Chapter 2
Literature Review

2.1 General

In India, bord and pillar is one of the most popular underground mining methods for coal extraction. The stability of underground workings in bord and pillar mining mainly depends on the stability of pillars and the roof of the gallery. The focus of the study is to assess the stability of the gallery roof rock. The immediate roof in a coal mine development heading generally consists of coal, sandstone, clay, or shale. A coal roof is reliable with occasional dressing down of the hanging portions and does not prove dangerous when it stands for a long period. A sandstone roof bends slightly before breaking and gives enough warning before a fracture. It is also, therefore, reliable and is considered good for mining. Shale roof, on the contrary, is the most unreliable roof from a stability viewpoint. It rarely gives any warning before collapsing. The fall of a roof provides valuable information regarding the behaviour of the immediate roof strata under specific geo-mining conditions. Investigating roof falls uncovers the principal causes of these incidents, and it is also crucial to establish or develop an appropriate support design based on the analysis of site-specific rock mass conditions (Mark & Molinda, 2005). There are various methods, such as empirical, numerical, and analytical, for the assessment of the roof stability. The site-specific characteristic behaviour of the rock is well understood by rock mass classification systems. Thus, the rock mass classification system is important for evaluating the behaviour of rock mass as it provides information about strength, deformation, and rock mass characteristics for support estimation (Bieniawski, 1989; Mark & Molinda, 2005).

Rock mass classification systems, along with numerical simulations, are jointly carried out for geotechnical analysis at places where the availability of data is scarce. It requires the strength of the roof rock and stress on the roof. Assessment of the stresses on the roof for the development stage can be done by the numerical simulation method.

Therefore, an extensive literature review regarding roof stability has been carried out to compile the existing knowledge base in the areas pertinent to this research work. The salient findings and prevailing knowledge gap have been identified from this work to formulate a suitable research methodology for developing the requisite know-how for a meaningful realisation of the research objectives. The chapter is organised into two main sections: rock mass classification and roof strata behaviour by numerical simulation.

The pie chart Fig. 2.1 illustrates the distribution of major accident causes in the mining industry based on recorded incidents. Ground movement accounts for the largest proportion at 41%, indicating its dominant role in mine-related hazards. Transport of machinery (other than winding in shafts) is the second most significant cause, contributing 30% of incidents.

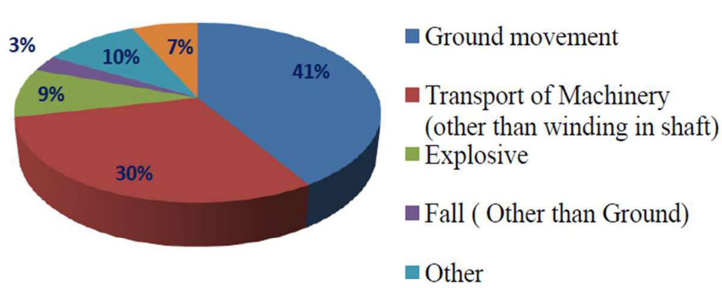

Fig. 2.1: Analysis of the causes of accidents in India's underground coal mines (1973 -2014) (Paul, 2020).

This distribution highlights that ground movement is the primary safety concern, underscoring the need for accurate roof stability assessment and effective support design in underground mining operations (Dash et al., 2016).

2.2 Rock Mass Classification

Many rock mass classification systems have been developed for different strata conditions, and accordingly, roof support guidelines were proposed. Quantification of inherent properties of the rock mass and assessment of its behaviour using relevant properties is the key aim of the rock mass classification. The determination of the influence of external loading on the rock mass is an additional objective of the classification. The first rock mass classification system, expressed by Terzaghi (1946), was specified based on rock loads carried by steel sets. Classifications of rock masses are commonly employed to estimate the required amount of rock support during pre-construction and to quantitatively assess the quality of the rock mass. As shown in Table 2.1, numerous rock mass classification systems have been developed for both general and specialized uses. The different engineering, geological, and geotechnical parameters are given varying degrees of emphasis by these classification systems.

It is crucial to choose relevant factors for classifying rock masses. There is not a single index or characteristic that can be used in engineering to quantitatively define the jointed rock mass. Aggregating the relevant parameters would help in comprehensively representing the rock mass. Geological features generally dominate at shallow depths of working. On the contrary, virgin and induced stresses are more important for designing deep underground workings. Thus, all the designers and engineers need to know the site-specific physico-mechanical properties, stress conditions, and associated geological factors of the rock mass so that suitable solutions can be given for engineering problems.

Table 2.1: Various Rock Mass Classification Systems.

Sr. No.	Name of Classification	Author and Year	Country of origin	Application Areas	Remarks
1	Rock Load Theory	Terzaghi, 1946	USA	Tunnels with steel supports	Unsuitable for modern tunnelling
2	Stand-up time	Lauffer, 1958	Austria	Tunnelling	Conservative
3	Rock Quality Designation (RQD)	Deere et al., 1967	USA	Core logging, tunnelling	Sensitive to the joint orientation effect
4	Rock Structure Rating (RSR)	Wickham et al., 1972	USA	Tunnelling, Hard rocks	Not useful with steel fibre shotcrete
5	Rock mass rating (RMR-system)	Bieniawaski, 1974	South Africa	Tunnels, Underground mines, etc.	Unpublished base case records
6	Tunnelling quality index (Q-system)	Barton et al., 1974	Norway	Tunnels, large chambers	Particularly for hard rock, jointed rock mass
7	Rock mass strength (RMS)	Stille et al., 1982	Sweden	Metal mining	Modified RMR
8	CMRI-RMR	Venkateswarlu et al., 1989	India	Coal Mining	
9	Modified Rock Mass Rating (M-RMR)	Ünal and Özkan, 1990	Turkey	Mining	
10	Rock mass index (RMi)	Arild Palmstrom, 1995	Norway	Rock engineering, communication, characterization	
11	Rock mass Number (N)	Goel et al., 1995	India	Tunnels	Stress-free Q-system
12	*CMRR*	Molinda & Mark, 1995	USA	Coal Mining	
13	Geological Strength Index (GSI)	Hoek et al., 1995	Canada	Mines, tunnels	
14	RMCR (Rock Mass Classification System)	Palmström (2001)	Norway	For various engineering purposes	Incorporates volumetric joint count (Jv), RQD, and strength
15	RMQR (Rock Mass Quality Rating)	Zhang (2016)	China	applicable to various excavation types	Quantitative classification incorporating multiple geomechanical factors
16	CRI (Coal Roof Index)	Present Study (2024)	India	Underground coal mine	Multiplicative index for coal mine roof stability; linked to Hoek–Brown parameters and numerical modelling.

The following geological characteristics are crucial for the stability of below-ground excavation:

- The strength of the intact rock mass
- Including roughness, separation, continuity, weathering, and filling; the orientation of discontinuities in the rock; and the spacing of discontinuities in the rock mass
- The rock quality designation (*RQD*)
- *In-situ* stress field
- Groundwater conditions

Moreover, the weatherability index and *LTH* are other crucial factors in coal mining.

2.3 Common Rock Mass Classification Systems

2.3.1 Rock quality designation (*RQD*)

The *RQD* was developed by Deere, (1967) to provide a quantitative estimation of rock mass quality, as shown in Fig. 2.2. The estimation was done by taking the drill core logs of NX size (54.7 mm diameter) and mapping its recovery.

The estimation of the *RQD* from the number of discontinuities per unit volume was suggested by Arild Palmstrøm, (1982). He proposed a method of determining *RQD* by traces of visible discontinuities on the exposed surface without any core. The method is applicable only for the clay-free rock masses. The relationship suggested is given below:

$$RQD = 115 - 3.3\,J_v\ (2.5) \qquad (2.1)$$

Where, *Jv* is termed as the volumetric joint count and determined as the sum of the number of joints per unit length for all joint (discontinuity) sets. Directional dependency can be reduced by the use of the volumetric joint count. There is an indirect representation of *RQD*. Priest & Hudson, (1976) Proposed a relationship between *RQD* and *LTH* (λ) given as:

$$RQD=100\ (0.1\lambda+1)\ e^{-0.1\lambda} \tag{2.2}$$

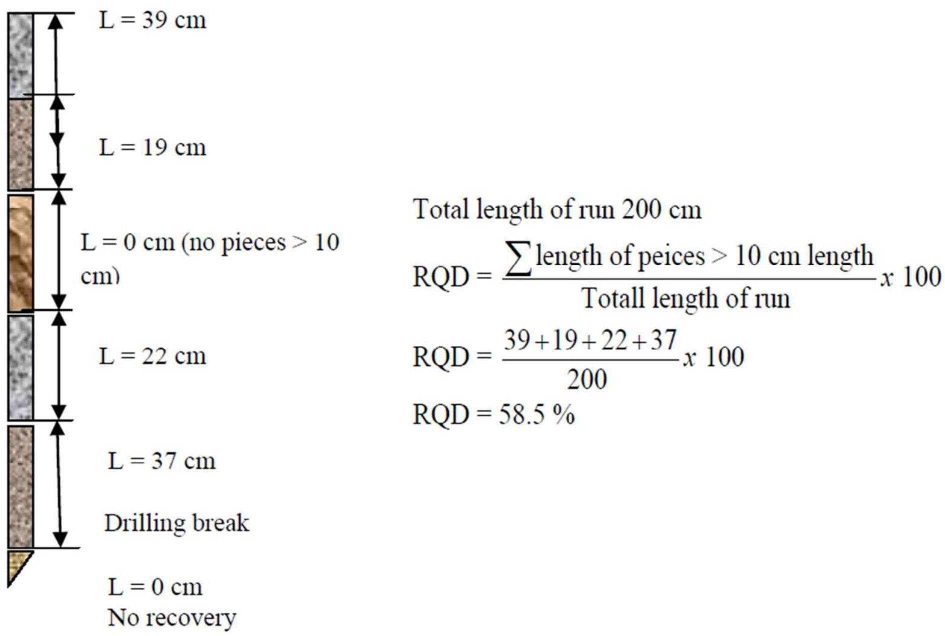

Fig. 2.2: The core piece length Measurement and calculation of *RQD* (D. U. Deere, 1967).

Fig. 2.2 demonstrates the procedure for calculating the *RQD* from a core run. The diagram shows individual core pieces obtained during drilling, with their lengths measured. Only pieces longer than 10 cm are considered in the *RQD* calculation, as shorter fragments are excluded from the quality assessment. The total length of such qualifying pieces (39 cm, 19 cm, 22 cm, and 37 cm) is summed and divided by the total core run length (200 cm), then multiplied by 100 to express the result as a percentage. In this example, the calculated *RQD* is 58.5%, indicating fair rock quality according to standard classification guidelines.

2.3.2 *RMR* Classification

Bieniawski, (1979) devised the Geomechanics Classification, also known as the Rock Mass Rating method, has been successively refined by various researchers and is the most widely used. The following parameters were considered to classify a rock mass:

- Uniaxial compressive strength (*UCS*) of rock material
- Spacing of discontinuities
- *RQD*
- Groundwater conditions and
- Orientation of discontinuities
- Condition of discontinuities

All these parameters can be obtained both by measuring in the field and from borehole data. *RMR* value is obtained by rating each parameter. Three sets of discontinuities have been considered in the rating. The Stand-up time as a function of *RMR*-values and unsupported span is shown in Fig. 2.3.

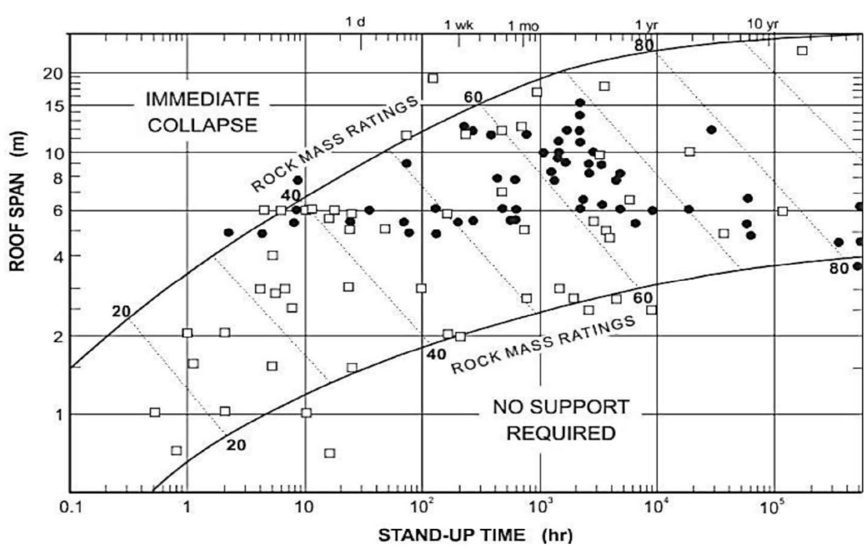

Fig. 2.3: Stand-up time estimation based on *RMR* value and unsupported span (Z. T. Bieniawski, 1979).

Rock masses are categorized using structural areas. The structural zones are delineated by the existence of faults or changes in the kind of rock. Major discontinuity spacing can occasionally separate the rock mass into smaller structural areas.

Another parameter that Bieniawski has suggested is adjustment for joint orientation. The *RMR* is calculated by adding the ratings acquired for the values of the various parameters and the changes. Table 2.2 shows the classification of the rock mass based on the *RMR* (Z. T. Bieniawski, 1979).

Table 2.2: *RMR* classification (Z. T. Bieniawski, 1979).

Class	Classification	*RMR*
I	Very Good	80- 100
II	Good	60 - 80
III	Fair	40 - 60
IV	Poor	20 - 40
V	Very Poor	0 - 20

2.3.2 Q-system

The Q-system for rock mass classification was developed at the Norwegian Geotechnical Institute (NGI). This is also called the Tunnelling Quality Index (N. Barton, 1974). Based on large case studies, *NGI* has developed the Q-System for determining the following two objectives:

1. Rock mass characteristics and

2. Tunnel Support Requirements

The Q system is a classification method used in geotechnical engineering to evaluate the quality of rock masses. Developed by Norwegian engineers in the 1970s, the

Q system helps assess the stability and support requirements for tunnels and other underground excavations. The Q-value represents the stability of a rock mass surrounding an underground opening in jointed rock formations. A higher Q-value indicates better stability, while a lower value suggests weaker stability. The calculation of the Q-value relies on six parameters, with values typically falling between 0.001 and 1000. In rare instances, extreme parameter combinations may result in values outside this range, but in those cases, 0.001 and 1000 can be used as practical limits for determining support. The Components of the Q System are as follows:

1. *RQD :* A measure of the degree of jointing or fracturing in the rock, expressed as a percentage of core recovery with a minimum length of 10 cm. Higher RQD values indicate better quality rock.
2. *Jn (Joint Set Number):* The number of joint sets or families of joints intersecting the rock mass. More joint sets generally imply lower rock mass quality.
3. *Jr (Joint Roughness Number):* A measure of the roughness of the joint surfaces. Joint roughness affects the shear strength of the rock mass.
4. *Ja (Joint Alteration Number):* The extent of alteration or weathering of the joint surfaces. Altered joints are weaker and can reduce the stability of the rock mass.
5. *Jw (Joint water reduction factor):* adjusts the classification to reflect the effects of water on rock stability.
6. *SRF (Stress Reduction Factor):* A factor that accounts for the influence of stress conditions on rock mass stability. It adjusts for the effects of high stress or stress relaxation on the rock mass.

The Q index is calculated using the following formula:

$$Q = \frac{RQD}{Jn} X \frac{\mathrm{Jr}}{\mathrm{Ja}} X \frac{Jw}{SRF} \tag{2.3}$$

The individual parameters are determined during geological mapping using tables that give numerical values to be assigned to a described situation. Paired, the six parameters express the three main factors that describe the stability in underground openings:

RQD/ Jn = Degree of jointing (or block size)

Jr/ Ja = Joint friction (inter-block shear strength)

Jw /SRF = Active stress

Interpreting the Q Index

- *High Q Values:* Indicate good rock mass quality, typically suggesting stable conditions that require less support.
- *Low Q Values:* Indicate poor rock mass quality, which may require additional support and stabilization measures.

The Q-value in Fig. 2.4 is related to the total amount of support (temporary and permanent) in the roof. The diagram is based on numerous tunnel support cases. Ratings of the excavation support ratio (*ESR*) have been tabulated in Table 2.3 (N. Barton, 1974).

Table 2.3: Ratings of the ESR (N. Barton, 1974)

Type or use of underground opening	**ESR**
Nuclear power plants, railroad stations, sports arenas, etc.	0.8
Power stations, road and railway tunnels with heavy traffic, civil defence shelters, etc.	1.0
Storage caverns, road tunnels with little traffic, access tunnels, etc.	1.3
Water tunnels, permanent mine openings, adits, and drifts	1.6
Vertical shafts, rectangular and circular, respectively	2.0 - 2.5
Temporary mine openings	3.5

Applications

The Q system is widely used in tunnel design, slope stability analysis, and other underground construction projects. It helps engineers determine the appropriate type and amount of support needed based on the quality of the rock mass.

By systematically classifying rock masses, the Q system enables engineers to make informed decisions about design and construction, improving safety and efficiency in underground projects.

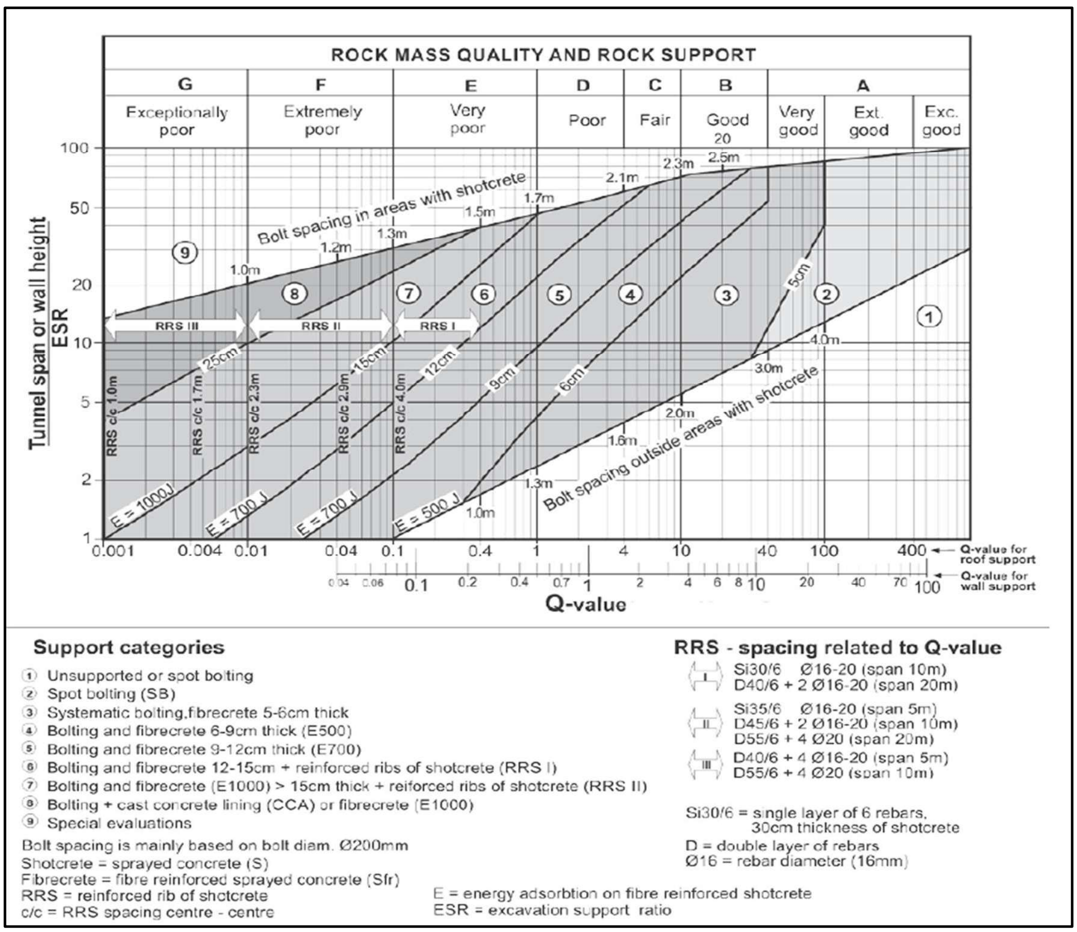

Fig. 2.4: The Q system chart for rock support estimate, developed by the *NGI*, (based on www.ngi.no, 2014)

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2.4 Coal Roof Classification

The rock geology of coal mines has a layered form. The above-mentioned classifications are not considered *LTH* of the stratum in the roof. Therefore, the application of these classification systems to coal mines is not appropriate. But there are two classifications as *CMRI-RMR* (Venkateswarlu et al., 1989) and *CMRR* (Molinda & Mark, N .D. 1994). These are the most accepted rock mass classifications for the roof of the underground coal mines. *CMRI-RMR* rock mass classification, developed by Venkateswarlu et al., has been used in Indian coal mines for the assessment of the roof support system for the last 3 decades. *CMRR* System is most popular in the USA, Australia, South Africa, and other regions. The development approach of both classifications is similar to Bieniawski's *RMR* system. The final rating value in both systems ranges from 0 to 100 (Brook et al., 2020). The better quality of roof rock is indicated by a higher *RMR* value. These are briefly explained in the subsequent section.

2.4.1 *CMRR* System

Molinda & Mark developed the *CMRR* over 25 years ago. This classification system's basic idea has a similarity to Bieniawski's *RMR*. The ultimate *CMRR* value ranges from 0 to 100. The ultimate rating is determined by adding up all of the individual ratings for the following five parameters: (a) the intact rock's *UCS*; (b) the rock's shear strength (cohesion and roughness); (c) the intensity (spacing and persistence) of the bedding and other discontinuities; (d) the existence of a strong bed in the bolted interval; and (e) the rock's moisture sensitivity (Hill, 2007).

The *CMRR* calculation procedure is split into steps. Unit ratings are first established for each layer after the mine roof is divided into structural units. Although a structural unit typically consists of a single lithologic layer, it is possible to combine multiple rock layers if they share similar engineering properties (Mark, C, Molinda, 2003). The second step involves applying the proper adjustment factors and averaging all unit ratings within the bolted zone, with the contribution of each unit weighted by its thickness, to determine the *CMRR* (Mark & Molinda, 2005). Fig. 2.5 illustrates the process for estimation of the *CMRR* value (Mark et al., 2004).

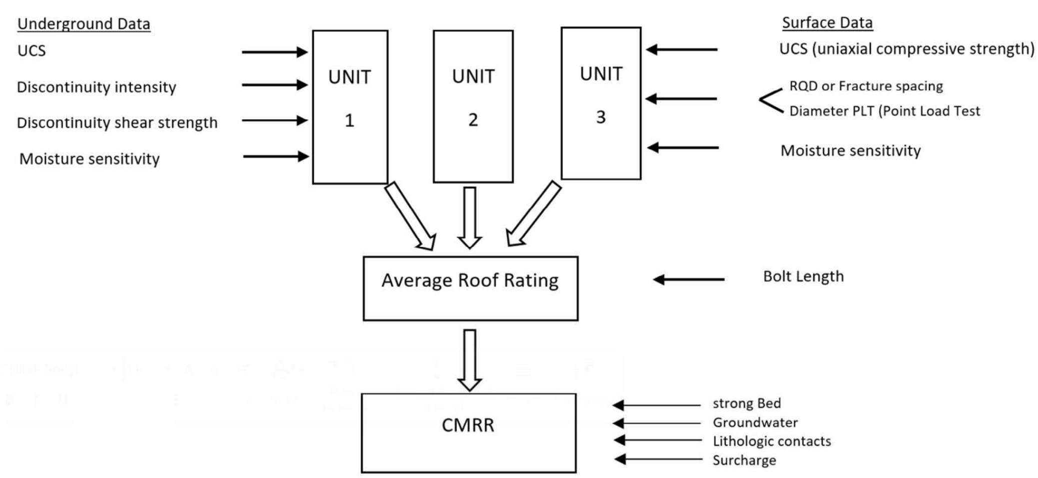

Fig. *2*.5: *CMRR* flow chart (Mark et al., 2004)

2.4.2 *CMRI-RMR* Classification

The *CMRI-RMR* classification is a practical and straightforward method of estimating the roof conditions of an underground coal mine. It had been developed by statistical analysis of the various geotechnical data obtained from the Indian underground coal mines. Details of the geotechnical and geo-mining data have been in scientific reports. The five most important influencing parameters were identified: *LTH*, structural characteristics, weatherability, rock's *UCS*, and groundwater flow rate (*Gw*). Through the use of factor analysis and principal component analysis (*PCA*), relationships between various parameters were discovered. *LTH* plays a substantial role in delamination, and it is a cause of roof deterioration. The *LTH* can be measured by measuring the layers' thickness within the bed. *SF* are the cause of roof degradation, which include cracks, major faults, joints, slips, etc. In Indian coal mines, Gw is a significant issue. Because many coal measure rocks deteriorate or disintegrate as a result of weathering, particularly when water is present, weatherability is crucial. It is determined by the slake durability apparatus, and the 1st cycle slake durability index (*SDI*) was considered for analysis. The rock's *UCS* has been calculated in the laboratory as per the standards of the Indian Bureau. The point load Index obtained from an irregular piece of rock is converted to estimate the *UCS* using the empirical relation: $UCS = 14\ I_p$. The groundwater seepage rate is measured by drilling a long hole in the roof (1.5-1.8 m) and collecting the percolated water through the hole. This water flow is expressed in mL/min. All the geological features are recorded through geotechnical mapping.

The *CMRI-RMR* classification system is derived from the sum of the ratings of five distinct parameters. The *CMRI-RMR* value is calculated as follows:

$$CMRI\text{-}RMR = R_{CMRI\_1} + R_{CMRI\_2} + R_{CMRI\_3} + R_{CMRI\_4} + R_{CMRI\_5} \tag{2.4}$$

Where R_{CMRI_1}, R_{CMRI_2}, R_{CMRI_3}, R_{CMRI_4}, and R_{CMRI_5} are the ratings for *LTH*, structural characteristics, weatherability, *UCS* of roof strata, and flow rate of groundwater, respectively.

Table 2.4: *CMRI RMR* parameters (Venkateswarlu et al., 1989).

Parameter	**Maximum rating**
Gw (ml/min).	10
UCS (Kg/cm2)	15
Weatherability (1st cycle SDI)	20
Structural characteristics	25
LTH (cm) `	30

The weighted average value of *CMRI-RMR* shall be determined for a case of multiple layers. The maximum rating of the parameters is provided in Table 2.4. It has been observed that the rating of the single parameters is low compared to other parameters in most of cases. This indicates that the influence of a single weak parameter is sufficient to reduce the strength of the roof mass. Therefore, the summation of the ratings for estimation of the overall rating of the rock mass is not appropriate.

2.5 Support Design Methods

Support design in mining and geotechnical engineering refers to the process of selecting and designing systems to stabilize underground structures such as tunnels, mines, and shafts. The purpose of support design is to ensure the safety of the workers and the long-term stability of the excavation. Support methods are often categorized into three broad approaches: empirical, analytical, and numerical methods(Kushwaha et al., 2010; Mark et al., 2021; Singh et al., 2005; Stille et al., 1989). Each has its own advantages and limitations depending on the specific project requirements. Analytical methods involve using mathematical models and formulas based on principles of mechanics, equilibrium, and material behavior to design support systems (Cai et al., 2015; Nguyen & Nguyen, 2015). These methods are more rigorous than empirical and numerical methods. The analytical methods make assumptions that may not fully represent the real-world conditions, such as assuming the homogeneity of materials or simple loading conditions.

Generally, the rock mass has heterogeneity in nature. Therefore, analytical methods are rarely used in the mining industry. Here's an overview of empirical methods and numerical methods.

2.5.1 Empirical Method

Empirical methods are based on practical experience, observational data, and results from past projects. These methods provide guidelines and rules for support design based on historical observations and case studies rather than a detailed understanding of the underlying mechanics. They are often used when there is limited data available or for situations where complex analysis is not feasible.

In Indian coal mines, the *CMRI-RMR* classification is used for the early assessment of appropriate support design (Paul, Kumar, et al., 2020). Therefore, firstly, the rock load is calculated by the suggested formula, and accordingly, support resistance is assessed by considering FOS 1.5. The *CMRI-RMR* system has some adjustment factors for gallery span, depth, method of extraction, induced stresses, and lateral stress. These adjustments account for their neutral, negative, and positive contributions to *CMRI-RMR* values. The following equations are used to calculate the rock load in junctions and galleries using adjusted *CMRI-RMR* (Ghosh & Ghose, 1992; Paul et al., 2014; Paul, Mallika, et al., 2020; Singh et al., 2009).

$$\text{Rock load in junctions (t/m}^2\text{)} = 5 \cdot B^{0.3} \cdot (1 - RMR/100)^2, \tag{2.5}$$

$$\text{Rock load in roadways (t/m}^2\text{)} = B \cdot D \cdot (1.7 - 0.037RMR + 0.0002RMR^2), \tag{2.6}$$

Where D is the density (t/m^3) of the roof rock, and B is the width of the gallery (m).

2.5.2 Numerical Method

Numerical methods are the most advanced and flexible of the three, involving the use of computer-based models to simulate the behavior of rock masses and support systems under various loading conditions. These methods solve complex equations that describe the physical behavior of the system and are particularly useful in complex scenarios where both the material properties and geometry are highly variable (Cai et al., 2015).

Numerical methods are very popular and acceptable in underground gallery design. They are flexible and can quickly analyze the effect of numerous geometric and geotechnical variables of the material (H. Wang et al., 2011; M. Wang et al., 2021). The rock mass classification is a prime input parameter in the numerical method. Stability assessment by numerical modelling technique is mainly based on the following approaches:

1) Stress redistribution and critical displacement approach, in this approach zone of stress or displacement has been studied taking into consideration with factor of safety. The ratio of stress to the rock's peak strength is the simplest way to define the factor of safety (*FOS*) for the rock mass surrounding an underground excavation. In underground rock engineering, numerous studies have reported that a *FOS* greater than 1 is generally considered indicative of a stable excavation, while an *FOS* of 1 denotes the critical threshold between stability and failure, and values below 1 represent failed conditions. However, several researchers advocate for a more conservative threshold, suggesting that the critical *FOS* should be in the range of 1.4–1.5 to account for uncertainties in material properties, loading conditions, and geological variability (Huang et al., 2023; Kaiser et al., 2001; Martin et al., 2003). Few authors give the limit of critical displacement for unstable cases; if the observed displacement in the numerical tool is beyond that limit, the failure is taken into account. The stress-based failure criteria are mostly used for higher depths of underground openings in numerical analysis.

2) True roof bolt response approach, it depends on the load coming on the bolt or the displacement observed in the bolt (Kong et al., 2023; S. Yu et al., 2019; W. Yu et al., 2024).

3) The critical strain behaviour approach is a recent practice for underground stability assessment (Cui et al., 2017; Sinha et al., 2016). A typical immediate roof deflection curve over an excavation would be expected to differ with and without roof supports, with associated horizontal strains at each point. It is also known that when the critical strain values are exceeded, the tensile, compressive, or shear strain will cause the immediate roof stratum to fail (Sinha et al., 2018).

It is easy to determine the material properties for the numerical modeling. To fully benefit from numerical modeling, a deeper comprehension of constitutive models and related input parameters is necessary, as models get more sophisticated (R. Kumar et al., 2015; Mandal et al., 2008; S.C li, 2015; Singh et al., 2011). For an underground working simulation to be successful, the physico-mechanical characteristics of the entire rock tested in the lab must be scaled to the rock mass. Choosing a suitable constitutive model for the strata, especially coal, is the most crucial stage in numerical modeling. In numerical analysis, two fundamental constitutive models of coal measure strata are taken into consideration: the elasto-plastic and elastic behavior of strata.

2.5.2.1 Elastic model

In this model, coal measure strata are assumed to behave elastically, meaning that they deform under stress but fully recover when the stress is removed. This model follows Hooke's Law, where the strain is proportional to the applied stress. It is simpler and computationally less expensive. This model is often used when the strata are relatively undisturbed and do not experience significant plastic (irreversible) deformations. It does not account for inelastic behavior, such as yielding or strain softening, which can occur in real-life coal measure strata due to factors like high stress or geological discontinuities (Pirhadi et al., 2023).

2.5.2.2 Elasto-plastic / strain softening model

It does not account for inelastic behavior, such as yielding or strain softening, which can occur in real-life coal measure strata due to factors like high stress or geological discontinuities. It provides a more realistic representation of coal measure strata under high-stress conditions. It can simulate behaviors like failure, fracturing, and post-peak softening, which are common in mining environments, particularly in the context of tunnel or mine roof stability (Cui et al., 2015). It is more complex to model and requires more computational effort. It also involves material parameters like yield stress, hardening/softening characteristics, and plastic flow rules, which need to be calibrated with experimental data (Gadepaka et al., 2024).

Using *FLAC3D* in the strain-softening mode, the different strengths and elastic constants of the rock mass are as follows: a) elastic constants b) peak and residual strength and its function with plastic shear strain for rock mass, c) the peak and residual friction angle and how it changes with plastic shear strain, d) the in-situ stress situation (Liu et al., 2018) and dilation behavior was studied concerning confinement and plastic shear strain.

2.5.2.3 Hoek-Brown failure criteria

For the majority of geo-materials, the Hoek–Brown strain-softening model (E. Hoek et al., 1980) is the most extensively used. Several researchers (Abdellah et al., 2018; Sinha & Associates, 2015; Sinha et al., 2016, 2018) have considered the Hoek–Brown strain-softening model when designing a gallery. The Hoek–Brown failure criterion is as follows:

$$\sigma_1 = \sigma_3 + \sigma_c \left(m \frac{\sigma_3}{\sigma_c} + s \right)^a \tag{2.7}$$

where σ_1 and σ_3 are maximum and minimum principal stresses, σ_c is the *UCS*, and *m*, *s*, and *a* are Hoek-Brown strength parameters.

2.5.2.4 Sheorey failure criteria

The shear strength and the friction angle for rock masses are estimated using Sheorey's failure criterion [35] for the rock masses. This criterion uses the 1976 version of *RMR* of Bieniawski for reducing the laboratory strength parameters to give the corresponding rock mass values. Sheorey proposed the following non-linear failure criterion for Indian coal (Murali Mohan et al., 2001):

$$\sigma_1 = \sigma_{cm} \left(1 + \frac{\sigma_3}{\sigma_{tm}} \right)^{bm} \tag{2.8}$$

where σ_{cm}, σ_{tm}, and b_m are rock mass strength parameters, defined as :

$$\sigma_{cm} = \sigma_c \, exp\left(\frac{RMR - 100}{20} \right) \tag{2.9}$$

$$\sigma_{tm} = \sigma_t \, exp\left(\frac{RMR - 100}{27} \right) \tag{2.10}$$

$$b_m = b^{RMR/100} \tag{2.11}$$

where σ_c and σ_t are the *UCS* and tensile strength of the coal specimen, respectively. The *RMR* of Indian coal varies from 55 to 65. In the above equations, b = 0.5 and σ_c / σ_t = 15 was considered (Murali Mohan et al., 2001).

2.5.2.5 In-situ Stress condition

Equations (2.12) and (2.13) provided by Sheorey (1994) were used in this work to estimate the vertical and horizontal in situ stress as well as initialize the in-situ stress gradient concerning the depth of the working in the model. Several researchers (Das et al., 2017; Loui et al., 2007; Murali Mohan et al., 2001; Singh et al., 2011) have previously adopted this pre-mining stress initialization method, with reasonable results for Indian geo-mining conditions. To get to an equilibrated state, the model was then solved.

$$\sigma_v = \gamma H \quad \text{(in MPa)} \tag{2.12}$$

$$\sigma_h = 2.4 + 0.01H \quad \text{(in MPa)} \tag{2.13}$$

Whereas σ_v =Vertical stress, σ_h= Horizontal stress, and H = Depth of gallery.

Equation (2.12) is based on the overburden load, and 0.025 MPa/m is used as the rock unit weight. The formula for the average in-seam horizontal stress, which Sheorey et al., (1989) presented based on the thermo-elastic shell model of the earth, is the source of equation (2.13).

2.6 Time-Dependent Strength Reduction

A convergence curve is the standard illustration of the relationship between cumulative roof convergence and time when examining the time-dependent behavior of a mine roof. This convergence curve is extremely similar to a creep curve and often includes three stages, as was previously explained. When the deformation approaches a threshold value, the deformation velocity typically first rapidly reduces, then remains steady, and finally may accelerate to eventual failure. The failure mechanism and time-dependent deformation are quite similar to the intact rock's creep characteristics. Because of this, time-dependent roof displacement is a commonly used indicator for mine roof time-dependent stability (Iannacchione et al., 2004). Several studies are concentrating on figuring out the creep property of intact rock and identifying an appropriate constitutive model to depict the observed creep behaviors.

Limited research has been done on the time-dependent strength deterioration of the coal roof by strain-softening constitutive models.

The failure criterion has taken into consideration how time affects strength in order to explain time-dependent failure. As demonstrated by the literature review, the relationship between strength and strain rate can be employed for compression testing, and the time-to-failure for creep circumstances can indicate the influence of time. The failure criterion, however, is unable to include these ideas. When using time-dependent strength or strength decline, failure criteria typically include time-dependent failure.

2.7 Research Gap, Limitations of Previous Studies, and Novelty of the Present Work

2.7.1 Limitations of Previous Studies

The review of existing literature on rock mass classification systems and stability assessment methods for coal mine roofs reveals several limitations, as summarised in Table 2.5.

Table 2.5: Existing approaches for stability assessment.

Study / Approach	Main Contribution	Limitations
Bieniawski *RMR*, Barton Q-system	Widely used in civil & mining engineering for rock mass classification	Developed for hard rock; does not account for *LTH*; limited applicability to coal measure strata
CMRR (Molinda & Mark, 1994)	Includes bedding and moisture sensitivity; used in USA, South Africa, Australia	Requires detailed stratigraphic mapping; lacks adjustment for gallery span and stress conditions; limited validation for Indian coalfields
CMRI-RMR (Venkateswarlu et al., 1989)	Designed for Indian coal mines; simple to apply	Summation-based rating masks the effect of a single weak parameter; poor correlation with stand-up time in short-term stable cases; does not explicitly account for stress–dimension effects
Empirical rock load formulas (CMRI, Ghosh & Ghose, 1992)	Easy to implement in early design stages	Based on limited datasets; does not integrate time-dependent strength reduction; not adaptable to varying mining geometries
Numerical modelling with generic parameters	Can simulate stress redistribution, deformation, and plastic zones	Accuracy highly dependent on quality of input parameters; often uses generic classification values without site-specific calibration
Time-dependent modelling studies (various authors)	Recognises creep and strength deterioration over time	Mostly applied to hard rock or non-layered strata; limited work on coal roof behaviour under Indian conditions

2.7.2 Research Gap

- No existing classification system for coal mine roofs in India adequately accounts for the multiplicative effect of weak parameters or integrates stress conditions and gallery dimensions.
- Limited work combines site-specific classification with numerical modelling for back-analysis and calibration of Hoek–Brown strength parameters in coal measure strata.
- Time-dependent deterioration models have rarely been developed and validated for layered, moisture-sensitive coal roofs in Indian mines.
- Empirical rock load formulations are not directly linked to modern classification approaches, reducing their predictive reliability.

2.7.3 Importance and Novelty of the Present Work

This research addresses the identified gaps by:

1. Developing a new *CRI* classification that uses multiplicative rating of key parameters and adjusts for stress–dimension effects via a Rock Load Factor (*RLF*).
2. Establishing direct correlations between *CRI* and stand-up time for Indian coal mines, improving prediction accuracy over *CMRI-RMR*.
3. Integrating *CRI* with numerical modelling to estimate Hoek–Brown strength parameters through the critical plastic strain technique.
4. Developing a time-dependent elastic-perfectly plastic-strain softening model for coal roof strata, enabling prediction of strength degradation and creep behaviour and cut-out distance optimisation for continuous miner operations.

5. Proposing a *CRI*-based empirical rock load equation that can be applied in support design.

2.8 Concluding Remark

This chapter reviewed the various rock mass classifications (i.e., *RQD*, *RMR*, Q system, *CMRR*, and *CMRI-RMR*) and support design methods. Traditional and current approaches (*CMRI-RMR*-based) for coal mine gallery stability have also been reviewed. The traditional approach for gallery stability is based on empirical rock load estimation. In this regard, various approaches for support design have been reviewed. The numerical method seems the most plausible choice for understanding the behaviour of coal roof strata. Details of the Hoek-brown strain softening constitutive behaviour model are given in this chapter.