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Original Research Article

New Empirical Relations for Determination of Rock Slope Safety Factor in Fully Drained Conditions on Section RS06 of Block five in Sungun Copper Mine

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Abstract: During the past centuries, more methods with many involved imperfections and limitations in various conditions suggested for slopes stability analysis specially based on different rock mass classification systems. However, RMi (rock mass index) classification system did not use for scrutinized assessments until now in accordance with its precise studies upon discontinuities system as joint characteristics, also block size as three-dimensional block volumes and intact rock strength to represent the rock mass uni-axial compressive strength. In this paper, for the first time, several empirical relations recommend only for the section RS06 in the block five of Sungun Copper Mine for 37° overall slope angle in fully drained conditions which presents some RMi applications in determining rock slopes safety factor in these situations to show multidimensional importance existence relations between factor of safety and RMi. Meanwhile, computed safety factors values from proposed equations compared with results of the limit equilibrium slope stability evaluation of SLOPE/W software. Using the earth's strength limit fundamental key parameter in offered formulas, which assays and indicates the major gravitational effects in slope stability procedures, increasing predictability of earth's materials behaviors that enabled us to calculate safety factor through RMi, simply, comprehensively and accurately in reasonable short-duration. Also, leads to better analyzing, showing moreover, understanding of the relationships between safety factor and RMi. Furthermore, this factor helps open new horizons and bridge between various sciences such as basis and applied physics, rock mechanics, rock engineering and engineering geology to indicating extended complicate correlations of all sciences' components that play role in the entire word wide. Research procedures in this article planned based on present mathematical relations via main properties of the slopes, earth and the universe. Low root-mean-square error, percent of variance, standard deviation and variation coefficient also great efficiency and correlation coefficient of calculated safety factor through new suggested empirical equations in its comparison with SLOPE/W results indicated high accuracy of proposed formulas.

Keywords: Strength Limit Number of Earth, Safety Factor Determination, RMi Classification System, Fully Drained conditions, Empirical Relations, Sungun Copper Mine, Slopes Stability Engineering, Earth Sciences

1. Introduction

Any part of existence such as key fundamental factors enable scientists to discover knowledge and solve the greatest challenges which faces to accelerate finding out new advanced intelligence researches, innovations and technologies that are changing lives every day.

The slope stability analysis has many applications in the engineering projects such as dams, roads, open pits, embankments, excavations, landfills and other engineering structures. These analyses studies carried out to prevent, eliminate and minimize the occurrences of slope failure, landslide or other results of instabilities. Slope stability investigation in open pit mines and design of these mines' walls at different stages of mining is important for the safe and economic mining operations.

The slope stability analyses usually implemented at the beginning, and sometimes throughout the lifetime of projects during planning, design, construction, improvement, rehabilitation, and maintenance by planners, engineers, geotechnical engineers, geologists, contractors, technicians, and maintenance workers, which become involved in this process [Adopted from 2].

Increasing construction of engineered cut and fill slopes in projects has necessitated to fully understanding analytical methods, investigative tools, and stabilization methods specialty construction techniques with contemplating limitations of them which must be modeled in realistic ways to solve slope stability problems have been faced.

Many scientists and researchers have focused on slopes stability assessment however, now slope instabilities problems are still presents an important and significant challenge for designers in mining, civil, rock mechanics, structural geology, geotechnical engineering and engineering geology [Adopted from 115, 166].

A common technique for the primary estimates of rock slopes stability even during implementing the work is empirical design using empirical methods and in the other words using of classification systems and empirical formulas [Adopted from 106]. Different rock mass classification systems used to assess the stability of slopes therewith for the design of slopes too. These systems worked based on empirical approaches, which means relations between rock mass parameters and various parameters of slopes such as height and dip of the slope [Adopted from 275]. Used calculation methods and parameters seem always not suited to any particular condition for slopes stability and should improve [Adopted from 106].

Construction materials commonly used in mining and civil engineering mostly characterized by their strength properties. This basic property of the material used in the engineering and design. In rock engineering, no such specific strength characterization of the rock mass is in common use. Most engineering carried out using various descriptions, classifications and un-quantified experience [221, 222]. Hoek and Brown (1980), Nieto (1983), Bieniawski (1984) and therefore several other authors have represented the necessities for rock masses' strength characterization. The Rock Mass index (RMi) developed to characterize the strength of the rock mass for construction purposes. The RMi is determined based on inherent parameters of the rock mass. Rock Mass index calculated by combining the intact rock compressive strength and jointing parameter which definitely and clearly represents the main jointing features, namely block volume (or density of joints) and joint characteristics as given by joint roughness, joint alteration and joint size. Using parameters in the RMi is an important issue in engineering [Adopted from 31, 117, 206, 221, 222].

In order to deficiency of different rock mass classification systems, must use a system which has a complete approach to slopes stability evaluation specialized in factor of safety determination also cover important and affecting items in calculations moreover emends limitations.

RMi classification system has many applications in different fields like rock mechanics and rock engineering as follows [Adopted from 221]:

a) RMi parameters applied in practical design and engineering (rock design and engineering) such as: stability, rock support calculations, TBM progressive evaluation, rocks blasting and fragmentation.
b) Application in systems for rock support evaluations such as: Q-system, RMR system and NATM.
c) Applied to input in rock mechanics likes: hoek-brown failure criterion, numerical modeling, rock mass deformation modulus, ground response curves.
d) Used as input data in other engineering methods.
e) Applied in communication.

The Assessment of RMi classification system applicability in rock slopes stability is implementing for the first time by author. One part of these applications are presents related terms, relations and formulas between safety factors of slopes with rock mass index in fully drained conditions. In fact, the principle and particular goal of this scrutiny is present the relationship between rock slopes safety factor with RMi classification system in fully drained conditions. And present relevant equations and formulas for slope stability assessments only for section RS06 of block five of Sungun copper mine as the largest open-cast copper mine of Iran for 37° overall slope angle in response to: Does any relation between factor of safety and RMi?

Using of RMi classification system in these empirical formulas has some benefits [Adopted from 221, 223, 224]:

(1) The three-dimensional block volume and joint parameters for rock masses uses in RMi determination that will generally meliorate the rock mass's characterization (description of the features of rock mass) and hence lead to better estimates.

(2) Using of RMi classification system cover most important ground parameters that led to better assessments.

(3) The RMi system covers a vast spectrum of rock mass variation. Therefore, mentioned system has eventualities for extensive usages rather than other today's rock mass classification and characterization systems.

(4) The RMi can used for difficult estimates easily when access limited information about the ground condition.

(5) The RMi suggests a suitable platform and framework for engineering judgments.

(6) The RMi will give principal meaningful improvements in the use of geological input data.

(7) The RMi very likely describes also determine the qualities of a wider range of materials more than most classification systems.
(8) Relatively quick and inexpensive to implement.

Some of the main characteristics, which affecting on slopes failures that could considered for more comprehensive slopes' design are as follows:

1) Fundamental factors of nature: Gravity, etc.; 2) Geology: Geological Conditions; 3) State of all stresses: like as shear stresses, current stresses in the rock and soil; 4) State of strains: strain softening; 5) Geometrical data: Geometry of the slope: Height and angle of the slope, bench height and angle; 6) Slope Characteristics: Position of the slope; 7) Loads: External loading, applied loads, cyclic loading, additional loads at the top of the slope, seismic loadings (loads), groundwater load, soil net weight load, ongoing load on the surface, concentrated horizontal and vertical loads, loads coming from structures in the region, constant loads; 8) External forces: reinforcing elements, seismic accelerations; 9) Site characteristics: site surface and subsurface conditions, Site topography; 10) Geology: Geological information: properties of soil and rock mass (Soil parameters: soil cohesion, internal friction angle of the soil, the power functional relationship for the soil), Soil and rock strength (soil type, stratification); 11) Geology: Geological Discontinuities (the presence and of structural discontinuities) such as: Fault, Joint, bedding plane, crack (crack width); 12) Forming Materials: Slopes forming materials; 13) Characteristics of structural discontinuities: Discontinuity orientation, discontinuity spacing; 14) Alternation of materials by faulting, joint or discontinuity systems; 15) Movements and tension in joints; 16) Cracking; 17) Swelling; 18) Erosion;

19) Weathering, progressive weathering; 20) Leaching; 21) Decomposition of clayey rock fills; 22) Hydrology: Water: ground water conditions (a. Alter the cohesion and frictional parameters and b. Reduction of the normal effective stress), water content, drainage pattern, rainfall, permeability, aquifer, pore pressure, water pressure in cracks at the top of the slope, hydrostatic forces' effects; 23) Strength: shear strength (shear strength parameters), compressive strength, and tensile strength; 24) Geotechnical parameters: gran size, moisture content, atterberg limit, cohesion and unit weight etc.; 25) Method of construction: shovel, dumper, BWE or combination; 26) Possible effects of proposed construction; 27) Excavations: excavation at the bottom of the slope; 28) Disturbances: Excavation disturbances; 29) Critical geological conditions; 30) Dynamic forces: blasting; 31) Seismicity: Seismic velocity, seismic activities (earthquake activity); 32) Displacements: Displacements caused by earthquakes; 33) Temperature; 34) Creep: creep under sustained loads, progressive creep; 35) Brittle fracture; 36) Liquefaction: Liquefaction of weaker soil layers; 37) Environment: Environmental influencers; 38) Deformations: Internal deformation, deformation limit values; 39) Properties of other materials; 40) Lithology; 41) Geological Structure: Amount and direction of dip, Intra-formational shear zones, Joints and discontinuities (a. Reduction of shear strength, b. Change permeability, c. Act as sub surface drain and plains of failure), Faults (a. Weathering and alternation along the faults, b. Act as ground water conduits, c. Provides a probable plane of failure); 42) Planes of weakness; 43) Climate: Climatic conditions; 44) Mining Methods and Equipment; 45) Seepage; 46) Vegetation; 47) Intact rock strength …

Because of the continuity of universe components there are different factors involve in cosmos and related process to them. Therefore, many parameters can use on FOS calculations, which act on a continuum procedure. In this paper, try to use some of the affecting and determining parameters in slope stability fields for FOS calculations. Strength limit factor of earth is one of these important parameters that help contemplate many other involving factors in scientific analysis potentially. One application of this factor will use to calculating safety factor of slopes on planets (on earth and space). All of the powers, coefficients and numbers serve in suggested formulas relevance with properties and characteristics of earth. In this paper, consider a new aspect in slope stability field that can extended these achieves to any sciences. Empirical relations suggested based on various parameters which shown in Table 8.

On the other hand, as the preliminary step toward to vast precise inter-disciplinary researches in sciences in the future, the main objectives in this study are showing the existence of relations between rock slopes engineering, rock mechanics and fundamental parameters and multi-disciplinary physics. Such as some of the existing relations between RMi rock mass classification system, factor of safety calculations and earth's strength limit key factor only for a defined section of one block of Sungun open pit mine under fully drained conditions.

The results provide a better understanding of the impact of strength limit of earth and RMi classification system in slope stability evaluations. In addition, results indicated that the suggested formulas have high accuracy in safety factor calculations, which contemplates safety, cost effectiveness, simplicity, time saving, widely adopting, easy understanding, training and calculating. Therewith, mentioned equations have no environmental impact also does not require to any equipment and assumptions. The obtained results by suggested relations for safety factor computation in fully drained conditions can then compare with any other assessment methods of slope stability even with result of any analysis software. Using of RMi system in safety factor determination implies the unity, correlation and dependency all of the existence parts (like the universe).

In this paper, Section 2 introduces a background of previous slope stability studies. Section 3 illustrate limitations of current design approaches for slope stability. Section 4 presents brief specifications of Sungun copper mine. Section 5 demonstrates basilar theory and various definitions of strength limit factor of earth. In Section 6, express specifications of variable strength limit of earth. Section 7 explains determination of cohesion variable limit number for slopes. Section 8 represents new procedural empirical investigations of safety factor computation formulas by RMi classification system for fully drained condition. Section 9 clarifies the validity check of recommended equations in comparison with stability analysis results of limit equilibrium SLOPE/W software. Section 10 describes discussion and Section 11 explain conclusion about the relationships between the factor of safety and RMi classification system.

2. Previous Studies on Slopes Stability Analysis Procedures

2.1. Review of available slope stability researches approaches

2.1.1. Background of slope stability researches in rock masses

It noted that the jointed rock masses strength was difficult to evaluate notoriously. Generally, mentioned rock masses are expressive inhomogeneous, discontinuous media composed of rock material and naturally occurring discontinuities such as joints, fractures and bedding planes. It is very difficult to make any analysis in presence of these features using simple theoretical solutions, like the limit equilibrium method. Therewith, the displacement finite element method is not suitable for analyzing fractured, breached, interrupted and discontinuous rock masses without including special interface or joint elements [Adopted from 166]. Jaeger (1971) and Goodman and Kieffer (2000) have outlined several simple methods and corroborated their limitations to overcome the problem of rock slope strength estimating and stability due to complicated failure mechanisms. In addition, many criteria have proposed by Hoek and Brown (1980a), Yu et al. (2002), Grasselli and Egger (2003), Sheorey (1997), and Yudhbir et al. (1983) for estimating rock strength. Currently the Hoek-Brown failure criterion is one recognized approach, which used to rock mass strength estimation widely [116, 119]. Merifield et al. (2006) pointed out that the Hoek-Brown failure criterion is one of the few non-linear criteria, which used by engineers to estimate rock mass strength [Adopted from 100, 102, 116, 119, 132, 166, 184, 262, 326, 327].

2.1.2. Past slope stability researches with physical model tests

Ladanyi and Archambault (Ladanyi and Archambault, 1970; Ladanyi and Archambault, 1972) carried out some of the uttermost remarkable model tests with relevance to probable failure mechanisms in large-scale slopes. Two types of failures observed in the tests in presence of discontinuous joints. Shear failure along a well-defined failure surface was the first type and shear zone was the second type of formation. However, shear failure was the big difference, which did not occur along the concrete bricks' interfaces, but instead appeared as shear failure through the intact material [Adopted from 148, 149, 166].

Model experimentations using a mix of gypsum plaster, water and celite carried out by Einstein et al. (1970). The purpose was to simulate a brittle rock of relatively high strength, like as granite and quartzite. A considerable result from these tests was that the confining stress forcefully impressed the mechanisms of failure in the samples under triaxial stress loadings. Failure happened by the side of the pre-existing joints in low confining stress (less than 10 MPa). However, failure occurred principally all over the intact material in higher confining stress. The test conclusions moreover represented that the jointed samples' overall strength was lower than the model material's intact strength however, failure happened throughout the intact material [Adopted from 80, 166].

Stacey (1973) was tested several different joint configurations according to centrifuge tests. The results reveal that failure was not observed in the tests via the intact model material but failure happened as sliding along pre-existing joints. The above studies reported some variation in the results from different model tests. This fact can likely clarify with the differences in loading conditions and model material. The conditions of loading alter from alone gravitational loading to biaxial and triaxial loading [Adopted from 166, 282].

A review of the literatures shows that there are very few small-scale experimental results presented for rock slopes. This is probably to be happened/ true because Stewart et al. (1994) highlighted that hard rock modelling could necessitate large capacity centrifuge equipment. Stewart et al. (1994) in their study are used centrifuge modelling to investigate mechanisms of rock slope failure. The collapse mechanism evident in the model compared well with flexural toppling failure observed in the field. Furthermore, it found that the rock masses with high stiffness would fail in a brittle fashion [Adopted from 166, 287].

Adhikary et al. (1994) presented a set of rock slopes stability charts due to flexural toppling failures. These charts regulated based on the resultants from centrifuge tests and the limiting equilibrium method. However, Adhikary and Dyskin (2007) were found that the rock slope stability through the chart solutions tends to overestimate in the case of joint angles of less than 20˚-25˚. Furthermore, Adhikary and Dyskin (2007) demonstrated that the fractures could observe from the toe and (1) in high joint friction angle cases; fractures propagate back into the slope immediately and (2) in low joint friction angle cases, fractures distribute progressively back into the slope [Adopted from 3, 6, 166].

2.1.3. Past researches based on the limit equilibrium method

The limit equilibrium (LE) approach created and has developed since the 1930s, for circular and non-circular surfaces' slopes instabilities problems. Fellenius in 1936 presented the Swedish circle that can only use for circular slip surfaces. Bishop (1955) enlarged a revised method for circular slip surface assessment, which made better the accuracy of the FOS calculations and was suitable for automated computer methods. Janbu's method and Janbu's simplified method (1956) is usually used for non-circular slips surface. After that, the Morgenstern-Price (1965) contemplates the normal and tangential also the moment equilibrium for each slice in circular and non-circular slip surfaces. Janbu's Generalized Procedure of Slices (GPS) (1957, 1973) is applying vertical slices for any shape of slip-surface that consider vertical and horizontal force and moment equilibrium of the slices and moment equilibrium of the entire slide mass. Spencer's method (1967) basically proposed for circular slip surfaces analysis also it is developed for non-circular slip surfaces through adopting a frictional center of rotation. Sarma's (1973) methods have since been proposed which accounted both force and moment equilibrium. These advanced LE methods improve the safety factor computations' exactitude [Adopted from 7, 39, 84, 134, 135, 136, 180, 194, 250, 280].

Currently, engineers trying to predict rock slopes stability using Hoek and Bray (1981) stability charts typically. These chart solutions take the water table in to account and are suited to uniform rock and rockfill slopes. In addition, Zanbak (1983) suggested a set of rock slopes stability charts suited for cases, which capable of accepting or permitting no resistance against toppling failure based on the limit equilibrium theorem. However, the conventional rock masses Mohr-Coulomb parameters (c' and Ø') or block interfaces are required as input for these two sets of chart solutions [Adopted from 115, 166, 328].

Sonmez et al. (1998) gained parameters of rock slope strength through back analysis of slope failures. In their study, rock mass classification's applicability and a practical procedure for estimating the mobilized shear strength based on the Hoek-Brown yield criterion clarified. They derived that determination of the shear strength is very difficult for jointed rock masses, particularly due to the scale effect [Adopted from 166, 278].

Chen et al. (2003) investigated rock slope stability under earthquake loadings through conducted a series of back-analyses for a case history. They found that could reasonably simulate the slope condition in the field when using the reduction factor of 2/3 for peak strength parameters. These peak strength parameters were attained from the laboratory testing in terms of c' and Ø′. Moreover, they indicated that the vertical ground acceleration was an important factor for inducing rockslide under near field conditions [Adopted from 52, 166].

Day and Seery (2007) in accordance to the failure mechanism's back-analysis emphasized that a major geological structure is the key factor, which controls slope failure. Thus, it cannot ignore that the slip surface follows the structural features in slope stability analysis. Harman et al. (2007) adopted a supposed case and investigated the permeability influences on the stability of rock slope. They found that the factor of safety increase with increasing permeability. As regards the fact that several types of rock have a low permeability, Harman et al. (2007) are indicated in their researches if these types to be and behave as intact rock then rock slopes should have no pore pressure and hydraulic continuity [Adopted from 67, 112, 166].

2.1.4. Past researches based on the numerical analysis

Zienkiewicz and Taylor (2000), Stead et al. (2001), Liu and Quek (2003), Hartmann and Katz (2007) efficiently contributed to cognizing theoretical background and the applications of numerical methods through computer software solutions [Adopted from 113, 171, 214, 286, 330]. Stead et al. (2001), Stacey et al. (2003), Singh (2011) and Nutakor (2012) applied numerical analysis for slope stability calculations [Adopted from 214, 215, 274, 283, 286].

Buhan et al. (2002) according to the numerical analysis found that the ultimate results of a slope stability analysis might affected by scale-effects in rock masses. Hoek et al. (2000), Wang C. et al. (2003), Eberhardt et al. (2004), and Stead et al. (2006) have been used a range of numerical methods in previous assays of progressive failures and/or rock slopes safety factor assessment. These include the continuum methods (finite element method and the finite difference method), the discontinuum methods (distinct element and discontinuous deformation analysis), and finite-discrete-element codes. Specifically, Elmo et al. (2007) study modelled a large-scale open pit mine in 2D and 3D analyses using finite-discrete-element codes. It certified that 3D large scale analysis of the fracturing process is currently limited because of the weak memory and processing capacity of computer hardware [Adopted from 43, 78, 81, 122, 166, 285, 305].

Stewart et al. (1994) applied the finite difference method (FDM) to investigate rock slope stability. In their study, the strain softening found to be an important factor in some slope stability situations. Adhikary et al. (1995) obtained stress concentration pattern around the slope's toe for the cases of frictionless joints from the finite element analysis. Hence, the mechanism of failure is progressive for the slope with frictionless joints. In addition, they implied that the frictional sliding along the joints tended to re-propagate the instant stresses rather evenly over a large area. This area expands inside from the slope's toe and thus, the slope fails immediately. Adhikary and Dyskin (2007) in their study found that the joint friction plays the greatest importance role in the toppling failure mechanism. However, the joint cohesion does not have a same result on the mechanism of failure [Adopted from 4, 6, 166, 287].

Clough and Woodward (1967), Kulhawy and Duncan (1972), Adikari et al. (1982) and Veiga Pinto and Neves (1985) in their analyses was used the finite element method [Adopted from 5, 57, 147, 302].

2.1.5. Past researches according to limit analysis theorems

Siad (2003) created 2D charts in accordance to the upper bound approach, which can apply for rock slopes with earthquake effects. A range of parameters includes slope angle, joint inclination, shear strength of rock masses, joints, etc. considered in this analysis study. In addition, Chen et al. (2001a) investigated about 3D slope stability of rocks. In their study, the critical failure mode can be found by optimization routines; however, the failure surface requires still to be assumed in advance. Moreover, the presented solutions in the above studies need conventional Mohr-Coulomb soil parameters, cohesion (c') and friction angle (Ø'), as input data [Adopted from 54, 166, 265].

Tangential strength parameters (C_t and $\emptyset_t$) from the $\sigma - \tau$ planes of nonlinear failure criteria adopted by Collins et al. (1988), Drescher and Christopoulos (1988) and Yang et al. (2004a) for slope stability estimations. After the Yang et al. (2004b) study, the latest version of the Hoek-Brown failure criterion used to conduct slope stability analyses. The effects of the seismic loadings [321-323] and pore pressure [324] on the rock slope stability considered. The studies of Yang et al. (2004b), Yang et al. (2004b) and Yang and Zou (2006) indicate the only endeavor at being preparing slope stability factors to estimating rock slope stability [Adopted from 59, 76, 166, 321, 322, 323, 324].

2.1.6. Previous slope stability researches by the pseudo static (PS) method

Seed (1979) tries to estimate stability of rock slope with contemplating of earthquake effects. Seed (1979) proposed the adopted seismic coefficient versus Richter's earthquake magnitude based on the PS analysis. Luo et al. (2004) found that ground water maybe considerably decrease stability of slopes during earthquakes which excite (or stimulate) surrounding ground of slopes where the maximum seismic coefficient obtained by up to 60% changes. Sepúlveda et al. (2005a) indicated that the topographic amplification effects such as slope orientation and seismic wavelength might influence the stability assessment of rock slopes. Chen et al. (2003) in their case study found the vertical ground acceleration as to be an important factor leading to rockslide under near field conditions [Adopted from 52, 166, 173, 251, 258].

PS method applied and extended to evaluate the induced ground movement through an earthquake by Newmark (1965). Sepúlveda et al. (2005b), Huang et al. (2001) and Ling and Cheng (1997) has been accepted this approach and extensively used to study earthquake triggered landslides and rockslides. Pradel et al. (2005) in particular, obtained a good agreement of slope crest displacement between the calculated and observed results. In their study, used strength parameters in analyses specified via repeated direct shear testing and back analysis [Adopted from 124, 166, 169, 203, 231, 259].

Newmark (1965), Seed (1979), Baker et al. (2006), Ling et al. (1997), and Loukidis et al. (2003) has been applied the PS approach in number of investigations, mainly due to its simplicity. Baker et al. (2006) and Loukidis et al. (2003) particularly provide chart solutions for soil slopes through have adopted the PS method respectively in limit equilibrium analysis and limit analysis. By using complicated dynamic response analysis coupled with appropriate constitutive laws, a more precise seismic evaluation for slopes can obtained. However, the PS method is still applicable and recommended as a screening procedure to identify any requirement for more sophisticated dynamic analyses. Although Cotecchia (1987) and Kramer (1996) represented that the PS approach has a number of limitations, generally conservative approach purposed as to be the considered methodology. This approach is the one most often used in current practice [Adopted from 16, 61, 146, 166, 170, 172, 203, 251].

The seismic coefficients in generally are determined from experience by using the maximum horizontal acceleration or peak ground acceleration of a designed earthquake. It should consider that, current design employing PS method is mostly according to a horizontal seismic coefficient (k_h). Therefore, this research is firstly centralizing on investigating the effects of earthquakes on stability of rock slopes by applying a range of horizontal seismic coefficients. Seed (1979) proposed that the PS analysis was applicable in evaluating the efficiency of embankments, which built from materials that do not endure considerable strength loss during earthquakes by reference to the magnitude of k_h[Adopted from 166, 251]. It is recommended to utilize k = 0.1 for earthquakes of Richter's magnitude 6.5, and k = 0.15 for earthquakes of Richter's magnitude 8.5. For both cases, a safety factor $F \geq 1.15$ is required for design [166, 251].

The Hynes-Griffin and Franklin's (1984) suggestion proposed one of the extensively used and accepted methods for determining a suitable value of k_h. They recommended that a PS analysis could apply for preliminary slope stability evaluation, where a seismic coefficient equal to one-half, the measured bedrock acceleration adopted. If the obtained safety factor is greater than 1.0, then the provided slope design can accept. Hynes-Griffin and Franklin (1984) proposed that a more thorough numerical analysis need to be performed for safety factors of less than 1. Because the magnitude of k_h is related to the measured bedrock acceleration as discussed above. However, the PS method may not account for the site amplification induced by the underlain stratum [28] or topography [259] etc. [Adopted from 28, 131, 166, 259].

The California Division of Mines and Geology (1997) summarized a diagram to select an appropriate PS coefficient for a given site. This provides the recommendations in regard to the seismic coefficient (k_h), versus a required factor of safety [Adopted from 44, 166]. From this diagram, it can recognize that the recommended k_h values do not exceed 0.375. Therefore, the range of the seismic coefficients adopted in the present study will be between k_h = 0.0 and k_h = 0.375 [44, 166].

2.1.7. Kinematic analysis

Hoek and Bray (1981), was explained the kinematic analysis method, which developed by Goodman (1989) and modified by Wyllie and Mah (2004). The investigation of potential planar, wedge and toppling failure conceives through this method via assumes that only friction involved in the sliding surface' shear strength and the cohesion is zero [Adopted from 99, 115, 316].

2.1.8. Three-dimensional analysis

Anagnosti (1969) more advanced the safety factor specification of the potential sliding mass for different shapes, which is a development of the 2-D Morgenstern-Price method (1967), via setting equilibrium equations for the series of thin vertical slices and assuming limit equilibrium conditions on sliding sides of each slice. Hovland (1977) approach is a development of the assumptions involved in the two-dimensional ordinary method of slices but columns used instead of slices. This is inaccurate because it assumes zero normal stress on vertical surfaces. Chen (1981) and Chen and Chameau (1982) according to extended Spencer and finite element methods and Chen and Chameau (1983) based on developed Spencer method expanded a pervasive study of the three-dimensional effects on the stability of slopes for vast types of soil parameters. Baligh and Azzouz (1975), also Baligh, Azzouz, and Ladd (1977) extended the concept of the two-dimensional circular arc method for simple loaded slopes with revolution slip surfaces to evaluate the end effects of the three-dimensional slip surface developed in a cohesive slope. Azzouz and Baligh (1976) present design charts for three-dimensional stability of cohesive slopes subjected to surcharge loads. Azzouz, et al. (Azzouz, Baligh, and Ladd) (1981) extended 3-D slope stability analysis based on extended Swedish circle for real embankments with slip surfaces of revolutions. Azzouz and Baligh (1983) developed 3-D stability analysis for slopes with loads on top in accordance to extended Swedish circle. Leshchinsky, Baker and Silver (1985) is according to limit equilibrium and variational analysis propound a 3-D mathematical approach for slope stability analysis, which represented by Kopacsy (1957). They proponed a quantified formulation of a given slope's safety margin relative to its available shear strength and therefore allowed a limiting condition's application (i.e., the adjusted Coulomb's failure criterion) for stable slopes. Hungr (1987) uses a microcomputer program (CLARA-3) (based on extended Bishop's modified) suggested a 3-D method which is a direct development of the assumptions accompanied by Bishop's (1954) 2-D simplified method. Hungr et al. (1989) applied a 3-D way, which was a development of the assumptions in Bishop's (1954) simplified and simplified two-dimensional Janbu models, and they displayed comparisons for number of solutions. Other efforts on developing methods of three-dimensional analysis of slopes stability are as follows:

1) Giger and Krizek (1975) and Giger and Krizek (1976) based on upper bound theory of perfect plasticity. 2) Hutchinson and Sarma (1985) based on three-dimensional limit equilibrium. 3) Dennhardt and Forster (1985) based on assumed S on slip surface. 4) Ugai (1985), Leshchinsky and Baker (1986) and Baker and Leshchinsky (1987) based on limit equilibrium and variational analysis. 5) Cavounidis (1987) based on limit equilibrium. 6) Leshchinsky and Mullet (1988a, 1988b) based on limit equilibrium and variational analysis. 6) Gens et al. (1988) based on extended Swedish

circle. 7) Xing (1988a, 1988b) based on limit equilibrium. 8) Ugai (1988) based on extended ordinary method of slices. 9) Bishop's modified, Janbu and Spencer, Michalowski (1989) based on kinematic theorem of limit plasticity. 10) Seed et al. (1990) based on Ad hoc 2-D and 3-D. 11) Leshchinsky (February 1992) and Leshchinsky and Huang (October 1992, November 1992) based on limit equilibrium and variational analysis. 12) Cavounidis and Kalogeropoulos (1992) based on 3D method. 13) Azzouz and Baligh (1978) and Lam and Fredlund (1993) based on 2D general limit equilibrium. 14) Yamagami and Jiang (1996, 1997) based on Simplified Junbu's methods (1954). 15) Huang and Tsai (2000) based on limit equilibrium and two-directional factor of safety. 16) Huang et al. (2002) based on Junbu's method and two-directional factor of safety. 17) Chen et al. (2003) based on Spencer's method. 18) Jiang and Yamagami (2004) based on Spencer's method according to variational analysis. 19) Cheng and Yip (2007) based on Bishop's, Junbu's and Morgenstern- Price's methods. 20) Zheng (2009) based on limit equilibrium. 21) Sun et al. (2012) based on Morgenstern-Price's method (for generalized slip surface) [Adopted from 7, 8, 9, 10, 11, 12, 15, 17, 18, 38, 45, 46, 48, 49, 50, 53, 55, 75, 91, 93, 94, 123, 125, 126, 128, 129, 130, 133, 137, 139, 144, 150, 158, 159, 160, 161, 162, 163, 164, 185, 195, 252, 289, 295, 296, 317, 318, 319, 320, 329].

2.1.9. Other investigations

Goldscheider et al. (2010), Martens et al. (2011), Petri and Stein (2012) and Nutakor (2012) consider the slope stability analysis using slices [Adopted from 96, 179, 214, 228].

Baker and Gaber (1978) applied the variational calculus to locating the critical slip surface. Baker (1980) utilized dynamic programming for this purpose. Celestino and Duncan (1981) also Li and White (1987) implemented critical noncircular slip surface through alternating variable ways. The Monte Carlo technique used by Greco (1996) and Malkawi et al. (2001) to characterize the critical slip surface. Goh (1999), McCombic and Wilkinson (2002), Das (2005), and Zolfaghari et al. (2005) represented the application of genetic algorithm for slope stability analysis in critical surface recognition [Adopted from 13, 14, 47, 66, 95, 103, 138, 167, 174, 182, 331].

Yarahmadi Bafghi and Verdel (2005) and Hack et al. (2003) was applied the probabilistic analytical method to identify the rock slope potential failure key-group and estimate the probability of failure. It must be attended that rock slope stability evaluations via the Slope Stability Probability Classification that proposed by Hack et al. (2003) does not require cohesion and friction angle factors as input for assessment. Furthermore, Wang et al. (2000) and Hack et al. (2007) predicted the failure risk of rock slopes and investigated the influence of earthquakes on rock slope stability through reliability analysis [Adopted from 107, 109, 166, 306, 325].

2.2. Empirical modeling

Analytical, empirical, and numerical are three design strategies in rock engineering. Empirical methods (i.e. rock mass classification,) are widely applied for feasibility and pre-design studies, also mostly used for the final design [Adopted from 36, 106, 225, 226, 238, 268, 269].

2.2.1. Rock Classification Systems

Rock mass classification systems applied in various engineering design and stability analysis projects. These systems created based on relations between rock mass parameters and engineering applications, such as tunnels, slopes, foundations, and other excavations empirically. The marcher step gaits in rock mass classification as the first system in geotechnical engineering field for steel set support of tunnels that suggested in 1946 [Adopted from 36, 106, 225, 226, 238, 268, 269].

In order to rock mass complexity, many researchers try to involve rock mass parameters in rock slopes designs and generate specific relation between them. Many empirical methods (rock mass classification systems) adjusted and generally these are using in primary designs techniques and sometimes use in final design such as: RMR, GSI, SMR and CSMR

Some of these classification systems, basically developed for underground excavations, also have been used for slopes (e.g., Q and RMR system) or have been modified for slopes (e.g., the RMS, SMR, SRMR and CSMR systems comprise modifications of the RMR system) [Adopted from 226].

In the recent several decades, rock mass classification systems have been propounded in accordance to cuttings with high risk of failure are recognized, also in this status preventative measures prioritized effectively [Adopted from 226].

Classification systems have played an indispensable role in engineering for centuries [29, 36]. The rock mass classification today forms an essential part of the most common design approach, the empirical design methods.

However, modern rock classifications have been never intended as the ultimate solution to design problems, but only a means towards this end [Adopted from 29, 36, 279].

Field engineers through the years have been attempting to describe the ground condition using the rock or rock mass properties such as petrologic descriptions, general rock type, or one or a few of the physiomechanical (physico-mechanical) properties. As a result, several methods have come into usage describing the same rock in different ways. Most of the earlier systems were "intact rock classifications", that is, systems based on laboratory properties determined on a sample of rock. On the other hand, "rock mass classifications" consider discontinuities and large-scale ground features [279]. The rock mass classifications main objectives (After Bieniawski 1989) categorized to [Adopted from 279]:

1. Find the most significative parameters which affecting on rock mass behavior [Adopted from 279].
2. Divide a particular rock mass formation into groups of similar behavior, that is, rock mass classes of varying quality [279].
3. Provide a basis for understanding the characteristics of each rock mass class [279].
4. Relate the experience of rock conditions at one site to the conditions and experience encountered at others [279].
5. Derive quantitative data and guidelines for engineering design [279].
6. Provide common fundament to communicate between engineers and geologists [Adopted from 279].

In addition, the rock mass classifications' principal advantages are as follows [Adopted from 36, 106, 225, 226, 238, 268, 269]:

- Making better the qualitative conditions of site investigations by using the least input information as classification factors.
- Preparing quantitative data for design aims.
- Enabling better engineering judgement
- Providing more efficient communicating in engineering projects.

Rock mass classification is a tool to deliberate the performance of rock cut slopes based on the most important inherent, fundamental and structural parameters [Adopted from 101]. The classification systems must be providing flexible consistent means to describing the rock mass conditions qualitatively and quantitatively even under uncertainty conditions. A detailed list of the existing developed empirical rock mass classification methods presented in Table 1.

Deere et al. (1967) developed the Rock Quality Designation index (RQD) [Adopted from 72, 304].

Bieniawski presented a rock mass classification system for stability assessment of cuttings in 1974. In this work, a ''rough'' adjustment factor which represents discontinuity orientation was added to the basic Rock Mass Rating (RMR) system's five pre-existing parameters to make it appropriate for slope stability problems. Since then, a number of rock mass classification systems have been proposed [Adopted from 30, 36, 226].

RMR had deficiencies in assessment of very poor quality rock masses. Stille et al. (1982) offered a modification of the RMR system that named as the rock mass strength (RMS) classification. Romana (1985) was introduced Slope Mass Rating (SMR) to assess slopes' behaviour. Hoek et al. (1995) was reported the geological strength index (GSI) as a complement of their generalized rock failure criterion. Hoek et al. (2002) was implemented several minor revisions for the original GSI [Adopted from 119, 120, 239, 288, 304].

In addition, Varnes proposed landslide classification that modified by Cruden and Varnes has been adopted [Adopted from 64, 65, 226, 299, 300].

Rock mass classification systems use particularly factors relevant to the condition of cuttings. Hence, further comparison with any rating systems of rock fall hazard or risk assessment system would not be appropriate, as the first consist of both hazard and consequence factors and the second estimates risk for failure instead of hazards for failure [226]. Substantial caution can exercise in using RMi classification system (same as others) to all rock-engineering problems such as slope stability problems accordance to above review.

Table 1: Detailed list of the existing developed empirical rock mass classification methods - modified and augmented [Adopted from 226 and modified by 60, 105, 106, 223, 224, 235, 249].

Name of the system	Abbreviation	Authors	Application	Comments	References
-	-	Ritter	Tunnels	The preliminary endeavor to formalize an empirical approach for the design of tunnel (1879).	Pantelidis (2009); Ritter (1879); Grasselli and Egger (2003).
Rock load	-	Karl Von Terzaghi	Tunnels (Tunnels with steel Support)	The rudimental resource to the application of rock mass classification to tunnel support design (1946).	Adopted from Pantelidis (2009); Terzaghi (1946); Kulhawy and Duncan (1972); Hack (1998).
Stand-up time	-	Lauffer	Tunnels	Linked to an unsupported tunnel excavation's stand-up time (1958), Applications in tunneling design.	Adopted from Pantelidis (2009); Lauffer (1958); Pantelidis (2009); Pantelidis (2009); Pantelidis (2009); Hack (1998).
Rock Quality Designation	RQD	De Deere et al.	General , core logging, tunneling	Incorporator parameter in many classification systems (1967, 1988, 1989).	Adopted from Pantelidis (2009); Pantelidis (2009); Deere (1963); Deere (1964); Deere (1989); Deere DU and Deere DW (1988); Deere, Hendron, Patton and Cording (1967); Deere and Miller (1967); Pantelidis (2009); Pantelidis (2009); Hack (1998).
New Austrian Tunneling Method	NATM	Rabcewicz, Muller and Pacher, Rabcewicz and Golser, Pacher et al, Pacher, Muller, Kovari	Tunnels	The system solely designed for tunneling (Rabcewicz, 1964/1965, 1972; Muller and Pacher 1964; Pacher et al., 1974; Muller, 1978; Kovari, 1993). (Rabcewicz and Golser 1973; Pacher, 1975), Uses in excavation and design of incompetent (overstressed) ground.	Hack (1998); Hack (1998); Hack (1998); Kovari (1993); Müller (1978a); Müller (1990); Muller and Pacher (1964); Pacher (1975); Pacher, Rabcewicz and Golser (1974); Rabcewicz (1964); Rabcewicz and Golser (1973); Hack (1998); Hack (1998).
Rock classification for rock mechanical purposes	-	Patching and Coates	General	Application as input in rock mechanics. Descriptive (1968).	Patching and Coates (1968).
The unified classification of rocks(and soils)	-	Deere et al.	General	Descriptive. According to particles and blocks for communication (1969).	Deere, Merritt and Coon (1969).
Rock Structure Rating (Rock Structure concept)	RSR	Wickham et al.	Small tunnels	First rock masses rating system Developed by Wickham, Tiedemann and Skinner in the 1970s (1972,	Adopted from Pantelidis (2009); Wickham, Tiedemann and

				1974).	Skinner (1972); Wickham, Tiedemann and Skinner (1974); Hack (1998).
Size Strength Classification for rock masses (Strength - Block size)	-	Franklin	Tunnel stability, Mining	Development of a slope stability classification system for tunnel stability and potential failure mechanisms assessment (Franklin 1986). According to rock strength and block diameter. (Franklin, 1970; Cottiss, Dowel and Franklin 1971; Franklin, Broch and Walton 1971; Franklin, Louis and Masure 1974; Franklin 1975; Franklin 1976; Franklin, J. A., 1986).	Hack (1998); Cottiss, Dowel and Franklin (1971a); Cottiss, Dowel and Franklin (1971b); Franklin (1970); Franklin (1975); Franklin (1976); Franklin (1986); Franklin, Broch and Walton (1971); Franklin, Louis and Masure (1974).
Rock Mass Rating	RMR	Bieniawski	Tunnels, cuttings, Mines, Slopes and Foundations Design	A raw rating regulation, modification and correction for discontinuity orientation to apply in slopes that was added in the 1979 version of the RMR system (1973, 1974, 1976, 1984, 1988, 1989), Last modification (1989).	Adopted from Pantelidis (2009); Bieniawski (1973); Bieniawski (1974); Bieniawski (1984); Bieniawski (1976a); Bierntowski (1976b); Bieniawski (1979); Bieniawski (1988); Bieniawski (1989); Hack (1998).
Rock Tunneling Quality Index (Rock Mass Quality)	Q	Barton et al.	Tunnels, mines, foundations, Chambers	They are the most commonly used classification systems for tunnels (1974, 1976a, 1980, 1988, 1999, 2002). Last modification 2002. Use in design of support in underground excavations.	Pantelidis (2009); Barton (1976); Barton (1988); Barton (2002); Barton and Grimstad (1994); Barton (1999); Barton and Grimstad (2004); Barton, Lien and Lunde (1974a); Barton, Lien and Lunde (1974b); Barton, Loset, Lien and Lunde (1980); Hack (1998).
RMR system extension	-	Weaver	Rippability	(1975).	Weaver (1975).
RMR system extension	-	Laubscher	Hard Rock Mining	(1976).	Laubscher and Taylor (1976).
Mining Rock Mass Rating	MRMR	Laubscher	Mines	Based on RMR (1973), (1977, 1981, 1984, 1990).	Pantelidis (2009); Laubscher (1977a); Laubscher (1977b); Laubscher, (1984); Laubscher (1990); Laubscher and Page (1990); Pantelidis (2009); Hack (1998).
The typological classification	-	Matula and Holzer	General	Descriptive, (1978).	Matula and Holzer (1978).
Terrain Index (Slope Stability	-	Vecchia	Stability of hillsides and	A simple terrain index for the stability assessment of	Hack (1998); Vecchia (1978).

System of Vecchia			scarps - Natural slopes	hillsides and scarps. A classification system designed to quantify the stability of a hillside and scarp, e.g. natural slopes (1978).	
RMR system extension	-	Olivier	Weatherability	(1979).	Olivier (1979).
Rock Mass Strength (Geomorphic Rock Mass Strength Classification)	RMS	Selby, Moon and Selby	Cuttings	In accordance to natural slope database (1980, 1982), (1990).	Pantelidis (2009); Hack (1998); Moon and Selby (1990); Selby (1980a); Selby (1980b); Selby (1982a); Selby (1982b); Selby (1987).
Unified Rock Mass Classification System (Unified classification)	URCS	Williamson	General	The input to the system mainly is based on descriptions. Main applications in communication, (1980, 1984, 1988).	Williamson (1980); Williamson (1984); Williamson and Kuhn (1988).
Basic geotechnical classification	BGD	Brown	General	Use for general applications, (1981).	Brown (1981).
RMR system extension	-	Ghose and Raju	Coal mining	(1981).	Ghose and Raju (1981).
RMR system extension	-	Moreno Tallon	Tunneling	(1982).	Moreno and Talion (1982).
Rock Mass Strength	RMS	Stille et al.	Metal mining	(1982).	Stille, Groth and Fredriksson (1982).
Q-system development	Q	Kirsten	Excavatability,	Applications as Excavatability (1982).	Kirsten (1982).
Q-system development	Q	Kirsten	Tunneling	Applications as Tunneling (1983).	Kirsten (1983)
RMR system extension	-	Kendorski and Cummings	Hard Rock Mining	(1983).	Kendorski, Cummings, Bieniawski and Skinner (1983).
RMR system extension	-	Nakao et al.	Tunneling	(1983).	Nakao, Iihoshi and Koyama (1983).
RMR system extension	-	Serafim and Pereira	Foundations	(1983).	Serafim and Pereira (1983).
RMR system extension	-	Gonzalez de Vallego	Tunneling	(1983, 1985).	Gonzalez de Vallejo (1983); González de Vallejo (1985).
RMR system extension	-	Ünal	Coal mine roof bolting	(1983).	Unal (1983).
Slope Mass Rating	SMR	Romana, Romana et al.	Cuttings	Based on RMR (1979). The utmost widespread classification system that used for slopes (1985, 1991, 1993, 1995, 2003).	Adopted from Pantelidis (2009); Romana (1985); Romana (1991); Romana (1993); Romana (1995a); Romana (1995b); Romana, Seron and Montalar (2003); Hack (1998).
RMR system extension	-	Newman	Coal mining	(1985, 1986).	Newman (1985); Newman (1985);

					Newman and Bieniawski (1986).
RMR system extension	-	Sandbak	Boreability	(1985).	Sandbak (1985).
RMR system extension	-	Smith	Dredgeability	(1986).	Smith (1986).
RMR system extension	-	Venkateswarlu	Coal mining	(1986).	Venkateswarlu (1986).
Slope Rock Mass Rating	SRMR	Robertson	Cuttings, Slope Stability	Based on RMR. The classification provided for weak altered rock mass materials from drill-hole cores (1988).	Pantelidis (2009); Robertson (1988); Hack (1998).
Communication Weakening Coefficient System	WCS	Singh	Coal mining	(1986, 1987, 1988, 1989).	Brown, Denby and Singh (1988); Brown and Singh (1987); Singh, Brown, Denby and Croghan (1986); Singh, Denby and Brown (1985); Singh and Gahrooee (1989); Singh, Reed and Hughes (1987).
Rockfall Hazard Rating System	RHRS	Developed by Pierson, Davis, and Van Vickle	Rock fall hazard assessment of slopes	One of the most widely recognized, accepted and used methods for assessment of slopes' rockfall hazards such as sides of highways (1990).	Adopted from Pierson, Davis and Van Vickle (1990); Hack (2002); Federal Highways Administration (1993).
Haines System (Slope Stability System of Haines - Modified Laubscher)	-	Haines & Terbrugge	Rock slopes stability in open pits	A rock mass classification system for introductory estimation of the stability of rock slopes (1991).	Adopted from Hack (1998); Hains and Terbrugge (1991).
Rock Engineering Systems	RES	Hudson	Natural slopes instability assessment	Rock mass characterization applied to assess natural slopes instability assessment (1992).	Hack (1998); Hudson (1992).
Coal Mine Roof Rating	CMRR	Molinda and Mark	Coal mining	Based on RMR format (1994).	Molinda and Mark (1994).
Natural Slope Methodology	NSM	Shuk	Natural slopes stability	Statistical analysis of existing natural slopes to predict rock mass and soil parameters, and the probability of slopes stability (1994a, 1994b, 1994c, 1994d).	Hack (1998); Shuk (1994).
Chinese Slope Mass Rating	CSMR	Chen	Cuttings	Adjustment parameters have been used to the SMR system for the discontinuity condition and slope height (1995).	Adopted from Chen (1995); Pantelidis (2009).
The Rock Mass index system	RMi	Palmström	General, Design of support, TBM progress, Rock engineering, Characterization	The numerical input parameters are ratings according to their character. Application in general characterization, design of support, TBM progress,	Palmstrom (1995).

				(1995).	
Modified Rock Mass Rating	M-RMR	UnaI	Mines, Weak rock, coal	For weak, stratified, anisotropic and clay bearing rock masses, (1996).	Pantelidis (2009); Ünal (1996).
-	-	Mazzoccola and Hudson	Natural slopes	An example for determining natural slopes instability following the rock engineering system methodology; A rock mass characterization method to show instabilities of natural slopes (1996).	Adopted from Mazzoccola and Hudson (1996); Pantelidis (2009); Hack (1998).
Geological Strength Index	GSI	Hoek, Hoek, Kaiser and Bawden, Hoek and Brown	General	Based on RMR (1976). Data for correlation with modulus of deformation (1997). Hoek 1994, Hoek, Kaiser and Bawden 1995, Hoek and Brown 1997.	Hoek (1994); Hoek and Brown (1997); Hoek, Kaiser and Bawden (1995); Pantelidis (2009).
Rock slope Deterioration Assessment	RDA	Nicholson and Hencher, Nicholson et al., Nicholson	Cuttings	For shallow, weathering-related breakdown of excavated rock slopes. (1997, 2000, 2002, 2003, 2004).	Nicholson (2000); Nicholson (2002); Nicholson (2003); Nicholson (2003); Nicholson (2004); Nicholson and Hencher (1997); Nicholson, Lumsden and Hencher (2000); Pantelidis (2009).
Geological Strength Index	GSI	Hoek et al., Marinos and Hoek, Marinos et al.	General	Use for non-structurally controlled failures. For all underground excavations, 1998, 2000, 2001, 2005. Application as design of support in underground excavations. Rock mass characterization. Hoek, Marinos and Benissi, 1998, Marinos and Hoek, 2000, 2001. Marinos V., Marinos P. and Hoek 2005.	Hoek, Marinos and Benissi (1998); Hoek, Read and Karzulovic (2000); Marinos and Hoek (2000); Marinos and Hoek (2001); Marinos V., Marinos P. and Hoek (2005); Pantelidis (2009).
Slope Stability Probability Classification	SSPC	Hack, Hack et al.	Cuttings	Probabilistic evaluation of independently various mechanics of failure, (1998, 2003).	Adopted from Pantelidis (2009); Hack (1998); Hack and Price (1993); Hack, Price and Rengers (2003).
Index of rock mass basic quality	BQ	Lin	-	(1998)	Lin (1998).
Q-system development	Q	Barton	TBM tunneling	(1999, 2000)	Barton (2002); Barton (1999); Barton and Grimstad (2004).
Volcanic Rock Face Safety Rating	VRFSR	Singh and Connolly	Cuttings [temporary excavations]	For volcanic rock slopes to define the safety of excavations on construction sites (2003).	Adopted from Pantelidis (2009); Singh and Connolly (2003).
Falling Rock Hazard Index	FRHI[a]	Singh	Cuttings [temporary excavations]	Enlarged for stable excavations to distinguish the dangers degree for	Adopted from Pantelidis (2009); Singh (2004).

				workers (2004).	
Deutsche Steinkohle	-	Witthaus	Coal mining	(2006).	Witthaus (2006).
Rock Mass Excavability	RME	Bieniawski et al.	TBM tunneling	(2006).	Bieniawski, Celada, Galera Fernández and Hernández Álvarez (2006).
RMR system extension	-	Pakalnis et al.	Weak rock mining	(2007).	Pakalnis, Brady, Hughes, Caceres, Ouchi and MacLaughlin (2007).
Slope Stability Rating classification system	SSR	Taheri and Tani	Rock slopes sites	A rock mass classification system for anticipatory slope stability assessment (2010).	Adopted from Rai; Rai, Taheri and Tani (2010); Taheri and Tani (2007).
Rock Slope Rating	RSR	-	Rock slopes stability	A system that evaluates the probability of failures for plane, wedge sliding, toppling and circular failures.	Rai.
Mine Dump Slope Classification (Dump Mass Rating or Dump Slope Rating)	DSR	Rahul et al.	Coal mine waste dump slope	A dump classification system for coal mine waste dump slope (2010, 2011, 2017).	Rai; Rahul (2011); Sharma and Rai (2017).

[a] Based on Singh, prior to the use of FRHI, it is essential to proceed for rock slope stability assessment using the previously suggested VRFSR system.

3. Limitations of Current Slope Stability Design Approaches

Empirical methods (models and formulas) of analysis such as rock mass classification; conventional methods of analysis such as basic analysis (Lorimer's method), stereo-graphic kinetic analysis and kinematic analysis tools, limit equilibrium (Swedish Circle (ø = 0) method; logarithmic spiral method; friction circle method; method of slices: such as ordinary method of slices, simplified bishop method, Janbu’s simplified method, Janbu’s generalized procedure of slices (GPS), Spencer’s method, Morgenstern and Price’s method, Sarma’s method), limit analysis, rockfall simulators; numerical methods of analysis such as continuum modeling; the dis-continuum modeling; hybrid/coupled modeling; statistical & probabilistic methods; Observational and analytical techniques and tools; new methods such as intelligent models, fuzzy logics, artificial neural network, genetic algorithm, slope stability radar and laser systems such as the 3-D laser mapping; the 3-D terrestrial laser scanning & ground penetrating radar, etc. are various slope stability analysis design approaches and modelling methods of slope behaviors.

It is very consequential which designers perceive and comprehend the assumptions and limitations before applying a design method and approach. The relative deserves and deficiencies of the currently existing design methods are summarized as follows [Adopted from 166]:

(1) Limit equilibrium methods (LEM) include the drawbacks, which are the assumptions of (i) The soil or rock masses behaving and acting as a rigid material and (ii) the shear strength being mobilized at the same time along the entire failure surface.

(2) The true failure load can bound by using both of the upper and lower bound limit analysis. However, in this condition the displacement of the slope cannot predict.

(3) Numerical modeling in rock mechanics analyses via standard software does not contemplate fracture propagation via intact material. Also, new developments in this field have not yet reached full maturity for practical applications in slope design. Sjöberg (1999) found that it was not possible to simulate smaller block sizes, as the models were very (computer) memory consuming and took a long time to run. His study also indicated that a reduced block or element size alone might not be sufficient to increase the ability of the rock mass to fracture [263].

(4) Empirical design charts only provide general design advice and process too. Because of limited data, the establishment of more detailed design rules is not possible.

(5) Although physical model tests can be useful for determining fundamental failure mechanisms and for the verification of analytical and numerical methods, they are not a true design method for simulating the correct loading conditions and modeling rock mass properties accurately. Centrifuge testing of rocks also requires somewhat larger

model dimensions, compared to soil testing in order to include discontinuities in the model. Larger model dimensions require a centrifuge that besides a high acceleration, also can handle a large mass. Unfortunately, these two objectives do not easily meet simultaneously.

(6) Probabilistic methods necessitate very extensive numbers of input values and suppositions regarding the distribution functions. Furthermore, probabilistic design methods worked often based on the limit equilibrium methods. Thus, these are exposing to the same limitations as limit equilibrium methods (LEM).

Integrating costs into design methods through probabilistic methods can be some extent to accomplish. Nevertheless, the large amount of required input data has difficultly rendered the use of these cost-benefit-methods in practical applications [Adopted from 166].

4. Study Area: Sungun Copper Mine

4.1. General characteristics of Sungun copper mine

Sungun copper deposit is located in east Azarbaijan province of Iran in mountainous area and 75 km North West of the provincial town of Ahar at Latitude 38° 41' 34" north and Longitude 46° 41' 54" east (Fig. 1) [281]. Mine linked to Tabriz city via approximately a 125-km road. The ore body is situated straightly and nearly west of the Sungun River [Adopted from 281] a deeply incised valley in mountainous country with topography locally ranging from 1700 meters to 2450 meters. This deposit is in the middle of Qarabagh Mountains that highest altitude of the area from open sea is about 2390 meters [281]. Pakhir and Sungun Rivers are streaming through the mine's area after joining to Mian-cafe River which they make Ilgene-chai River. Climate condition in winter is cold. In the summer, the weather status is moderate. The area is compactly covered by plants and jungle as a result of its humidity. Maximum temperature in summer is 33° and minimum is -16 ° in winter [Adopted from 186].

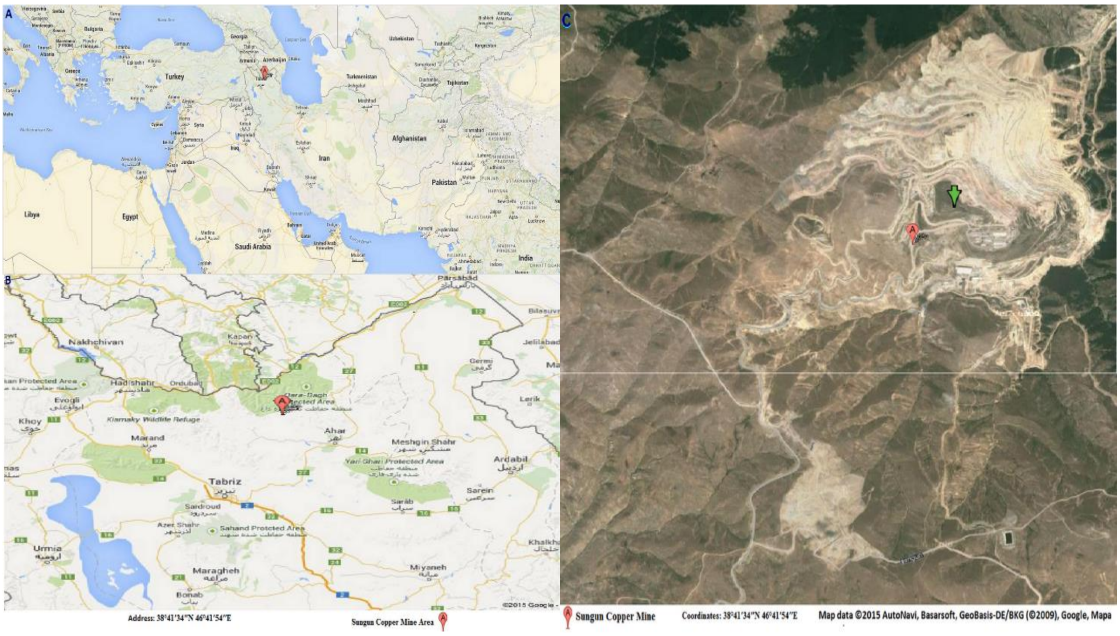


Figure 1. Location of Sungun copper mine: map and satellite view, A. Overall map view, B. Detailed map view, C. Detailed satellite view [175].

Historical evidence and documents confirm long-term and ancient mining traditional activities in this area. NICICO Company (Taskmaster is National Iranian Copper Industries Company) in Sungun Copper Project.) is operating the Sungun Copper mine. That is a world-class project of great magnitude and complexity. The Sungun copper porphyry deposit would exploit through this open pit mine which will produce in four or five pushbacks. This roughly semi-circular pit has high walls on three sides and a low wall to the east in the Sungun Valley. The ultimate open pit's initial design indicated a maximum slope height of 765 meters. The interim pit approximately has a maximum height of 675 meters. Therefore, this pit categorized to be a high to a very high slope and considered too. Height of the bench is about 12.5 meters, and width of main road is 30 meters; safety bench width is 11 meters, bench slope angle is 70° and final pit slope angle is 37° that design for different walls [Adopted from 1, 111, 281, 292].

4.2. Geological settings of Sungun copper mine

The Azarbaijan application area is part of the comprehensive and universal copper belt of ALP-HIMALIA and the orebody surrounding region placed in the Alpine-Himalayan metallogenic belt. Location of the project in major porphyry copper-gold occurrences belt in Iran shown in Fig. 2. Sub-volcanic rocks have intruded into cretaceous limestone and andesite volcanics. Mineralization in monzonite and volcanic host rocks is disseminated, and stock works in nature. Characteristic vertical zoning of porphyry deposits as Leached, Supergene, Hypogene and Skarn zones have been recognized as being significant in regard to grade. Three ages of dykes exist and injected into the ore-body that considered waste [Adopted from 82, 111, 292].

In briefly, Sungun copper ore deposit is Porphyry that encompass via dikes. Mine's area in the lithological approach divided in to Sungun Porphyry ore body, dyke A, dyke B and skarn. Type of rock in porphyry ore body is Monzonite up to Quartz-Monzonite. Rock type in dyke A is Monzonite and in dyke B is Diorite. Fig. 3 presents a snap shot of the 3-dimensional dyke model. Green represents type 1A dykes (DK1A), whereas brown indicates DK1B and DK1C dykes. There are kinds of alteration like Phyllic (sericite, kaolinite, quartz and pyrite), Potassic (potassic feldspar, quartz and secondary biotite), Propylitic (epidote, chlorite and calcite), Argillic (kaolinite, clays, sericite) and Silicic (mainly fine-grained quartz) in the mine's area [Adopted from 111]. Mine's area divided in to six blocks that Situation of these blocks of Sungun copper mine presents in Fig. 4. In addition, general features of block five of Sungun mine shown in Table 2.

Block five of Sungun copper mine is selected for this purpose because only is suitable for fully drained conditions and other blocks face with slightly, partially or whole considerable water problems (partially drained, un-drained conditions) which a set of water influence factor should be contemplated in study formulas. Other conditions will consider in other articles. Table 3 shown the lithological log summary of bore hole 15 (GT15) in the section RS06 of block five of Sungun copper mine.

Table 2: General features of block five of Sungun copper mine.

Block number	Lithology	Average of rock mass Compressive strength (MPa)	Average of rock mass tensile strength (MPa)	Average of rock mass young modulus (MPa)	Average of cohesion (kPa)	Internal friction angle(°)
5	Monzonite and Diorite Dykes	4.184	0.007	1683	220-240	33-35

Table 3: Summary of lithological log of bore hole15 (GT15) in section RS06 of block five of Sungun copper project.

LITHOLOGY			ALTERATION			MINERALIZATION			ZONE		
FROM	TO	TYPE	FROM	TO	TYPE	FROM	TO	TYPE	FROM	TO	TYPE
0.00	1.60	CORE LOSS	0.00	1.60	-	0.00	1.60	-	0.00	1.60	-
1.60	5.50	SP	1.60	30.90	PHY	1.60	43.10	PYY-FEX-MNX	1.60	5.50	LEA
5.50	6.10	DK1a	30.90	41.20	-	43.10	52.40	PYY-FEX-CHA	5.50	6.10	DY
6.10	30.90	SP	41.20	43.10	PHY	52.40	78.00	PYY-FEX	6.10	12.00	OXI
30.90	41.20	Ex-Ep	43.10	52.40	-	78.00	122.00	PYY-FEX-CHA	12.00	30.90	LEA
41.20	43.10	SP	52.40	162.00	PHY	122.00	144.00	PYY-FEX-CHA-CPY-BOR-MDL	30.90	41.20	-
43.10	52.40	Ex-Ep	162.00	192.70	POT	144.00	192.70	PYY-CPY-MDL	41.20	43.10	LEA
52.40	192.70	SP	192.70	194.65	PHY	192.70	194.65	-	43.10	52.40	-
192.70	194.65	DK1b	194.65	198.95	POT	194.65	198.95	PYY-CPY-MDL	52.40	88.00	LEA
194.65	198.95	SP	198.95	203.35	PHY	198.95	203.35	-	88.00	122.00	SUP
198.95	203.35	DK1b	203.35	300.00	POT	203.35	210.00	PYY-CPY-MDL-BOR	122.00	142.00	SUP-HYP
203.35	300.00	SP				210.00	300.00	PYY-CPY-MDL	142.00	192.00	HYP
									192.00	194.00	DY
									194.00	198.95	HYP
									198.95	203.35	DY
									203.35	300.00	HYP

Note 1. TYPE OF LITHOLOGY- SP: Sungun Porphyry (Monzonite to Quartz Monzonite); DK1a: Dyke 1a (Diorite Porphyry, Very altered, Similar to Sungun Porphyry); Ex-Ep: Exoscarn-Epidote Zone; DK1b: Dyke 1b (Diorite Porphyry, Altered, Argillic Alteration).

Note 2. TYPE OF ALTERATION- POT: Potassic Altered; PHY: Phyllic (quartz-sericite) altered.

Note 3. TYPE OF MINERALIZATION- PYY: Pyrite; FEX: Iron Oxide; MNX: Manganese Oxide; CHA: Chalcocite; CPY: Chalcopyrite; MDL: Molibdonite.

Note 4. TYPE OF ZONE- LEA: Leached; DY: Dyke; OXI: Oxide; SUP: Supergene; SUP-HYP: Supergene-Hypogene; HYP: Hypogene.

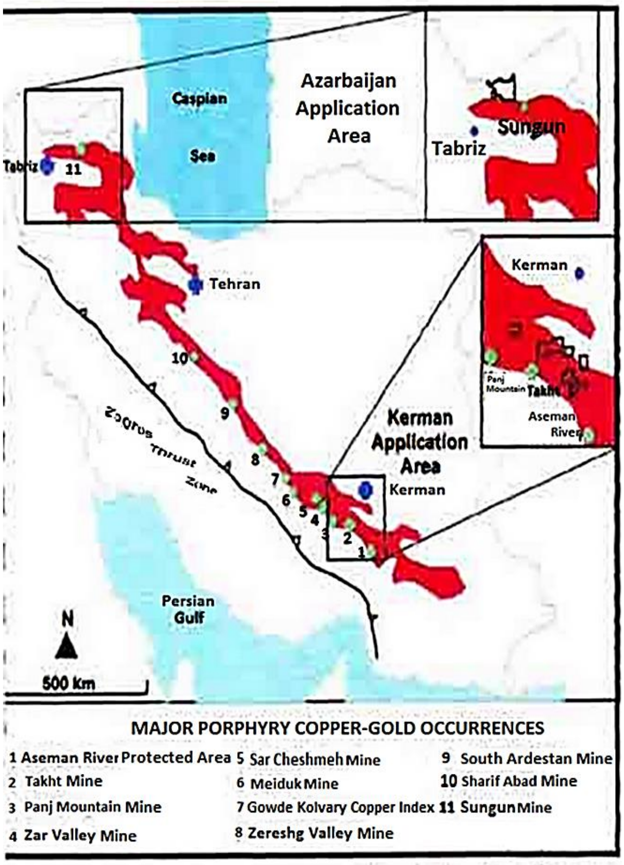


Figure 2. Location of project in major porphyry copper-gold occurrences belt in Iran – modified [199].

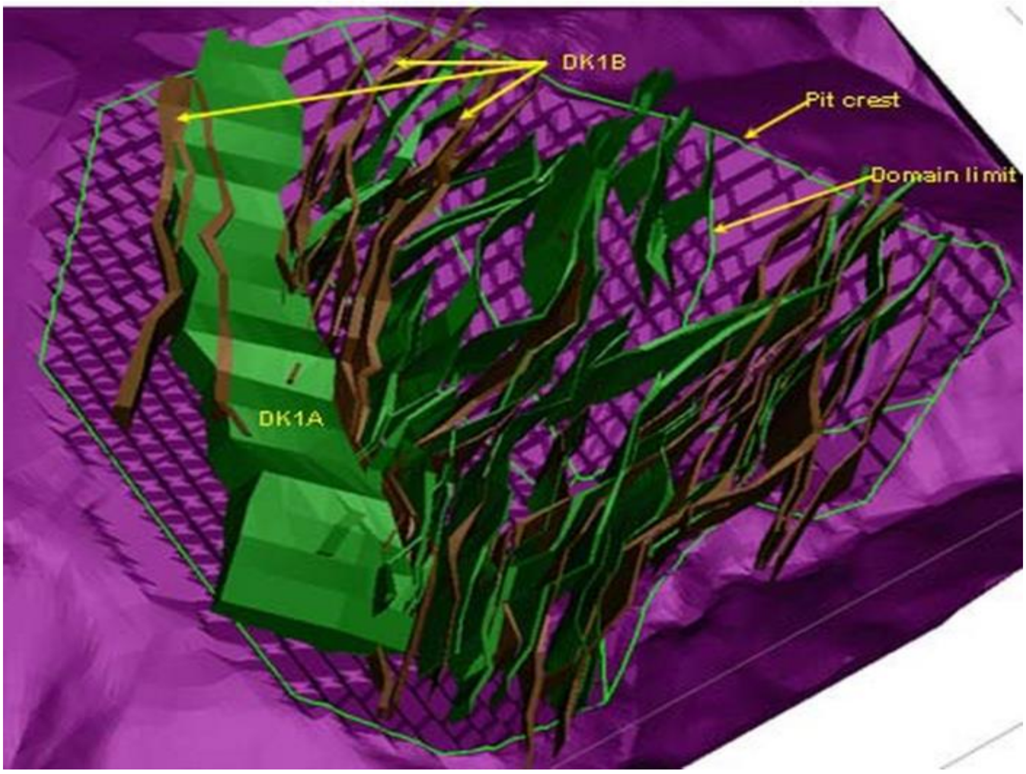


Figure 3. Three-dimensional lithological model of dykes (DK1A, DK1B and DK1C) in mine area looking northwest [281].

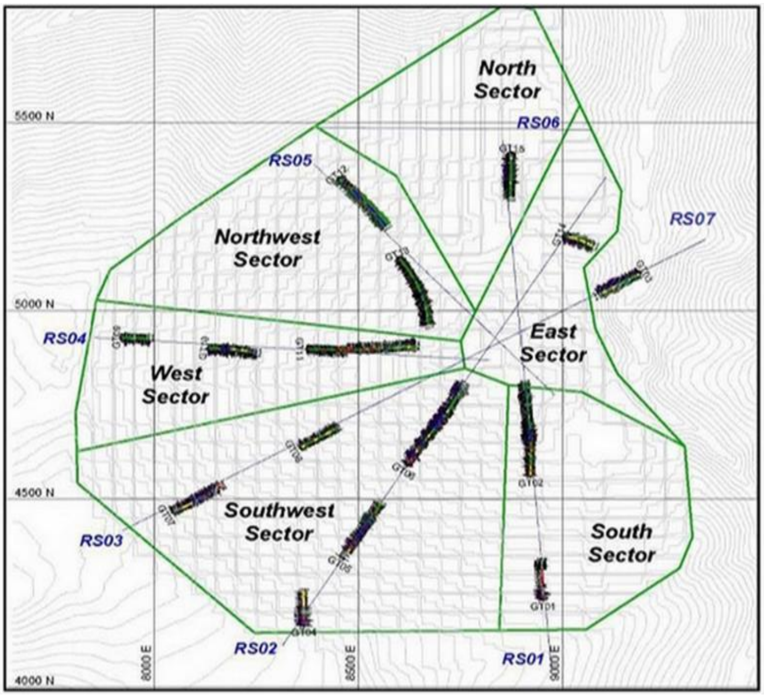


Figure 4. Domains, Pit Sectors and Cross Section Locations of Sungun Copper Mine [281].

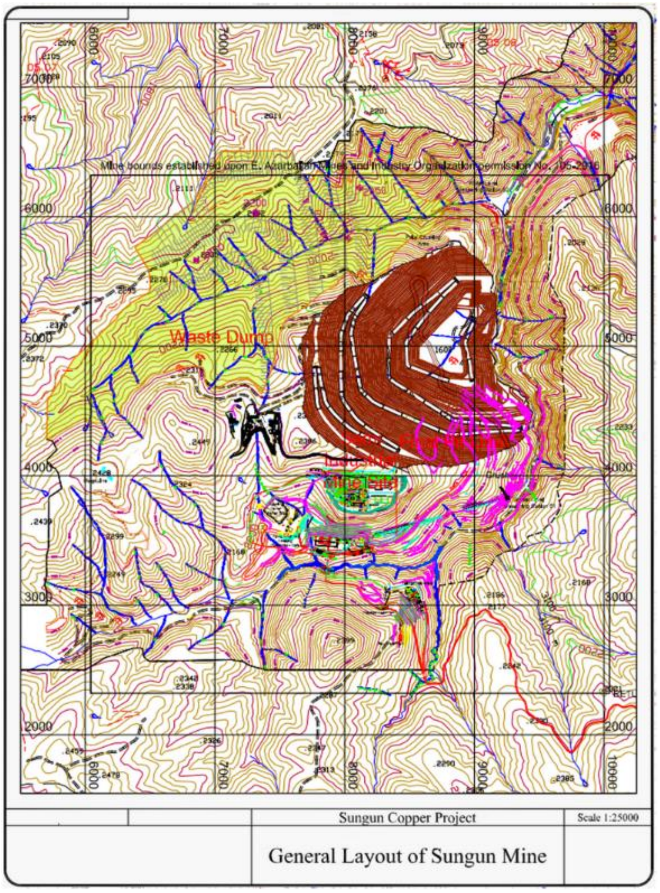


Figure 5. General layouts of Sungun copper mine [281].

General layout of Sungun copper mine shown in Fig. 5. Also, geological section of borehole 15 (GT15) in block five of Sungun copper mine presented in Fig. 6. LS in Fig. 6 means loss of core.

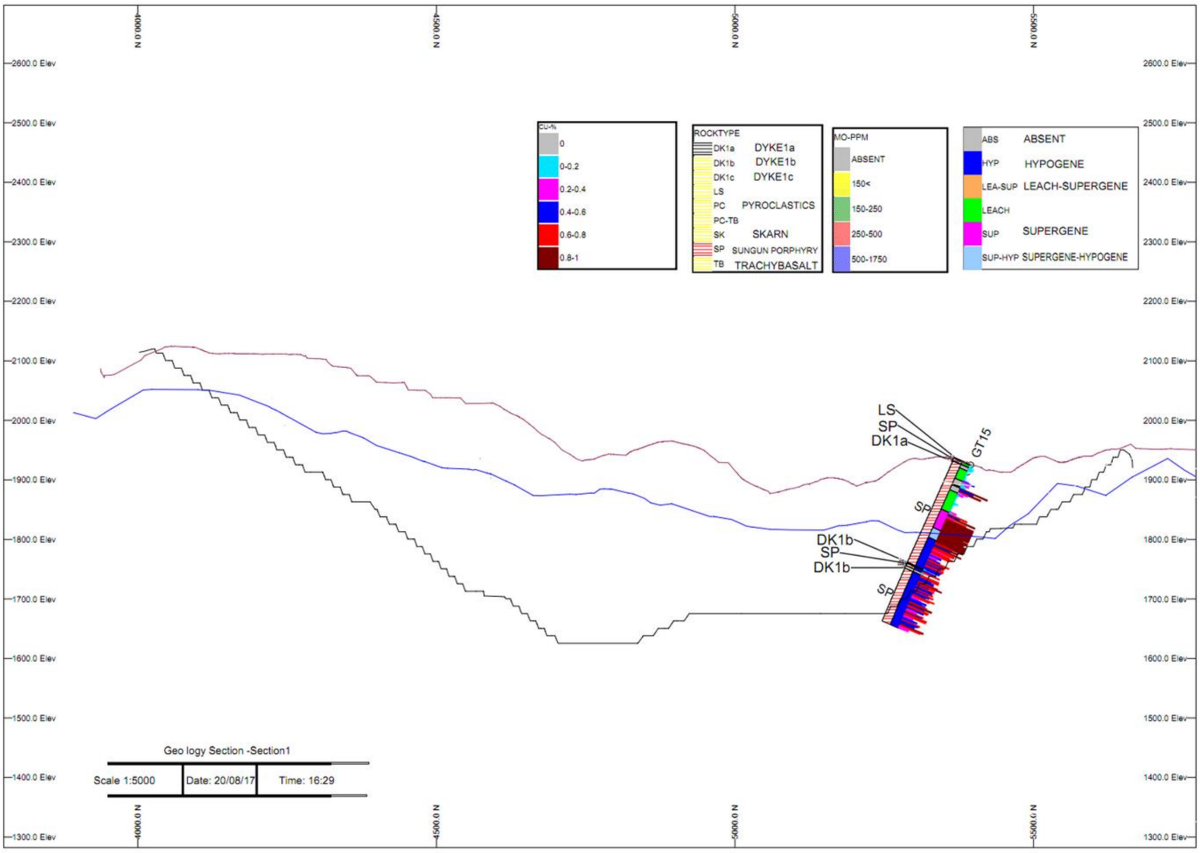


Figure 6. Geological section for borehole 15 (GT15) in block five of Sungun copper mine.

5. Basic Theory and Calculation: Strength Limit of Earth; a New Basis Parameter

5.1. Strength limit of earth's explanation

Earth's strength limit is a key basis parameter for earth perceptions and interpretations. Materials' cohesion between particles varies through different reasons like structural problems, tectonic actions, geological structures and engineering geology problems. If these values are less than about, 65 percent (exactly equal to 64.71215717224341917241044931966 1 %) of own first value therefore instability changes begin. In such condition, micro cracks create and develop. Then, finally different results of instability such as failures occur, e.g., in mines' slopes and on side of the roads. Also in luminaries or other structures. Strength limit of earth represented that how much value remain active from one MPa or one percent of any strength or set of all resisting forces. Strength limit is the cohesion value of different materials, particles, elements, factors and parameters on earth and the stability limit of earth and all-constitutive particles and elements of earth and earth's visible or invisible constitutive components in presence of all disturbance factors such as different stresses. This Actual Multi-Interaction Strength (A.M.I.S) in existence illustrates the equilibrium and equilibrium limit between resisting and disturbance forces acting on earth during universe stability. Cohesion limit number is constant ($X_{C-Constant}$ or X_{CCLN}) instantly and on the other hand is variable that called cohesion variable limit number ($X_{C-variable}$ or X_{CVLN}). When strength limit parameter decreases continuously and reach to $[(X_{C-Earth}) \times 100]$%, $\left([\frac{X_{c-Earth}+G.S.Ns}{2}] \times 100\right)$ % and (Golden section numbers$\times$ 100) % of theirs first own value then instabilities of structures like earth begin. The Golden section number is 0.6180339887. Thus, in critical stages' procedures strength limit parameter pass from $[(1\text{-}G.S.Ns) \times 100]$ % and $\left([1-\frac{X_{c-Earth}+G.S.Ns}{2}] \times 100\right)$ % of its primary value and ruptures begin in structures. Therefore, complete decomposing times and procedures occur during and after that strength limit parameter reduce yet and its value pass from $[(1\text{-}X_{C-Earth}) \times 100]$ % of preliminary value. Variations diagram of this interactional resistance realizes from nature and exactly from shape of the mountains. Strength limit can define for every structure and everything in various fields of science [Adopted from 246].

5.2. Definitions of strength limit of earth ($X_{C-Earth}$)

Scrutinizing researches about gravitation's effects, its relations and influences on other sciences are major works, which required for future of sciences perfection. One branch of this basis project should be carried out with respect to connect strength limit factor such as Earth's Strength Limit and rock strength especially strength of various rock(s) types like as intact, highly strength also jointed and altered rocks. Strength Limit of Earth is an identifier parameter, which linked other sciences to rock engineering and geology sciences also slope engineering too, to illustrate extensible detailed complicated relations of all sciences' parts, which acquit roles in the entire existence.

Determining the strength limit of earth parameter can organize, also adjusts and completes more theories. Therefore, this essential and efficient factor helps response to different unsolved questions in various fields of sciences.

5.2.1. Independent determination of strength limit by area density and standard gravity for earth

One of the principal basilar tensional forces that affect materials in the world is gravity. Gravitational effects are one of the regulator forces of all things in the universe. Gravity of earth denotes to space and time with fundamental parameters. Strength limit factor of earth is the strength of gravity forces also is the structural resistance of gravitational fields. Mainly, strength limit of earth indicates the equilibrium or equilibrium limit between resisting and disturbing forces in the universe gravitational field and plays considerable and indispensable key roles in cosmos universal stability field that is the reason to applying this factor in slope stability engineering analysis. In addition, Napier's or Euler's constant is a key number in universe explanation. Therefore, gravity and Napier's constant can use in definitions of earth's strength limit. Strength limit number of the earth also is the cohesion constant limit number of the earth that characterized through standard gravity and Euler's constant by below equation [Adopted from 246]:

$$X_{C-Constant} = X_{C-earth} = \frac{g_0}{\mathrm{e}^{\mathrm{e}}} = g_0 \times \frac{1}{\exp.^{\mathrm{e}}} = gravity \times density = 0.64712157172243419172410449319661 \text{ (MPa or \%)} \quad (1)$$

Where $\boldsymbol{g_0}$ is the standard gravitational acceleration or Earth's standard surface gravity that's equal to 9.80665 m/s^2 [284, 294], e is the simply Euler's constant or Napier's constant, $\boldsymbol{X_{C-earth}}$ [Adopted from 246] is the cohesion constant limit number of earth or strength limit parameter or strength of existence global gravitation (in MPa), e is Napier's number, $\boldsymbol{\frac{1}{e^e}}$ is area density (A.D or D_A) or surface density (S.D or D_S) and equal to 0.06598803584531253707679018759685 Ggr/m^2 65.98803584531253707679018759685 ton/m^2.

$\boldsymbol{e^e}$ is equal to 15.15426224147926418976043027263 m^2/Ggr which means every value of gravitation is effect on e^e square meter of every part of earth or everything near the earth with one Ggr mass. In-fact e^e is the impact surface area of gravity on one Ggr mass. Surface density means every value of gravitation is effects on approximately 0.066 Ggr in one square meter of every part of earth or everything near the earth. In fact, area density is 65.98803584531253707679018759685 ton in one square meter also equal to 1000 tons (1 Ggr) in 15.15426224147926418976043027263 square meter (m^2). Then, area density defined by [Adopted from 246]:

$$A.D = Area\ Density = S.D = Surface\ Density = \frac{1}{\mathrm{e}^{\mathrm{e}}} = \frac{1}{\sqrt{\mathrm{e}^{\mathrm{e}}} \times \sqrt{\mathrm{e}^{\mathrm{e}}}} = \frac{X_{C-earth}}{g_0} = \frac{Strength\ Limit\ of\ Earth}{Standard\ Gravity}. \quad (2)$$

Mathematical and numerical roots of strength limit of earth have different meanings. Eq. (1) represent that strength limit of earth approximately is equal to 0.65 (MPa or %) from every 1 (MPa or %) unit scale of any strength forces or set of all kinds of resisting forces. However, this parameter approximately is equal to 64.7122 (MPa or %) in every 100 (MPa or %) unit scale(s) of any strength forces or set of all kinds of resisting forces. In other words, earth's strength limit is equal to 0.647122 MPa or 64.7122 % approximately from any strength forces or set of all resisting forces [Adopted from 246].

Why 32 decimal digits used for some key numbers? "Because" an extremely small discrepancy in key numbers causes making great differences in features of various sciences [Adopted from 247]**.**

5.2.2. Specification of strength limit by Newton's second law of motion and Newton's law of universal gravitation and standard gravity

Briefly, Definition of Newton's Second Law is the involved net force F of a body is directly equal to the scalar multiplication of the mass m and the acceleration ("a") of mentioned body [Adopted from 204, 205].

According to Newton's second law of motion and universal gravitation law below relation is feasible based on Eq. (1) [Adopted from 58,79,141,165,187,188,189,190,204,230,308]:

$$F = ma = mg = m\frac{X_{C-earth}}{A.D} = \frac{M \times G \times E}{R^2}, \quad (3)$$

The mass of the earth is equal to [Adopted from 246]:

$$E = ED \times V_{Earth} \quad (4)$$

Then, cohesion constant limit number of the earth with impact of earth density according to Eqs. (3) and (4) written as [Adopted from 246]:

$$X_{C-Earth} = \frac{M}{m} \times \frac{G \times E}{R_{Earth}{}^2 \times e^e} = \frac{M}{m} \times \frac{G \times ED \times V_{Earth}}{R_{Earth}{}^2 \times e^e} = \frac{M}{m} \times \frac{G \times ED \times V_{Earth} \times A.D}{R_{Earth}{}^2} \cong 0.647122 \quad (5)$$

Many fundamental laws can be deriving and clarifying from substantial sciences by Strength limit factor of earth such as laws which put parallel beside Newton's notable laws [Adopted from 246].

Then, standard gravity expressed by below relation without deleting m and M from two sides of Eq. (3) [Adopted from 247]:

$$g_0 = \frac{F}{m} = \frac{M}{m} \times \frac{G \times E}{R^2} = R^{'} \times \frac{G \times E}{R^2} = \frac{G \times E}{R^{''}} = \frac{X_{C-Earth}}{A.D} = \frac{X_{C-earth}}{\frac{1}{exp.^e}}. \quad (6)$$

Furthermore, follow relation deduced via the Eq. (6) [Adopted from 247]:

$$\frac{g_0 \times R^2}{G} = E \times R^{'} = ED \times V_{Earth} \times R^{'} = 242892793676594.55573513891084111 \text{ Kg.m.} \quad (7)$$

The ratio of every point mass M to mass m in calculations is equal to the distance between the masses only from quantitative aspect and can be written as [Adopted from 247]:

$$\frac{M}{m} = R^{'} = \frac{R_{Earth}{}^2}{R^{''}} = \frac{g_0 \times R_{Earth}{}^2}{G \times E} = 40656957195372.527825506161635217. \quad (8)$$

$$R^{''} = \frac{R_{Earth}{}^2}{R^{'}} = \frac{R_{Earth}{}^2}{\frac{M}{m}}. \quad (8.1)$$

Where,

$$R^{'} = R_{Earth-Dimensionless} = \text{The Quantitative Value of R} = \text{Dimensionless R}$$

Cohesion limit number of the earth with impact of earth density clearly expressed as [Adopted from 247]:

$$X_{C-Earth} = \frac{M}{m} \times \frac{G \times E}{R_{Earth}{}^2} \times A.D = \frac{G \times E}{R^{''} \times exp.^e} = \frac{G \times ED \times V_{Earth}}{R^{''}} \times A.D = 0.64712157172243419172410449319661. \quad (9)$$

$$X_{C-Earth} = G \times ED \times V_{Earth} \times \frac{R_{Earth-Dimensionless}}{R_{Earth}{}^2} \times A.D = G \times ED \times V_{Earth} \times \frac{R_{Earth-Dimensionless}}{R_{Earth}\ (in\, m) \times R_{Earth}\ (in\, m)} \times A.D = \frac{G \times ED \times V_{Earth}}{R^{'}_{Earth}} \times A.D. \quad (10)$$

Where [Adopted from 247] $\boldsymbol{F}$ is the force between the masses (in N), **G** is the Newtonian gravitational constant or Newtonian constant of gravitation that's approximately equal to $6.67384 \times 10^{-11} \pm 0.00080 \times 10^{-11}$ $m^3kg^{-1}s^{-2}$ [104] or 6.67384×10^{-11} N m^2 kg^{-2} [187, 188, 189, 190], **E** (also identified by $\boldsymbol{\mathcal{M}_{\oplus}}$) is the mass of the earth that's equal to $\mathcal{M}_{\oplus} = 5.9742 \times 10^{24} Kg$ [56], **M** is the every point mass (in Kg), $\mathbf{R_{Earth}}$ **or R** is the distance between the masses (in m) which equal to 40656957195372.527825506161635217 m, **ED** is the earth density (in Kg/m^3) that's equal to 5.5152740465837649209294596615615 gr/cm^3, $\mathbf{V_{earth}}$ is the volume of earth (in m^3) and equal to 1.08321×10^{12} km^3 or 1.08321×10^{27} cm^3 [56] (2.59876×10^{11} cu mi [311]), $\mathbf{R'}$ or $\mathbf{R_{Earth-Dimensionless}}$ is the ratio of every point mass M to mass m or is the quantitative radius value of earth without unit (dimensionless) and equal to 40656957195372.527825506161635217 **(**dimensionless**)**, $\mathbf{R''}$ is equal to 40656957195372.527825506161635217 m^2, $\mathbf{R'_{Earth}}$ is equal to 40656957195372.527825506161635217 m^2.

6. Variable Strength Limit of Earth ($\mathbf{X_{C-variable}}$ or $\mathbf{X_{CVLN}}$)

In the Theory section, it is necessary to give a theoretical or methodical basis that will be required for obtaining results of the paper.

The universe has been expanding and earth has twitching therefore density, volume, mass and radius of earth varying. In addition, Earth's gravity, mass, density, volume and strength limit of earth have direct proportion together. Thus, cohesion variable limit number for earth (variable strength limit number of earth) must be defined. Variable strength limit number of earth changing according to earth's variations of features defined by below equations:

$$X_{C-variable} = \frac{M}{m} \times \frac{G \times E_{V-Earth}}{R_{V-Earth}{}^2 \times exp.^e} = \frac{M}{m} \times \frac{G \times ED_{V-Earth} \times V_{V-Earth}}{R_{V-Earth}{}^2 \times exp.^e} = \frac{M}{m} \times \frac{G \times ED_{V-Earth} \times V_{V-Earth} \times A.D}{R_{V-Earth}{}^2}. \quad (11)$$

$$X_{CVLN} = X_{CCLN} \times \left(\frac{r_e}{r_e + h}\right)^2. \quad (12)$$

Therefore, according to Eq. (1) will have:

$$X_{C-variable} = X_{CVLN} = g_V \times A.D = \text{Variable Gravity} \times \text{Area Density} \quad (13)$$

Where $\mathbf{X_{CVLN}}$ or $\boldsymbol{X_{C-variable}}$ is the variable strength limit of earth or cohesion variable limit number of earth (in MPa), $\mathbf{X_{CCLN}}$ is the cohesion constant limit number (in MPa), $\mathbf{g_V}$ is the variable gravity (in m/s^2), $\mathbf{E_{V-Earth}}$ is the variable mass of the earth (in Kg), $\mathbf{R_{V-Earth}}$ is the variable distance between the masses (in m), $\mathbf{ED_{V-Earth}}$ is the variable earth density (in Kg/m^3), $\mathbf{V_{V-Earth}}$ is the variable volume of earth (in m^3).

Area density of earth and gravitational factor would change in accordance to changing the earth's features if the changes did not perform with constant proportion. Hence, variable strength limit based on Eqs. (11) and (13) defined as:

$$X_{C-variable} = \frac{M}{m} \times \frac{G_V \times E_{V-Earth} \times AD_V}{R_{V-Earth}^{\;2}} = \frac{M}{m} \times \frac{G_V \times ED_{V-Earth} \times V_{V-Earth} \times AD_V}{R_{V-Earth}^{\;2}} = \frac{M}{m} \times \frac{G_V \times ED_{V-Earth} \times V_{V-Earth} \times AD_V}{R_{V-Earth}^{\;2}}. \quad (14)$$

$$X_{C-variable} = X_{CVLN} = g_V \times AD_V = Variable\,Gravity \times Variable\,Area\,Density \quad (15)$$

Where $\mathbf{G}_{V}$ is the gravitational variation, $\mathbf{g}_{V}$ is variable standard gravity, $\mathbf{AD}_{V}$ is variable area density of earth.

7. Definition of Cohesion Variable Limit Number for Slope

For the reason of importance, complicating, different and wide applications of X_C parameter in existence descriptions like earth and various sciences, in this article only presents a specific definition of this parameter about slope stability field. X_C parameter according to one approach is variable for different parts of earth like a slope that's call cohesion variable limit number for the slope ($\mathbf{X}_{\mathbf{CV-Slope}}$ or $\mathbf{X}_{\mathbf{CV(Slope)}}$ or $\mathbf{X}_{\mathbf{CVLN-Slope}}$), even for different parts of a mine (or different blocks of a mine). In the other words, if a mine is divided into eight blocks this parameter is different for every mine's block also is variable from one block to another block. Earth has a constant or standard strength limit number as same as its standard gravity. In addition, different parts of earth have different and variable strength limit number as same as various things near or on earth which has variable gravity in diverse situations. Therefore, several parts of slopes have varied strength limit number. Thus, variable strength limit number for a slope defined by:

$$\mathrm{X}_{\mathrm{CVLN-Slope}} = \mathrm{X}_{\mathrm{CV-Slope}} = A - B + C - X_1 + D - E = (A + C + D) - (B + X_1 + E) \quad (16)$$

A, B, C, D and E variables in above relation are as follows:

$$A = \frac{\overline{\varnothing}_i \times C_{imin}}{\overline{C}_i \times \varnothing_{imin}} \quad (17)$$

$$B = \left(MEARS \times \left(\frac{\overline{C}_i \times \varnothing_{imin}}{\overline{\varnothing}_i \times C_{imin} \times C_{imax}} \right) \right) = \left(MEARS \times \left(\frac{1}{A \times C_{imax}} \right) \right) \quad (18)$$

$$B = \left(10 \times \left(\frac{\overline{C}_i \times \varnothing_{imin}}{\overline{\varnothing}_i \times C_{imin} \times C_{imax}} \right) \right) = \left(10 \times \left(\frac{1}{A \times C_{imax}} \right) \right) \quad (19)$$

$$C = \left(\frac{1}{100} \times \frac{\overline{C}_i \times \varnothing_{imin}}{\overline{\varnothing}_i \times C_{imin}} \right) = \left(\frac{1}{100} \times \frac{1}{A} \right) \quad (20)$$

$$D = \frac{1}{1000} \left(\overline{\mathrm{V}} \times \left(\frac{\overline{C}_i \times \varnothing_{imin} \times \varnothing_{imax}}{\overline{\varnothing}_i \times C_{imin} \times C_{imax}} \times \mathrm{MEARS} \right) \right) = \frac{1}{1000} \left(\overline{\mathrm{V}} \times \left(\frac{1}{A} \times \frac{\varnothing_{imax}}{C_{imax}} \times \mathrm{MEARS} \right) \right) \quad (21)$$

$$D = \frac{1}{1000} \left(\overline{\mathrm{V}} \times \left(\frac{\overline{C}_i \times \varnothing_{imin} \times \varnothing_{imax}}{\overline{\varnothing}_i \times C_{imin} \times C_{imax}} \times 10 \right) \right) = \frac{1}{1000} \left(\overline{\mathrm{V}} \times \left(\frac{1}{A} \times \frac{\varnothing_{imax}}{C_{imax}} \times 10 \right) \right) = \frac{1}{100} \left(\overline{\mathrm{V}} \times \left(\frac{1 \times \varnothing_{imax}}{A \times C_{imax}} \right) \right) \quad (22)$$

$$E = [(10^{-2} \times \left(\overline{T} + \frac{\frac{H+D}{2} + \frac{\beta}{100}}{1000} \right)) \times (H + D + \frac{\beta}{100})] = \left[\left(10^{-2} \times \left(\overline{T} + \frac{100(H+D) + 2\beta}{200000} \right) \right) \times \left(H + D + \frac{\beta}{100} \right) \right] \quad (23)$$

$$E = \left(\frac{(2 \times 10^5 \times \overline{T}) + 100(H+D) + 2\beta}{2 \times 10^7} \right) \times (H + D + \frac{\beta}{100})] = \left(\frac{\left[\left(2 \times 10^5 \times \overline{T} \right) + 100(H+D) + 2\beta \right] \times [100H + 100D + \beta]}{2 \times 10^9} \right] \quad (24)$$

Then, simple forms of Eq. (13) according to Eqs. (16), (17), (18), (20), (21) and (23) are as below:

$$\mathrm{X}_{\mathrm{CVLN-Slope}} = \mathrm{X}_{\mathrm{CV(Slope)}} = A + \frac{1}{A} \left[\left(\frac{1}{100} \right) + \left(\frac{\overline{\mathrm{V}} \times \varnothing_{imax} \times \mathrm{MEARS}}{1000 \times C_{imax}} \right) - \left(\frac{\mathrm{MEARS}}{C_{imax}} \right) \right] - X_1 - E \quad (25)$$

$$\mathrm{X}_{\mathrm{CVLN-Slope}} = \mathrm{X}_{\mathrm{CV(Slope)}} = A + \frac{1}{A} \left[\left(\frac{1}{100} \right) + \left(\frac{\overline{\mathrm{V}} \times \varnothing_{imax}}{100 \times C_{imax}} \right) - \left(\frac{10}{C_{imax}} \right) \right] - X_1 - E \quad (26)$$

Where ***MEARS*** is the minimal effective action and reaction unit of strength and equal to 10 MPa, [Adopted from 246] $\boldsymbol{X_{CVLN-Slope}}$ or $\boldsymbol{X_{CV-Slope}}$ or $\boldsymbol{X_{CV(Slope)}}$ is the cohesion variable limit number for a slope (in MPa), $\overline{\mathbf{V}}_{\mathbf{Assessing\ Mine\ Block}} = \overline{\mathbf{V}} = \overline{\mathbf{V}}_{\mathbf{MB}}$ is the rock mass index values average to sum of the rock mass index values of the assessing mine block ratio, $\boldsymbol{X_1}$is the detector parameter (dimensionless), $\boldsymbol{\overline{C}_i}$ is the Cohesion values average of slope components (in kPa), $\boldsymbol{C_{imax}}$ is the Maximum Cohesion value of slope components (in kPa), $\boldsymbol{C_{imin}}$ is the Minimum Cohesion value of slope components (in kPa), $\boldsymbol{\overline{\varnothing}_i}$ is the internal friction angle values average of slope components (in degree), $\varnothing_{\boldsymbol{imax}}$is the Maximum internal friction angle of slope components (in degree), $\varnothing_{\boldsymbol{imin}}$ is the Slope components' minimum internal friction angle (in degree), $\boldsymbol{\overline{T}}$ is the average of the rock mass tensile strength of the assessing mine block (in MPa), $\boldsymbol{H}$ is the seismic acceleration, $\boldsymbol{D}$ is the disturbance factor (dimensionless), $\boldsymbol{\beta}$ is the overall slope angle (in degree).

X_1in Eqs. (16), (25) and (26) is a detector parameter that is given by below equations:

$$\overline{\mathrm{V}} = \frac{\overline{RMI}}{RMI_{Sum}} \quad (27)$$

$$exp.^{\overline{V}} = 1 + \overline{V} + X_1 \tag{28}$$

Then, X_1 factor obtained by:

$$X_1 = exp.^{\overline{V}} - \overline{V} - 1 \tag{29}$$

For example:

$$IF\ \overline{V} = 0.0042 \Rightarrow e^{\overline{V}} = exp.^{0.0042} = 1 + 0.0042 + 0.000008832 \Rightarrow X_1 = 0.000008832.$$

In Eq. (28) e is the Napier number. In addition**,** the minimal effective action and reaction unit of strength obtained by [Adopted from 246]:

$$MEARS = \frac{þ}{Decay\,Constant\left(yr^{-1}\right)} = \frac{þ}{D.C\left(yr^{-1}\right)} = 10\,MPa. \tag{30}$$

Where [Adopted from 246] ***MEARS*** is the minimal effective action and reaction unit of strength and equal to 10 MPa, $\boldsymbol{D.C}$ is the decay constant of gravitational particles and currently exact value for the decay constant of gravitational particles which close to the decay constant of ^{87}Rb and is equal to $1.42219049734677361280579263193 35\times 10^{-11}$ yr^{-1}, þ is strength variations coefficient in 10^{10} years and equal to $þ = 1.4221904973467736128057926319335 \times 10^{-10}$ in $\frac{\text{MPa}}{\text{Year}} = \frac{\text{MN}}{\text{m}^2\text{Year}} = \frac{\text{Ggr}}{\text{Year m s}^2}$.

Strength variations coefficient calculated through one of the relations between age of the earth and strength limit number of earth accordance to Eq. (2) as below equations [Adopted from 246]:

$$X_{C-earth} = AOE \times þ. \tag{31}$$

$$þ = \frac{X_{C-earth}}{AOE} = \frac{g_0 \times \frac{1}{exp.^e}}{AOE} = \frac{g_0 \times A.D}{AOE}. \tag{32}$$

Where [Adopted from 246] $\boldsymbol{X_{c\text{-}earth}}$ is strength limit number of earth and equal to 0.64712157172243419172410449319661 MPa (this value calculated according to old standard gravity ($\boldsymbol{g_{0-Old}}$) which is equal to 9.80665 $\frac{\mathbf{m}}{\mathbf{s^2}}$), $\boldsymbol{AOE}$ is the age of the earth's abbreviation and old age of the earth according to old standard gravity is equal to 4.5501750498945017745329167415034 Billion Years.

Age of the earth according to old standard gravity ($\mathbf{AOE_{Old}}$) is equal to 4.5501750498945017745329167415034 Billion Years also the strength limit parameter of the earth according to old standard gravity ($\mathbf{X_{C-Earth,Old}}$) is equal to 0.64712157172243419172410449319661 MPa [Adopted from 246].

Strength limit factor of earth changed by changing gravity of earth. Above values for $\mathbf{X_{c\text{-}earth}}$ and **AOE** computed according to old standard gravity. Accurate or exact age of the earth according to present standard gravity at present time ($\boldsymbol{AOE_{Exact}}$) is equal to 4.7303652362904 Billion Years. Then, strength limit parameter of earth or strength of existence global gravitation according to present standard gravity ($\boldsymbol{X_{C-Earth,Now}}$) is equal to 0.6727480488031732255208148942682 MPa. The present value of standard gravity ($\mathbf{g_{0-Now}}$) is equal to 10.195000354006777301014825930477 m/s^2 [Adopted from 246]**.**

What are the meanings of old and present values of gravity, age and strength limit of the earth?

Creation procedures of the earth started in 4.7303652362904 Billion Years ago with 0.673 MPa strength limit value and 10.2 m/s^2 gravity. Earth created completely (fully whole) in 4.5501750498945017745329167415034 Billion Years ago with approximately 0.65 MPa value of strength limit and 9.80665 m/s^2 of the gravity in whole creation point. Gravity value along the 180190186.4 years' creation process of the earth from 4.7304 Billion Years ago to 4.56 Billion Years ago was higher than now and equal to 10.2 m/s^2. These momentous and epochal discoveries destine also desquamates the covered key of the sciences' fast perfections.

Meanwhile, all above formulas true based on old and present values of standard gravity, strength limit and age of the earth therefore according to Eqs. (31) and (32) have:

$$X_{C-Earth,Old} = AOE_{Old} \times þ. \tag{33}$$

$$X_{C-Earth,Now} = AOE_{Exact} \times þ. \tag{34}$$

$$þ = \frac{X_{C-Earth,Old}}{AOE_{Old}} = \frac{g_{0-Old} \times \frac{1}{exp.^e}}{AOE_{Old}} = \frac{g_{0-Old} \times A.D}{AOE_{Old}}. \tag{35}$$

$$þ = \frac{X_{C-Earth,Now}}{AOE_{Exact}} = \frac{g_{0-Now} \times \frac{1}{exp.^e}}{AOE_{Exact}} = \frac{g_{0-Now} \times A.D}{AOE_{Exact}}. \tag{36}$$

Earth's historical features involve in calculation of variable strength limit for slopes and its component factors. In addition, we have below equations in accordance to Eqs. (18), (16), (30), (35), and (36):

$$B=\left(\frac{\text{þ}=\frac{X_{C-Earth,Old}}{AOE_{Old}}=\frac{g_{0-Old}\times\frac{1}{exp.^{e}}}{AOE_{Old}}=\frac{g_{0-Old}\times A.D}{AOE_{Old}}}{Decay\,Constant\left(yr^{-1}\right)}\times\left(\frac{\overline{C}_i\times\varnothing_{imin}}{\overline{\varnothing}_i\times C_{imin}\times C_{imax}}\right)\right)\tag{37}$$

$$B=\left(\frac{\text{þ}=\frac{X_{C-Earth,Now}}{AOE_{Exact}}=\frac{g_{0-Now}\times\frac{1}{exp.^{e}}}{AOE_{Exact}}=\frac{g_{0-Now}\times A.D}{AOE_{Exact}}}{Decay\,Constant\left(yr^{-1}\right)}\times\left(\frac{1}{A\times C_{imax}}\right)\right)\tag{38}$$

$$D=\frac{1}{1000}\left(\overline{\mathrm{V}}\times\left(\frac{\overline{C}_i\times\varnothing_{imin}\times\varnothing_{imax}}{\overline{\varnothing}_i\times C_{imin}\times C_{imax}}\times\mathrm{MEARS}\right)\right)=\frac{1}{1000}\left(\overline{\mathrm{V}}\times\left(\frac{1\times\varnothing_{imax}}{A\times C_{imax}}\times\frac{\text{þ}=\frac{\mathrm{X}_{\mathrm{C-Earth,\,Old}}}{\mathrm{AOE_{Old}}}=\frac{\mathrm{g_{0-Old}}\times\frac{1}{\mathrm{exp.}^{e}}}{\mathrm{AOE_{Old}}}=\frac{\mathrm{g_{0-Old}}\times\mathrm{A.D}}{\mathrm{AOE_{Old}}}}{\mathrm{Decay\ Constant}\left(\mathrm{yr}^{-1}\right)}\right)\right)\tag{39}$$

$$D=\frac{1}{1000}\left(\overline{\mathrm{V}}\times\left(\frac{1}{A}\times\frac{\varnothing_{imax}}{C_{imax}}\times\frac{\text{þ}=\frac{X_{C-Earth,Now}}{AOE_{Exact}}=\frac{\mathrm{g_{0-Now}}\times\frac{1}{\mathrm{exp.}^{e}}}{AOE_{Exact}}=\frac{\mathrm{g_{0-Now}}\times\mathrm{A.D}}{AOE_{Exact}}}{\mathrm{Decay\ Constant}\left(\mathrm{yr}^{-1}\right)}\right)\right)\tag{40}$$

Therefore, variable strength limit for slopes in accordance to Eqs. (25), (27), (28), (35), and (36) and also based on old and present value of earth's strength limit with RMi effects obtained by:

$$X_{CVLN-Slope}=A+\frac{1}{A}\left[\left(\frac{1}{100}\right)+\left(\frac{\frac{\overline{RMI}}{RMI_{Sum}}\times\varnothing_{imax}\times\frac{\text{þ}=\frac{X_{C-Earth,Old}}{AOE_{Old}}=\frac{g_{0-Old}\times A.D}{AOE_{Old}}}{Decay\,Constant\left(yr^{-1}\right)}}{1000\times C_{imax}}\right)-\left(\frac{\frac{\frac{X_{C-Earth,Now}}{AOE_{Exact}}=\frac{g_{0-Now}\times A.D}{AOE_{Exact}}}{Decay\,Constant\left(yr^{-1}\right)}}{C_{imax}}\right)\right]-exp.^{\overline{V}}-\overline{V}-1-E\tag{41}$$

$$X_{CVLN-Slope}=\frac{\overline{\varnothing}_i\times C_{imin}}{\overline{C}_i\times\varnothing_{imin}}+\frac{1}{\frac{\overline{\varnothing}_i\times C_{imin}}{\overline{C}_i\times\varnothing_{imin}}}\left[\left(\frac{1}{100}\right)+\left(\frac{\frac{\overline{RMI}}{RMI_{Sum}}\times\varnothing_{imax}\times\frac{\text{þ}=\frac{X_{C-Earth,Old}}{AOE_{Old}}=\frac{g_{0-Old}\times A.D}{AOE_{Old}}}{Decay\,Constant\left(yr^{-1}\right)}}{1000\times C_{imax}}\right)-\left(\frac{\frac{\frac{X_{C-Earth,Now}}{AOE_{Exact}}=\frac{g_{0-Now}\times A.D}{AOE_{Exact}}}{Decay\,Constant\left(yr^{-1}\right)}}{C_{imax}}\right)\right]$$
$$-exp.^{\frac{\overline{RMI}}{RMI_{Sum}}}-\frac{\overline{RMI}}{RMI_{Sum}}-1-\left(\frac{\left[\left(2\times10^{5}\times\overline{T}\right)+100\left(H+D\right)+2\beta\right]\times[100H+100D+\beta]}{2\times10^{9}}\right)\tag{41-1}$$

8. Procedures of Empirical Investigations: New Empirical Relations for Calculation of the Safety Factor in Fully Drained Conditions

Various relations between involved factors in existence such as rock engineering factors could use to slope stability analysis procedures.

The height of the slopes has a direct proportion to the changes of the stress levels in the slopes' rock mass also has face-to-face influence on it [Adopted from 106]. A high slope may also let more opportunities to evince of discontinuities associated failures to act as instabilities because the discontinuities' quantity which intersected by the slope is larger. High stress levels may lead to the slopes failures due to failure of the intact rock compared to the intact rock strength (Gama, 1989). Although slope's height is momentous in slope stability systems, but only the Haines and Shuk [264] system includes the slope height [Adopted from 106, 264]. High slope height included high-pressure causes instabilities to the slope structure, which needs to considered in a slope stability's assessment system. The overall slope angle, number of slope components and slope's characteristics and configurations are related to dewatering and drainage proportionally as a result of slope stability analysis. The overall slope angle has great importance application in drained, partially drained, fully drained even in un-drained conditions therefore play its roles in any conditions also slope configurations too.

If the slopes are not dewatered effectively, therefore, the overall slope angles will reduce significantly. The influence of ground water is often crucial to the stability of large open pit mine slopes [281]. Overall slope angle is an intra-factor, which carries the conditions of ground and underground water inside self and illustrates it too. The disturbance parameter (D) is one of the strength reduction factors, which accounts for the disturbance and strength reduction of the in-situ rock mass by blasting or stress relaxation.

A D factor of less than or equal to 0.7 is applicable for free dig material. A D factor of one indicates maximum disturbance by blasting and is appropriate for open pit conditions [281]. Similar to many other porphyry deposit regions, the Sungun Copper Mine situated in an area with high seismicity. The relative importance of seismic accelerations is very significant in terms of the stable slope angles. Therefore, it is important to determine the appropriate earthquake accelerations for design's input [281]. A horizontal seismic acceleration coefficient indicates the one of the critical conditions for the open pit mines particularly for Sungun copper mine to specify damages during an earthquake in the lifetime of the mine. Thus, this specialty must consider precisely and carefully.

This factor in Sungun seismic study obtained from a re-interpretation of a seismic hazard evaluation report. Seismic hazard analysis for Sungun mine carried out based on relevant seismo-tectonic information for determine the peak ground acceleration (**PGA**) at an appropriate design level. The most stringent seismic design criterion is the Maximum Credible Level (**MCL**), and in further decreasing, order the levels are: Maximum Design Level (**MDL**), the Design Basis Level (**DBL**), and the Construction Level (**CL**)) [Adopted from 281]. Overall slope height ($\mathbf{h_{OS}}$) is required to estimate the weight of the rock column overlying the estimated failure surface. This is also required to estimate $\boldsymbol{\sigma_3}$ to calculate various parameters of the Hoek-Brown equation and for calculation of equivalent Mohr-Coulomb strength parameters, cohesion (**c**) and friction angle (**phi**) [281].

In presented equations in this part, the parameters requiring for safety factor calculation shown in Table 4:

Table 4: Input involved parameters for safety factor calculation in presented equations.

Factors	**Explanations**	**Units**
FS_{fd}	Factor of safety for fully drained conditions.	Dimensionless
$OSH = H_{OS} = h_{OS}$	Overall slope height or is the maximum overall slope height.	Meter (m)
n	Number of components in a slope (number of slope's components).	Dimensionless
$\overline{RMi}$ or $RMI_{(av)}$	Average of rock mass index values for assessing mine block.	MPa
RMi_{Sum}	Sum of rock mass index values obtained from every approved components of a discovery boreholes in a slope for assessing mine block.	MPa
e	Napier number.	Dimensionless
H	Seismic acceleration.	Dimensionless
D	Disturbance factor.	Dimensionless
OSA = β	Overall slope angle.	Degree
$IRA = \alpha$	Inter ramp angle.	Degree
$BFA = \Delta = \delta$ i	Bench face angle.	Degree
$BFH = H_{BF}$	Bench face height.	Meter (m)
$SP.BW = W_{SP.B}$	Spill berm width.	Meter (m)
$SBW = W_{SB}$	Safety berm width.	Meter (m)
$IRH = H_{IR}$	Inter ramp height.	Meter (m)
$GBW = W_{GB}$	Geotechnical berm width.	Meter (m)
$\bar{C}_{components\ of\ slopes\ (c\ of\ s)}$	Average of cohesion values of slope components (average of a slope components cohesion values).	MPa
C_{imin}	Minimum cohesion value of slope components.	kPa
$\bar{\emptyset}_{c\ of\ s}$	Average of internal friction angle values of slope components.	Degree
$\bar{T}$	Average of rock mass tensile strength in assessing mine block.	MPa
γ	Unit weight of rock in assessing mine block.	$\frac{MN}{m^3}$
$MEARS$	Minimal effective action and reaction unit of strength.	MPa
X_{CVLN}	Cohesion variable limit number.	MPa

8.1. Safety factor computation via RMi and slope's properties using strength limit with an independent approach

Earth's strength limit factor can help identify and solve slope stability engineering problems. This feature of earth shows the equilibrium limit and limit equilibrium between resisting and disturbing forces in universal gravitational field and universe's overall stability field. Therefore, more required detailed data involved in this parameter that will uncover as a hidden potential. In addition, strength limit by these uncapping can apply the included equivalency between more influencer factors on slopes failures engineering analysis to solving slope stability problems without doing excessive long time-consuming laboratory and field researches.

Numbers like strength limit and Napier are Principal Fundamental Pivot Numbers (PFPN), which shows the equivalency and continuity in the entire world. Also, functions like $e^x$and $e^{2x}$are linear independents on every interval. Therefore, this number can be applying for setting the equivalency between factors of rocks slopes stability engineering. Euler's constant in fact derived from earth characteristics and overhand. Napier pivotal and fundamental number (e) reach science to finding the balance point between earth's features (in accordance to Eq. (1)) also, slope properties, average of rock mass index and variable strength limit of the slope, seismic acceleration and disturbance factor as an importance result which are obtained as follow:

$$e = ln\left(\frac{g_0}{X_{C-earth}}\right) = ln(\frac{1}{A.D}) = \sqrt{\frac{h_{OS} \times X_{CVLN} \times \left(H+D+\frac{\beta}{100}\right) \times (H+D)}{\overline{RMi} \times FS_{fd} \times [\frac{(\alpha+\beta+\delta)}{100} + \frac{(H_{IR}+H_{BF})}{100} + \frac{(W_{GB}+W_{SP.\,B}+W_{SB})}{100}]}} \tag{42}$$

Strength limit show how properties of slopes, RMi, seismic acceleration and Napier number can play role parallel for safety factor estimation independently like as:

$$FS_{fd} = \frac{h_{OS}}{\overline{RMi} \times exp.^2 \times [\frac{(\alpha+\beta+\delta)}{100} + \frac{(H_{IR}+H_{BF})}{100} + \frac{(W_{GB}+W_{SP.\,B}+W_{SB})}{100}]} \times X_{CVLN} \times \left(H+D+\frac{\beta}{100}\right) \times (H+D) \tag{43}$$

Eq. (43) determined from results of Napier's number definition based on Eq. (42). Eq. (43) can use to check the reliability of designed slopes' stabilization. Unit equilibrium in Eq. (43) display as:

$$\frac{m}{MPa \times m} \times MPa = dimensionless$$

8.2. Calculation of the safety factor using impact of the unit weight of rock in assessing mine block

Strength limit of earth is a phenomenal parameter that enables us to compute essential factors with minimum data such as rock engineering properties. One of the applications of this factor is calculating the unit weight of rock in assessing mine's block ($\boldsymbol{\gamma}$) as:

$$2 \times \gamma = \frac{X_{C-earth}}{g_0} \times \left[\frac{\left(H+D+\frac{\beta}{100}\right)}{n} + \frac{IRA}{100}\right] = \frac{1}{exp.^e} \times \left[\frac{\left[100 \times \left(H+D+\frac{\beta}{100}\right) + (\,\right] n \times IRA)}{100n}\right] = Area\,Density \times \left[\frac{\left(H+D+\frac{\beta}{100}\right)}{n} + \frac{IRA}{100}\right] \tag{44}$$

Hence, the unit weight of rock according to Eqs. (4) and (44) given by:

$$\gamma = \frac{\frac{1}{exp.^e} \times \left[\frac{\left(H+D+\frac{\beta}{100}\right)}{n} + \frac{IRA}{100}\right]}{2} = \frac{A.D \times \left[\frac{\left(H+D+\frac{\beta}{100}\right)}{n} + \frac{IRA}{100}\right]}{2} \tag{44.1}$$

$$4\gamma^2 = (A.D)^2 \times \left[\frac{\left(H+D+\frac{\beta}{100}\right)}{n} + \frac{IRA}{100}\right]^2 \tag{44.2}$$

Eq. (44.2) defined in accordance to Eq. (44.1) and below procedures of Eq. (2):

$$\frac{1}{exp.^{2e}} = \frac{1}{exp.^e \times exp.^e} = A.D^2 \tag{45}$$

$$exp.^{2e} = \frac{1}{A.D^2} \tag{45.1}$$

$$e = \frac{ln(\frac{1}{A.D^2})}{2} \tag{45.2}$$

$$e = ln(\frac{1}{A.D}) \tag{45.3}$$

$$exp.^2 = \left(ln(\frac{1}{A.D})\right)^2 \tag{45.4}$$

Hack et al. (1982), Hack et al. (1990), Cervantes (1995) and Hack (1998) imply that the behavior of a seismic wave in a rock mass and the relationships between the rock mass parameters and the seismic parameters not known in all details. Consequently, the interpretation is often ambiguous. Strength limit of earth, area density and gravity uncover the relationship between rock mass parameters like unit weight of rock and the seismic parameters such as seismic acceleration on a slope (Eqs. (44) and (44.1)). Inter ramp angle, average of rock mass tensile strength, the sum of rock mass index values obtained from every approved component of discovery boreholes in a slope, unit weight of rock mass and minimum and average of cohesion values of slope components in assessing mine's block prescribed other definitions of Euler’s basis number as:

$$e=\frac{ln\left(\dfrac{\mathrm{h_{OS}}^{2}\times \mathrm{X_{CVLN}}\times[\dfrac{\left(H+D+\frac{\beta}{100}\right)}{n}+\dfrac{IRA}{100}]^{2}\times(H+D)\times\overline{T}\times 0.888}{\overline{\overline{RMi}}\times \mathrm{FS_{fd}}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}}\right)}{2}=\sqrt[2e]{\left(\dfrac{\mathrm{h_{OS}}^{2}\times \mathrm{X_{CVLN}}\times[\dfrac{\left(H+D+\frac{\beta}{100}\right)}{n}+\dfrac{IRA}{100}]^{2}\times(H+D)\times\overline{T}\times 0.888}{\overline{\overline{RMi}}\times \mathrm{FS_{fd}}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}}\right)} \tag{46}$$

$$e=\frac{ln\left(\dfrac{\mathrm{h_{OS}}^{2}\times \mathrm{X_{CVLN}}\times\dfrac{4\gamma^{2}}{A.D^{2}}\times(H+D)\times\overline{T}\times 0.888}{\overline{\overline{RMi}}\times \mathrm{FS_{fd}}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}}\right)}{2}=\left(\dfrac{\mathrm{h_{OS}}^{2}\times \mathrm{X_{CVLN}}\times\dfrac{4\gamma^{2}}{A.D^{2}}\times(H+D)\times\overline{T}\times 0.888}{\overline{\overline{RMi}}\times \mathrm{FS_{fd}}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}}\right)^{\frac{1}{2e}} \tag{46.1}$$

Therefore, safety factor based on another key specification of Euler’s determinant number with respect to slope stability properties can discovered from the following relations according to Eq. (46):

$$\mathrm{FS_{fd}}=\frac{\mathrm{h_{OS}}^{2}}{\overline{\overline{RMi}}\times exp.^{2e}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}}\times \mathrm{X_{CVLN}}\times[\frac{\left(H+D+\frac{\beta}{100}\right)}{n}+\frac{IRA}{100}]^{2}\times(H+D)\times\overline{T}\times 0.888 \tag{47}$$

Moreover, safety factor in accordance to Eqs. (2), (44.1), (46.1) and (47) given by:

$$\mathrm{FS_{fd}}=\frac{\mathrm{h_{OS}}^{2}}{\overline{\overline{RMi}}\times exp.^{2e}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}}\times \mathrm{X_{CVLN}}\times\frac{4\gamma^{2}}{A.D^{2}}\times(H+D)\times\overline{T}\times 0.888 \tag{47.1}$$

$$\mathrm{FS_{fd}}=\frac{\mathrm{h_{OS}}^{2}}{\overline{\overline{RMi}}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}}\times \mathrm{X_{CVLN}}\times 4\gamma^{2}\times(H+D)\times\overline{T}\times 0.888 \tag{47.2}$$

The quantitative factor (0.888) in Eqs. (46), (46.1), (47), (47.1) and (47.2) determined by:

$$0.888=\left(X_{1}\times 10^{5}\right)+\left(\frac{\varnothing_{i\max}}{10^{4}}\right)\cong\left(X_{1}\times 10^{5}\right)+\overline{\mathrm{V}} \tag{47.3}$$

Other approximations based on Eqs. (47), (47.1) and (47.2) for safety factor determination given by:

$$\mathrm{FS_{fd}}=\frac{\mathrm{h_{OS}}^{2}}{\overline{\overline{RMi}}\times exp.^{2e}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}\times\left(H+D+\frac{\beta}{100}\right)}\times \mathrm{X_{CVLN}}\times[\frac{\left(H+D+\frac{\beta}{100}\right)}{n}+\frac{IRA}{100}]^{2}\times(H+D)\times\overline{T} \tag{48}$$

$$\mathrm{FS_{fd}}=\frac{\mathrm{h_{OS}}^{2}}{\overline{\overline{RMi}}\times e^{2e}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}\times\left(H+D+\frac{\beta}{100}\right)}\times \mathrm{X_{CVLN}}\times 4\gamma^{2}\times exp.^{2e}\times(H+D)\times\overline{T} \tag{48.1}$$

$$\mathrm{FS_{fd}}=\frac{\mathrm{h_{OS}}^{2}}{\overline{\overline{RMi}}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}\times\left(H+D+\frac{\beta}{100}\right)}\times \mathrm{X_{CVLN}}\times 4\gamma^{2}\times(H+D)\times\overline{T} \tag{48.2}$$

Design of pit slope of the section RS06 with this slope’s specifications illustrated in Fig. 7. Number of slope components factor (n) which used in Eqs. (47) and (48) showed based on Fig. 7.

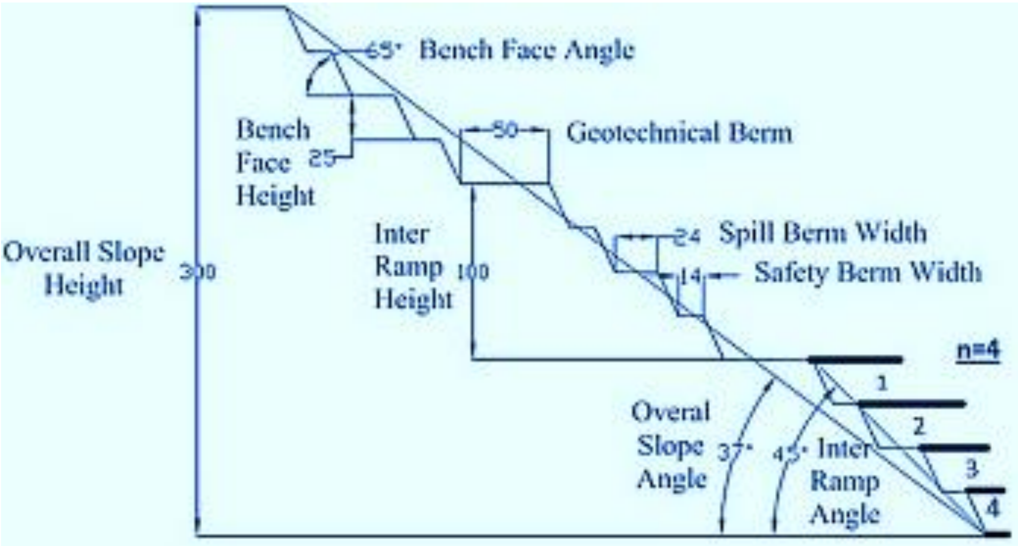


Figure 7. Pit slope design of north sector (section RS06) of Sungun copper project, scale 1:5000.

8.3. Empirical definition of safety factor via RMi classification system using limit equilibrium analysis approach

The simplex form of limit equilibrium analysis conventional methods only satisfied the equilibrium of the known forces. The sum of the known forces acting to persuade sliding of slope's parts compared with the sum of the identified forces available to resist against instabilities such as failure. The ratio of the total resisting actions to the sum of driving actions (ratio between two sums) defined as the factor of safety that present as Eq. (49):

$$FS = \frac{\sum(resisting\ actions)}{\sum(driving\ actions)} \tag{49}$$

Limit equilibrium can't become efficient if the slope failure occurs by complex mechanisms such as internal deformation, progressive deformation, brittle fracture, progressive creep, liquefaction of weaker soil layers, discontinuity orientation, progressive weathering, excavation disturbances, extensive internal disruption of slope mass, etc. [Adopted from 77]. In these conditions engineer trepan to more sophisticated numerical modelling techniques. Toward resolving these inadequacies, this simple definition of the factor of safety can interpreted in many modes, types, forms and ways. It could explain in terms of loads, forces, moments, works, action and reaction, etc. This concept of above definition also can use as a ratio of one influencing unit of total resisting forces to one unit of the total disturbing forces. The safety factor for fully drained condition given by below formulas:

$$FS_{fd} = \frac{reaction\ of\ resisting\ forces}{action\ of\ disturbing\ forces} = \frac{\frac{\mathrm{h_{OS}}^{n-2}}{RMi_{Sum} \times \overline{\varnothing}_{c\ of\ s}} \times \left(\overline{C}_{components\ of\ slopes\ (c\ of\ s)}\right)^2}{1MN} \tag{50}$$

Eq. (50) indicate that what values of resisting forces react when 1 MN of disturbing forces act equivalently and simultaneously. Unit's equilibrium of Eq. (50) given as follows:

$$\frac{\frac{m^2}{MPa}(MPa)^2}{MN} = \frac{m^2\,MPa}{MN} = \frac{m^2\,\frac{MN}{m^2}}{MN} = \frac{MN}{MN} = Undimentioned$$

In addition, safety factor for fully drained condition with impact of the unit weight of rock in assessing mine's block and effective unit of strength given by below formula:

$$FS_{fd} = \frac{\left[\frac{\mathrm{h_{OS}}^{n}}{RMi_{Sum} \times \overline{C}_{components\ of\ slopes\ (c\ of\ s)} \times \overline{\varnothing}_{c\ of\ s} \times 10^3}\right] \times \gamma_{ROCK}{}^2 \times \mathrm{MEARS}}{1MN} \tag{51}$$

Equilibrium of units in Eq. (51) is as follows:

$$\frac{\frac{m^4}{MPa \times MPa}\left(\frac{MN}{m^3}\right)^2 \times MPa}{1MN} = \frac{\frac{m^4}{MPa \times MPa}\left(\frac{MN}{m^2} \times \frac{1}{m}\right)^2 \times MPa}{1MN} = \frac{\frac{m^4}{MPa \times MPa}\left(MPa \times \frac{1}{m}\right)^2 \times \frac{MN}{m^2}}{1MN} = \frac{MN}{MN} = dimentionless$$

8.4. Safety factor computing through disturbing work in a defined slope using limit equilibrium approach

This approach clarifies what values of resisting forces react based on 1 Mj disturbing work of disturbing forces (or what values of resisting works is carried out reactionary and responsively when 1 Mj of disturbing work is carried out). On the other words, this scrutiny shows what values of resisting works of resisting forces react when 1 Mj disturbing work of disturbing forces act in slope stability procedures. Thus, safety factor value obtained based on involved work approach in a slope stability action and reaction as:

$$\mathrm{FS}_{fd}=\frac{reaction\,works\,value\ of\ resisting\ forces}{action\,works\,value\ of\ disturbing\ forces}=\frac{\dfrac{\mathrm{h_{OS}}^{n}}{RMi_{Sum}\times\overline{C}_{components\ of\ slopes\,(c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times\left(H+D+\dfrac{\beta}{100}\right)^{n+1}}\times \mathrm{X_{CVLN}}^{n-2}\times\gamma_{ROCK}}{1Mj\ Disturbing\ work} \tag{52}$$

Equilibrium of units in Eq. (52) is as:

$$\frac{\dfrac{m^4}{MPa\times MPa}\times(MPa)^2\times\dfrac{MN}{m^3}}{1Mj}=\frac{MN\times m}{1Mj}=\frac{Mj}{Mj}=dimentionless$$

8.5. Rapid calculation of the safety factor through a strength reduction factor (by a simple approach in presence of minimum data via overall slope angle using seismic acceleration and disturbance factor)

Overall slope angle, seismic acceleration, disturbance factor and number of slope components can be help rapid computation of safety factor in emergency unseen cases as follows:

$$FS_{fd}=\left(H+D+\frac{\beta}{100}\right)^{n+2} \tag{53}$$

8.6. Calculating the RMi values for the assessing zone

$\overline{\mathrm{RMi}}$ factor in the sections 8.1 and 8.2 is the acting RMi that rise from the interacting effects' intersecting of the RMi values of the slope's components. This value also obtained by averaging RMi values of the slope components. The RMi values for every component of the assessing zone calculated from correlation formulae between RMi and GSI, which proposed by G. Russo based on real field data and presented in Appendix 1.

In 2009 a new approach for a quantitative assessment of the Geological Strength Index (GSI) proposed by Russo. Based on the conceptual affinity of the GSI with the Joint Parameter (JP), which used in the RMi, (that introduced by Palmstrom in 1996). A relationship between the two indexes derived, exploited and proposed in order to obtain a reliable, quantitative assessment of the GSI by means of the basic input parameters for the determination of the RMi (i.e. the elementary block volume and the joint conditions). According to the RMi and GSI systems, Russo proposed below formulas [Adopted from 245]:

$$RMi:\ \sigma_{cm}=\sigma_c\times JP \tag{54}$$

$$GSI:\ \sigma_{cm}=\sigma_c\times s^a \tag{55}$$

Where s and a are the Hoek and Brown constants. Therefore, JP should be numerically equivalent to s^a and given that for undisturbed rock masses [245]:

$$s\ =\ exp.\left[(GSI-100)/9\right] \tag{56}$$

$$a=(1/2)+(1/6)\times\left[exp.(-GSI/15)-exp.(-20/3)\right] \tag{57}$$

Then, a direct correlation between JP and GSI obtained from [245]:

$$JP=\left[exp.((GSI-100)/9)\right]^{(1/2)+(1/6)\times[exp.(-GSI/15)-exp.(-20/3)]} \tag{58}$$

Therefore, we have the following relations according to Eqs. (42), (46), (54), (55), (56), (57) and (58):

$$RMi:\ \sigma_{cm}=\sigma_c\times JP=\sigma_c\times s^a=\sigma_c\times(exp.^{[(GSI-100)/9]})^{((1/2)+(1/6)\times[exp.^{(-GSI/15)}-exp.^{(-20/3)}])} \tag{59}$$

$$e=ln\left(\frac{\mathrm{g_0}}{\mathrm{X_{C-earth}}}\right)=ln\left(\frac{1}{A.D}\right)=1^{\frac{\frac{logS}{GSI-100}}{9}}=\sqrt{\frac{\mathrm{h_{OS}}\times\mathrm{X_{CVLN}}\times\left(H+D+\dfrac{\beta}{100}\right)\times(H+D)}{\overline{RMi}\times\mathrm{FS_{fd}}\times\left[\dfrac{(\alpha+\beta+\delta)}{100}+\dfrac{(H_{IR}+H_{BF})}{100}+\dfrac{(W_{GB}+W_{SP.\ B}+W_{SB})}{100}\right]}}$$

$$=\sqrt[2e]{\left(\frac{\mathrm{h_{OS}}^{2}\times\mathrm{X_{CVLN}}\times\left[\dfrac{\left(H+D+\dfrac{\beta}{100}\right)}{n}+\dfrac{IRA}{100}\right]^{2}\times(H+D)\times\overline{T}\times0.888}{\overline{RMi}\times\mathrm{FS_{fd}}\times RMi_{Sum}\times\overline{C}_{components\ of\ slopes\,(c\ of\ s)}\times\overline{\varnothing}_{c\ of\ s}\times C_{imin}}\right)} \tag{60}$$

Eq. (60) can use in reliability procedures of engineering decisions as a proof of relations in Section 8.1 and Section 8.2.

9. Validation of Recommended Formulas: Comparison of SLOPE/W Predictions and Experimental Formulations Results

All offered empirical equations can use to ensure the reliability of slope stability assessment by other methods. For this purpose, limit equilibrium SLOPE/W stability analysis of the slope carried out based on the section RS06 with contemplate various rock mass materials. FOS as a discriminant of stability limit help to determine suitable slope angles that slope consider actively fully drained in this study. The resulting FOS by SLOPE/W is high (1.97) indicating stable conditions for the slope stability analysis. Furthermore, safety factor values obtained from recommended equations in this part are compared with acquired results from SLOPE/W software slope stability analysis (that was implemented by SRK consulting engineers and scientists group in 2008) for validation of presented formulas for the section RS06 of block five of Sungun copper mine with 37° overall slope angle.

Presence of water causes to create various instabilities conditions for slopes. This feature input and involves water-affecting factors in safety factor calculations. Effective fully drainage of the pit slopes will be the most critical parameter to ensure that the pit slopes remain stable. Block five of North sector (Section RS06) of Sungun Copper mine selected for the survey because this section is suitable for fully drained conditions.

Program SLOPE/W regulated and formularized in phrases of moment and force equilibrium of safety factor equations. Limit equilibrium methods used in this software includes Morgenstern-Price, General limit equilibrium, Spencer, Bishop, Ordinary, Janbu methods, etc. This program allows integration with other applications like finite element code [Adopted from 276]. Limit equilibrium analyses using the commercial software SLOPE/W carried out to reach follow scenario: • Slope fully drained to the toe. Standard deviation, percent of variance, variation coefficient, percent of variation coefficient, mean square error, root-mean-square error, correlation coefficient and percent of efficiency are used to comparison and validation of suggested empirical equations. The differences between obtained safety factor values through suggested equations correspond to the computational accuracy of the used parameters. Results of overall slope stability analysis of Sungun pit's north domain by SLOPE/W software demonstrate that the section RS06 is stable up to 37° overall slope angle.

Limit equilibrium SLOPE\W stability analysis model of overall pit slope on the section RS06 in block five of Sungun copper mine for fully drained condition by Janbu method and Mohr-Coulomb strength function with auto search of critical slip surface shown in Fig. 8. Table 5 presents the obtained FOS values from SLOPE/W software for the section RS06 of block five of Sungun copper mine for fully drained conditions and with $\beta=37^{\circ}$. Rock mass input parameters for SLOPE/W stability analysis of the section RS06 in the selected block of Sungun copper mine reported in Table 6. The overall slope angles and slope configurations for north pit domain (section RS06) recommended as a result of SLOPE/W stability analysis, which is included in Table 7. Used factors in FOS calculations through recommended formulas (for the section RS06 in block five of Sungun copper mine) displayed in Table 8. Comparisons results between factors of safety values of recommended equations with SLOPE/W software stability analysis presents in Table 9. Statistical equations and factors that used in validation shown in Table 10.

In Table 6, must note that Sungun Porphyry, Dyke 1a, Dyke 1b and Faults are main geological units of the section RS06 (in its cross-section) [226]. Main geological units of the section RS06 of Sungun copper project in slope stability analysis cross-section SLOPE/W model of overall pit slopes for fully drained condition (which shown in Fig. 8) were colour coded. Sungun Porphyry (SP) was marked green. Dykes 1a (DK1a) were marked blue. Dykes 1b (DK1b) was marked yellow. Faults (FLT) were marked red. Skarn (SK) was marked aqua. Statistical measuring in Table 9 used to measure the differences between predicted values by suggested estimator formulas and the values actually observed by SLOPE/W software. The results of the SLOPE/W analysis for the north domain in the section RS06 for fully drained conditions are as Table 5:

Table 5: Factor of safety values obtained from SLOPE/W software.

Section	Pit Domain	OSA: Overall slope angle (°)	Fully Drained Factor of Safety (FS)
RS06	North	37	1.97

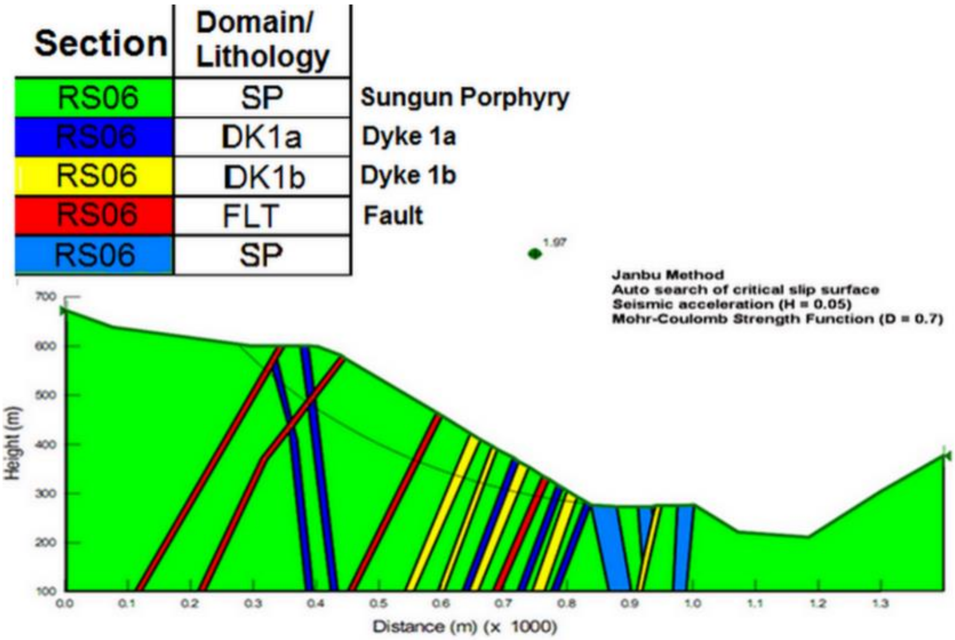


Figure 8. Limit equilibrium Slope/W stability analysis model of overall pit slope for section RS06 in fully drained condition with Janbu method.

Table 6: Rock mass input parameters for limit equilibrium SLOPE/W stability analysis.

Domain / Lithology		**Dyke 1a (DK1a)**	**Dyke 1b (DK1b)**	**Sungun Porphyry (SP: POT[b])**	**Fault (FLT)**
Mean uniaxial compressive strength of the intact rock material, pieces and elements, *UCS* mean, $\overline{sigci}$, $\overline{\sigma_{ci}}$ (MPa)	**Section RS06**	105	130	82	30
Geological strength index mean, *GSI Mean*		42	50	40	24
Intact rock parameter: A material constant of the intact rock, *mi* (Dimensionless)		23	20	18	18
Disturbance Factor, *D*		0.7	0.7	0.7	0.7
Unit weight of rock, γ (MN/m³)		0.024	0.024	0.023	0.023
Slope Height (m)		150	150	150	150
Cohesion, *C* (kPa)		931	1198	727	319
Friction angle, ∅ - Phi (°)		44	48	39	24
Rock mass tensile strength, Sig_t - σ_t (MPa)		-0.025	-0.072	-0.025	-0.004
Uniaxial compressive strength of the rock mass, Sig_{cm} - σ_{cm} (MPa)		13.09	19.00	9.03	2.54
Rock mass young's modulus, E_{rm} (MPa)		2743	5578	2142.6	512

[b] Potassic altered (potassic feldspar, quartz, secondary biotite). The rock mass at Sungun strongly affected by hydrothermal alteration. Potassic alteration is of considerable volume and importance to affect significantly the rock mass strength [80].

Note: The strength of intact rock obtained through statistical analysis of uniaxial compressive strength data is derived from rock core's laboratory testing and simple field strength estimates. The intact rock strength refers to the strength of a finite piece of rock, free of any defects, such as veins, cemented or un-cemented joints, micro fissures, or faults.

This intact rock strength usually derived from testing the unconfined compressive strength (UCS) of hand specimen or drill core samples. Intact rock strength, for which average values have been calculated using field estimate backed up by a comprehensive laboratory testing programme of UCS and point load tests [281].

Table 7: Recommended overall slope angles and slope configurations for fully drained slope in section RS06.

Pit Domain	North
Section	**RS06**
Max. Overall Slope Height (m)	300
Overall Slope Angle (°) - crest to toe	37
Inter Ramp Height (m)	100
Inter Ramp Slope Angle (°) crest to toe	45
Bench Face Angle	65
Geotechnical Berm Width (m)	50
Spill Berm Width (m)	24
Safety Berm Width (m)	14
Bench Height (m)	25

Table 8: Input factors of slope stability analysis for suggested computational formulas in section RS06.

Factors	For Section RS06	Factors	For Section RS06
$\overline{RMI}$	3.122937975 Mpa	H	0.05
RMI_{sum}	740.1363 Mpa	D	0.7
h_{OS}= OSH	300 m	e	2.718281828
n	4 components	$\bar{T} = \bar{\sigma}_t$	0. 007 Mpa
$\bar{C}_i$	793.75 kPa	X_1	0.000008832
C_{imax}	1198 kPa	α= IRA	45°
C_{imin}	319 kPa	δ= BFA	65°
$\bar{\emptyset}_i$	38.75°	β= OSA	37°
$\emptyset_{imax}$	48°	H_{IR}	100 m
$\emptyset_{imin}$	24°	H_{BF}	25 m
$\bar{\gamma}_{rock}$	0.0235 MN/m^3	W_{GB}	50 m
$\bar{V} = \frac{\overline{RMi}}{RMI_{Sum}}$	0.00421940928312798602095316768006≅ 0.0042	$W_{SP.\ B}$	24 m
$X_{C-Variable} = X_{CVLN}$	0.65133593217841457973135724564648 Mpa	W_{SB}	14 m

Table 9: Comparison and validation results between two calculated safety factors (slope stability analysis by new empirical formulas using RMi system and SLOPE/W stability assessment by Janbu method) for section RS06 with $\beta = 37^{\circ}$.

Section	Equations	FS obtained from suggested equations	FS from Slope/w	FS mean ($\overline{FS}$)	Standard Deviation (SD or S)	Variance (VAF or Var)	Variation coefficient (V)	Percent of Variation coefficient (V%)	Mean Square Error (MSE)	Root Mean Square Error (RMSE)	Coefficient Of Correlation (R^2)	Efficiency % (E)
RS06	Eq. (43)	1.9758354	1.97	1.9729177	0.0029177	8.51E-06	0.0014789	0.15%	3.41E-05	0.0058354	0.9941646	99.42%
	Eq. (47)	1.9701806	1.97	1.9700903	9.03E-05	8.16E-09	4.58E-05	0.0046%	3.26E-08	0.0001806	0.9998194	99.98%
	Eq. (48)	1.980957	1.97	1.9754785	0.0054785	3.00E-05	0.0027733	0.28%	1.20E-04	0.010957	0.989043	98.90%

	Eq. (50)	1.9770906	1.97	1.9735453	0.0035453	1.26E-05	0.0017964	0.18%	5.03E-05	0.0070906	0.9929094	99.29%
	Eq. (51)	1.9649595	1.97	1.9674797	0.0025203	6.35E-06	0.001281	0.13%	2.54E-05	0.0050405	0.9949595	99.50%
	Eq. (52)	1.9706294	1.97	1.9703147	3.15E-04	9.90E-08	1.60E-04	0.02%	3.96E-07	0.0006294	0.9993706	99.94%
	Eq. (53)	1.9738227	1.97	1.9719113	0.0019113	3.653E-06	9.69E-04	0.10%	1.461E-05	0.0038227	0.9961773	99.62%

Table 10: Used statistical equations and factors in validation.

Statistic parameters	Statistic formulas
Mean factor of safety	$\overline{FS} = \frac{1}{n}\sum_{i=1}^{n} FS_i$
Standard Deviation = SD = S	Standard Deviation $= \sqrt{Variance} = \sqrt{\overline{FS^2} - \overline{FS}^2}$
Variance = VAF = Var = S^2	$Variance = \overline{FS^2} - \overline{FS}^2$
Variation coefficient = Coefficient of variation = V =CV	Variation coefficient $= \frac{\mathrm{SD}}{\overline{FS}}$
Percent of variation coefficient = V%	Percent of variation coefficient $= \frac{\mathrm{SD}}{\overline{FS}} \times 100$
Mean Square Error = MSE	MSE $= \frac{1}{n}\sum_{i=1}^{n}\left(FS_{Suggested} - FS_{SLOPEW}\right)^2$ [c]
Root Mean Square Error = RMSE	RMSE $= \sqrt{\frac{1}{n}\sum_{i=1}^{n}\left(FS_{Suggested} - FS_{SLOPEW}\right)^2}$ [c]
Correlation Coefficient =R^2	Correlation Coefficient = 1- RMSE [d]
Efficiency = E	Efficiency = [Correlation Coefficient] × 100 [e]

[c] $FS_{Suggested}$ is the factor of safety predicted by suggested empirical equations and FS_{SLOPEW} is the calculated (observed) safety factor by SLOPE/W software.

[d] This is the correlation coefficient in accordance to root mean square error.

[e] This is the real efficiency which result by correlation coefficient.

Variance is a good criterion to show the dispersion and variability of calculated safety factor through proposed relations and SLOPE/W results in comparison with mean safety factor. In Fig. 9, minimum variances shown that demonstrates all safety factors are close to mean FOS. The mean square error (MSE) of the calculator relations evaluates the average of the squares of the "errors", which is the difference between the estimator formulas and SLOPE/W assessment results. Fig. 9 has shown a risk function that related to the anticipated value of the squared error loss or quadratic loss. Less mean square error or deviation values illustrate more accurate slope stability assessment because these empirical relations account the information that could produce a more efficient and precise estimation. Root-mean-square deviation (RMSD) is the second instant of the error between the supposed formulas and SLOPE/W estimation results which incorporates both the variance of the estimator formulas, SLOPE/W results and their bias.

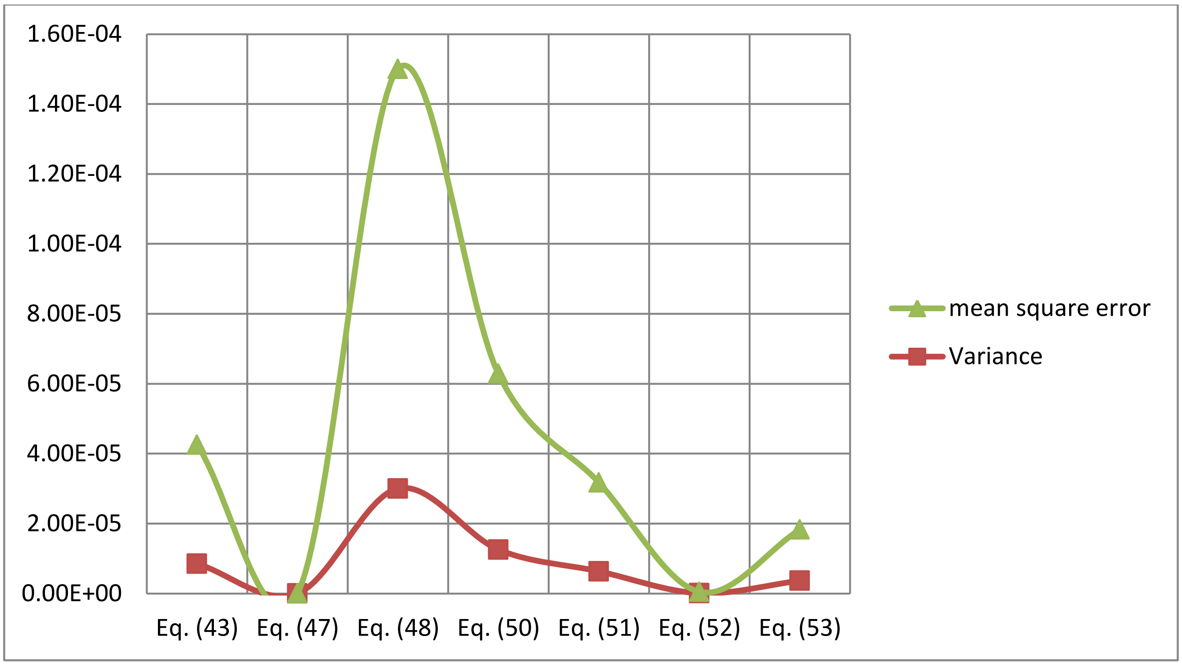


Figure 8. Limit equilibrium Slope/W stability analysis model of overall pit slope for section RS06 in fully drained condition with Janbu method.

In Fig. 10, less root mean square error as a quantitative comparison demonstrates a good measure of accuracy of presented formulas. RMSE express the typical standard deviation of the differences between predicted values by relations and computed values by SLOPE/W. Standard deviation is a measure that is used to quantify the amount of variation or dispersion of computed FOS values by empirical relations and SLOPE/W software. Obtained standard deviations is close to zero, which indicates the computed safety factors through equations and SLOPE/W tend to be very close to the mean FOS values that are shown in Fig. 10.

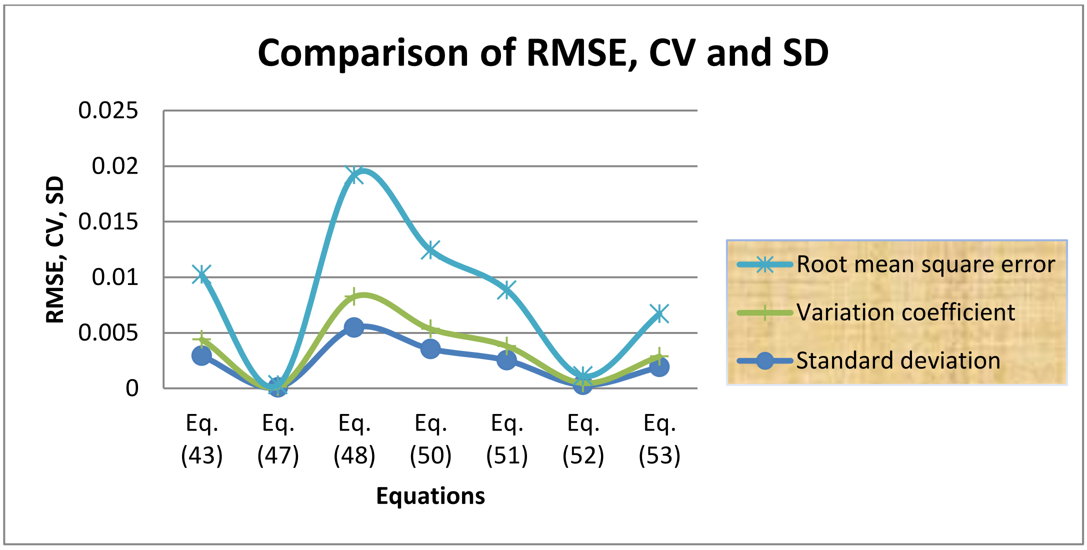


Figure 10. Scatter curves for the standard deviation, variation coefficient and root mean square error according to comparison between suggested empirical relations and SLOPE/W results with that listed in Table 8.

The coefficient of variation (CV) which is also known as unitized risk is a standardized measure of dispersion of safety factors computed by relations and SLOPE/W results. Less percent of variation coefficient that defines low relative standard deviation (RSD) proves these relations are qualified for safety factor calculation, which displayed in Figs. 11(a) and 11(b).

Comparison between SLOPE/W results and mentioned equations illustrate less variation coefficients percent of them as Figs. 11(a) and 11(b) that presents high efficiency of offered formulas as shows as Fig. 12.

Efficiency is often determined the measure of desirability of suggested empirical relations. It measures empirical estimator formulas' optimality in slope stability procedures. More efficient estimator formulas of safety factor achieve a given performance of slope stability analysis.

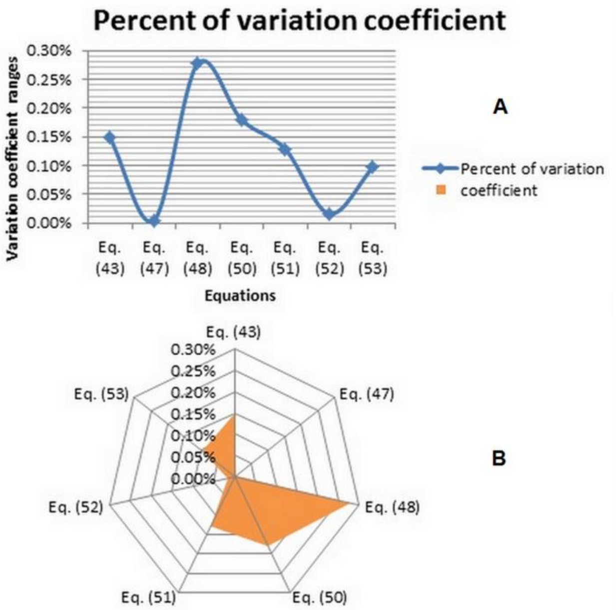


Figure 11. Percent of variation coefficient stacked curve (A) and filled radar chart (B) according to comparison between suggested empirical relations and SLOPE/W result, which listed in Table 8.

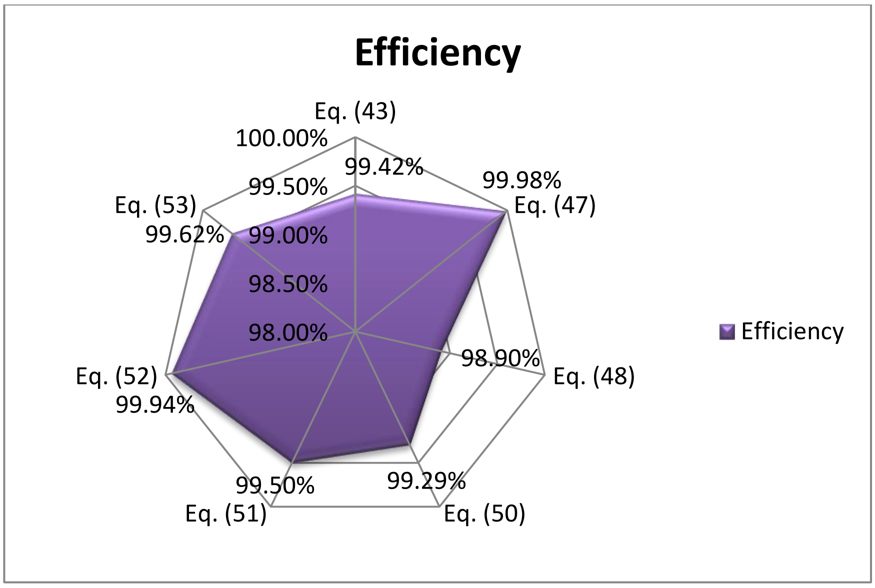


Figure 12. Efficiency chart based on comparison between suggested empirical relations and SLOPE/W result that listed in Table 8.

The strength of association between proposed relations and SLOPE/W analysis result, which defined as correlation coefficient shown in Fig. 13**.** This curve indicates large amounts of correlation between new relations and SLOE/W results. All correlation coefficients are close to one, it would indicate that the relations are positively related and the scatter plot falls almost with a positive slope.

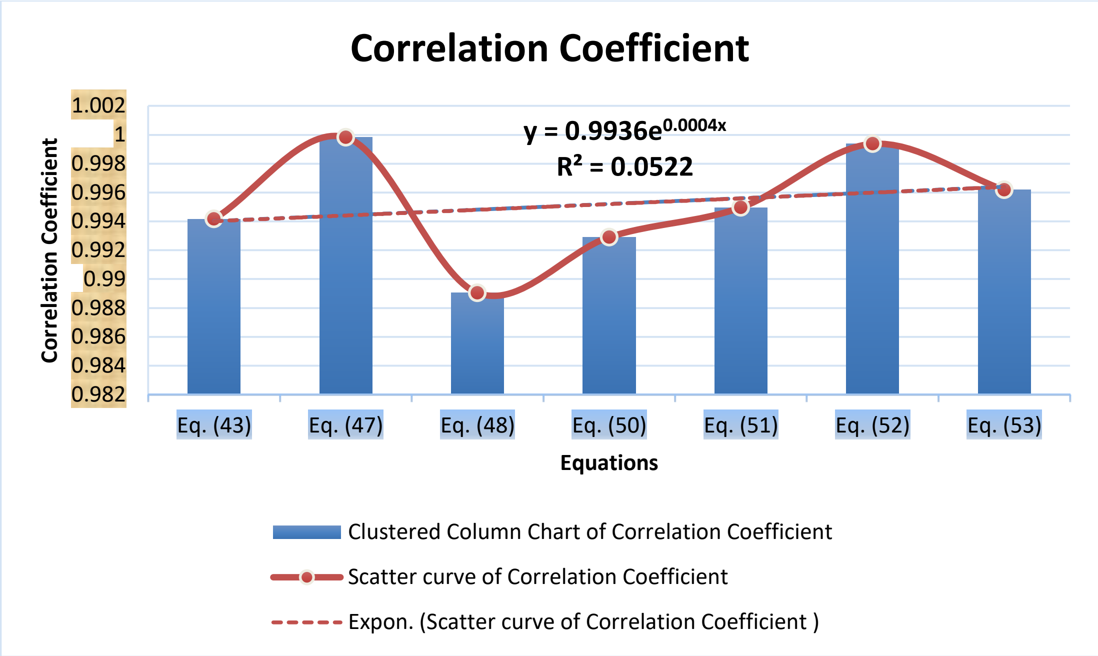
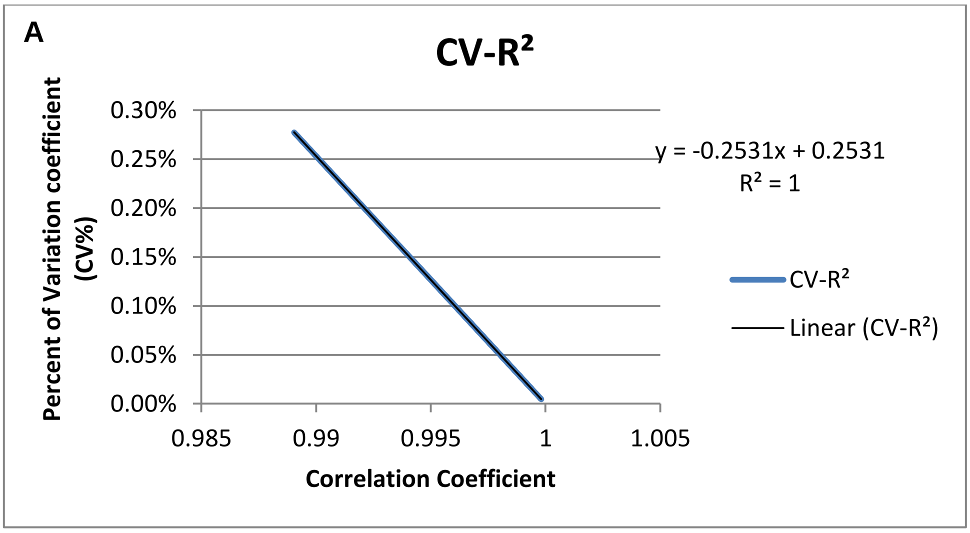
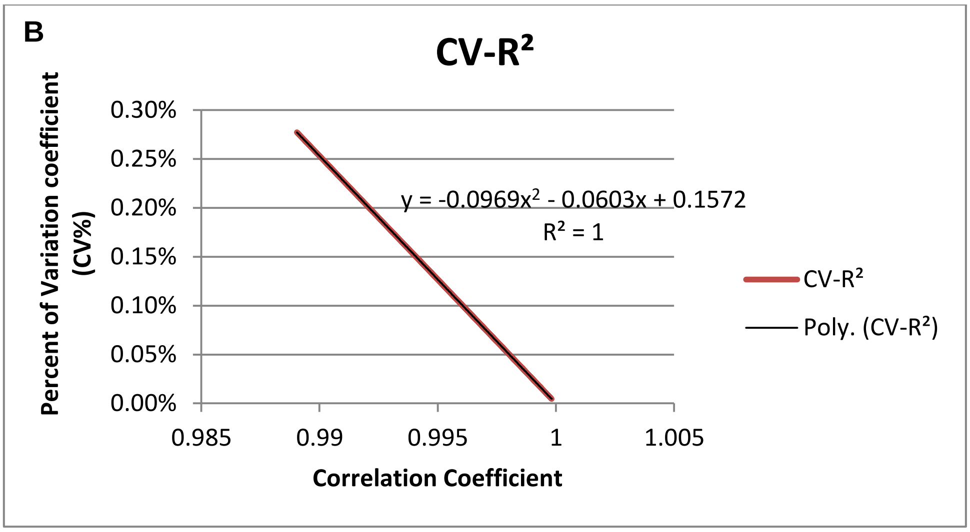


Figure 13. Correlation coefficients scatter curve and column charts based on comparison between suggested empirical relations and SLOPE/W result, which listed in Table 8.

High correlation between percent of variation coefficient and correlation coefficient with linear, logarithmic and polynomial trend line (regression) in Fig. 14 shows very high accuracy of recommended relations in calculations.





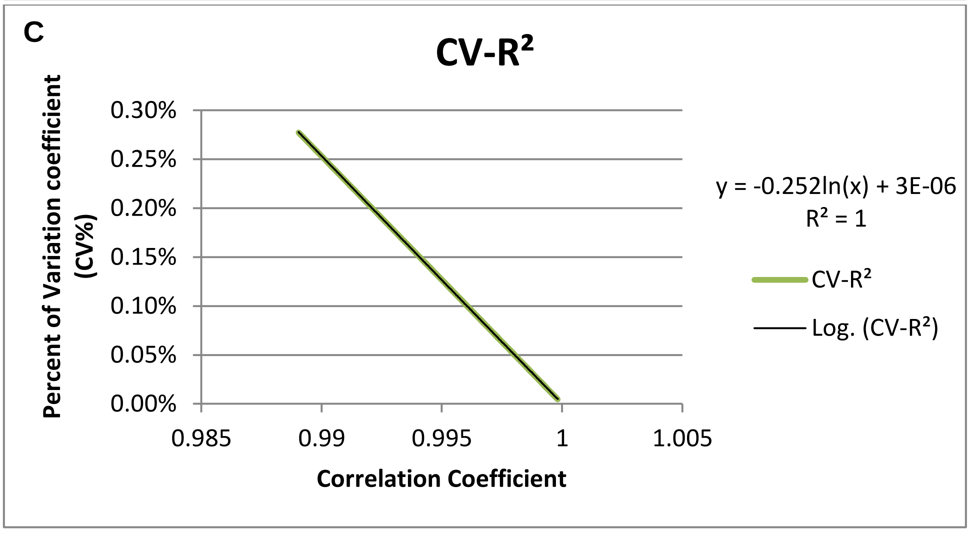
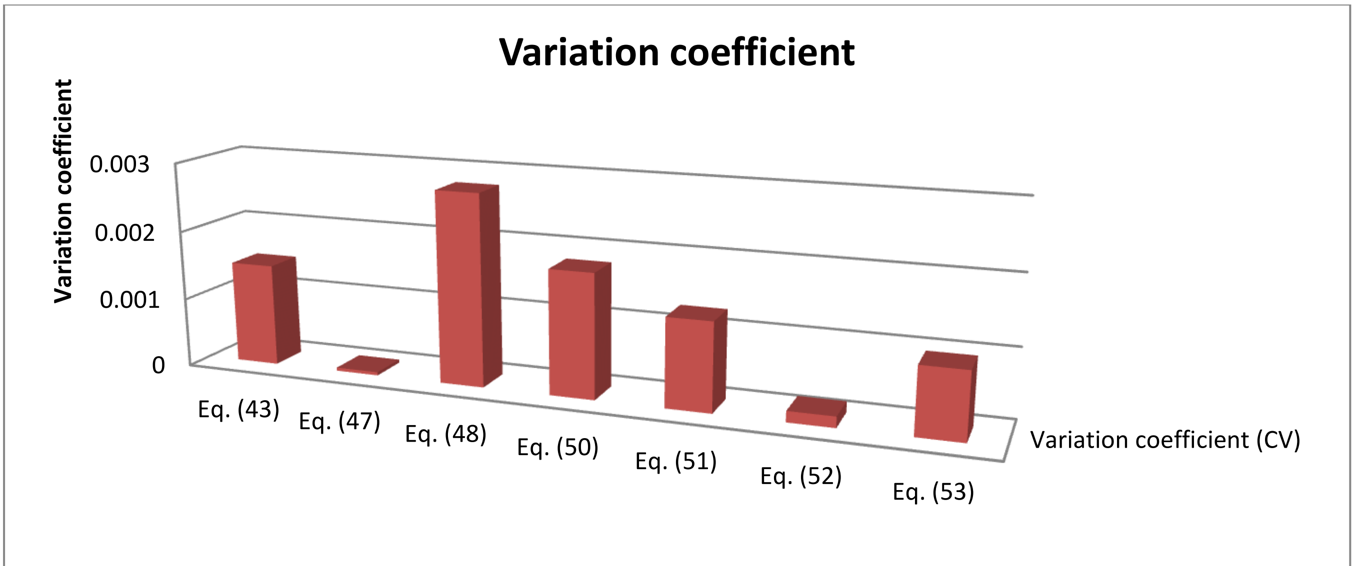


Figure 14. Correlation between percent of variation and correlation coefficients in accordance to comparison between suggested empirical relations and SLOPE/W result, which listed in Table 8, A. Linear regression, B. Polynomial regression, C. logarithmic regression.

High correlation coefficient and low CV indicate very negligible unitized risk of supposed relations and high reliability of obtained results that represented in Figs. 13 and 15.



	Eq. (43)	Eq. (47)	Eq. (48)	Eq. (50)	Eq. (51)	Eq. (52)	Eq. (53)
Variation coefficient (CV)	0.001478867	4.58E-05	0.002773261	0.001796415	0.001280965	1.60E-04	9.69E-04

Figure 15. 3-D column chart of variation coefficients based on comparison between suggested empirical relations and SLOPE/W result, which listed in Table 8.

10. Discussion of Results

In order to increase investments in mines' fields also heighten the open pit mines depth, whereby mines extraction tends to grow the factors of safety with a principle, effective and rational reason that is slopes stability problems intensification. Slope stability analysis through empirical relations require to quick analysis and primary estimates in different stages of design particular in primary design, design revision, modification, adjustment and optimization even encounter with unforeseen cases and emergency conditions. Besides, the deficiency of different classification systems in mentioned situations, it requires implements complete and quick analysis of slopes' safety factor through basis earth's strength limit specifications which covers all conditions or at least assess all-important and affecting conditions and items in calculations without any limitations toward to reduce error possibility. Moreover, the intended significant central importance effects of the earth strength limit functional factor's applications besides the covering some of the most important ground parameters therefore, using the RMi classification system in empirical safety factor calculator formulas lead to better computations for different reasons. Such as considering of the three-dimensional block volume in RMi determination that will universally improve the description of the features of the rock mass.

Strength limit of earth shows how fundamental factors of nature such as gravity also rock mass classification systems like RMi, impact on scrutinizing slopes stability assessments. This elementary factor and Napier's constant (or Euler's constant) uncovers the relationship between fundamental physics, geo-mechanics and rock slope engineering also other sciences too (For more information refers to: Salehi Alashti, 2017).

The slope stability against failure risk represented by safety factor, which is determined using recommended relations and limit equilibrium method by SLOPE/W software. It is found that the factor of safety in selected area is higher than 1.9 when encounters with fully drained condition and seismic acceleration effects, which also displays slope is still stable. The minimum value obtained for FOS is about 1.96748. The following discussions derived from the presented results:

1) The stability of the section RS06 is acceptable because of the minimum factor of safety is more than 1.95.
2) Analysis the stability of rock slopes by these procedural equations' method provides detailed information and an independent approach for determining the safety factor of the selected section even based on both total and effective driving actions.
3) The case study that presented above illustrates that application of these relational processes made to reduce the risk of failure.
4) These easily calculable relations can use in uncertainties occurrence moments even help predict the appearance of them and uncover the silent risks.

5) The equations give some precise results for factors of safety calculation in comparison with the SLOPE/W limit equilibrium method predictably.

11. Summary and Conclusions

In this paper, previous slope stability investigations and approaches reviewed based on different methods, especially existing rock mass classification systems. This found mentioned methods contain various limitations and lacks evolutions. Therefore, these systematic procedures could improve through substantial factors like strength limit of earth, which can be including all aspects. In addition, better-improved techniques needed for slope stability evaluation via rock mass classification systems. In the direction of reaching presented purposes, some definitions of the earth's strength limit substantial factor described in detail also discussed precisely. Then, new empirical relations suggested by contemplating the beneficial roles of RMi classification system based on strength limit of earth fundamental factor which envisage most principal effectors to decrease errors and limitations for analysis of slope stability accurately, quickly, safely and efficiently, especially in un-seeming, un-detected and unforeseen conditions. This could influence slope stability engineering, stimulation, design and modelling also leads to entirely different approaches for rock engineering design particular rock slope engineering. Offered relations can apply in reliability analysis of slopes designs projects.

The explanation about cognizing the seismic wave's behaviour in rock mass and the connections between the rock mass parameters and the seismic factors are often vague. Strength limit, area density and gravity of earth help discovers the relations between the unit weight of rock as rock mass parameters and the seismic acceleration as seismic parameters on a slope.

Less percent of variance in recommended equations show that the results of calculated safety factors estimate with high precision. Low values of variation coefficient, root-mean-square error, standard deviation and variance illustrate errors reduction, which cause bettering safety factor calculations. High value of correlation coefficient tend to one indicates high accuracy and solidarity of these formulas. Low values of mean square error, percent of variance, standard deviation and percent of variation coefficient also high efficiency and correlation coefficient, represents that obtained factor of safety from recommended equations have high correlation and very low dispersion in its comparison with results of SLOPE/W software's limit equilibrium stability analysis. Therefore, engineers can be relying on the presented formulas, which has a high reliability degree.

Recommended formulas for safety factor calculation in fully drained conditions on section RS06 in block five of Sungun copper mine with 37° overall slope angle can apply to assess the reliability of the obtained factors of safety by other various methods. These applications have some benefits as follows:

(1) Specialist knowledge of rock engineering not required. However, as a minimum, the inspectors would need to be mine engineers or and engineering geologists or geotechnical engineers. (2) Quick (time saver) — typically takes no more times for calculations. (3) Cost effectiveness. (4) Safer. (5) Labor-saving. (6) Simple. (7) More environmentally friendly. (8) These applications have many practical advantages and includes precise approaches. (9) Easy for engineering geologists, mine engineer and geotechnical engineers to understand — no need for external training course, all necessary training information is available online and an engineering geologist or mine engineer or geotechnical engineer would be able to train themselves via self-directed learning. (10) Widely adopted. (11) Designed to accommodate both harder and weaker rock slopes. (12) The used RMi system recommended by Palmström also cited in other publications. (13) Using of RMi classification system cover most important ground parameters, which lead to better estimates. (14) Research into the system is ongoing which demonstrates its wide acceptance, viability and potential for long-term improvement. (15) This process does not require any equipment. (16) The process has been refined over many times to its present level of development. (17) This process does not require assumptions.

Finally, and briefly, can note that:

1. An empirical formulated procedure originally developed based on the fundamental parameter to assess (assessing) the stability of rock slopes such as mine's walls that has described to be capable of generating feasible rapid independent solutions for instability problems of rock slopes.
2. Importance of RMi introduces through earth's strength limit factor in rock slope engineering with precise simple equations.
3. A significant advantage of the suggested limit equilibrium empirical relations and independent equations compared with the analysis of SLOPE/W software, which have commonly applied for rock slope stability's problems that it is generally applicable, and will not require any assumptions to be made even about how forces act in the system.
4. The new ascertainable analytic formulas have discovered to provide suitable, acceptable, applicable, rational, albeit somewhat cautious safely estimates of the rock slopes stability.

5. New relations determine procedures that can provide solutions for fully drained conditions in a particular assessment mechanism of rock slope stability that have illustrated. When this process used in combination with other proposed valid methods and analysis procedures, these makes solutions that prosperously cover and bracket experimental data involved in design.

Strength Limit factor arise and enlarge considerable interests in various areas of sciences. Also, will using as a basis in new scientific discoveries. Moreover, this key substantial number applying for rock slopes engineering developments. Furthermore, this primordial fundament will be using as a bridge between fundamental applied physics and practical applications in the fields of geological engineering, geo-mechanics, rock mechanics, rock engineering, mining engineering, environmental hazards assessments and earth sciences.

Generally, application of strength limit of earth fundamental factor and RMi system in safety factor determination implies the unity, correlation, continuity and dependency all of the existence parts (like the universe and earth).

Highlights

- Earth’s strength limit (ESL) and RMi open new gates between fundamental physics and Earth sciences.
- Earth’s strength limit indicates the relationship between FOS and RMi.
- Strength Limit presents a rigorous mathematical approach to study slope’s problems.
- RMi provides slope stability analysis accurately, quickly, safely and efficiently.
- Strength limit of earth (SLE) displays determinant role of RMi in scrutinizing slope stability assessment.

Acronyms, Nomenclature, Abbreviations and Notations

Symbols: Meanings, SI Units;

$\mathbf{X_{c\text{-}earth}}$: Strength limit number of earth, MPa & % & Dimensionless;
ESL: Earth’s strength limit, MPa & % & Dimensionless;
SLE: Strength limit of earth, MPa & % & Dimensionless;
$\mathbf{X_{C-Earth,Old}}$: Strength limit parameter of the earth according to old standard gravity, MPa & % & Dimensionless;
$\mathbf{X_{C-Earth,Now}}$: Strength limit parameter of earth or strength of existence global gravitation according to present standard gravity, MPa & % & Dimensionless;
$\mathbf{X_{CV-Slope}}$ or $\mathbf{X_{CV(Slope)}}$ or $\mathbf{X_{CVLN-Slope}}$: Cohesion variable limit number for slope, MPa;
$\mathbf{X_{CCLN}}$: Cohesion constant limit number, MPa;
$\boldsymbol{g_0}$: Standard gravitational acceleration or Earth's standard surface gravity, m/s^2;
$\mathbf{g_{0-Old}}$: Old standard gravity, m/s^2;
$\mathbf{g_{0-Now}}$: Present value of standard gravity, m/s^2;
$\mathbf{AOE_{Old}}$: Age of the earth according to old standard gravity, years & Billion years;
$\mathbf{AOE_{Exact}}$: Accurate or exact age of the earth according to present standard gravity at present time, years & Billion years;
RMi: Rock Mass index, MPa;
FS or **FOS** or **SF** or **SOF**: safety factor;
MEARS: Minimal effective action and reaction unit of strength, MPa;
$\mathbf{D.C}$: Decay constant of gravitational particles;
þ: Strength variations coefficient in 10^{10} year, $\frac{\text{MPa}}{\text{Year}} = \frac{\text{MN}}{\text{m}^2\text{Year}} = \frac{\text{Ggr}}{\text{Year m s}^2}$;
$\mathbf{\overline{V}_{Assessing\ Mine\ Block}}$ or $\mathbf{\overline{V}}$ or $\mathbf{\overline{V}_{MB}}$: Rock mass index values average to sum of the rock mass index values of the assessing mine block ratio, Dimensionless;
$\mathbf{X_1}$: Detector parameter, Dimensionless;
FS mean ($\overline{FS}$): Mean factor of safety, Dimensionless;
$\mathbf{FS_{fd}}$; Factor of safety for fully drained conditions, Dimensionless;
OSH or $\mathbf{H_{OS}}$ or $\mathbf{h_{OS}}$: Overall slope height or is the maximum overall slope height, Meter (m);
n: Number of components in a slope (number of slope’s components), Dimensionless;
$\mathbf{\overline{RMI}}$ or $\mathbf{RMI_{(av)}}$: Average of rock mass index values for assessing mine block, MPa;
$\mathbf{RMi_{Sum}}$: Sum of rock mass index values obtained from every approved components of a discovery boreholes in a slope for assessing mine block, MPa;
e: Napier number, Dimensionless;
H: Seismic acceleration, Dimensionless;
D: Disturbance factor, Dimensionless;
OSA or $\boldsymbol{\beta}$: Overall slope angle, Degree;
IRA or $\boldsymbol{\alpha}$: Inter ramp angle, Degree;
BFA or Δ or $\boldsymbol{\delta}$: Bench face angle, Degree;

BFH or $\mathbf{H_{BF}}$: Bench face height, Meter (m);
SP.BW or $\mathbf{W_{SP.B}}$: Spill berm width, Meter (m);
SBW or $\mathbf{W_{SB}}$: Safety berm width, Meter (m);
IRH or $\mathbf{H_{IR}}$: Inter ramp height, Meter (m);
GBW or $\mathbf{W_{GB}}$: Geotechnical Berm Width, Meter (m);
$\bar{\mathbf{C}}_{\mathbf{components\ of\ slopes\ (c\ of\ s)}}$: Average of cohesion values of slope components (average of a slope components cohesion values), MPa;
$\mathbf{C_{imin}}$:Minimum cohesion value of slope components, kPa;
$\bar{\emptyset}_{\mathbf{c\ of\ s}}$: Average of internal friction angle values of slope components, Degree;
$\bar{\mathbf{T}}$: Average of rock mass tensile strength in assessing mine block, MPa;
$\boldsymbol{\gamma}$: Unit weight of rock in assessing mine block, $\frac{MN}{m^3}$;
$\mathbf{X_{CVLN}}$: Cohesion variable limit number, MPa;
C: Cohesion, MPa;
Phi: Friction angle, Degree;
$\mathbf{g_h}$: Gravity at height h above sea level, m/s^2;
$\mathbf{g_0}$: Standard gravity, m/s^2;
$\mathbf{r_e}$: Radius of earth;
h: Height from sea level;
F: Force between the masses, N;
G: Gravitational constant or Newtonian gravitational constant or Newtonian constant of gravitation, $m^3kg^{-1}s^{-2}$ & N $m^2\ kg^{-2}$;
$\boldsymbol{\mathcal{M}_{\oplus}}$ **or E**: Mass of the earth, Kg;
M: Every point mass;
R: Distance between the masses, Meter (m);
ED: Earth density, Kg/m^3 & gr/cm^3;
$\mathbf{V_{earth}}$: Volume of earth, m^3 & km^3 & cm^3 & cu mi;
A.D or $\boldsymbol{D_A}$ **or S.D or** $\boldsymbol{D_S}$: Area density, Ggr/m^2;
$\mathbf{R'}$**or** $\mathbf{R_{Earth-Dimensionless}}$**or Dimensionless R:** Ratio of every point mass M to mass m or Quantitative radius value of earth without unit, Dimensionless;
$\mathbf{R''}$**:** m^2;
$\mathbf{R'_{Earth}}$**:** m^2;
$\mathbf{g}_{\boldsymbol{V}}$**:** Variable standard gravity, m/s^2;
$\mathbf{E}_{\boldsymbol{V-Earth}}$**:** Variable mass of the earth, Kg;
$\mathbf{R_{V-Earth}}$**:** Variable distance between the masses, m;
$\mathbf{ED}_{\boldsymbol{V-Earth}}$**:** Variable earth density, Kg/m^3;
$\mathbf{V_{V-Earth}}$**:** Variable volume of earth, m^3;
$\mathbf{G}_{\boldsymbol{V}}$**:** Gravitational variation, $m^3kg^{-1}s^{-2}$ & N $m^2\ kg^{-2}$;
$\mathbf{AD}_{\boldsymbol{V}}$**:** Variable area density of earth, Ggr/m^2;
CV: Coefficient of variation;
MSE: Mean square error;
RMSD: Root-mean-square deviation;
SD or **S**: Standard Deviation;
VAF or **Var** or $\mathbf{S^2}$: Variance;
V or **CV**: Variation coefficient (Coefficient of variation);
V%: Percent of variation coefficient;
$\mathbf{R^2}$: Correlation Coefficient;
E: Efficiency;
NICICO: National Iranian Copper Industries Company;
PGA: Peak ground acceleration;
MCL: Maximum credible level;
MDL: Maximum design level;
DBL: Design basis level;
CL: Construction Level;
PFPN: Principle Fundamental Pivot Numbers;
LEM: Limit equilibrium methods.

Declarations

Interests

The author declares that he (author) is the 'owner' of all interests of this article forever.

Conflict of Interest

The author confirms that there are no known conflicts and competing of interest related to this publication.

Author's Descriptions

Mohammad Salehi Alashti (M.S.A) had the idea for the project, designed, performed and interpreted the research study. M.S.A analysed the data, drafted and wrote the paper also reviewed the manuscript. The author read and approved the final manuscript.

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APPENDIXES

Table A.1: Calculated RMi values for assessing zone via borehole 15 (GT15).

From	to	Length	GSI	S	a	$JP = S^a$	σ_c	RMi	RMi Classifications
0	1.6	1.6	0	1.49453E-05	0.666454561	0.000608173	82	0.04987	Low
1.6	3.3	1.7	0	1.49453E-05	0.666454561	0.000608173	82	0.04987	Low
3.3	4.2	0.9	23	0.000192473	0.535757075	0.010217029	82	0.837796	Medium
4.2	4.8	0.6	14	7.08068E-05	0.565328015	0.004507452	82	0.369611	Medium
4.8	6.4	1.6	20	0.000137913	0.543720751	0.007962014	82	0.652885	Medium
6.4	6.6	0.2	24	0.000215092	0.533437314	0.011058214	82	0.906774	Medium
6.6	7.2	0.6	61	0.013123729	0.502643629	0.113253978	82	9.286826	High
7.2	8.4	1.2	26	0.000268617	0.529236969	0.012887416	82	1.056768	High
8.4	9.9	1.5	15	7.91279E-05	0.561101135	0.004995147	82	0.409602	Medium
9.9	11	1.1	14	7.08068E-05	0.565328015	0.004507452	82	0.369611	Medium
11	12.1	1.1	0	1.49453E-05	0.666454561	0.000608173	82	0.04987	Low
12.1	12.7	0.6	15	7.91279E-05	0.561101135	0.004995147	82	0.409602	Medium
12.7	13.5	0.8	16	8.8427E-05	0.557146859	0.005516358	82	0.452341	Medium
13.5	14.7	1.2	20	0.000137913	0.543720751	0.007962014	82	0.652885	Medium
14.7	16.1	1.4	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
16.1	17.4	1.3	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
17.4	19.2	1.8	39	0.001138803	0.512166824	0.03107496	82	2.548147	High
19.2	19.8	0.6	30	0.000418942	0.522343775	0.01720296	82	1.410643	High
19.8	21.4	1.55	34	0.000653392	0.517064082	0.022554893	82	1.849501	High
21.4	22	0.6	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
22	23.3	1.35	42	0.001589327	0.509922905	0.037396812	82	3.066539	High
23.3	24.7	1.4	38	0.001019045	0.513020216	0.029183866	82	2.393077	High
24.7	25.6	0.9	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
25.6	26.2	0.6	12	5.66977E-05	0.574676055	0.003628069	82	0.297502	Medium
26.2	27	0.8	23	0.000192473	0.535757075	0.010217029	82	0.837796	Medium
27	28.1	1.05	26	0.000268617	0.529236969	0.012887416	82	1.056768	High
28.1	29.7	1.6	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
29.7	30.9	1.25	38	0.001019045	0.513020216	0.029183866	82	2.393077	High
30.9	31.2	0.3	39	0.001138803	0.512166824	0.03107496	105	3.262871	High
31.2	33.6	2.4	45	0.002218085	0.508085739	0.044825935	105	4.706723	High
33.6	35.2	1.55	37	0.000911882	0.513932441	0.027391375	105	2.876094	High
35.2	36.1	0.9	36	0.000815988	0.514907553	0.025692277	105	2.697689	High
36.1	37.2	1.15	34	0.000653392	0.517064082	0.022554893	105	2.368264	High
37.2	38	0.8	36	0.000815988	0.514907553	0.025692277	105	2.697689	High
38	39.6	1.6	34	0.000653392	0.517064082	0.022554893	105	2.368264	High
39.6	40.3	0.65	30	0.000418942	0.522343775	0.01720296	105	1.806311	High
40.3	41.2	0.95	35	0.000730178	0.515949889	0.024081659	105	2.528574	High

41.2	41.6	0.4	27	0.000300185	0.527337709	0.013880161	82	1.138173	High
41.6	43.1	1.5	39	0.001138803	0.512166824	0.03107496	82	2.548147	High
43.1	43.2	0.1	0	1.49453E-05	0.666454561	0.000608173	105	0.063858	Low
43.2	44.8	1.55	62	0.014666017	0.502459454	0.119852259	105	12.58449	Very High
44.8	46.3	1.55	57	0.008414677	0.503516356	0.090203301	105	9.471347	High
46.3	47.8	1.45	65	0.020468076	0.501975182	0.141971959	105	14.90706	Very High
47.8	49.3	1.55	55	0.006737947	0.50404815	0.080440238	105	8.446225	High
49.3	50.8	1.5	61	0.013123729	0.502643629	0.113253978	105	11.89167	Very High
50.8	52.3	1.5	48	0.003095587	0.506581595	0.053561975	105	5.624007	High
52.3	52.4	0.1	27	0.000300185	0.527337709	0.013880161	105	1.457417	High
52.4	53.8	1.4	26	0.000268617	0.529236969	0.012887416	82	1.056768	High
53.8	55.3	1.45	36	0.000815988	0.514907553	0.025692277	82	2.106767	High
55.3	56.5	1.2	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
56.5	58	1.55	16	8.8427E-05	0.557146859	0.005516358	82	0.452341	Medium
58	59.5	1.5	22	0.000172232	0.538236758	0.009421899	82	0.772596	Medium
59.5	60	0.5	21	0.00015412	0.540887388	0.008670843	82	0.711009	Medium
60	61.4	1.35	21	0.00015412	0.540887388	0.008670843	82	0.711009	Medium
61.4	62.4	1	14	7.08068E-05	0.565328015	0.004507452	82	0.369611	Medium
62.4	62.5	0.15	22	0.000172232	0.538236758	0.009421899	82	0.772596	Medium
62.5	63.9	1.35	26	0.000268617	0.529236969	0.012887416	82	1.056768	High
63.9	65.2	1.35	26	0.000268617	0.529236969	0.012887416	82	1.056768	High
65.2	66.6	1.35	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
66.6	68.2	1.65	30	0.000418942	0.522343775	0.01720296	82	1.410643	High
68.2	69.9	1.65	22	0.000172232	0.538236758	0.009421899	82	0.772596	Medium
69.9	71.4	1.5	39	0.001138803	0.512166824	0.03107496	82	2.548147	High

Table A.1: Continued.

From	to	Length	GSI	S	a	$JP = S^a$	σ_c	RMi	RMi Classifications
71.4	73.1	1.7	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
73.1	74.8	1.7	42	0.001589327	0.509922905	0.037396812	82	3.066539	High
74.8	76.4	1.65	23	0.000192473	0.535757075	0.010217029	82	0.837796	Medium
76.4	77.2	0.75	13	6.33607E-05	0.569846292	0.004052124	82	0.332274	Medium
77.2	79.2	2.05	0	1.49453E-05	0.666454561	0.000608173	82	0.04987	Low
79.2	79.6	0.35	0	1.49453E-05	0.666454561	0.000608173	82	0.04987	Low
79.6	80.4	0.85	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
80.4	80.8	0.4	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
80.8	81.1	0.25	0	1.49453E-05	0.666454561	0.000608173	82	0.04987	Low
81.1	81.8	0.75	24	0.000215092	0.533437314	0.011058214	82	0.906774	Medium
81.8	83.4	1.6	33	0.000584681	0.518255087	0.021107619	82	1.730825	High
83.4	84.2	0.8	17	9.88188E-05	0.553447606	0.006072306	82	0.497929	Medium
84.2	85.6	1.35	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
85.6	86.8	1.2	56	0.007529784	0.503773393	0.085188218	82	6.985434	High
86.8	88.8	2	64	0.018315639	0.502125972	0.134189287	82	11.00352	Very High
88.8	89.8	1.05	47	0.002770053	0.507049947	0.050490969	82	4.140259	High
89.8	91.9	2.1	60	0.011743628	0.502840501	0.107008536	82	8.7747	High
91.9	92.8	0.9	50	0.00386592	0.50573356	0.060227219	82	4.938632	High
92.8	94.8	2	33	0.000584681	0.518255087	0.021107619	82	1.730825	High
94.8	95.8	1	48	0.003095587	0.506581595	0.053561975	82	4.392082	High
95.8	97.6	1.8	45	0.002218085	0.508085739	0.044825935	82	3.675727	High
97.6	98.8	1.2	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
98.8	100	1.4	34	0.000653392	0.517064082	0.022554893	82	1.849501	High
100	102	1.5	39	0.001138803	0.512166824	0.03107496	82	2.548147	High
102	104	1.95	35	0.000730178	0.515949889	0.024081659	82	1.974696	High
104	105	1.15	35	0.000730178	0.515949889	0.024081659	82	1.974696	High

105	107	1.9	33	0.000584681	0.518255087	0.021107619	82	1.730825	High
107	109	2.25	40	0.001272634	0.51136847	0.033070186	82	2.711755	High
109	111	1.8	40	0.001272634	0.51136847	0.033070186	82	2.711755	High
111	113	2.4	44	0.00198483	0.508657785	0.042214911	82	3.461623	High
113	115	1.6	51	0.004320239	0.505350106	0.06384158	82	5.23501	High
115	116	0.8	34	0.000653392	0.517064082	0.022554893	82	1.849501	High
116	117	1.15	73	0.049787068	0.501071059	0.222414354	82	18.23798	Very High
117	119	2	40	0.001272634	0.51136847	0.033070186	82	2.711755	High
119	120	1	49	0.003459377	0.506143449	0.056804132	82	4.657939	High
120	122	2	44	0.00198483	0.508657785	0.042214911	82	3.461623	High
122	123	1	48	0.003095587	0.506581595	0.053561975	82	4.392082	High
123	124	1.1	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
124	125	1.6	50	0.00386592	0.50573356	0.060227219	82	4.938632	High
125	127	1.65	43	0.001776104	0.509269267	0.039740994	82	3.258762	High
127	129	1.65	49	0.003459377	0.506143449	0.056804132	82	4.657939	High
129	130	1.5	27	0.000300185	0.527337709	0.013880161	82	1.138173	High
130	132	1.5	46	0.002478752	0.507550586	0.047581868	82	3.901713	High
132	133	1.5	38	0.001019045	0.513020216	0.029183866	82	2.393077	High
133	135	1.55	45	0.002218085	0.508085739	0.044825935	82	3.675727	High
135	136	1.55	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
136	138	1.4	47	0.002770053	0.507049947	0.050490969	82	4.140259	High
138	139	1.2	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
139	139	0.35	45	0.002218085	0.508085739	0.044825935	82	3.675727	High
139	140	0.7	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
140	141	0.75	24	0.000215092	0.533437314	0.011058214	82	0.906774	Medium
141	141	0.6	30	0.000418942	0.522343775	0.01720296	82	1.410643	High
141	142	0.9	32	0.000523195	0.519528199	0.019735738	82	1.618331	High
142	144	1.5	35	0.000730178	0.515949889	0.024081659	82	1.974696	High
144	145	1.6	38	0.001019045	0.513020216	0.029183866	82	2.393077	High
145	147	1.4	46	0.002478752	0.507550586	0.047581868	82	3.901713	High
147	149	2.05	62	0.014666017	0.502459454	0.119852259	82	9.827885	High
149	150	0.95	49	0.003459377	0.506143449	0.056804132	82	4.657939	High
150	151	1.3	41	0.001422193	0.510621603	0.035175401	82	2.884383	High
151	153	1.7	36	0.000815988	0.514907553	0.025692277	82	2.106767	High
153	154	1.6	46	0.002478752	0.507550586	0.047581868	82	3.901713	High
154	155	1.15	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
155	157	1.95	34	0.000653392	0.517064082	0.022554893	82	1.849501	High
157	159	1.3	47	0.002770053	0.507049947	0.050490969	82	4.140259	High
159	160	1.5	36	0.000815988	0.514907553	0.025692277	82	2.106767	High
160	162	1.5	59	0.01050866	0.503050945	0.101096828	82	8.28994	High

Table A.1: Continued.

From	to	Length	GSI	S	A	$JP = S^a$	σ_c	RMi	RMi Classifications
162	164	2	48	0.003095587	0.506581595	0.053561975	82	4.392082	High
164	166	2.1	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
166	168	1.8	39	0.001138803	0.512166824	0.03107496	82	2.548147	High
168	169	1.55	53	0.005395326	0.504655794	0.071688517	82	5.878458	High
169	171	1.5	44	0.00198483	0.508657785	0.042214911	82	3.461623	High
171	172	1.6	41	0.001422193	0.510621603	0.035175401	82	2.884383	High
172	174	1.45	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
174	176	2	41	0.001422193	0.510621603	0.035175401	82	2.884383	High
176	177	1	42	0.001589327	0.509922905	0.037396812	82	3.066539	High
177	178	1.6	42	0.001589327	0.509922905	0.037396812	82	3.066539	High
178	180	1.45	48	0.003095587	0.506581595	0.053561975	82	4.392082	High
180	181	1.6	43	0.001776104	0.509269267	0.039740994	82	3.258762	High

181	183	1.35	45	0.002218085	0.508085739	0.044825935	82	3.675727	High
183	184	1.5	35	0.000730178	0.515949889	0.024081659	82	1.974696	High
184	186	1.5	33	0.000584681	0.518255087	0.021107619	82	1.730825	High
186	187	1.55	52	0.00482795	0.504991382	0.067658156	82	5.547969	High
187	189	1.5	57	0.008414677	0.503516356	0.090203301	82	7.396671	High
189	190	1.5	39	0.001138803	0.512166824	0.03107496	82	2.548147	High
190	192	1.45	50	0.00386592	0.50573356	0.060227219	82	4.938632	High
192	193	0.9	48	0.003095587	0.506581595	0.053561975	82	4.392082	High
193	193	0.6	69	0.031922492	0.5014632	0.177770456	130	23.11016	Very High
193	195	1.43	55	0.006737947	0.50404815	0.080440238	130	10.45723	Very High
195	195	0.07	40	0.001272634	0.51136847	0.033070186	82	2.711755	High
195	197	2	36	0.000815988	0.514907553	0.025692277	82	2.106767	High
197	198	1	36	0.000815988	0.514907553	0.025692277	82	2.106767	High
198	199	1.25	45	0.002218085	0.508085739	0.044825935	82	3.675727	High
199	200	0.75	40	0.001272634	0.51136847	0.033070186	130	4.299124	High
200	201	1	91	0.367879441	0.500174376	0.606424905	130	78.83524	Very High
201	203	2.7	77	0.077649082	0.500770706	0.278107553	130	36.15398	Very High
203	206	2.7	53	0.005395326	0.504655794	0.071688517	82	5.878458	High
206	208	1.7	49	0.003459377	0.506143449	0.056804132	82	4.657939	High
208	210	1.9	50	0.00386592	0.50573356	0.060227219	82	4.938632	High
210	212	2	40	0.001272634	0.51136847	0.033070186	82	2.711755	High
212	213	1.75	41	0.001422193	0.510621603	0.035175401	82	2.884383	High
213	215	1.2	32	0.000523195	0.519528199	0.019735738	82	1.618331	High
215	217	1.95	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
217	218	1.1	34	0.000653392	0.517064082	0.022554893	82	1.849501	High
218	219	1	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
219	220	1	27	0.000300185	0.527337709	0.013880161	82	1.138173	High
220	221	1.05	42	0.001589327	0.509922905	0.037396812	82	3.066539	High
221	222	1.5	53	0.005395326	0.504655794	0.071688517	82	5.878458	High
222	223	1.2	35	0.000730178	0.515949889	0.024081659	82	1.974696	High
223	224	0.55	30	0.000418942	0.522343775	0.01720296	82	1.410643	High
224	226	1.8	34	0.000653392	0.517064082	0.022554893	82	1.849501	High
226	227	0.8	22	0.000172232	0.538236758	0.009421899	82	0.772596	Medium
227	228	1.2	24	0.000215092	0.533437314	0.011058214	82	0.906774	Medium
228	228	0.25	18	0.000110432	0.54998693	0.006664289	82	0.546472	Medium
228	230	1.5	34	0.000653392	0.517064082	0.022554893	82	1.849501	High
230	231	1.25	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
231	232	1	22	0.000172232	0.538236758	0.009421899	82	0.772596	Medium
232	234	1.9	35	0.000730178	0.515949889	0.024081659	82	1.974696	High
234	235	1.5	38	0.001019045	0.513020216	0.029183866	82	2.393077	High
235	237	1.5	36	0.000815988	0.514907553	0.025692277	82	2.106767	High
237	238	1.4	35	0.000730178	0.515949889	0.024081659	82	1.974696	High
238	240	1.6	38	0.001019045	0.513020216	0.029183866	82	2.393077	High
240	241	1	21	0.00015412	0.540887388	0.008670843	82	0.711009	Medium
241	242	1.7	22	0.000172232	0.538236758	0.009421899	82	0.772596	Medium
242	243	0.6	38	0.001019045	0.513020216	0.029183866	82	2.393077	High
243	244	1.2	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
244	246	1.5	46	0.002478752	0.507550586	0.047581868	82	3.901713	High
246	247	1.5	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
247	249	1.5	19	0.00012341	0.546749443	0.007293695	82	0.598083	Medium
249	250	1.2	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
250	251	1.4	26	0.000268617	0.529236969	0.012887416	82	1.056768	High
251	253	1.55	41	0.001422193	0.510621603	0.035175401	82	2.884383	High
253	254	1.35	30	0.000418942	0.522343775	0.01720296	82	1.410643	High

Table A.1: Continued.

From	to	Length	GSI	S	a	$JP = S^a$	σ_c	RMi	RMi Classifications
254	256	1.5	26	0.000268617	0.529236969	0.012887416	82	1.056768	High
256	257	0.9	21	0.00015412	0.540887388	0.008670843	82	0.711009	Medium
257	257	0.7	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
257	259	1.9	32	0.000523195	0.519528199	0.019735738	82	1.618331	High
259	261	1.55	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
261	262	1.05	24	0.000215092	0.533437314	0.011058214	82	0.906774	Medium
262	263	1.1	23	0.000192473	0.535757075	0.010217029	82	0.837796	Medium
263	264	0.8	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
264	265	1.5	30	0.000418942	0.522343775	0.01720296	82	1.410643	High
265	266	0.8	19	0.00012341	0.546749443	0.007293695	82	0.598083	Medium
266	267	0.8	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
267	268	1	33	0.000584681	0.518255087	0.021107619	82	1.730825	High
268	269	1	32	0.000523195	0.519528199	0.019735738	82	1.618331	High
269	270	1 1	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
270	271	1.05	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
271	272	0.75	25	0.000240369	0.531267162	0.011947582	82	0.979702	Medium
272	272	0.85	27	0.000300185	0.527337709	0.013880161	82	1.138173	High
272	274	1.15	33	0.000584681	0.518255087	0.021107619	82	1.730825	High
274	275	1.1	37	0.000911882	0.513932441	0.027391375	82	2.246093	High
275	276	1.1	33	0.000584681	0.518255087	0.021107619	82	1.730825	High
276	277	1.3	24	0.000215092	0.533437314	0.011058214	82	0.906774	Medium
277	278	1	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
278	279	1.1	33	0.000584681	0.518255087	0.021107619	82	1.730825	High
279	280	0.4	47	0.002770053	0.507049947	0.050490969	82	4.140259	High
280	281	1.5	43	0.001776104	0.509269267	0.039740994	82	3.258762	High
281	282	0.9	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
282	283	1.4	26	0.000268617	0.529236969	0.012887416	82	1.056768	High
283	284	0.7	30	0.000418942	0.522343775	0.01720296	82	1.410643	High
284	285	0.9	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
285	286	1.45	35	0.000730178	0.515949889	0.024081659	82	1.974696	High
286	287	0.7	19	0.00012341	0.546749443	0.007293695	82	0.598083	Medium
287	289	1.45	23	0.000192473	0.535757075	0.010217029	82	0.837796	Medium
289	290	1.15	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
290	291	0.8	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
291	292	1.4	34	0.000653392	0.517064082	0.022554893	82	1.849501	High
292	293	1.1	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
293	294	0.95	24	0.000215092	0.533437314	0.011058214	82	0.906774	Medium
294	295	1.2	28	0.000335463	0.525560938	0.014928435	82	1.224132	High
295	296	0.95	29	0.000374886	0.523898757	0.016035038	82	1.314873	High
296	298	1.4	36	0.000815988	0.514907553	0.025692277	82	2.106767	High
298	298	0.8	35	0.000730178	0.515949889	0.024081659	82	1.974696	High
298	300	1.35	31	0.000468176	0.520889078	0.018435393	82	1.511702	High
300	300	0.3	36	0.000815988	0.514907553	0.025692277	82	2.106767	High