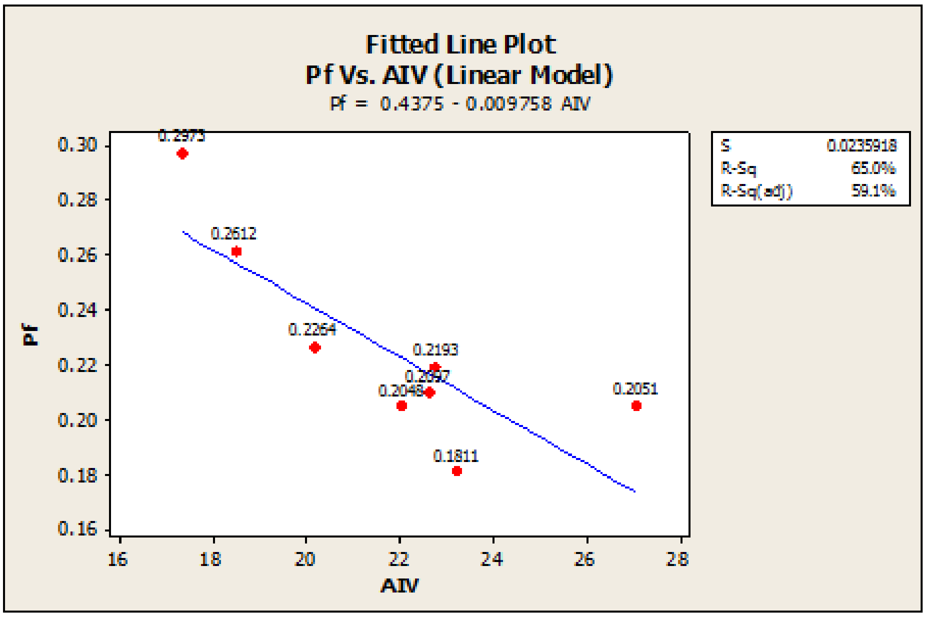

Figure 4.10: Powder Factor vs. Aggregate Impact Value (Linear Model)

As per the results of the above linear relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant linear relationship between Powder Factor versus Tensile Strength under 95% confidence level. There could be a linear relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.6.2 Quadratic model

Graph obtained for quadratic regression model in "Minitab" is shown in the Figure 4.11 below.

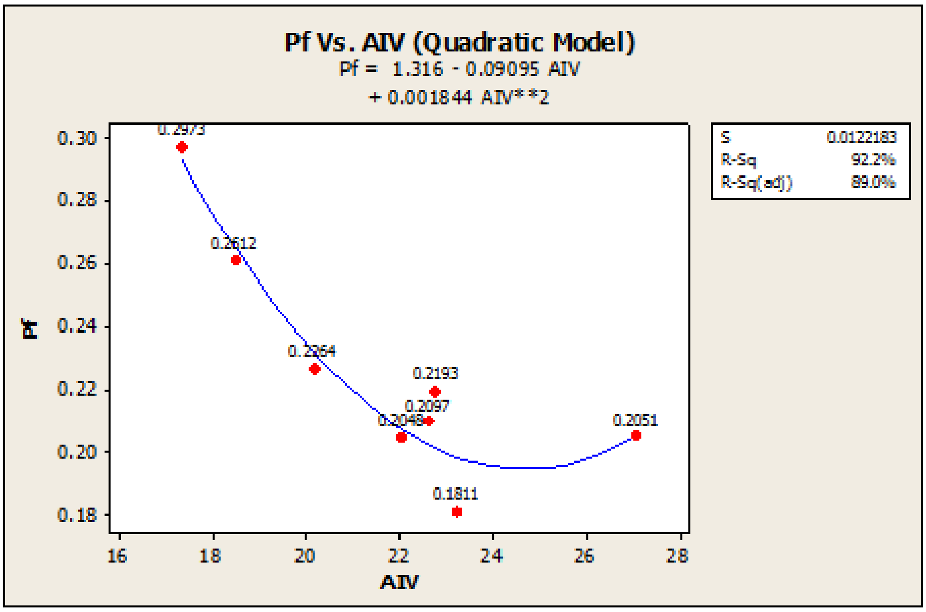

Figure 4.11: Powder Factor vs. Aggregate Impact Value (Quadratic Model)

As per the results of the above quadratic relationship analysis, P value is less than 0.05 and R^2value is greater than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) can be rejected. Also, there is a significant quadratic relationship between Powder Factor (Pf) versus Aggregate Impact Value (AIV) under 95% confidence level. Quadratic relationship between Powder Factor versus Aggregate Impact Value (AIV) can be expressed as equation 25 below.

$$Pf = 1.316 - 0.09095\ AIV + 0.001844\ AIV^2 \qquad \text{Eq (25)}$$

The above revealed quadratic relationship between two factors; Powder Factor versus Aggregate Impact Value (AIV) was validated using data gathered from three other quarry sites and details are tabulate in the table 4.7.

Table 4.7: Validation of results for existing quarries using quadratic model

Quarry and location	Mean AIV	Powder Factor (Predicted)	Powder Factor (Actual)
Quarry A at Oddusuddan	25	0.1948	0.20
Quarry B at Arankele	32	0.2939	0.28
Quarry C at Millennia	18	0.2764	0.26

Validation results confirm the above revealed relationship up to some extent but detail study has to be performed to confirm its behavior with incremental AIVs in future.

4.6.3 Cubic model

Graph obtained for cubic regression model in "Minitab" is shown in the Figure 4.12.

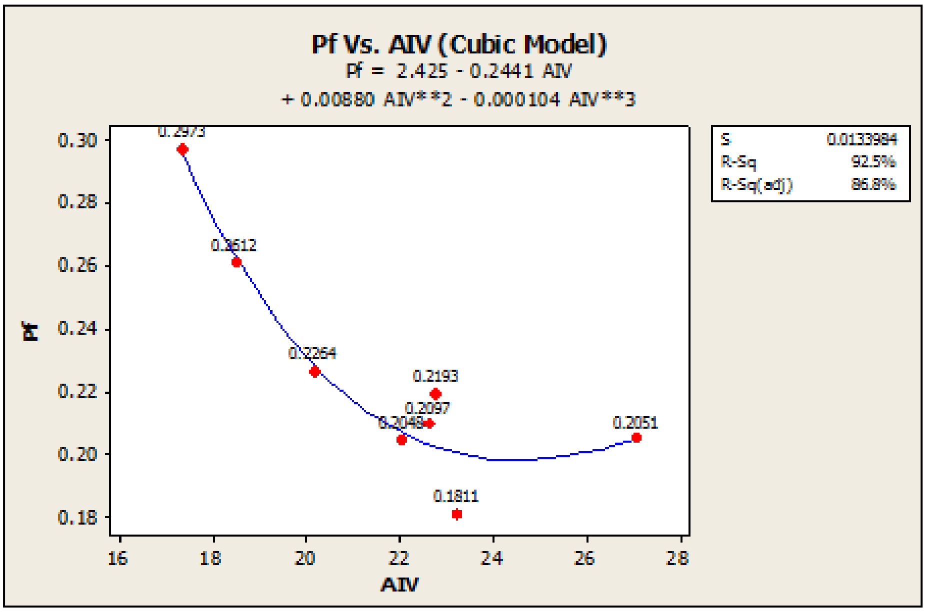

Figure 4.12: Powder Factor vs. Aggregate Impact Value (Cubic Model)

As per the results of the above cubic relationship analysis, P value is less than 0.05 and R^2value is greater than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) can be rejected. Also, there is a significant quadratic relationship between Powder Factor versus Aggregate Impact Value (AIV) under 95% confidence level. Cubic relationship between Powder Factor (Pf) versus Aggregate Impact Value (AIV) can be expressed as equation 26 below.

$$Pf = 2.425 - 0.2441\ AIV + 0.00880\ AIV^2 - 0.000104\ AIV^3 \qquad \text{Eq (26)}$$

The above revealed quadratic relationship between two factors; Powder Factor versus Aggregate Impact Value (AIV) was validated using data gathered from three other quarry sites and details are tabulate in the table 4.8 below.

Table 4.8: Validation of results for existing quarries using cubic model

Quarry and location	Mean AIV	Powder Factor (Predicted)	Powder Factor (Actual)
Quarry A at Oddusuddan	25	0.1975	0.20
Quarry B at Arankele	32	0.2171	0.28
Quarry C at Millennia	18	0.2759	0.26

Validation results confirm the above revealed relationship with better agreement than quadratic relationship but detail study has to be performed to confirm its behavior with incremental AIVs in future.

4.7 Powder Factor versus Rock Mass Rating relationship

Rock Mass Ratings recorded for each quarry is tabulated in table 4.9 shown in the following page. Data collected during the extensive field visits carried out during the research in order to determine Rock Mass Ratings for each quarry are enclosed as Appendix 01 to this report.

Table 4.9: Calculated RMR Values

Quarry	RMR
01	71
02	75
03	74
04	77
05	66
06	62
07	70
08	70

4.7.1 Linear model

Graph obtained for linear regression model in "Minitab" is shown in the Figure 4.13 below.

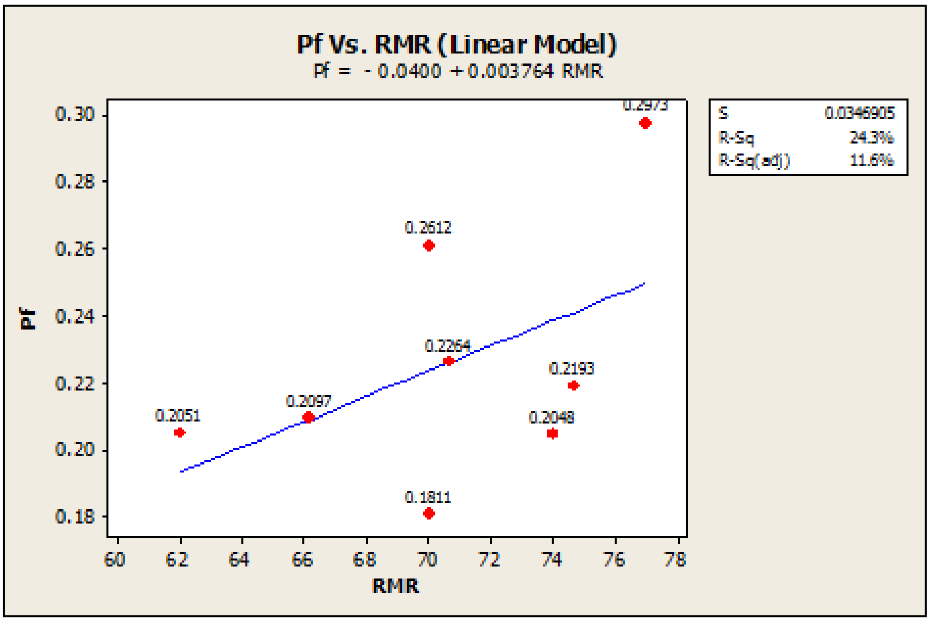

Figure 4.13: Powder Factor vs. Rock Mass Rating value (Linear Model)

As per the results of the above linear relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant linear relationship between Powder Factor versus Rock Mass Rating under 95%

confidence level. There could be a linear relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.7.2 Quadratic model

Graph obtained for quadratic regression model in "Minitab" is shown in the Figure 4.14 below.

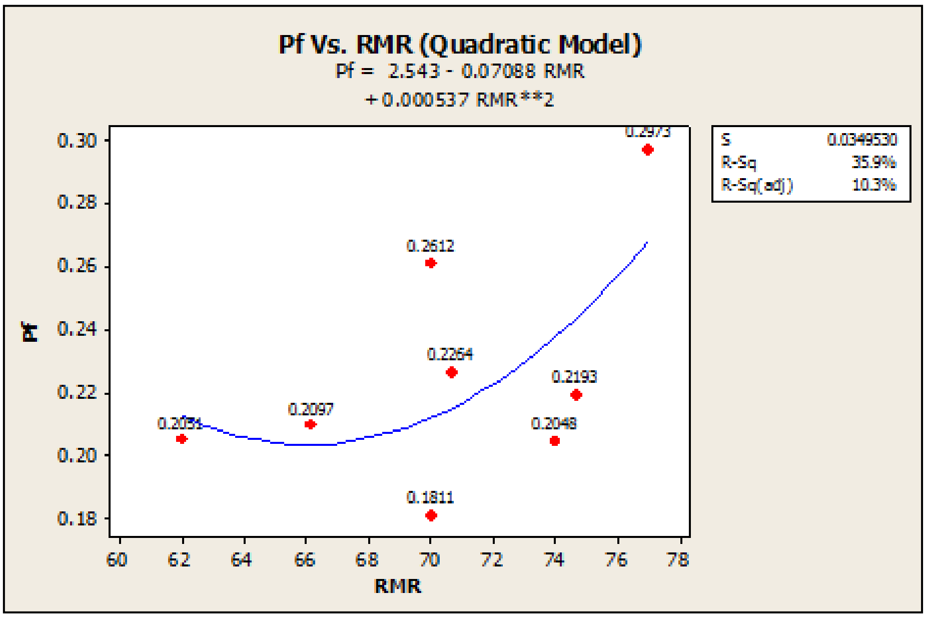

Figure 4.14: Powder Factor vs. Rock Mass Rating value (Quadratic Model)

As per the results of the below quadratic relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant quadratic relationship between Powder Factor versus Rock Mass Rating under 95% confidence level. There could be a quadratic relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.7.3 Cubic model

Graph obtained for cubic regression model in "Minitab" is shown in the Figure 4.15 below.

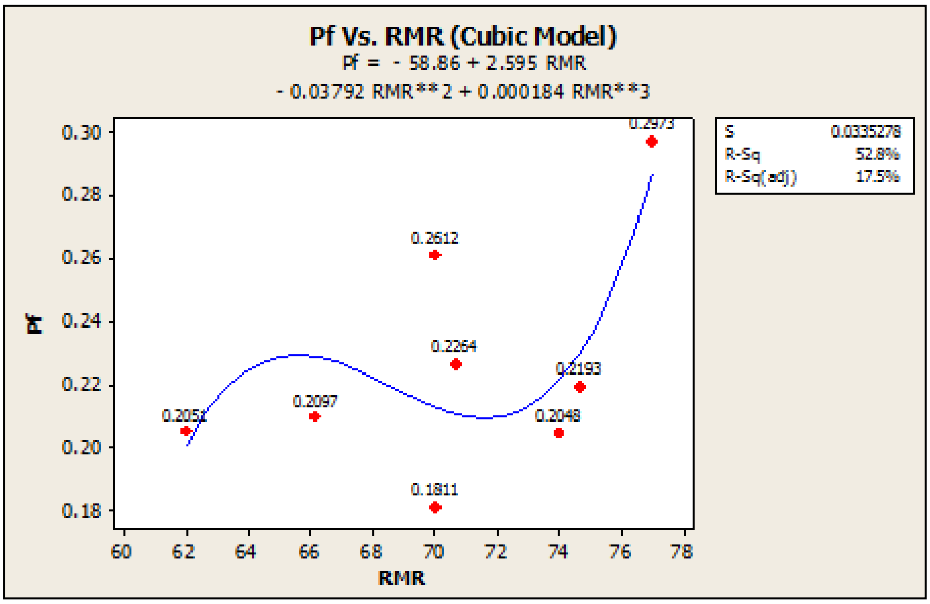

Figure 4.15: Powder Factor vs. Rock Mass Rating value (Cubic Model)

As per the results of the cubic relationship analysis, P value is greater than 0.05 and R^2value is less than 80 hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Also, there is no significant cubic relationship between Powder Factor versus Rock Mass Rating under 95% confidence level. There could be a quadratic relationship found between two factors with incremental sample test data points and the subjects is opened for further studies in future.

4.8 Pearson correlation analysis between each variable

Pearson correlation analysis results carried out using "Minitab" statistical software are tabulated in the table 4.10. According to the P values shown in the table 4.10, there is as significant correlation between powder factor & AIV values but correlation between other variables are in significant

Table 4.10: Correlations of Pf, SG, UCS, TS, AIV, RMR in "Minitab" output

	Pf	SG	UCS	TS	AIV
SG	0.150				
	0.723				
UCS	0.423	-0.017			
	0.297	0.968			
TS	-0.232	-0.245	-0.500		
	0.580	0.559	0.207		
AIV	-0.806	-0.293	-0.307	0.061	
	0.016	0.482	0.459	0.886	
RMR	0.493	0.050	0.094	-0.115	-0.696
	0.215	0.907	0.825	0.787	0.055

4.9 Multiple Regression Analysis

Multiple regression analysis was done to analyse the behavior of Powder Factor of rock blasting against rock properties; Specific Gravity, Uniaxial Compressive Strength, Tensile Strength, Aggregate Impact Value and Rock Mass Rating. Output of the "Minitab" analysis is shown in the table 4.11 below.

Table 4.11: "Minitab" output for correlation analysis

Predictor	Coef	SE Coef	T	P	VIF
Constant	0.9980000	1.2160000	0.82	0.498	
SG	-0.1399000	0.3348000	-0.42	0.717	1.4
UCS	0.0000190	0.0005177	0.04	0.974	1.9
TS	-0.0010080	0.0021140	-0.48	0.681	1.7
AIV	-0.0120040	0.0078500	-1.53	0.266	3.1
RMR	-0.0016580	0.0043980	-0.38	0.742	2.4

Where;

S = 0.0361148, R-Sq = 72.6%, R-Sq(adj) = 4.2%, PRESS = 0.124412 and

R-Sq (pred) = 0.00%

Results of variance analysis performed in "Minitab" statistical software are tabulated in table 4.12.

Table 4.12: "Minitab" output for variance analysis

Source	DF	SS	MS	F	P
Regression	5	0.006926	0	1.06	0.55
Residual Error	2	0.002609	0		
Total	7	0.009535			

Results of statistical significance analysis performed in "Minitab" statistical software are tabulated in table 4.13 below.

Table 4.13: "Minitab" output for statistical significance

Source	DFSeq	SS
SG	1	0.000214
UCS	1	0.001725
TS	1	0.000005
AIV	1	0.004796
RMR	1	0.000185

Durbin-Watson statistic is 3.44511for the above analysis.

The regression equation derived from the multiple analysis is given as equation 27 below;

$$P_f = 1.00 - 0.140\, SG + 0.000019\, UCS - 0.00101\, TS - 0.0120\, AIV - 0.00166\, RMR \qquad \text{Eq (27)}$$

Where;

Pf is the Powder factor of rock blasting, SG is the Specific Gravity of rock, UCS is the Uniaxial Compressive Strength of the rock, TS is the Tensile Strength of the rock, AIV is the Aggregate Impact Value of rocks and RMR is the Rock Mass Rating of the associated quarry.

In the above regression model equation, R^2 is less than 80% and only 73% of the variation in powder factor is explained by the regression line. P value of overall test is 0.55 which is much greater than 0.05 and concluding that model is insignificant hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) cannot be rejected. Since VIF is less than 10 “Multicollinearity” problems won’t be generated and errors are random since Durbin-Watson statistic is 3.44511 which is close to 2.

4.10 Powder factor versus explosive cost variation analysis

Primary explosives and secondary used in all the eight quarries selected for this research are similar in kind. They are PETN containing electric detonators, water-gel and Ammonium Nitrate plus fuel oil. Similarly, all the quarries are situated in Colombo district and explosive transportation cost variation is minimal. Therefore, variation of the explosive cost of every quarry is directly depending on metal production.

Explosive usages of six month period were extracted from the past records of selected quarries and explosive costs were calculated using market values of explosives. In situ rock volume subjected to each blast was considered as the volume of rock excavated by the blast. Table 4.14 shows the total explosive cost, in-situ rock volume subjected to blast, per cubic meter explosive cost and powder factor for eight quarries considered in this research.

Table 4.14: Cost Analysis for Explosive usages in selected locations

Quarry	Total Explosive cost (Rs)	In situ Volume (m3)	Explosive cost / m3	Powder factor (Kg/m3)
Q1	1,812,282.00	16109	112.49	0.22
Q2	1,731,369.00	22018	78.63	0.21
Q3	8,241,636.00	102051	80.76	0.20
Q4	12,140,712.00	49804	243.76	0.29
Q5	4,425,909.00	67360	65.70	0.20
Q6	1,108,605.00	20990	52.81	0.20
Q7	3,434,524.00	38455	89.31	0.18
Q8	4,503,010.00	48695	92.47	0.26

Variation of per cubic meter explosive cost against powder factor is shown in the graph in Figure 4.16.

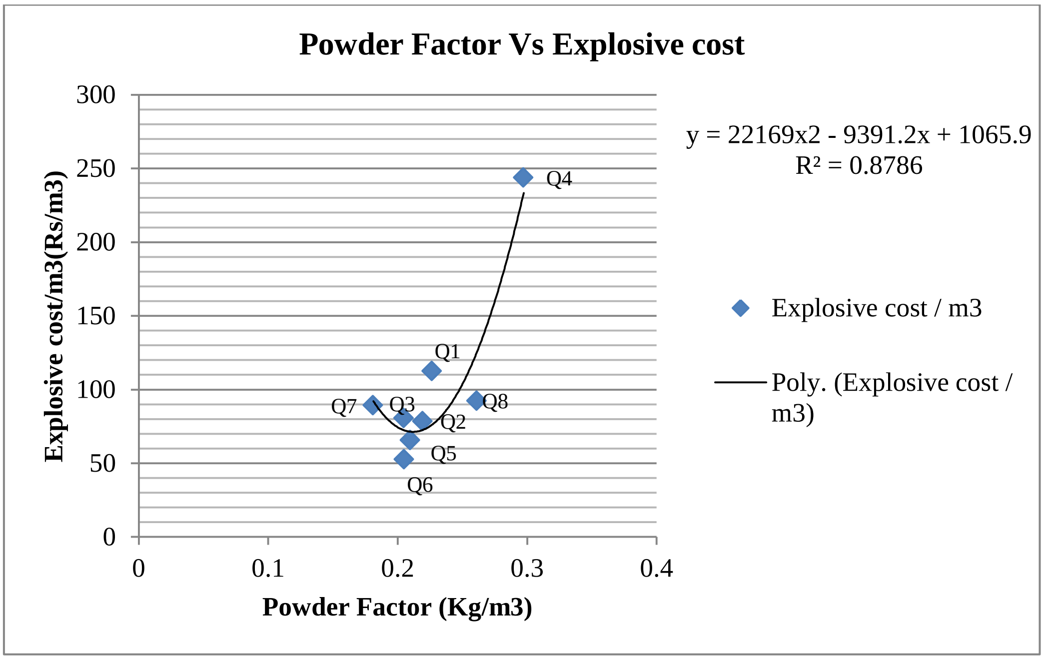

Figure 4.16: Explosive cost / m^3 vs. Powder factor variation

Cost analysis results shows that explosive cost range varies from Rs 52.13 to Rs 243.76 per cubic meter. Minimum per cubic meter explosive costs were recorded in the no 7, no 6, no 5, no3 and no 2 quarries.

Quarry no 07, the lowest powder factor recorded site reports comparatively higher explosive cost and it is due to high quartz percentage in the rock. Explosive consumption drastically increased with quartz percentage of the rock since drilling and blasting capability is very low in quartz. Other than that, it is confirmed that Q_2, Q_5 and Q_6 has minimum per cubic meter explosive cost hence lowest production cost when compared with other quarries selected for the research. Analysis shows that the drilling cost and labour cost is almost same for each quarry. Hence it can be concluded that the lowest powder factor recording quarries will incur lowest explosive cost subsequently minimum production cost when it come to the quarry metal production.

Summarized test results obtained for all the eight quarries are tabulated in the table 4.15 below.

Table 4.15: Summery of test results

Quarry	Powder Factor	Density (Kg/m3)	UCS (MPa)	Tensile strength (MPa)	AIV	RMR Value
01	0.22	2805.2	51	22	20	70.66
02	0.21	2653.7	77	14	22	74.66
03	0.20	2756.8	76	5	22	74.00
04	0.29	2721.4	124	16	17	77.00
05	0.20	2718.3	151	11	22	66.17
06	0.20	2701.6	62	16	27	62.00
07	0.18	2663.7	48	34	23	70.00
08	0.26	2696.6	81	15	18	70.00

5. CHAPTER 05: CONCLUSION AND RECOMMENDATION

Conclusions and recommendations relevant to this research are given below under this chapter 05.

5.1 Conclusion

Though the research model targeted five mechanical and aggregate rock properties such as Density, Uniaxial Compressive Strength (UCS), Tensile Strength, Aggregate Impact Value (AIV) and Rock Mass Rating (RMR) which thought to have direct relationships with the blasting powder factor, the statistical test outcome concludes that influence of all those rock property factors are insignificant other than AIV.

Model equation derived from multiple regression analysis result gave out, R^2 is less than 80% and only 73% of the variation in powder factor is explained by the regression line. P value of overall test is 0.55 which is much greater than 0.05 and concluding that model is insignificant hence standard null hypothesis of the test; that the coefficient is equal to zero (no effect) could not be rejected. Since VIF is less than 10 “Multicollinearity” problems won’t be generated and errors are random since Durbin-Watson statistic is 3.44511 which is close to 2. Hence, equation no 26 derive in section 4.9 which consist of all the considered mechanical and aggregate rock property factors had to reject from the research purview. However, Quadratic and Cubic relationships derived under section 4.6; behavior of blasting powder factor against AIV confirms statistical significances and a strong direct relationships. Figure 5.1 demonstrates the behavior of both quadratic and cubic relationships during the models validation process. Even though both quadratic and cubic relationships are agreeing with actual powder factor values very closely at low AIVs, predictions of cubic model deviate considerably away from the actual in case of higher AIVs as shown in the figure 5.1 below. Markers in blue represent quadratic model predicted powder factors while red markers represent powder factor values predicted by the cubic model. Markers in green indicate actual powder factors recorded in the quarries selected for the results validation process.

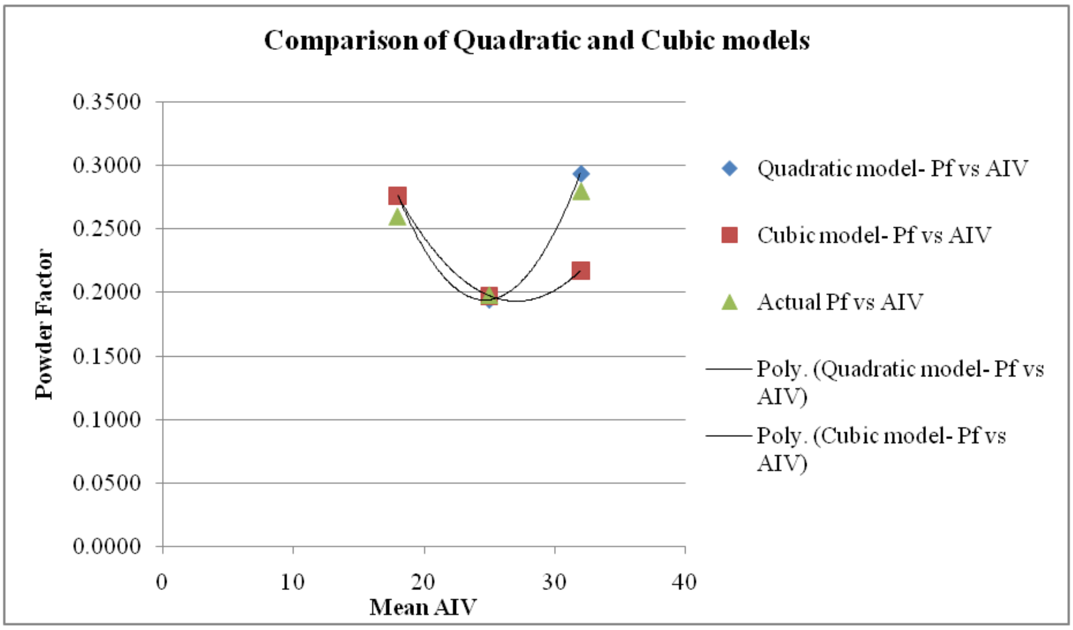

Figure 5.1: Comparison of Quadratic and Cubic models of Pf vs. AIV

Therefore, it can be concluded that the Aggregate Impact Value (AIV) influence on powder factor of blasting and furthermore affects economics of the quarry production. More importantly, quadratic formulae derived in this research for the relationship between powder factor versus Aggregate Impact Value (AIV) can be used predicting powder factor of a fresh rock with higher accuracy, even before conducting any blasting activity.

5.2 Recommendation

The research results inaugurates new dimension to select a suitable rock mass for a quarry site among several available options based on expected powder factor of rock blasting. The Quadratic relationship between Powder factor and Aggregate Impact Value (AIV) derived in this research recommend use by Mining Engineers to select rock which produce quality fragmentation while giving out low powder factor results in rock blasting operations. Nevertheless, use of the above model recommends for quarry investors to project their future potential operational costs and forecast cash flows as well as profits for the same site. Outcome is an indirect invention to control fragmentation as well as cost of the blasting operations while operational cost controlling methods for the metal quarry industry. Further, the model recommend to predict powder factors of any operating quarry using Aggregate Impact Value (AIV) and derive metal production using explosives consumption data hence will be useful tool for Mining Engineers working at the Geological Survey & Mines Bureau (GSMB) to forecast quarry productions with higher accuracy.

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7. APPENDICES

Appendix A: Quarries site locations in Western Provincial map

Appendix B: Quarries selected for the research and collected data

<u>Quarry no 01</u>: This quarry is situated at Leenawaththa, Padukka area in Colombo district and currently been mined as an active metal quarry. Garnet content of the quarry metal is comparatively high. Figure 7.1 shows view of the front working face of the quarry.

Figure 7.1: Front working face view of Quarry no 01

Explosive usage and drilling geometry data of quarry no 01 from January 2016 to July 2016; for consecutive 07 months are tabulated in the table 7.1 below.

Table 7.1: Explosive consumption and drilling geometry data for Quarry No 01

Month	Explosives		Spacing (m)	Burden (m)	Blasted depth (m)
	Water-gel (Kg)	ANFO (Kg)			
January	45.31	304.8	1.3	1.1	1557.27
February	72.25	476.2	1.3	1.1	2612.61
March	26.50	173.7	1.3	1.1	900.90
April	27.37	190.0	1.3	1.1	1033.89
May	16.37	122.7	1.3	1.1	561.99
June	90.75	695.5	1.3	1.1	3166.02
July	39.875	270.0	1.3	1.1	1432.86

Field data collected at the quarry no 01 which were used to calculate Rock Mass Rating (RMR) relevant to the site are tabulated in the table no 7.2 below.

Table 7.2: Field data sheet for RMR calculation in Quarry No 01

Quarry 01	**Location 01**	**Location 02**	**Location 03**
Strength of intact rock (UCS)	50 – 100 Mpa	50 – 100 Mpa	50 – 100 Mpa
Drill core Quality (RQD)	50% - 75%	50% - 75%	50% - 75%
Spacing of discontinuities	>2m	>2m	>2m
Condition Of Discontinuities			
Discontinuity length	10 – 20 m	1 – 3m	10 – 20 m
Separation	< 0.1 mm	None	0.1 – 1.0 mm
Roughness	Rough	Rough	Rough
Infilling	Hard Filling < 5 mm	Hard Filling < 5 mm	Hard Filling > 5 mm
Weathering	Slightly Weathered	Slightly Weathered	Slightly Weathered
Ground water condition	Damp	Damp	Damp
Effect of discontinuity strike and dip orientation in Excavation (Very favorable, favorable, unfavorable, etc…)	Strike perpendicular to blasting axis Drive against Dip 20^0 - 45^0 Unfavorable	Strike perpendicular to blasting axis Drive against Dip 20^0 - 45^0 Unfavorable	Strike perpendicular to blasting axis Drive against Dip 20^0 - 45^0 Unfavorable

<u>Quarry no 02</u>: Charnokitic rock is been mine in this quarry which is situated in Kaduwela area. The rocks encountered in and around the project area are granitic gneiss, charnockitic biotite gneiss and quartz. This quarry has recorded proper Aggregate Impact Value for the road projects. Access ramp and side view of the quarry is shown in the figure 7.2 below.

Figure 7.2: Access ramp and side view of the quarry no 02

Explosive usage and drilling geometry data of quarry no 02 from January 2016 to March 2016; for consecutive 03 months are tabulated in the table 7.3 below.

Table 7.3: Explosive consumption and drilling geometry data for Quarry No 02

Month	Explosives		Spacing (m)	Burden (m)	Blasted depth (m)
	Water-gel (Kg)	ANFO (Kg)			
January	172.00	1670.0	1.5	1.2	4506.94
February	112.99	895.0	1.5	1.2	2925.41
March	162.83	1816.0	1.5	1.2	4799.87

Field data collected at the quarry no 02which were used to calculate Rock Mass Rating (RMR) relevant to the site is tabulated in the table no 7.4 below.

Table 7.4: Field data sheet for RMR calculation in Quarry No 02

Quarry 02	**Location 01**	**Location 02**	**Location 03**
Strength of intact rock (UCS)	50 - 100 MPa	50 - 100 MPa	50 - 100 MPa
Drill core Quality (RQD)	90% - 100%	50% - 75%	75% - 90%
Spacing of discontinuities	0.6m - 2m	0.6m - 2m	0.6m - 2m
Condition Of Discontinuities			
Discontinuity length	10m - 20m	10m - 20m	10m - 20m
Separation	< 0.1m	< 0.1m	< 0.1m
Roughness	Slightly Rough	Slightly Rough	Slightly Rough
Infilling	Hard Filling<5mm	Hard Filling<5mm	Hard Filling<5mm
Weathering	Slightly Weathered	Slightly Weathered	Slightly Weathered
Ground water condition	Completely Dry	Completely Dry	Completely Dry
Effect of discontinuity strike and dip orientation in Excavation (Very favorable, favorable, unfavorable, etc…)	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair

<u>Quarry no 03</u>: This quarry is situated in a private land in Hanwella area in Colombo district. Surrounded are is highly populated and blasting activities carried out with utmost care. Part of the access ramp and working benches of the quarry no 03 can be seen in the figure 7.3 below.

Figure 7.3: Part of the access ramp and working benches of the quarry no 03

Explosive usage and drilling geometry data of quarry no 03 from January 2016 to May 2016; for consecutive 05 months are tabulated in the table 7.5 below.

Table 7.5: Explosive consumption and drilling geometry data for Quarry No 03

Month	Explosives		Spacing (m)	Burden (m)	Blasted depth (m)
	Water-gel (Kg)	ANFO (Kg)			
January	219.00	4420.0	1.8	1.7	7032.00
February	205.00	4201.0	1.8	1.7	6696.00
March	168.00	3887.0	1.8	1.7	6744.00
April	248.00	4515.0	1.8	1.7	7874.00
May	140.00	2891.0	1.8	1.7	5004.00

Field data collected at the quarry no 03 which were used to calculate Rock Mass Rating (RMR) relevant to the site are tabulated in the table no 7.6 below.

Table 7.6: Field data sheet for RMR calculation in Quarry No 03

Quarry 03	**Location 01**	**Location 02**	**Location 03**
Strength of intact rock (UCS)	50 – 100 MPa	50 – 100 MPa	50 – 100 MPa
Drill core Quality (RQD)	75% - 90%	75% - 90%	75% - 90%
Spacing of discontinuities	0.6 – 2 m	0.6 – 2 m	0.6 – 2 m
Condition Of Discontinuities			
Discontinuity length	3 – 10 m	3 – 10 m	3 – 10 m
Separation	0.1 - 1 mm	0.1 - 1 mm	0.1 - 1 mm
Roughness	Rough	Rough	Rough
Infilling	Hard Filling < 5 mm	Hard Filling < 5 mm	Hard Filling < 5 mm
Weathering	Slightly Weathered	Slightly Weathered	Slightly Weathered
Ground water condition	Completely Dry	Completely Dry	Completely Dry
Effect of discontinuity strike and dip orientation in Excavation (Very favorable, favorable, unfavorable, etc…)	Strike perpendicular to blasting axis Drive with Dip 45^0 - 90^0 Very Favorable	Strike perpendicular to blasting axis Drive with Dip 45^0 - 90^0 Very Favorable	Strike perpendicular to blasting axis Drive with Dip 45^0 - 90^0 Very Favorable

Quarry no 04: Quarry no 04 is situated in Meepe area of the Colombo district. Quarry metal found in this quarry has high garnet content it recorded higher UCS and TS values but low AI values. Overall rock mass shows higher stability hence recorded RMR is comparatively high. Quarry metal loading point and ramp to the upper bench of the quarry no 04 is shown in the figure 7.4 below.

Figure 7.4: View of quarry metal loading point and a ramp of quarry no 04

Explosive usage and drilling geometry data of quarry no 04 from January 2016 to June 2016; for consecutive 06 months are tabulated in the table 7.7 below.

Table 7.7: Explosive consumption and drilling geometry data for Quarry No 04

Month	Explosives		Spacing (m)	Burden (m)	Blasted depth (m)
	Water-gel (Kg)	ANFO (Kg)			
January	2,348.00	229.50	1.8	1.7	2865.26
February	2,521.25	226.50	1.8	1.7	2895.03
March	1,935.50	246.00	1.8	1.7	2627.98
April	2,886.00	267.13	1.8	1.7	3410.05
May	1,264.00	118.88	1.8	1.7	1521.07
June	2,544.50	222.00	1.8	1.7	2956.59

Field data collected at the quarry no 04 which were used to calculate Rock Mass Rating (RMR) relevant to the site are tabulated in the table no 7.8 below.

Table 7.8: Field data sheet for RMR calculation in Quarry No 04

Quarry 04	**Location 01**	**Location 02**	**Location 03**
Strength of intact rock (UCS)	100 – 250 MPa	100 – 250 MPa	100 – 250 MPa
Drill core Quality (RQD)	75% - 90%	75% - 90%	50% - 75%
Spacing of discontinuities	> 2m	> 2m	> 2m
Condition Of Discontinuities			
Discontinuity length	10 - 20m	3 - 10m	3 - 10m
Separation	0.1 – 1m	0.1 – 1m	0.1 – 1m
Roughness	Rough	Rough	Rough
Infilling	None	None	None
Weathering	Slightly Weathered	Un weathered	Un weathered
Ground water condition	Wet	Wet	Wet
Effect of discontinuity strike and dip orientation in Excavation (Very favorable, favorable, unfavorable, etc…)	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair

<u>Quarry no 05</u>: Quarry no 05 is situated in Kaduwela area of the Colombo district. It recorded comparatively higher RMR value because of low joint intensity in the rock mass. Quarry has excavated below surface level and a reserve to be excavated is comparatively poor. Current blasting face of the quarry no 05 is shown in the figure 7.5 below.

Figure 7.5: View of current blasting face of quarry no 05

Explosive usage and drilling geometry data of quarry no 05 from January 2016 to May 2016; for consecutive 05 months are tabulated in the table 7.9 below.

Table 7.9: Explosive consumption and drilling geometry data for Quarry No 05

Month	Explosives		Spacing (m)	Burden(m)	Blasted depth (m)
	Water-gel (Kg)	ANFO (Kg)			
January	234.00	2700.0	1.8	1.5	5107.20
February	253.50	2875.0	1.8	1.5	5635.50
March	182.00	2100.0	1.8	1.5	4074.78
April	214.50	2475.0	1.8	1.5	4733.85
May	247.00	2850.0	1.8	1.5	5396.84

Field data collected at the quarry no 05 which were used to calculate Rock Mass Rating (RMR) relevant to the site are tabulated in the table no 7.10 below.

Table 7.10: Field data sheet for RMR calculation in Quarry No 05

Quarry 05	**Location 01**	**Location 02**	**Location 03**
Strength of intact rock (UCS)	100 - 250 Mpa	100 - 250 Mpa	100 - 250 Mpa
Drill core Quality (RQD)	50% - 75%	50% - 75%	50% - 75%
Spacing of discontinuities	0.6 - 2m	0.6 - 2m	0.6 - 2m
Condition Of Discontinuities			
Discontinuity length	10 - 20m	10 - 20m	10 - 20m
Separation	1 - 5mm	1 - 5mm	1 - 5mm
Roughness	Rough	Rough	Rough
Infilling	Hard filling < 5 mm	Hard filling < 5 mm	Hard filling < 5 mm
Weathering	Slightly weathered	Slightly weathered	Slightly weathered
Ground water condition	Dripping	Wet	Wet
Effect of discontinuity strike and dip orientation in Excavation (Very favorable, favorable, unfavorable, etc…)	Strike perpendicular to blasting axis Drive with Dip 45^0 - 90^0 Very Favorable	Strike perpendicular to blasting axis Drive with Dip 45^0 - 90^0 Very Favorable	Strike perpendicular to blasting axis Drive with Dip 45^0 - 90^0 Very Favorable

Quarry no 06: Aggregate Impact Value of metal in this quarry is marginal for road construction projects. The quarry is situated in the Homagama area in the Colombo district. Partially excavated cliff of the quarry is shown in the Figure 7.6 below.

Figure 7.6: View of partially excavated cliff of the quarry no 06

Explosive usage and drilling geometry data of quarry no 06 from January 2016 to July 2016; for consecutive 07 months are tabulated in the table 7.11 below.

Table 7.11: Explosive usage and drilling geometry data for Quarry No 06

Month	Explosives		Spacing (m)	Burden (m)	Blasted depth (m)
	Water-gel (Kg)	ANFO (Kg)			
January	27.00	550.0	2.13	2.05	660.00
February	27.34	654.0	2.13	2.05	767.20
March	29.38	655.4	2.13	2.05	791.00
April	23.14	525.1	2.13	2.05	623.00
May	25.48	563.5	2.13	2.05	686.00
June	30.16	690.2	2.13	2.05	812.00
July	20.28	455.3	2.13	2.05	468.00

Field data collected at the quarry no 06 which were used to calculate Rock Mass Rating (RMR) relevant to the site are tabulated in the table no 7.12 below.

Table 7.12: Field data sheet for RMR calculation in Quarry No 06

Quarry 06	**Location 01**	**Location 02**	**Location 03**
Strength of intact rock (UCS)	50 – 100 MPa	50 – 100 MPa	50 – 100 MPa
Drill core Quality (RQD)	50% - 75%	50% - 75%	50% - 75%
Spacing of discontinuities	0.6 – 2 m	0.6 – 2 m	0.6 – 2 m
Condition Of Discontinuities			
Discontinuity length	>20 m	>20 m	10 – 20 m
Separation	< 0.1 mm	< 0.1 mm	< 0.1 mm
Roughness	Rough	Rough	Rough
Infilling	Hard Filling < 5 mm	Hard Filling < 5 mm	Hard Filling < 5 mm
Weathering	Slightly Weathered	Slightly Weathered	Slightly Weathered
Ground water condition	Wet	Dripping	Dripping
Effect of discontinuity strike and dip orientation in Excavation (Very favorable, favorable, unfavorable, etc…)	Strike parallel to the blasting axis Dip 45^0 - 90^0 Very unfavorable	Strike parallel to the blasting axis Dip 45^0 - 90^0 Very unfavorable	Strike parallel to the blasting axis Dip 45^0 - 90^0 Very unfavorable

<u>Quarry no 07</u>: Metal samples obtained from quarry no 07 contain comparatively higher amounts of quartz. It is situated in the Kaduwela area of the Colombo district. Single high bench in the operating quarry face is shown in the figure 7.7 below.

Figure 7.7: View of Single high bench in the operating face of quarry no 07

Explosive usage and drilling geometry data of quarry no 07 from January 2016 to June 2016; for consecutive 06 months are tabulated in the table 7.13 below.

Table 7.13: Explosive usage and drilling geometry data for Quarry No 07

Month	Explosives		Spacing (m)	Burden (m)	Blasted depth (m)
	Water-gel (Kg)	ANFO (Kg)			
January	367.50	1470.0	1.3	1.1	7056.78
February	253.50	1012.0	1.3	1.1	4858.90
March	60.00	240.0	1.3	1.1	1152.00
April	120.00	480.0	1.3	1.1	2304.00
May	225.00	900.0	1.3	1.1	4320.00
June	195.00	780.0	1.3	1.1	3744.00
July	180.00	720.0	1.3	1.1	3456.00

Field data collected at the quarry no 07 which were used to calculate Rock Mass Rating (RMR) relevant to the site are tabulated in the table no 7.14 below.

Table 7.14: Field data sheet for RMR calculation in Quarry No 07

Quarry 07	**Location 01**	**Location 02**	**Location 03**
Strength of intact rock (UCS)	25 - 50 MPa	25 - 50 MPa	25 - 50 MPa
Drill core Quality (RQD)	75% - 90%	75% - 90%	75% - 90%
Spacing of discontinuities	> 2m	> 2m	> 2m
Condition Of Discontinuities			
Discontinuity length	10m - 20m	10m - 20m	10m - 20m
Separation	1mm – 5mm	< 0.1mm	< 0.1mm
Roughness	Rough	Rough	Rough
Infilling	Soft Filling <5mm	Hard Filling>5mm	Hard Filling>5mm
Weathering	Slightly Weathered	Slightly Weathered	Slightly Weathered
Ground water condition	Completely Dry	Completely Dry	Completely Dry
Effect of discontinuity strike and dip orientation in Excavation (Very favorable, favorable, unfavorable, etc…)	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair

<u>Quarry no 08</u>: This Quarry has high quality rock with low quartz content and situated in Homagama area. Though the site has high metal reserve capacity, mining activities already carried out below surface level hence cost of production is comparatively high. Figure 7.8 shows the front view of the quarry face currently been mined.

Figure 7.8: Front view of the location currently been mined at quarry no 08

Explosive usage and drilling geometry data of quarry no 08 from January 2016 to July 2016; for consecutive 07 months are tabulated in the table 7.15 below.

Table 7.15: Explosive usage and drilling geometry data of Quarry no 08

Month	Explosives		Spacing (m)	Burden(m)	Blasted depth (m)
	Water-gel (Kg)	ANFO (Kg)			
January	275.00	2091.0	1.5	1.2	5056.52
February	100.00	610.0	1.5	1.2	1592.72
March	77.00	645.0	1.5	1.2	1487.05
April	34.00	350.0	1.5	1.2	755.15
May	231.00	1935.0	1.5	1.2	4461.15
June	300.00	1830.0	1.5	1.2	4778.16
July	462.00	3870.0	1.5	1.2	8922.30

Field data collected at the quarry no 08 which were used to calculate Rock Mass Rating (RMR) relevant to the site are tabulated in the table no 7.16 below.

Table 7.16: Field data sheet for RMR calculation in Quarry No 08

Quarry 08	**Location 01**	**Location 02**	**Location 03**
Strength of intact rock (UCS)	50 – 100 MPa	50 – 100 MPa	50 – 100 MPa
Drill core Quality (RQD)	75% - 90%	75% - 90%	75% - 90%
Spacing of discontinuities	0.6m – 2m	0.6m – 2m	0.6m – 2m
Condition Of Discontinuities			
Discontinuity length	10m – 20m	10m – 20m	10m – 20m
Separation	< 0.1 mm	< 0.1 mm	< 0.1 mm
Roughness	Rough	Rough	Rough
Infilling	Hard Filling < 5mm	Hard Filling < 5mm	Hard Filling < 5mm
Weathering	Un weathered	Un weathered	Un weathered
Ground water condition	Damp	Damp	Damp
Effect of discontinuity strike and dip orientation in Excavation (Very favorable, favorable, unfavorable, etc…)	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair	Strike parallel to blasting axis Dip 20^0 - 45^0 Fair

Appendix C: GSMB Mining licensing process

Geological Survey and Mines Bureau (GSMB) is the Sri Lankan regulatory authority responsible for all mining related activities in the entire country. Ultimate responsibility of the GSMB is to achieve a balance between mineral resources requirement for the development activities in the country and socio-environmental well being.

GSMB is authorized to issue variety of licenses such as exploration licenses, mining licenses, trading licenses, transport licenses as well as export permits. Classification of mining licences currently been issued by the GSMB is shown in the Figure 7.9 below.

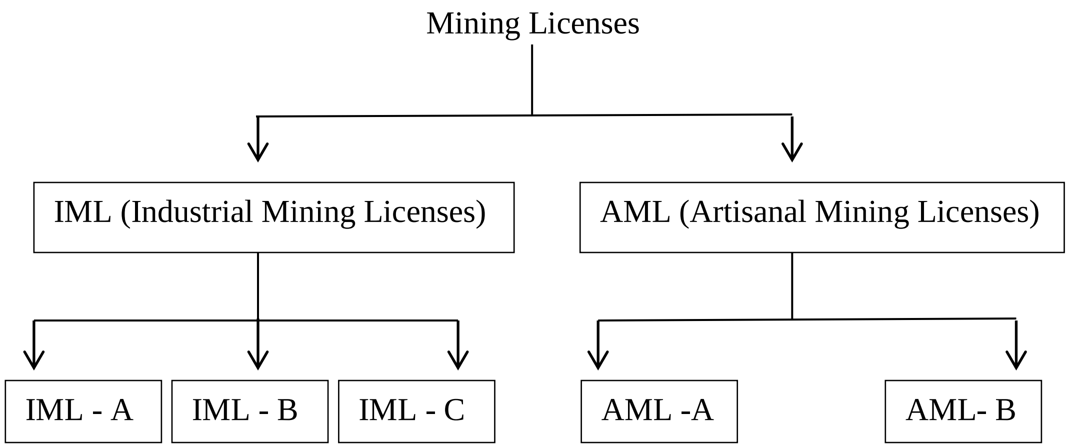

Figure 7.9: License categories

Industrial Mining License Category (IML)

Category “A” (IML – A)

In IML-A category, multi bore holes blasting is allowed using delay elements. The depth of bore holes can be more than 3.0 meters. However, bore hole depth is decided by Central Environmental Authority and GSMB after conducting a test blast. The production volume can be more than 1500 cubic meters per month. In this category mining license, mine machineries such as track drills, jack hammers, rock breakers, front end loaders and other machinery can be used.

Category “B” (IML – B)

This industrial mining licenses category allows only single bore hole blasting method. The depth of bore hole should not be less than 1.5 meters and should not be more than 3.0 meters. The production volume allows is not less than 600 cubic meters and not more than 1500 cubic meter per month. Only Jack hammers, excavator machinery are allowed using in this category.

Category “C” (IML – C)

Industrial Mining License “C” category license allows only single shot hole method. The depth of bore hole should be less than 1.5 meters and the production volume should be less than 600 cubic meters per month. Only jack hammers are allowed to use for mining activities.

Artisanal Mining License (AML) Category

Category “A” (AML – A)

This category licenses permits depth of bore holes less than 1.5 meters. The production volumes should not be less than 100 cubic meters and should not be more than 600 cubic meters per month. Under the Artisanal Mining Licenses, machinery cannot be used.

Category “B” (AML – B)

Artisanal Mining Licenses “B” category licenses permits bore hole depth less than 1.5 meters. The production volume should not be exceeding 100 cubic meters per month. Depth of excavation should not exceed 2 meters and machinery can not to be used for mining operations (Minister, 1993).