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1 INTRODUCTION

1.1 General

SM Consultant Limited was awarded the work of rock/soil investigation and consultancy services for technical and allied facilities related to 11 proposed tunnels in between Jeypore-Malkanagiri, Odisha.

Name of the Work is "Jeypore-Malkangiri New B.G Railway Line: Geotechnical investigation, Geological mapping and Desk Study of the tunnel alignment etc. between Tanginiguda Station to Govindapalli Station i.e., Tunnel-1 from Ch. 41592m to 42052m, Tunnel-2 from Ch.42870m to 43620m, Tunnel-3 from Ch.44181m to 44771m, Tunnel-4 from Ch.45460m to 45560m, Tunnel-5 from Ch.50177m to 53177m, Tunnel-6 from Ch.57503m to 59253m, Tunnel-7 from Ch.60778m to 61308m, Tunnel-8 from Ch.61529m to 61709m, Tunnel-9 from Ch.61919m to 62040, Tunnel-10from Ch.63080 m to 63200 m,Tunnel-11 from Ch.66440 m to 66890 m including other allied works in connection with construction of Jaypore-Malkanagiri new BG rail link project.

The geological / geotechnical data accessible from the site were very limited, hence there is an adequate scope for further improvement of the precision and presentation of the data. However, the geological data presented in the report are in well consultation with the previous works, in and around the area by Geological Survey of India and other workers along with robust field work at the site. The subsurface investigation comprised of core drillings, Ground investigations with in-situ tests in each borehole (slug-tests), laboratory tests and petrographic analysis on selected core specimen were done. The knowledge of outcrops along the tunnel alignment was recorded with field geological survey under supervision of a Geological/Geotechnical expert.

1.2 Scope of the Geotechnical Report (GTR)

This GTR presents the status of knowledge regarding the geological and structural conditions along the tunnel alignment. In order to optimize the design, a geological assessment was done as part of the work. The assessment included field and desk work. This report summarizes subsurface and site conditions that are expected to be encountered during the tunnelling works. The subsurface and site conditions are derived from geotechnical information and data gathered from field investigations.

The Geotechnical Report contains following test results (as per relevant IS code) and their analysis to suggest recommended values for tunnel designing: Moisture Content/Dry Density, Atterberg Limits, Specific Gravity, Grain size analysis including; Direct Shear Test; Natural Density, Consolidation Test, Unconfined Compression Test, Tri-axial Test, Density Test, Water Absorption & Porosity, Hardness, Unconfined Compression Test, Point Load Test, Modulus of Elasticity, Abrasion Testing, chemical analysis of ground water samples, tests on collected soil samples as per relevant IS code, deformability of Rock materials (Young's modulus, modulus of deformability, stress strain curve, failure energy) IS 9221, IS 9143 for rock samples, Tensile strength of Rock materials (indirect tensile strength, 'Brazilian test') IS 10082 for each sample, In -Situ Permeability Test in bore hole at Tunnel sites.

Geological mapping has been done for the Proposed Tunnels alignment for a corridor width of 200m (100m on either side) in scale 1:50,000 of critical area, longitudinal details in relevant areas (1:5000H and 1:500V) and cross section across the valley / nallah / streams and across other important geological features including survey work (by total station and DGPS) for geological mapping and geophysical survey. Desk Study of the tunnel alignment of proposed tunnels has been carried out in connection with doubling section, followed by site reconnaissance. Design of horizontal and vertical alignment of the proposed tunnels satisfying the required parameters for double line tunnels as being adopted for railway tunnels in India. Plans and L-section of alignment for tunnels on a suitable scale has been prepared. Geological interpretation of the area has been carried out based on available satellite data supported by limited/feasible ground verification for major features. After ground verification and satellite data interpretation, geological plan of the area of interest on 1:25,000 scale and L-section on the suitable scale to clearly depict the surface features has been prepared. Ground pattern/rock mass behaviour has been evaluated and geologically characterized based on various empirical approaches. Geotechnical Investigations, including Geo-Physical Studies (Seismic Refraction Survey) including data acquisition, data processing, and interpretation of sub surface strata has been carried out.

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2 LOCATION MAP AND TOPOGRAPHY

2.1.1 Locality & Accessibility

The mapped area extends between latitudes 18.508°N and 18.651°N and longitudes 82.253°E and 82.413°E, and falls within Survey of India toposheet No. 65J/6. Koraput, the district headquarters, and Jeypore, the principal commercial centre of Koraput district, are situated to the northeast of the area. The nearest railway station is at Salur, located about 112 km from Jeypore; however, the area is more conveniently accessed from Vizianagram Junction on the South Eastern Railway via regular bus services along National Highway NH-43. The distance between Vizianagram and Jeypore is approximately 170 km. A metalled road traverses the central part of the area (toposheet 65J/6), providing connectivity between Jeypore and Malkangiri.

2.1.2 Physiography

The north-western part of toposheet 65J/6 is relatively lower in elevation compared to the rest of the area, with an average altitude of about 1,800 ft (548 m), and is largely occupied by the Dharamgod Reserve Forest. This sector is characterized by isolated hillocks, whereas the south-eastern part comprises continuous hill ranges trending in a northeast–southwest direction, attaining a maximum elevation of 3,565 ft (1,085 m). The north-western portion forms part of the Jeypore uplands, while the south-eastern sector merges with the Koraput plateau.

Drainage in the area is dominated by the Machkund River, which flows southwestward along the south-eastern margin. South of Machkund (Chikamput, as marked on the map), the river has carved a deep gorge, resulting in the prominent Duduma Falls with a vertical drop of about 800 ft (243 m). The river water is harnessed under the Machkund Project for the generation of approximately 100 kW of hydroelectric power, supplying electricity to Andhra Pradesh and Odisha.

Among the minor rivers and streams, the Kamer Gedda, Dharam Gedda, Jam Nadi, and Bali Gedda drain towards the south and southwest, while the Kudda Gedda and Kurlu Nadi flow northwards.

2.1.3 Climate

The region experiences a tropical climate with occasional temperate conditions. Winters are severe, with night temperatures occasionally dropping to around 4°C. Summers, particularly in the Machkund area, are moderate and accompanied by cool nights. The monsoon generally sets in during early May and continues until September. The average annual rainfall, based on data from the Soil Conservation Department, Machkund (mean of the last five years), is approximately 135 cm.

2.1.4 Flora and Fauna

The reserved forests are predominantly covered by tree species such as Sal (Shorea robusta), Kasan (Terminalia tomentosa), Kasi (Bridelia retusa), Sisoo (Dalbergia sissoo), and Teak (Tectona grandis), with Sal accounting for nearly 85% of the forest cover. Other species including Mahul, Kendu, Mango, and Jackfruit are also present. To promote soil conservation, extensive plantations of cashew, sisal, and silver oak have been undertaken by the Forest Department and the Soil Conservation Department of Odisha. The forests support a diverse wildlife population comprising tiger, leopard, cheetah, bear, barking deer, bison, and wild boar. Crocodiles have also been reported from the waters of the Machkund River.

2.1.5 Seismicity

The Koraput–Jeypore region of Odisha falls within Seismic Zone II (Figure 2.1)., indicating low to moderate seismic hazard as per classifications monitored by the National Centre for Seismology (NCS), India. The area experiences occasional low-magnitude earthquakes, generally in the range of micro- to minor seismic events, while the occurrence of major destructive earthquakes is rare. The NCS regularly records micro-seismic activity in this broader region, including minor events of magnitude around 3–4, illustrating that low-intensity earthquakes do occur but destructive high-magnitude events are uncommon here.

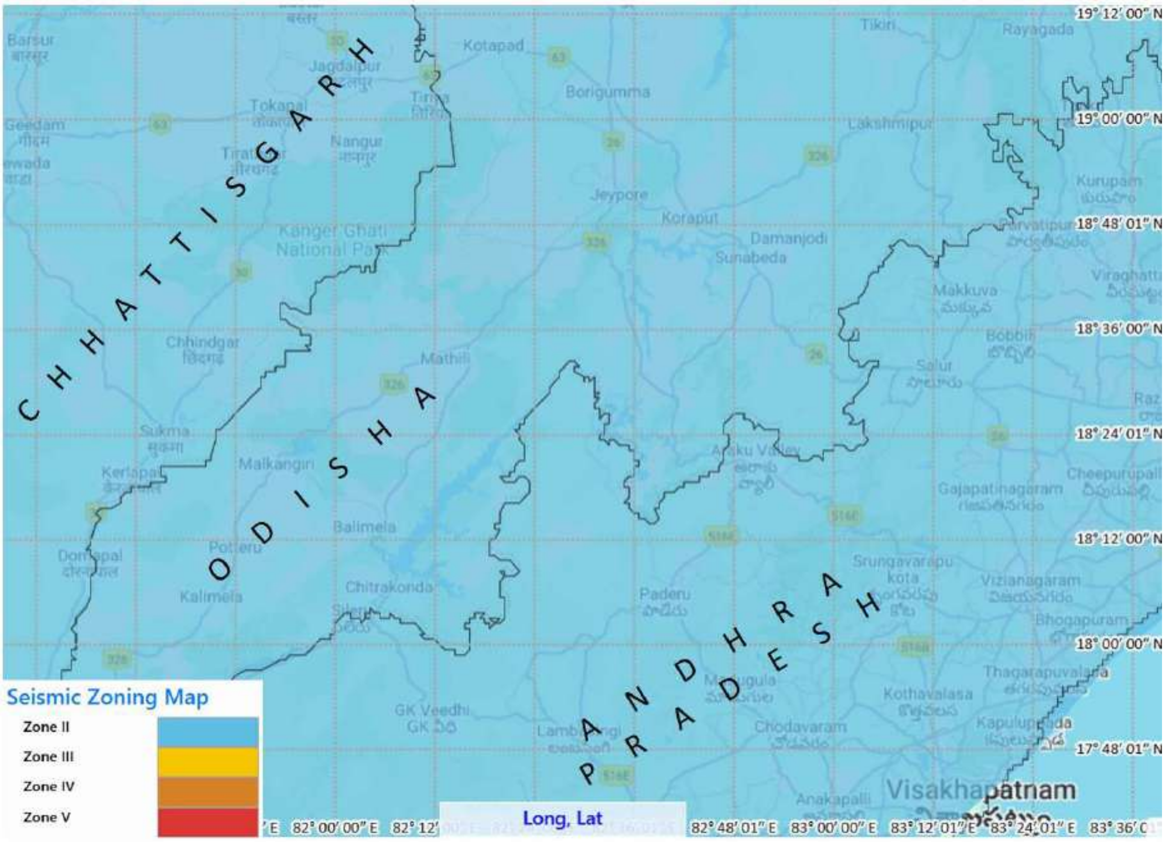

Figure 2.1: Seismic zonation map. (Adopted from National Centre for Seismology)

3 SITE GEOLOGY

3.1 Regional Geology

The site area lies within the eastern part of the Indian Shield and encompasses the contact zone between the Jeypore Province and the Eastern Ghats Province (EGP), two major crustal domains with contrasting lithological and metamorphic histories. The region is characterized by high-grade granulite facies rocks and records a complex tectono-metamorphic evolution associated with deep crustal processes.

3.1.1 Jeypore Province

The Jeypore Province, occupying the western part of the transect, is dominated by a suite of charnockitic rocks comprising greasy green, coarse- to medium-grained, equigranular charnockites, charno-enderbites, and enderbites (Subbarao et al., 1998; Kovach et al., 2001). These lithologies occur in close spatial association and are texturally and compositionally gradational, rendering them difficult to map as separate units at the regional scale. All these rocks exhibit well-developed gneissic banding and compositional layering, indicative of high-grade ductile deformation under granulite facies conditions. Mafic granulites occur as lenticular enclaves and disrupted bands within the charnockitic host rocks, suggesting syn-deformational incorporation of mafic material during high-temperature metamorphism.

3.1.2 Eastern Ghats Province

East of the Jeypore Province, the Eastern Ghats Province is dominated by strongly weathered garnet–sillimanite-bearing quartzofeldspathic gneisses, calc-silicate gneisses, and associated high-grade metamorphic rocks. These lithologies are intruded by orthopyroxene-bearing megacrystic granitoids, reflecting prolonged magmatic activity during high-grade metamorphism. Along the studied transect, the structural and metamorphic evolution of the EGP, including the Koraput Alkaline Complex (KAC), has been documented in detail by Gupta et al. (2005), Nanda et al. (2008), and Nanda et al. (2009).

These studies indicate that the EGP initially underwent granulite facies metamorphism accompanied by penetrative fabric development. This early high-grade event was followed by the emplacement of the Koraput Alkaline Complex, located approximately 3 km east of the contact between the EGP and the Jeypore Province. Subsequent retrogression of the granulite facies assemblages to amphibolite facies conditions occurred in response to hydrous fluid infiltration. This retrograde metamorphism was synchronous with regional-scale ductile shearing along a north-easterly trending, south-easterly dipping foliation. The region later experienced renewed thermal input and crustal loading, culminating in a second granulite facies metamorphic event with peak conditions estimated at ~8.0 kbar and ~700 °C (Nanda et al., 2008).

3.2 Stratigraphy of the area

Recent to sub-recent		Soil and alluvium
	Younger intrusive	Pegmatite, vein quartz and dolerite.
	Peninsular Gneiss Complex	Biotite gneiss, composite gneiss and migmatite.
Archaean	Older intrusive	Meta-dolerite, gabbro, amphibolite, hornblende a schist and ultramafic
	Charnockite suite	Charnockite (acid to intermediate) pyroxene granulite and banded-magnetite quartzite.
	older rocks	Quartz-felspathic granulite and hornblende porphyry

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3.3 Local Geology

Considering the regional geological framework and the field observations undertaken during the present investigation, the project area exhibits lithological and structural characteristics broadly consistent with the granulite–gneissic basement terrain of the Eastern Ghats Province. The alignment traverses highly deformed and metamorphosed rocks, which have been further influenced by multiple ductile and brittle deformation events.

General Layout and Mapping Framework

For geological mapping and interpretation, the project area has been subdivided into three layout sheets based on the tunnel distribution and required mapping scale:

- Layout I: Covers Tunnels 1 to 4, which occur within the western segment of the alignment.
- Layout II: Covers Tunnel 5, located centrally and significantly affected by structural deformation and faulting.
- Layout III: Includes Tunnels 6 to 11, situated in the eastern part of the alignment, showing comparatively more stable lithological continuity.

Each layout was prepared at an appropriate scale to accommodate detailed geological mapping, structural data plotting, and correlation with subsurface borehole information.

Structural Features

The area displays a well-developed joint network comprising four prominent joint sets, observed consistently along rock exposures, slopes, and borehole cores. The orientation and characteristics of these joint sets are summarized below:

Joint Set	General Trend	Dip / Dip Direction	Remarks
J_1	NNE–SSW	Steep (70°–80°)	Dominant joint set parallel to regional foliation; major structural control.
J_2	NNW–SSE	Moderately steep (60°–70°)	Secondary joint set; typically open to tight, rough surfaces.
J_3	WNW–ESE	Moderate (50°–60°)	Discontinuous; locally shear-filled, contributing to block disintegration.
J_4	E–W (Subhorizontal)	Gentle (10°–15°)	Subhorizontal joint/fracture plane; may influence overbreak and water seepage in tunnel crown.

These joint sets collectively contribute to a blocky to moderately jointed rock mass, particularly where intersected by weathered or sheared zones.

Fault Zone and Structural Discontinuities

A major fault zone trending NNE–SSW has been delineated in the central part of the project area, based on both field mapping and geophysical (SRT) data interpretation. This fault coincides spatially with the Tunnel 5 alignment and is anticipated to significantly influence tunnelling operations in this section.

Field evidences supporting this structure include:

- Slickenside lineations and polished fault planes observed on outcrop surfaces.
- Local modification and offset of drainage patterns, consistent with fault-controlled geomorphology.
- Zones of intense crushing and brecciation with clay gouge, indicating brittle reactivation along an earlier ductile shear.
- Reduced seismic wave velocity in SRT profiles, confirming the presence of a fractured zone.

This fault zone is interpreted as a rejuvenated shear zone, potentially associated with the regional NE–SW tectonic grain of the Eastern Ghats Belt. The alignment of the fault coincides with the transition between Khondalite–Migmatite units and associated Restitic bands and Dolerite dykes, suggesting a litho-structural control on the deformation.

Lithology

Detailed field mapping and borehole core logging reveal that the rock types exposed along the alignment include:

- Khondalite: Predominantly garnet–sillimanite–quartz–feldspar gneiss, medium- to coarse-grained, locally migmatitic; represents the dominant country rock.
- Migmatite: Leucocratic and biotite-rich varieties observed; often intruded by pegmatitic and quartz veins.
- Restite: Occurs as dark, mafic segregations within the migmatite; locally enriched in biotite and garnet.
- Dolerite Dykes: Fine- to medium-grained, trending parallel to the regional foliation, showing chilled margins and occasional shearing along contacts.

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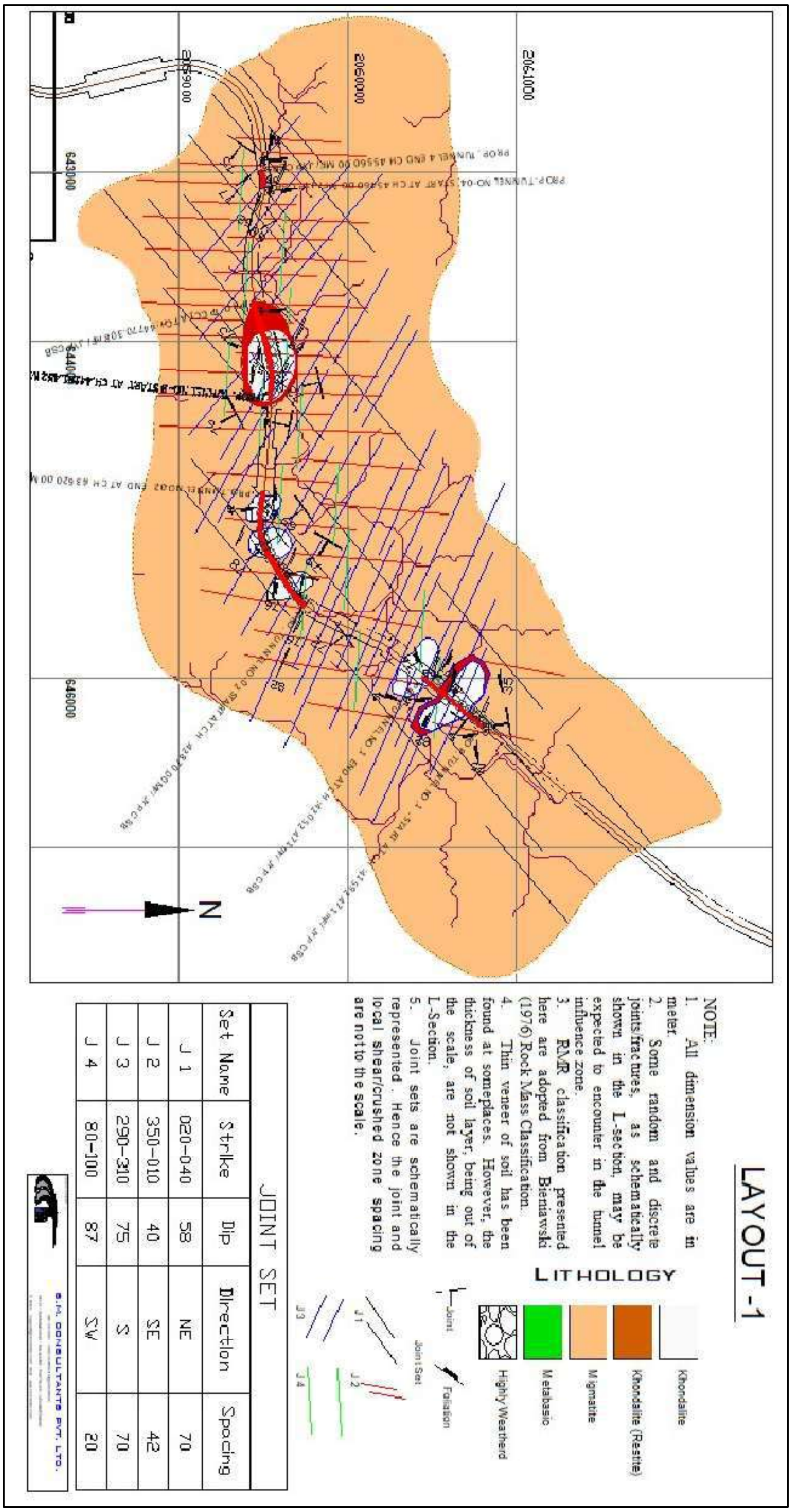
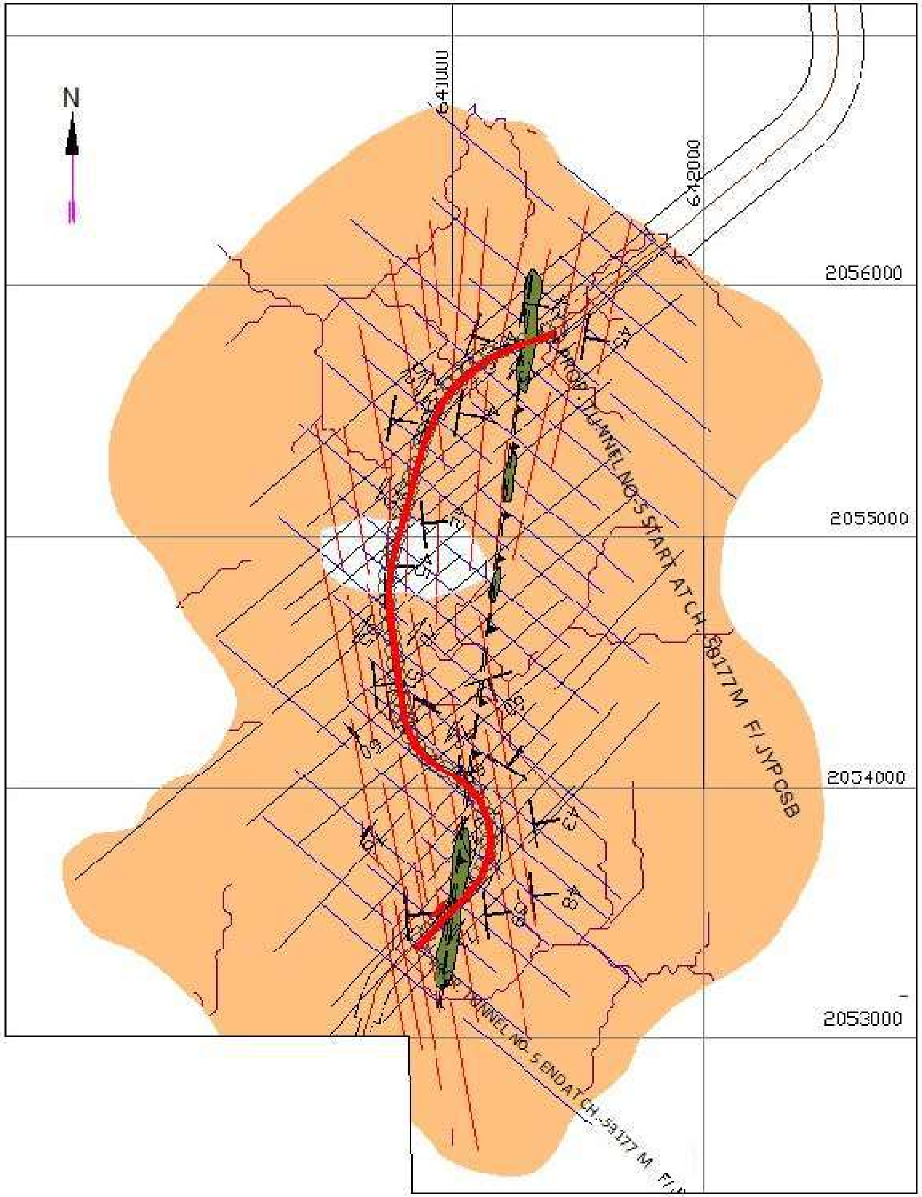
The alignment of the fault zone corresponds closely with the outcrop trace of the doleritic intrusive, implying that the structural discontinuity was preferentially exploited by these later intrusions and subsequent brittle deformation.

3.3.1 Geological Map

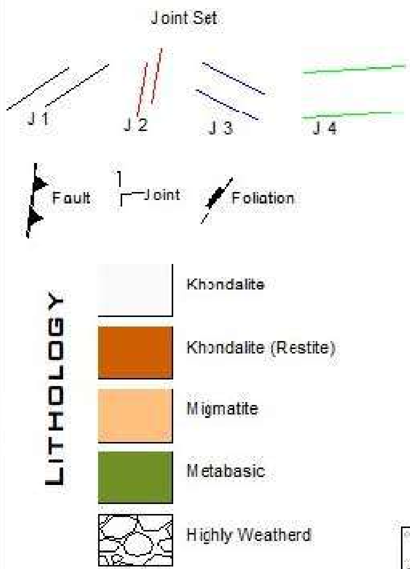




LAYOUT-2

NOTE:

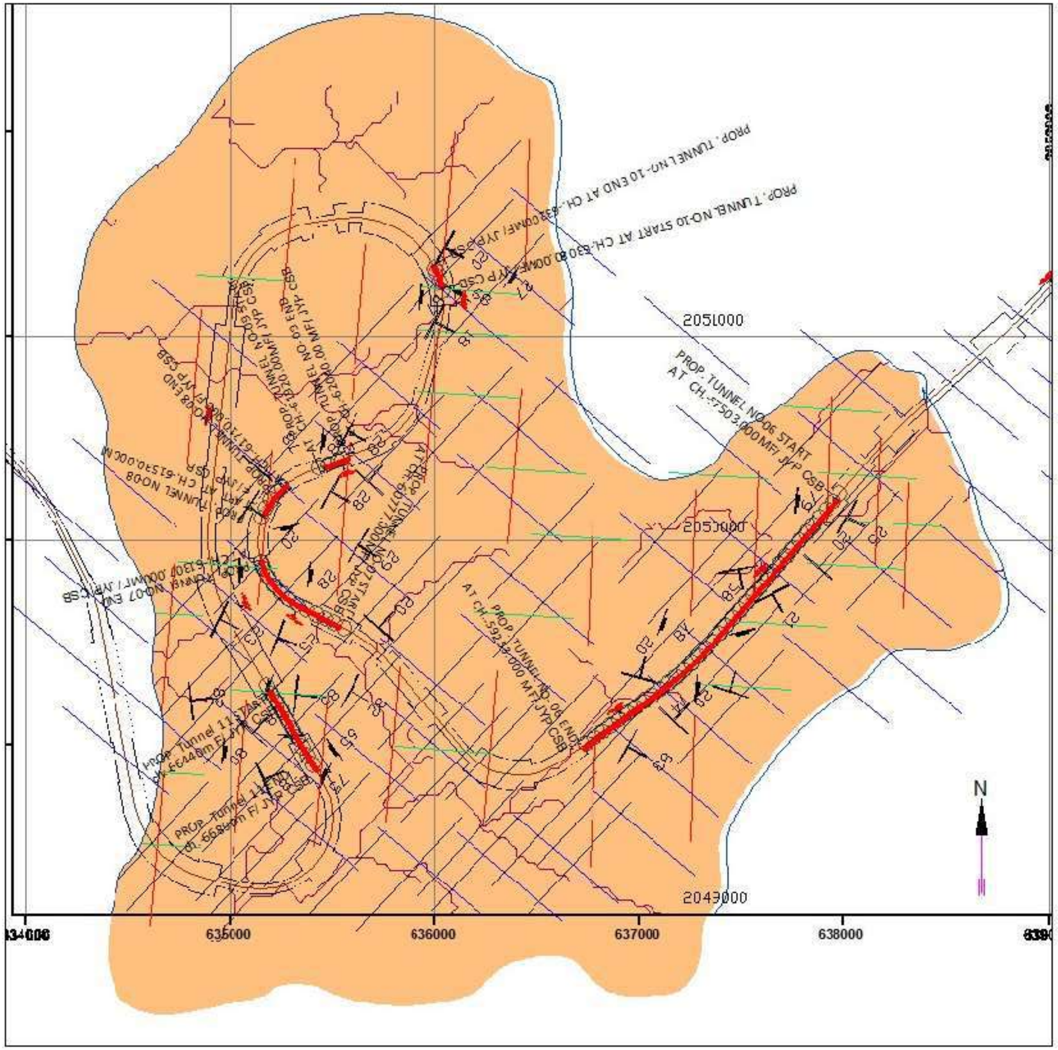
1. All dimension values are in meter.
2. Some random and discrete joints/fractures, as schematically shown in the L-section, may be expected to encounter in the tunnel influence zone.
3. RMR classification presented here are adopted from Bieniawski (1976) Rock Mass Classification.
4. Thin veneer of soil has been found at someplaces. However, the thickness of soil layer, being out of the scale, are not shown in the L-Section.
5. Joint sets are schematically represented . Hence the joint and local shear/crushed zone spacing are not to the scale.



JOINT SET

Set Name	Strike	Dip	Direction	Spacing
J 1	020-040	58	NE	70
J 2	350-010	40	SE	42
J 3	290-310	75	S	70
J 4	80-100	87	SW	20

LAYOUT-3



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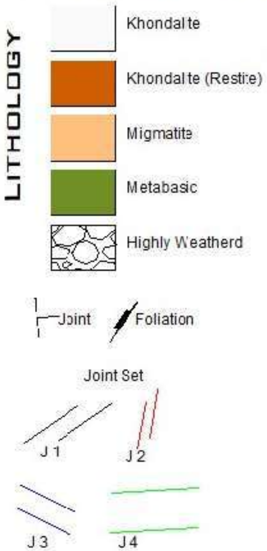

JOINT SET				
Set Name	Strike	Dip	Direction	Spacing
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J 2	350-010	40	SE	42
J 3	290-310	75	S	70
J 4	80-100	87	SW	20

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3.3.2 Field Photographs

4 GEOMECHANICAL CLASSIFICATION

4.1 Rock Quality Designation (RQD)

Rock Quality Designation (RQD) is a measure of quality of rock core taken from a borehole. RQD signifies the degree of jointing or fracture in a rock mass measured in percentage, where RQD of 75% or more shows good quality hard rock and less than 50% show low quality weathered rocks. RFQ is calculated by taking a rock core sample from a borehole and lengths of all sound rock pieces which are minimum 100 mm long are summed up and are divided by the length of the core run. Only those pieces of rocks are considered which are hard and good quality. Weathered rocks which do not meet soundness requirements and whose lengths are not greater than 100mm are not considered for calculation of RQD. The length of core pieces is measured along center line of the pieces. RFQ test provides assessment of soundness of the rock and damages caused due to weathering.

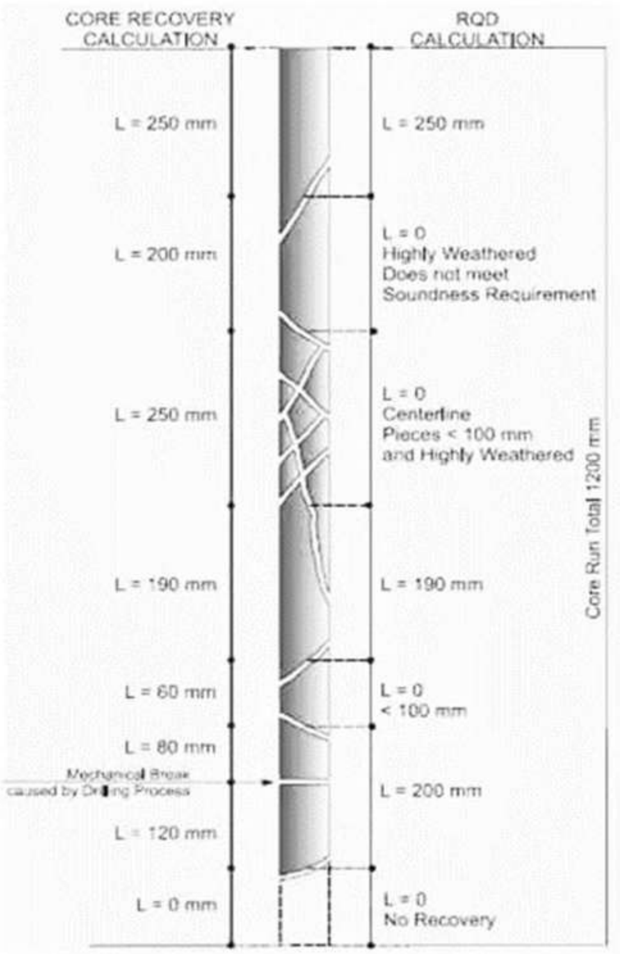

Table 4.1 Categories of rock with respect to their RQD value

Rock Quality	RQD (%)
Very poor (Completely weathered rock)	<25%
Poor (weathered rocks)	25 to 50%
Fair (Moderately weathered rocks)	51 to 75%
Good (Hard Rock)	76 to 90%
Very Good (Fresh rocks)	91 to 100%

4.1.1 Rocks Core Recovery and RQD Calculations:

Core recovery (CR) is calculated by following formula:

$$CR = \left[\frac{total\ length\ of\ rock\ recovered}{Total\ core\ run\ length} \times 100\right]\%$$

$$RQD = \left[\frac{Length\ of\ core\ pieces > 10cm}{Total\ core\ run\ length} \times 100\right]\%$$

ROCK CLASSIFICATION BASED ON RQD		Joint Frequency	RQD (%)
A	VERY POOR ROCK	>27 joints per m^3	0-25
B	POOR	20-27 joints per m^3	25-50

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C	FAIR	13-19 joints per m^3	50-75
D	GOOD	8-12 joints per m^3	75-90
E	EXCELLENT	0-7 joints per m^3	90-100
Note: i) Where RQD is reported or measured as ≤ 10 (including 0) the value 10 is used to evaluate the Q-value ii) RQD-intervals of 5, i.e. 100, 95, 90, etc., are sufficiently accurate.			

4.1.2 Methodology

The drill cores (NX & NQ size) were properly logged and stored in the GI core boxes specifically designed as per the standard specifications. The cores are aligned systematically according to the core run and all the relevant information regarding the core recovery, Rock Quality Designation (RQD), fracture pattern was observed from the geotechnical logging of the drilled holes. The details of the rock type obtained from each drill hole runs were systematically recorded and summarized in the standard Geotechnical logging format.

Core samples were collected from the drill holes at different depth intervals to represent the envisaged strata conditions of the proposed crown and invert portion of different tunnel types.

The assessment of rock mass has been carried out based on the geotechnical investigation, observation of the core logs, joint orientation with reference to the proposed tunnel orientation and physico -mechanical properties of rock cores. The rock mass is classified in to RMR Geo-mechanics Classification system.

The data, thus obtained from geotechnical inputs has been analysed by using both the standard Rock Mass Classification systems. Pre-investigations for underground excavations often include core-logging.

However, special attention has been addressed to the following aspects:

I. Only a small section of each joint surface will usually be available, particularly for joints intersecting the borehole at an obtuse angle. Evaluation of the roughness coefficient (Jr) may therefore be difficult. Particularly the large and medium scale undulation may be difficult to estimate. As water is used during drilling, fillings like clay minerals may be washed out, making it difficult to evaluate in some cases.

II. The drilling direction of the borehole influences the number of joints that are intersected by the borehole. Sub-parallel joints to the borehole will be under represented in the cores, and this will give too high RQD-values and too low Jn values. Whereas, RQD is often calculated for every meter, Jn must usually be estimated for sections of several meters.

III. In massive rock it is impossible to estimate SRF (Stress Reduction Factor) from drill cores. However, in rock intersected by weakness zones, it may be possible to give some suggestions about SRF. In massive rock, SRF can be estimated partially based on the overburden, height of a mountain side, stress measurements carried out in the borehole, or experiences from nearby construction sites.

IV. In general, a core log should only contain data obtained from the cores or measurements carried out in the borehole itself. However, by using the log data combined with estimates of J_w and SRF, it will be possible to get a rough impression of the Q-values of the cores, and these could be helpful during planning phase. Water-loss tests are often carried out during core drilling. The results are normally given in Lugeon (Lugeon = the loss of water in litters per minute and per meter borehole at an over-pressure of 1 MPa), and form the basis for evaluation of the J_w-value. One also must take into account whether the rock mass is going to be grouted or not in order to estimate the Q-value as a basis for rock support after excavation.

V. It is always important to evaluate how representative the cores are. Boreholes are often drilled just in order to investigate zones. It is then imperative to consider how much of the total rock masses these zones represent. If a borehole is orientated along a fracture zone, the parameter values for this zone will be determined.

4.2 Rock Mass Rating Index (RMR):

Bieniawski (1976) published the details of a rock mass classification called the Geomechanics Classification or the Rock Mass Rating (RMR) system. Over the years, this system has been successively re- fined as more case records have been examined and the reader should be aware that Bieniawski has made significant changes in the ratings assigned to different parameters. The discussion which follows is based upon the 1989 version of the classification (Bieniawski, 1989). The following six parameters are used to classify a rock mass using the RMR system:

i) Uniaxial compressive strength of rock material.
ii) Rock Quality Designation (RQD).
iii) Spacing of discontinuities.
iv) Condition of discontinuities.
   a) Length, persistence
   b) Separation
   c) Smoothness
   d) Infilling
   e) Alteration / weathering
v) Groundwater conditions.
vi) Orientation of discontinuities.

All of these are measurable in the field and can also be obtained from borehole data. The rating of each of these parameters is summarized to give a value of RMR. All parameters are measurable in the field and some of them may also be obtained from borehole data.

To apply the RMR classification, the rock mass along a tunnel route is divided into several structural regions, i.e., zones in which certain geological feature are more or less uniform. The above six classification parameters are determined for each structural region from measurements in the field. Once the classification parameters are determined, the ratings are assigned to each parameter according to Table 4.2.

Table 4.2: RMR Classification Parameters and Their Ratings

	PARAMETER		Range of values // ratings						
1	Strength of intact rock material	Point-load strength index	> 10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range Uniaxial compr. strength is preferred		
		Uniaxial com- pressive strength	> 250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 -5 MPa	< 1 MPa
	RATING		**15**	**12**	**7**	**4**	**2**	**1**	**0**
2	Drill core quality RQD		90 - 100%	75 - 90%	50 - 75%	25 - 50%	< 25%		
	RATING		**20**	**17**	**13**	**8**	**5**		

3	Spacing of discontinuities		> 2 m	0.6 - 2 m	200 - 600 mm	60 - 200 mm	< 60 mm
	RATING		**20**	**15**	**10**	**8**	**5**
4	Condition of discontinuities	Length, persistence	< 1 m	1 - 3 m	3 - 10 m	10 - 20 m	> 20 m
		Rating	**6**	**4**	**2**	**1**	**0**
		Separation	None	< 0.1 mm	0.1 - 1 mm	1 - 5 mm	> 5 mm
		Rating	**6**	**5**	**4**	**1**	**0**
		Roughness	very rough	Rough	slightly rough	smooth	Slickensided
		Rating	**6**	**5**	**3**	**1**	**0**
		Infilling (gouge)	None	Hard filling		Soft filling	
			-	< 5 mm	> 5 mm	< 5 mm	> 5 mm
		Rating	**6**	**4**	**2**	**2**	**0**
		Weathering	unweathered	slightly w.	moderately w.	highly w.	Decomposed
		Rating	**6**	**5**	**3**	**1**	**0**
5	Ground water	Inflow per 10 m tunnel length	None	< 10 litres/min	10 - 25 litres/min	25 - 125 litres/min	> 125 litres /min
		p_w / □1	0	0 - 0.1	0.1 - 0.2	0.2 - 0.5	> 0.5
		General conditions	completely dry	Damp	Wet	dripping	Flowing
	RATING		**15**	**10**	**7**	**4**	**0**
p_w = joint water pressure; □1 = major principal stress							

In this respect the typical, rather than the worst conditions, are evaluated. Furthermore, it should be noted that the ratings, which are given for discontinuity spacing, apply to rock masses having three sets of discontinuities. Thus, when only two sets of discontinuities are present, a conservative assessment is obtained.

Table 4.3: Rating Adjustment for Discontinuity Orientations

		Very favorable	Favorable	Fair	Unfavorable	Very unfavorable
RATINGS	Tunnels	**0**	**-2**	**-5**	**-10**	**-12**
	Foundations	**0**	**-2**	**-7**	**-15**	**-25**
	Slopes	**0**	**-5**	**-25**	**-50**	**-60**

Table 4.4: Rock Mass Classes Determined from Total Ratings

Rating	100 - 81	80 – 61	60 – 41	40 - 21	< 20
Class No.	I	II	III	IV	V
Description	VERY GOOD	GOOD	FAIR	POOR	VERY POOR

Table 4.5 Significance of Rock Mass Classes

Class No.	I	II	III	IV	V
Average stand-up time	10 years for 15 m span	6 months for 8 m span	1 week for 5 m span	10 hours for 2.5 m span	30 minutes for 1 m span
Cohesion of the rock mass	> 400 kPa	300 - 400 kPa	200 - 300 kPa	100 - 200 kPa	< 100 kPa
Friction angle of the rock mass	< 45°	35 - 45°	25 - 35°	15 - 25°	< 15°

Table 4.6: RMR Classification Guide for Excavation and Support in Rock Tunnels

Rock mass class	**Excavation**	**Support**		
		Rock bolts (20 mm diam., fully bonded)	Shotcrete	Steel sets
1. Very good rock RMR: 81-100	Full face: 3 m advance	Generally, no support required except for occasional spot bolting		
2. Good rock RMR: 61-80	Full face: 1.0-1.5 m advance; Complete support 20 m from face	Locally bolts in crown, 3 m long, spaced 2.5 m with occasional wire mesh	50 mm in crown where required	None
3. Fair rock RMR: 41-60	Top heading and bench: 1.5-3 m advance in top heading; Commence support after each blast; Commence support 10 m from face	Systematic bolts 4 m long, spaced 1.5-2 m in crown and walls with wire mesh in crown	50-100 mm in crown, and 30 mm in sides	None
4. Poor rock RMR: 21-40	Top heading and bench: 1.0-1.5 m advance in top heading; Install support concurrently with excavation - 10 m from face	Systematic bolts 4-5 m long, spaced 1-1.5 m in crown and walls with wire mesh	100-150 mm in crown and 100 mm in sides	Light ribs spaced 1.5 m where required
5. Very poor rock RMR < 21	Multiple drifts: 0.5-1.5 m advance in top heading; Install support concurrently with excavation; shotcrete as soon as possible after blasting	Systematic bolts 5-6 m long, spaced 1-1.5 m in crown and walls with wire mesh. Bolt invert	150-200 mm in crown, 150 mm in sides, and 50 mm on face	Medium to heavy ribs spaced 0.75 m with steel lagging and fore poling if required. Close invert

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In applying this classification system, the rock mass is divided into a number of structural regions and each region is classified separately. The boundaries of the structural regions usually coincide with a major structural feature such as a fault or with a change in rock type. In some cases, significant changes in discontinuity spacing or characteristics, within the same rock type, may necessitate the division of the rock mass into a number of small structural regions or domains. The Rock Mass Rating system is presented in Table 4.2, giving the ratings for each of the six parameters listed above. These ratings are summed to give a value of RMR.

5 ENGINEERING PROPERTIES

5.1 Laboratory Testing of Rock Mass

Laboratory tests were also carried out on rock samples, the details of different laboratory tests conducted as part of the project are given in the table below

Table 5.1: The laboratory tests conducted for rock.

Laboratory tests conducted for rock	1. Specific Gravity
	2. Dry Density
	3. Water Absorption Test
	4. Porosity
	5. Hardness
	6. Unconfined Compression Test
	7. Point Load Test
	8. Brazilian Tensile Test
	9. Modulus of Elasticity
	10. Abrasion Test

5.1.1 Selection of Core Sample

Representative core samples (NX and NQ size) are collected for covering the crown and invert section and other portions above the crown. The samples were properly labelled and packed carefully and sent NABL accredited Bhubaneswar laboratory for determining the physico-mechanical properties.

The physico-mechanical properties like unit weight, water absorption, specific gravity, uniaxial compressive strength (UCS), tensile strength (TS), modulus of elasticity, is determined. Simultaneously, the specific heat, and petrography tests are also conducted on the rock samples representing to the tunnel influence zone. The following laboratory tests have been conducted to determine intact rock properties.

5.1.2 Tensile Strength

Brazilian test is intended to measure the tensile strength of a rock sample in the form of specimens of regular geometry. The test is mainly intended for strength classification and characterization of intact rock. The test specimens are right circular cylinders having a length to diameter (L: D) ratio approximately equal to 0.5 and a diameter shall not be less than 45 mm. This method of determining tensile strength is an indirect method, and is popularly known as Brazilian method. The indirect tensile strength is calculated as follows:

$$\sigma_t = \frac{2P}{\pi D t}$$

Where:

σ_t = Brazilian tensile strength (MPa); D = Diameter of the core sample (mm);

P = Maximum failure load (N); t = Thickness or Length of the sample (mm)

5.1.3 Unconfined Compressive Strength

UCS test is intended to determine the unconfined compressive strength of a rock sample in the form of specimens of regular geometry. The length to diameter ratio of cylindrical specimen shall preferably be 2 to 3. If the ratio is less than 2, usual correction shall be applied taking standard slenderness ratio as 2. Load on the specimen shall be applied continuously at a constant stress rate such that failure will take place in about 5 to 15 minutes of loading. Alternatively, the stress rate shall be within the limits of 0.5 MPa/s to 1 MPa/s. The unconfined compressive strength of the specimen has been calculated by dividing the maximum load carried by the specimen during the test, by the average original cross-sectional area.

5.1.4 Density, Specific Gravity, Water Absorption

These tests are performed as per relevant standard. The Bulk volume is obtained by buoyancy technique and the pore volume is obtained by water saturation. It may also be applied to a sample in the form of specimen of irregular geometry.

Based on the tests conducted, the unit weight has been calculated

Specific gravity has been estimated for core samples picked up from different borehole. The true specific gravity has been expressed as a numerical value and shall be based on average of three determinations.

5.1.5 Modulus of Elasticity

This test is intended to determine the Modulus of Elasticity of cylindrical rock specimen in compression. Circumferential and axial deformations or strains may be determined from data obtained by electrical resistance strain gauges, compress meters, optical devices, or other suitable means. The design of the measuring device shall be such that the average of at least two circumferential and two axial strain measurements can be determined for each increment of load. Measuring positions shall be equally spaced around the circumference of the specimens close to the mid height. They should not fall within D/2 of the specimen ends, where D is the diameter.

5.1.5.1 Calculation

The axial strain (ε_a) and the diametric strain (ε_d) may be recorded directly from strain indicating equipment or may be calculated from the measured deformation depending upon the type of apparatus or instrument used.

The axial (ε_a) and diametric (ε_d) strains shall be calculated as follows:

$$\varepsilon a = \Delta l / l$$

$$\varepsilon d = \Delta d / d$$

Where l = original axial length before deformation,

d = original diameter before the deformation,

Δl = change in measured axial length (positive for a decrease in length), and

Δd = change in diameter (positive for an in increase in diameter).

Since,

$$C = \pi d$$

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$$\Delta c = \pi \Delta d$$

The circumferential and diametric strains are related as follows:

$$\varepsilon c = \Delta c / c$$

$$= \pi \Delta \mathrm{d} / \pi \mathrm{d}$$

$$= \Delta d / d$$

$$= \varepsilon d$$

Where, c and d are circumference and diameter of the specimen respectively. The compressive stress in the test specimen σ shall be calculated from compressive load P and the θ initially computed cross-sectional area A, as follows:

$$\sigma = \frac{P}{A}$$

The stress versus axial and lateral strain shall be plotted as a curve.

5.1.6 Hardness Test

Hardness test is intended to determine the hardness number of a rock sample. The length of the sample should be at least 60 mm. Test locations shall be separated by at least twice the diameter of the plunger.

5.1.6.1 Calculation

The correction factor is calculated as: Correction factor=

$$\frac{Specified\ standard\ value\ of\ the\ anvil}{Average\ of\ 10\ reading\ on\ calibration\ anvil}$$

The measured test values for the sample should be tabulated in descending order. The lower 50 percent of the values should be discarded and the average obtained of the upper 50 percent values. This average shall be multiplied by the correction factor

5.1.7 Abrasiveness Test

Abrasiveness test is intended to determine the wear or loss of material which the rock produces on contact with another material.

5.1.7.1 Calculation

Abrasiveness is calculated by following formula

$$\text{CAI or CAIs}=\frac{1}{10\in}\sum_{1}^{10} di$$

CAI or CAIs = Cerchar index for natural or saw cut surface respectively and d_i is diameter of the abraded flat area measured in units of 0.1 mm. If Saw cut specimen is tested, then calculated CAIs of Eq. 1 it is advised to be normalized using Eq.2

CAI= 0.99 CAIs + 0.48

CAI = Cerchar index for natural surface; CAIs= Cerchar index for smooth surface

5.1.8 Petrography Test

The prime requirement of petrography test is the thin section, of 0.3mm thickness, prepared out of the rock sample. The different mineral shows different optical phenomena when they are exposed to transmitted light. The purpose of the preparation of thin section is to make light to transmitted through the rocks. The minerals comprised within the rock are identified from the thin section with the microscope. A keen identification of minerals is required to determine mineral assemblage of rock sample.

5.2 Seismic Refraction Test

5.2.1 Equipment and Accessories:

Following equipment and accessories were used:

1. Seismograph : PASI model 16S24-U Signal enhancement type fully digital 24 channel seismograph

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2. Geophones: Piezoelectric type digital grade vertical geophones, Natural frequency 10 Hz
3. Cable: Geophone spread cables, 5m spacing and waterproof joints, made in Italy
4. Software: WinSism 14.7

5.2.2 Energy Source:

For the seismic refraction survey work to generate the P-wave, an alloy plate was been struck by a 20lb sledge-hammer.

5.2.3 Geophone and Geophone Spread:

Low frequency (10Hz) spike geophones were used to record seismic signals. The general layout of the seismic survey lines was supplied by the client in the form of chainage system, mark at field. Along each survey lines total five (5) hammer shots (two offset shots, two end shots and one centre shot) were taken to collect the field data. The geophones were spiked into the ground at 5 m spacing along the spread length and the spread length were chosen along any particular survey line depending on the site condition, accessibility issue etc. Depending on the spread length, 24 channels of the seismograph were used as per requirement. Proper care was taken to ensure that the pointed ends (spikes) of the geophones were fully embedded in the top-soil.

5.2.4 Seismograph:

Pasi 16S24-U Engineering Seismograph was used to record the field data. The seismograph itself has the capability of signal enhancement or stacking. The seismograph records the arrival of seismic waves through 12 / 24 channels. The seismic waves detected by each geophone are displayed simultaneously on the screen.

5.2.5 Seismic Refraction:

With this technique, in general, a stress is applied at the surface of an elastic media that creates a condition for the associated strains to propagate as elastic waves (P & S) through the subsurface material as pattern of particle deformation travelling with velocities that are dependent on the elastic properties and densities of the media through which they travel.

The basic principle behind seismic refraction survey is to initiate elastic waves at a point at or near the ground surface and to determine at a number of other positions the arrival times of the seismic energy that has traveled along the discontinuities or interfaces between surface layers and totally refracted back to the surface.

The seismic energy source creates the seismic elastic waves. The travel times of these elastic waves are detected by a series of geophones, placed in line within the ground and connected to the seismograph, via. the geophone cables. The seismograph will register these arrival times and display them as traces of time for individual geophones.

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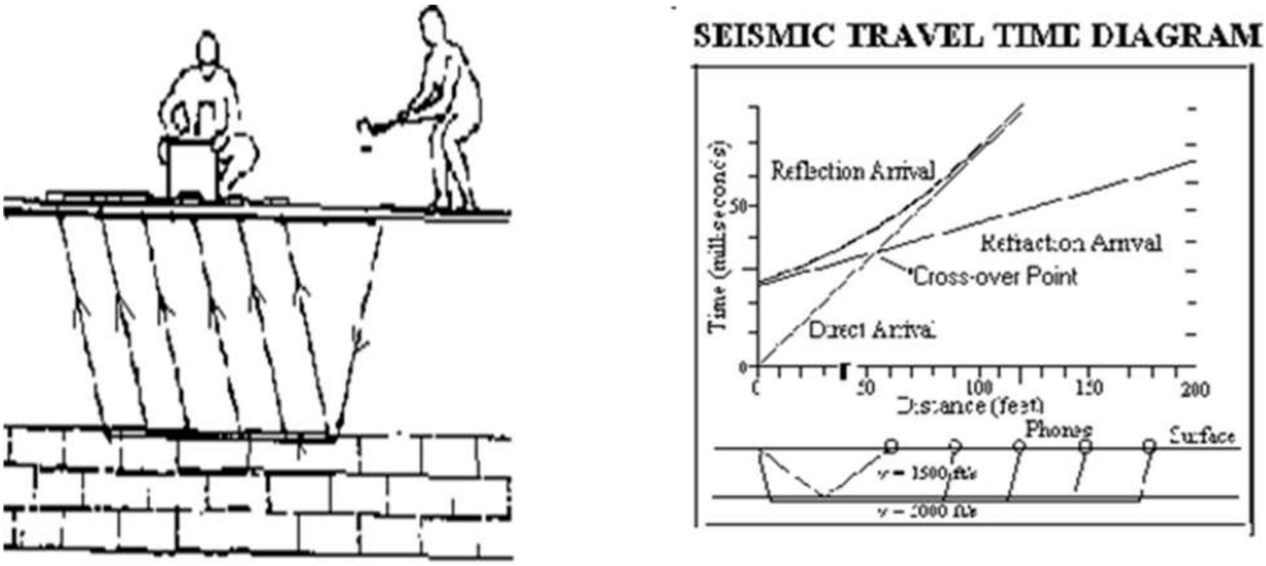
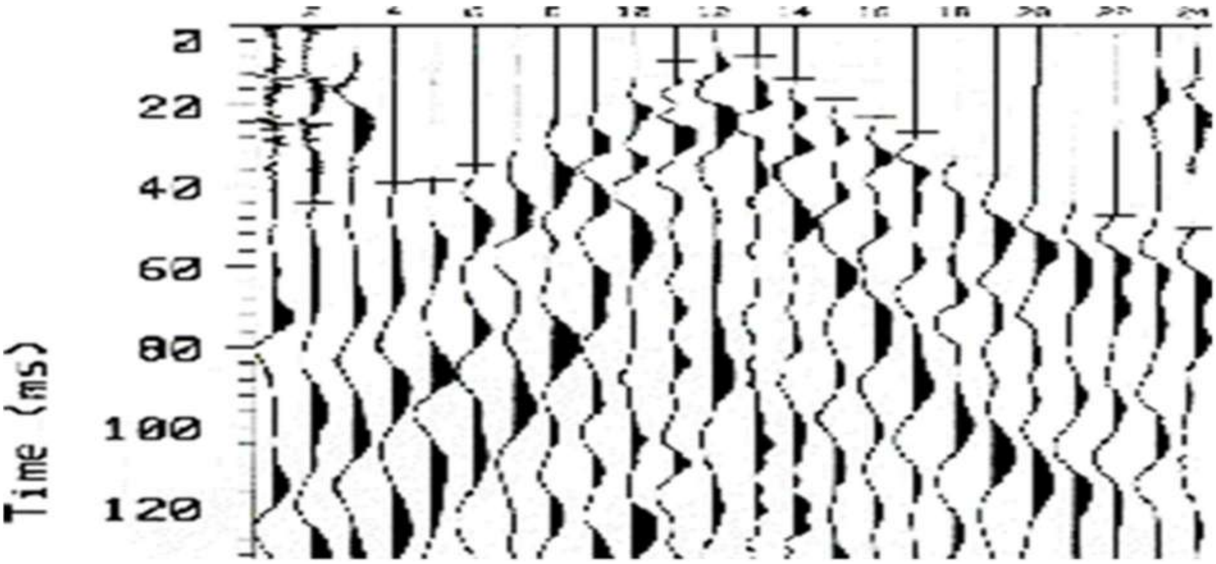


Figure 5.1: Working principle of SRT.

By processing these arrival times, the characteristic velocity and the thickness of the underlying layer can be calculated.



5.2.6 Data interpretation

5.2.6.1 Data processing:

The data is stored in the hard disk of the seismograph at the time of data acquisition. These data are transferred to the computer for further processing. The processing involves picking of the first arrivals, input of data with differential geophone elevations with the first arrivals and finally interpretation of the velocity model. In case of noisy field data, there are intermediate steps of data processing using filtering, amplitude corrections etc.

5.2.6.2 Picking of First Arrivals:

All geometrical data are being displayed under the field record panel on the grid. Now one can pick the first arrival time by clicking on the first break, the travel time is drawn automatically and the first break value is written on the grid. The analysis needs very precise time determination to ensure the exact travel time and a good seismic interpretation. Time scale can be modified by clicking the mouse button on the time scale to select a time range to display.

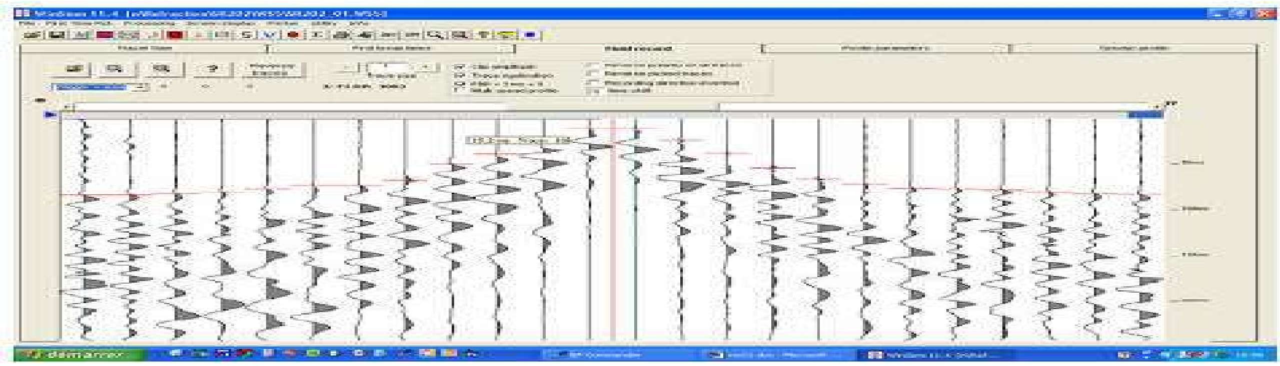


Figure 5.2: First Break Point:

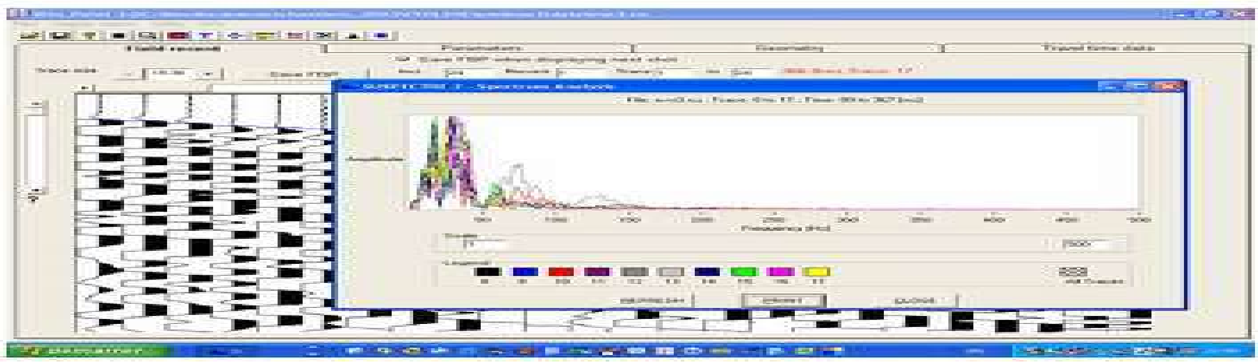


Figure 5.3: Frequency Spectrum

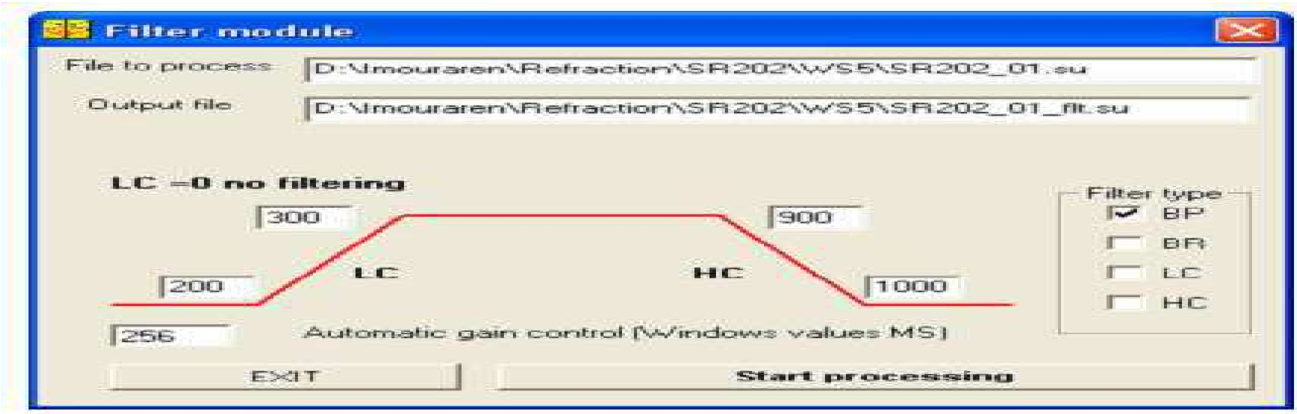


Figure 5.4: Automatic grain & Filtering

5.2.6.3 Velocity Model Development:

a) **Phantom Trace:** With WinSism one can move up and down very easily to any taken shot for comparing the travel times with the other shots. Intercept times can be measured and recorded. Phantoming is used to check whether the parts of two travel times correspond to the same layer or not. The intercept number and intercept time is obtained by giving input of the fixed shot number and moving shot number within the WinSism software.

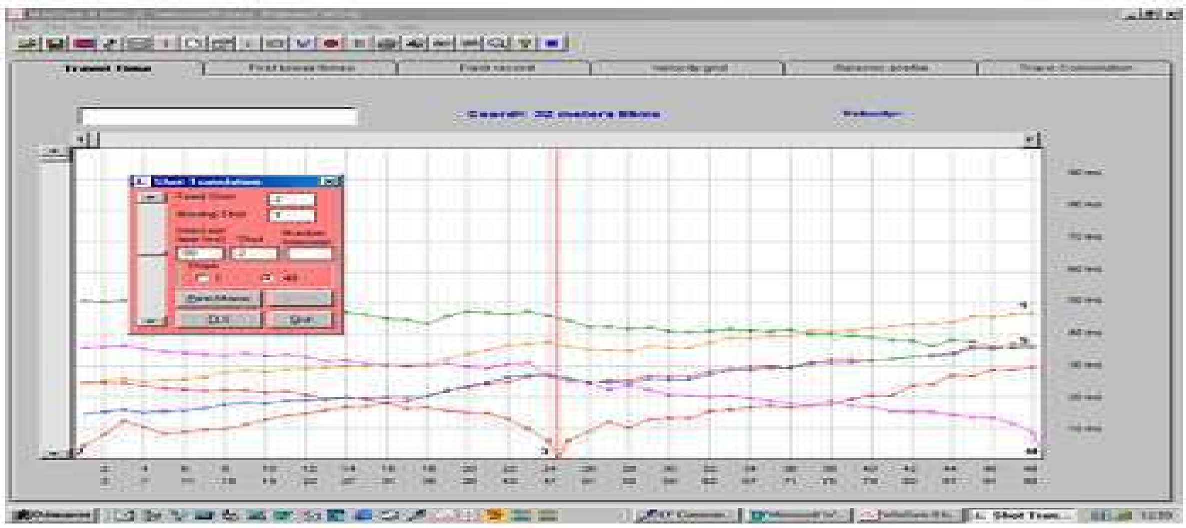


Figure 5.5: Snapshot showing Phantom trace in WinSim.

b) Linear Regression**:** Velocity is computed using linear regression between selected receivers.

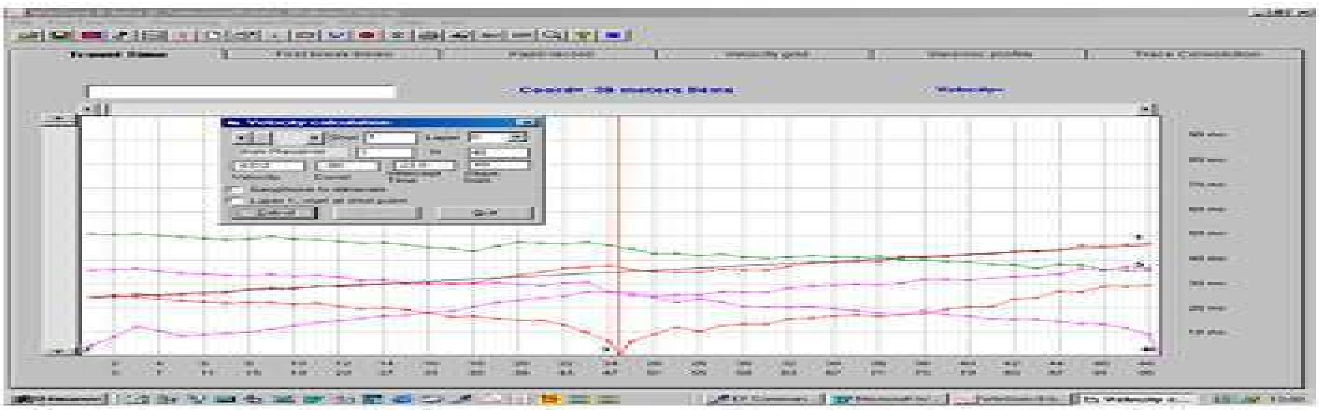


Figure 5.6: Snapshot showing Linear Regression in WinSim

c) True Velocity: One can compute the true velocity, even if the bedrock is not horizontal. In this case, two shots (first and last) and the 1st and last receivers are selected. In this way one will get the real velocity and the correlation factor.

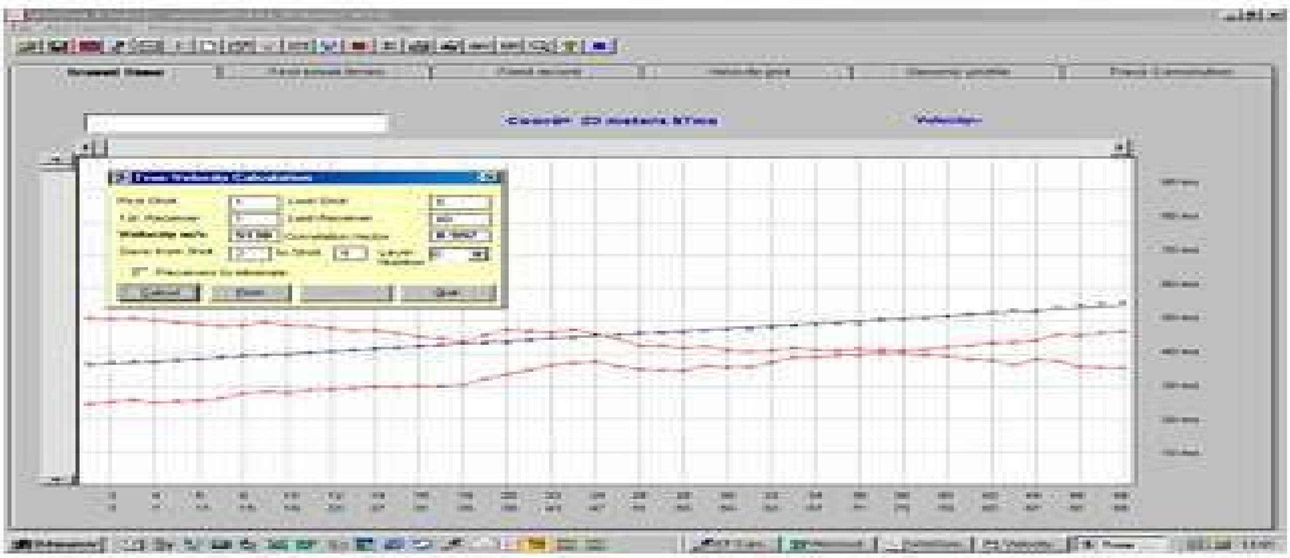


Figure 5.7: Snapshot showing window for evaluation of True Velocity in WinSim

d) Depth Computation: When all velocities and intercept times are recorded, user can compute thickness at shot point using the intercept time method.

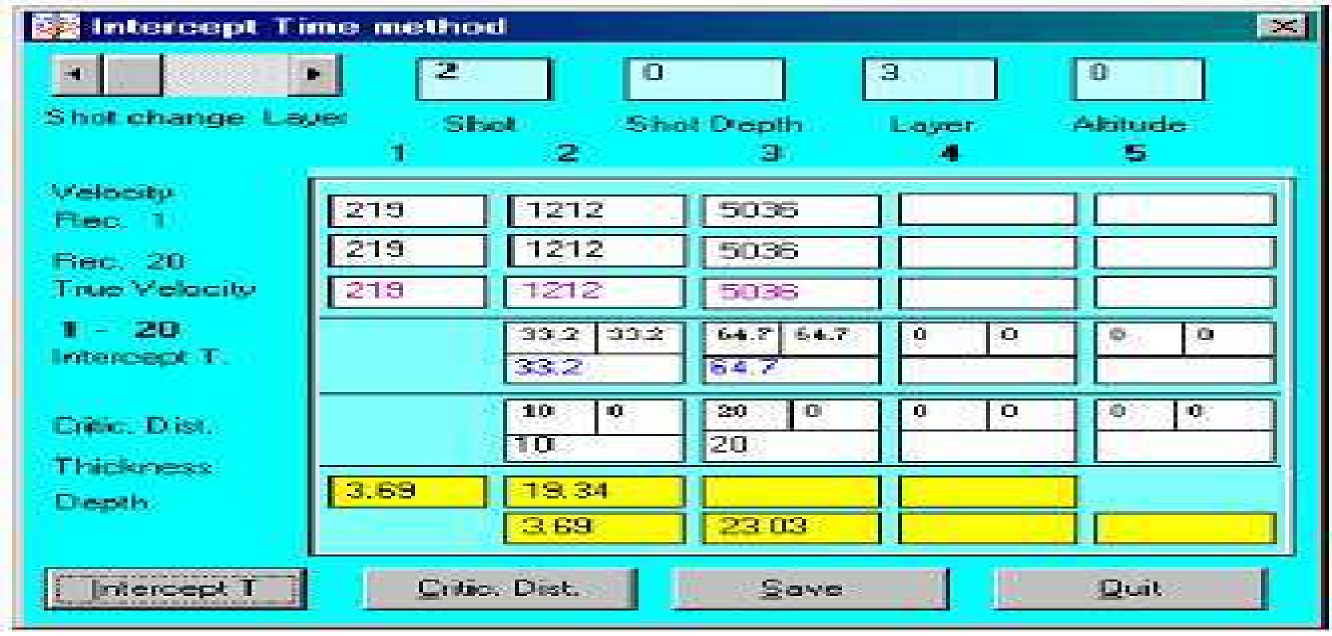


Figure 5.8: Snapshot showing window for depth computation in WinSim

5.2.6.4 *WinSism 14*

WinSism can accept any field geometry, receiver spacing, shot spacing. All distances are computed from the first end-of-line shot (End Shot-1). This shot must always be at distance 0 and is mandatory. First out-of-line shot (Offset Shot-1) must have a negative distance. Spread length will always be from first end-of-line shot to the last end-of-line shot. Usually, the first end-of-line shot distance to the first receiver is half of the receiver spacing. It is to be remembered that two offset shots are mandatory to assess the depth of bed rock.

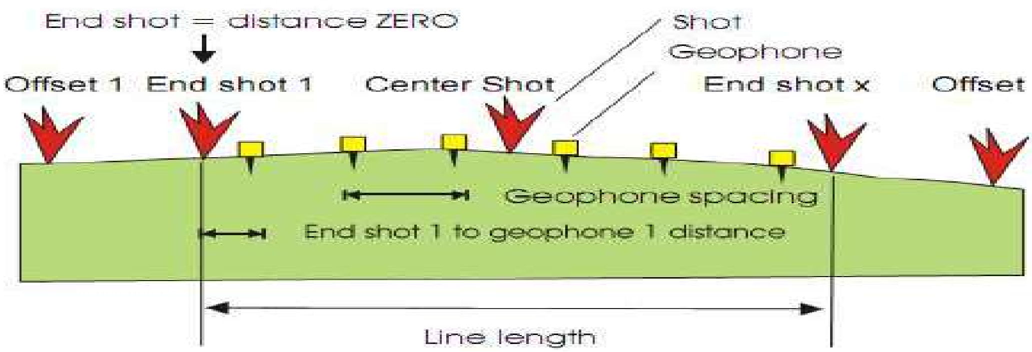
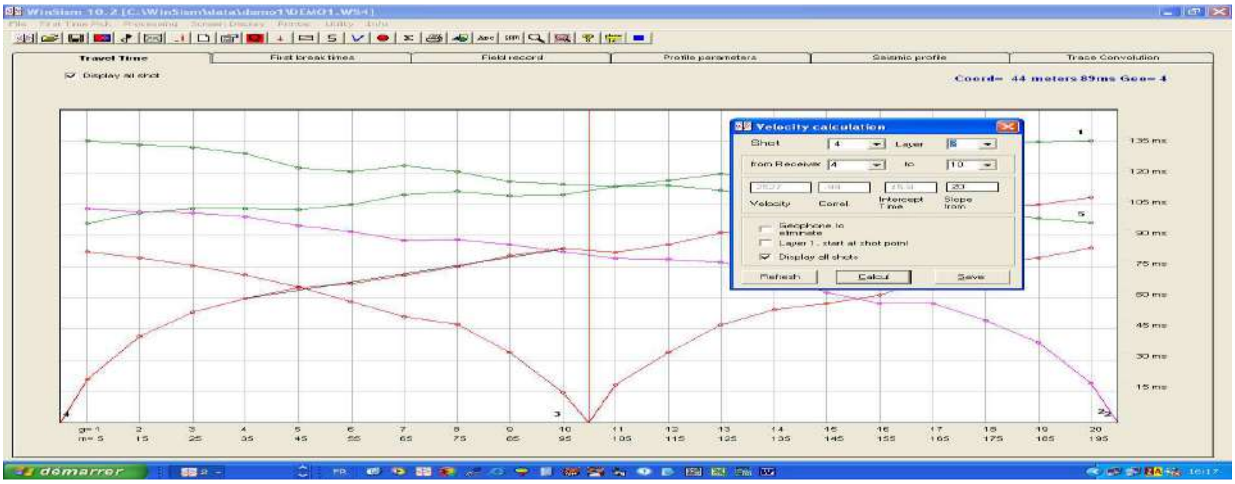


Figure 5.9: Arrangement for shots in SRT.



WinSism Travel time can also analyse more than one spread at a time. In this case, first pick all original records using picking module, without taking care of exact geometry. However, data recording with at least 5 shots, as mentioned below, for each spread are to be taken to compute depth below all geophones:

two offsets' shots (distance half spread length)

two ends shot (half receiver spacing from receiver one)

One centre shot

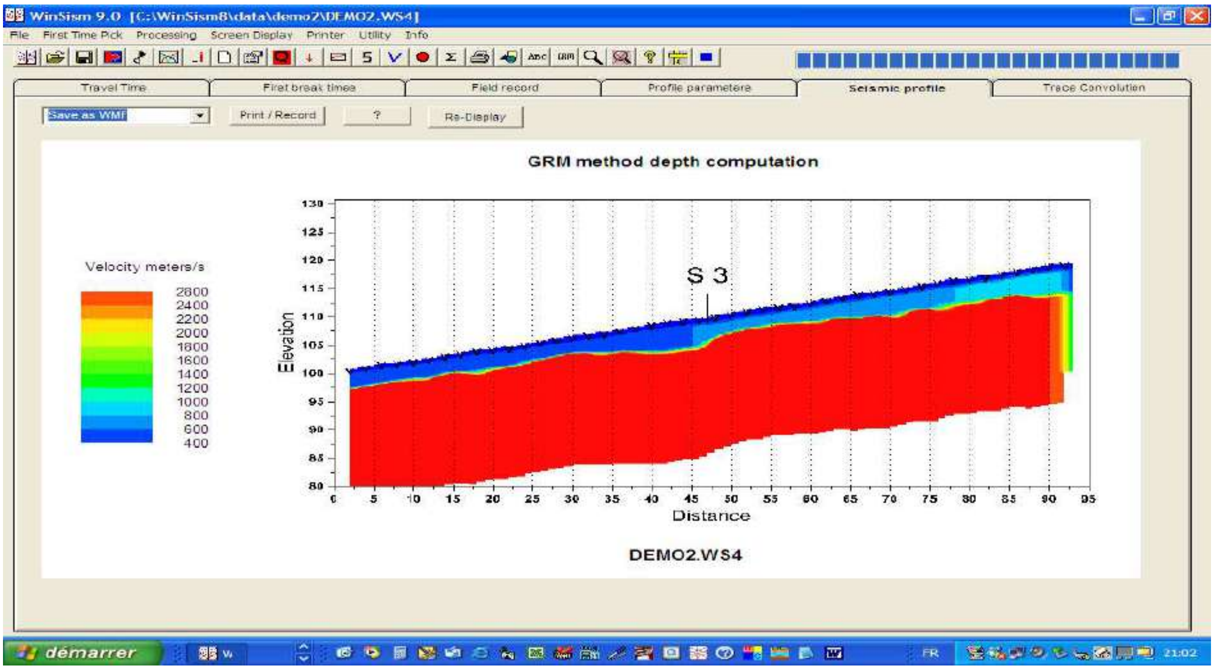


Figure 5.10: Snapshot showing result window in WinSism.

6 TUNNEL 1: GEOLOGICAL & GEOTECHNICAL ASSESMENT

6.1 Exploratory drillings

As per the requirement of scope of work outlined in the terms of reference,10 bore holes were drilled with a cumulative length of374.10 at different locations along the proposed alignment. Necessary care has been taken during drilling operations by deploying good quality diamond drill machines to obtain good core recovery to obtain RQD values. The locations of the boreholes were selected in such a way, so that these holes intersect the envisaged ground/ strata conditions at different depths. The location and details of boreholes drilled; total depth of drillings is shown in table below.

Chainage	BH Name	GL	FL	Depth
41592	BH-1	579.693	561.052	29.00
41642	BH-2	582.671	560.552	40.00
41692	BH-3	594.68	560.052	40.00
41742	BH-4	590.397	559.552	40.10
41792	BH-5	595.527	559.052	42.00
41842	BH-6	586.469	558.552	33.00
41892	BH-7	583.475	558.052	32.00
41942	BH-8	595.272	557.552	43.00
41992	BH-9	593.353	557.052	44.00
42052	BH-10	575.416	556.452	31.00

6.2 TUNNEL-1 SRT

6.2.1 Location:

Sr. No.	Chainage	Line	Spread	Location (T-01)		Length
				Start	End	In meter
1	41.592km to 42.052km	L1	S1 to S4	41.592km	42.052km	460

6.2.2 Seismic survey results and conclusion

Table 6.1: Summery of Tunnel-1 SRT test

Variation of maximum range of thicknesses below EGL (M)			Avg. V_p (m/sec)	Calculated V_s (m/sec)	Dynamic Young's Modulus (MPa)	Shear Modulus (MPa)
Layer	From	To				
Layer-I	0.50	3.50	700	327	466	172
Layer-II	3.50	5.00	2300	1183	8133	3081
Layer-III	5.00	25.00	3400	1879	21702	8477

Sample Calculation:

The Young's Modulus E is the uni-axial stress-strain ratio. Its dynamic value is expressed by the following equation:

$$E = \frac{\rho V_P^2 (1+\mu)(1-2\mu)}{1-\mu}$$

Where, E = Dynamic Young's Modulus in kN/m2

Vp = 700 m/sec

ρ = 1.6 gm/cc ≈ 1.60 kN.s2/m4 (mass density)

μ = 0.36

So, calculated E = 466480 kN/sqm ≈ 466 MPa

The Shear Modulus G is the stress-strain ratio for simple shear. Its dynamic value is obtained by the following:

$$G = \frac{E}{2(1+\mu)} = \rho V_S^2$$

So, Shear Modulus G comes out to be 171500 kN/sqm ≈ 172 MPa

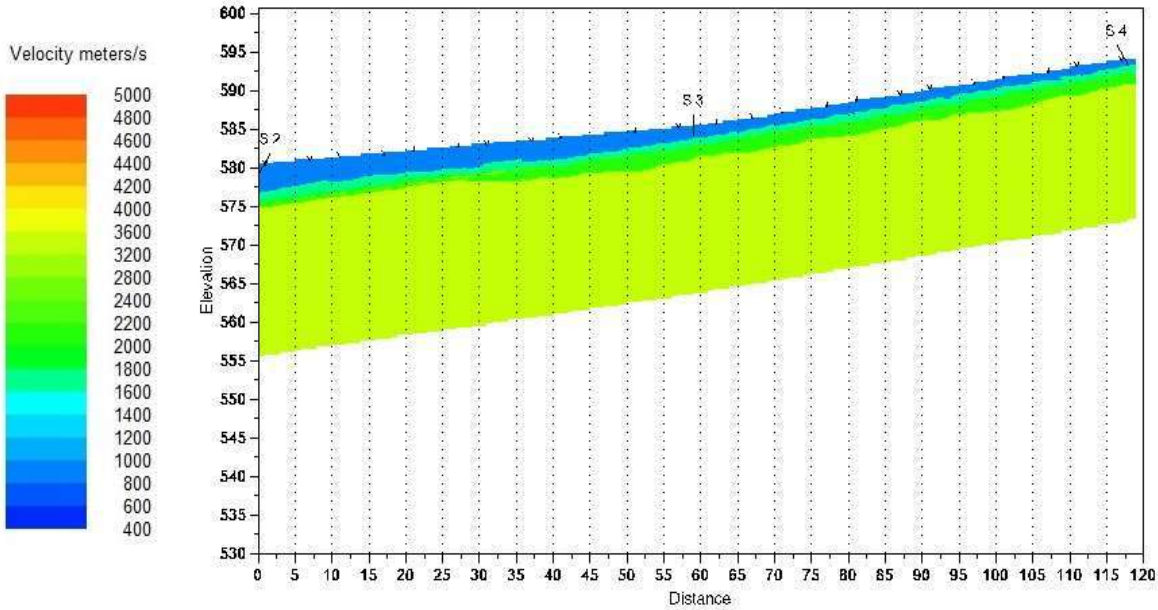
Again,

$G = \rho V_S^2$ giving Vs = √(G/ρ)

So, calculated Vs = 327.395 m/sec, say 327 m/sec

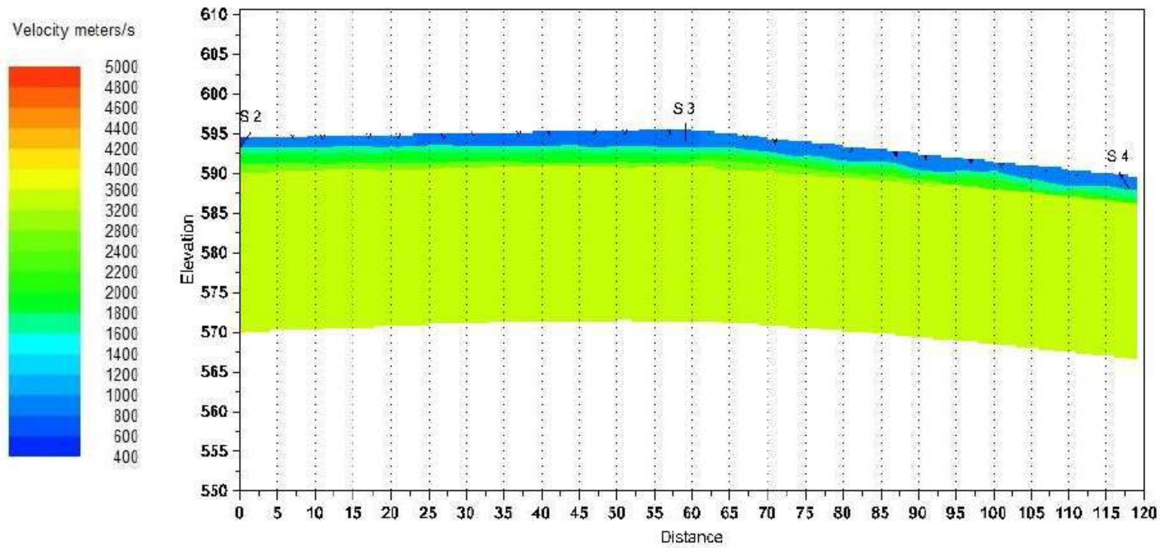
6.2.3 SEISMIC PROFILE(T01)

T1L1S1



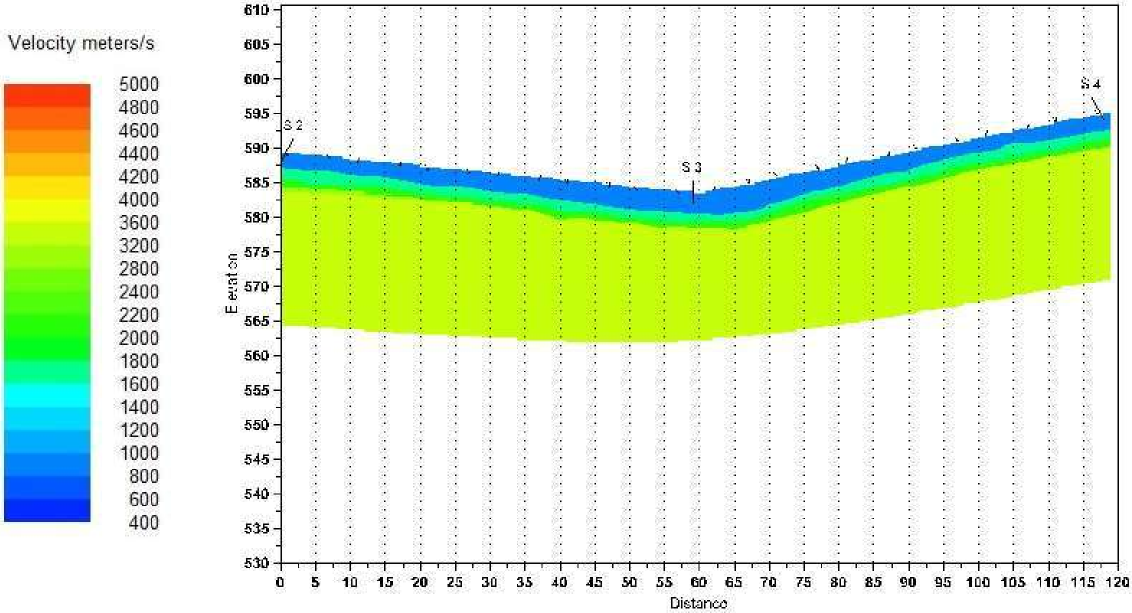
Shot point depth computation

T1L1S2



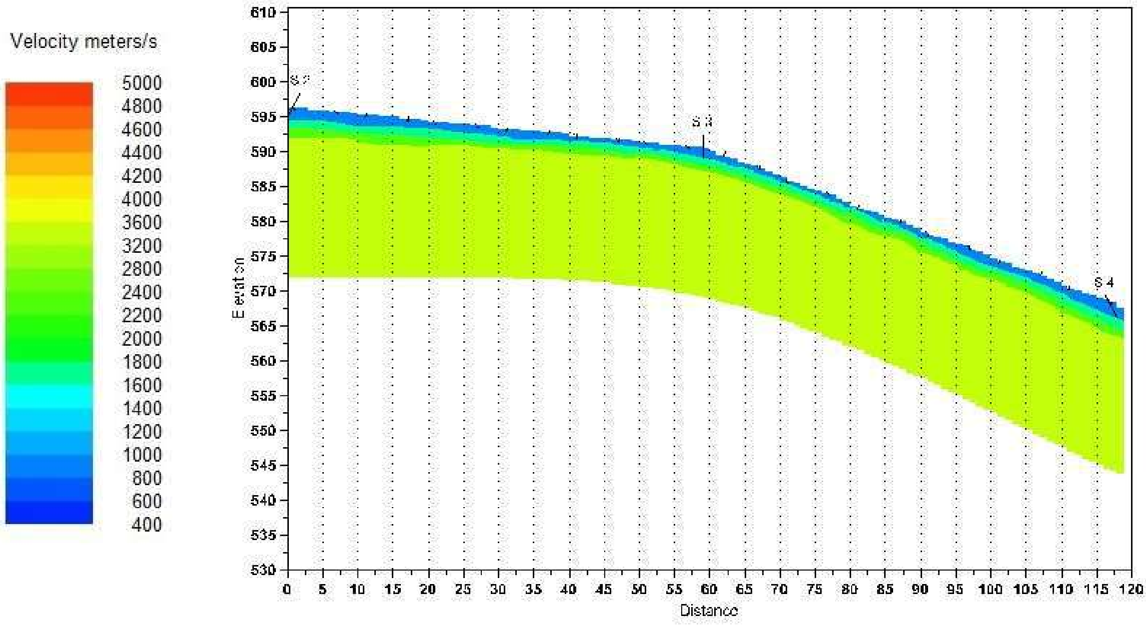
Shot point depth computation

T1L1S3



Shot point depth computation

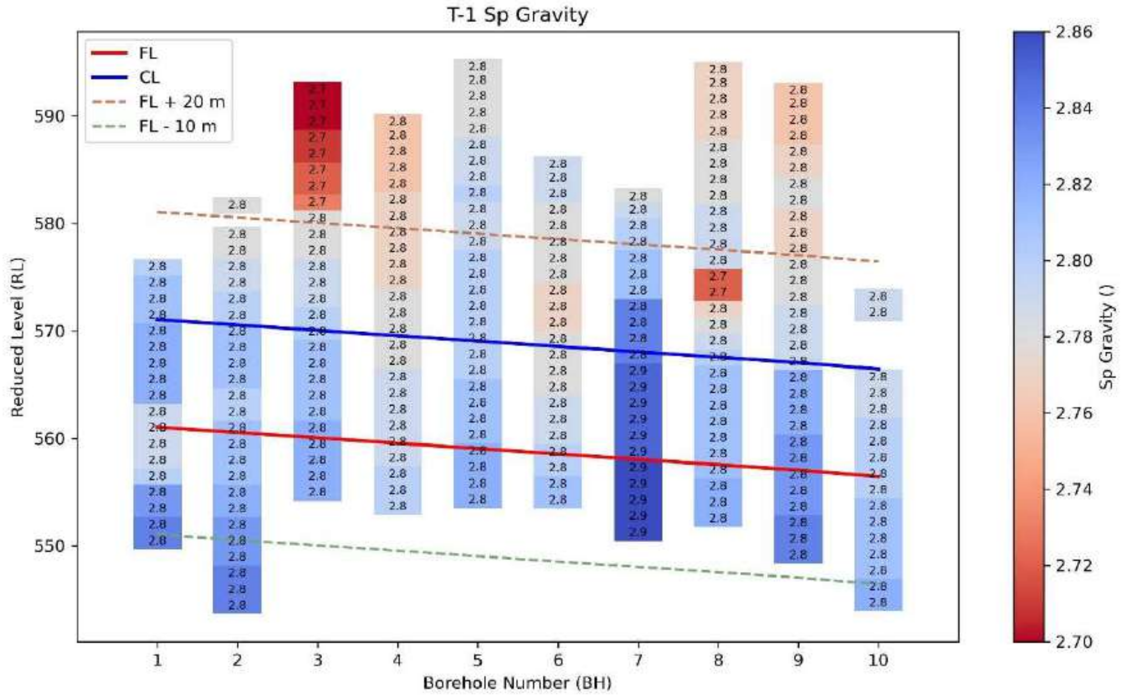
T1L1S4



Shot point depth computation

6.3 Assessment of the engineering properties of rock sample:

6.3.1 Specific Gravity



6.3.1.1 Mean and Standard deviation Specific Gravity considering 1D zone of influence of each borehole:

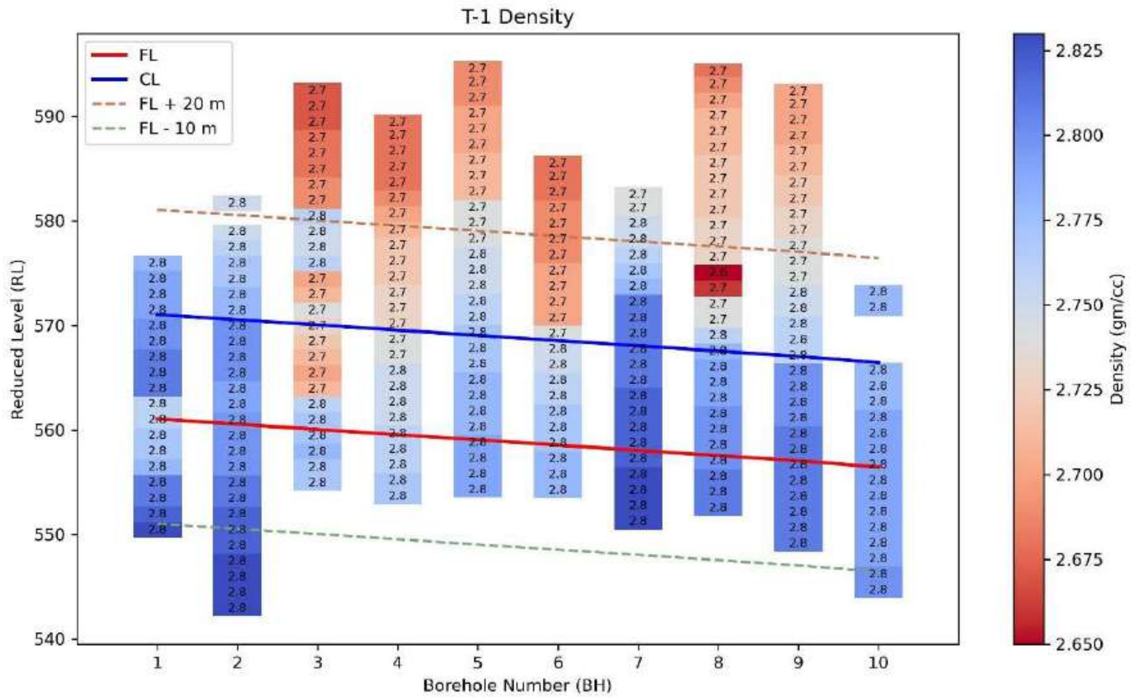
BH No.	Chainage	Mean	Std
BH01	41592	2.812308	0.017867
BH02	41642	2.811538	0.008006
BH03	41692	2.813636	0.006742
BH04	41742	2.790909	0.007006
BH05	41792	2.812	0.007888
BH06	41842	2.793	0.011595

BH No.	Chainage	Mean	Std
BH07	41892	2.853333	0.006513
BH08	41942	2.811818	0.00603
BH09	41992	2.829167	0.00793
BH10	42052	2.801538	0.008006

6.3.1.2 Recommended Specific Gravity considering 1D zone of influence:

Chainage	Statistical / Reduction Method (Sp = μ - σ across boreholes within ±10 m of FL and CL)	Reference Standards / Guidelines
41592 to 42052	2.79	IS 13030:1991; IS 1124:1974

6.3.2 Dry Density



6.3.2.1 Mean and Standard Deviation in Density considering 1D zone of influence of each borehole:

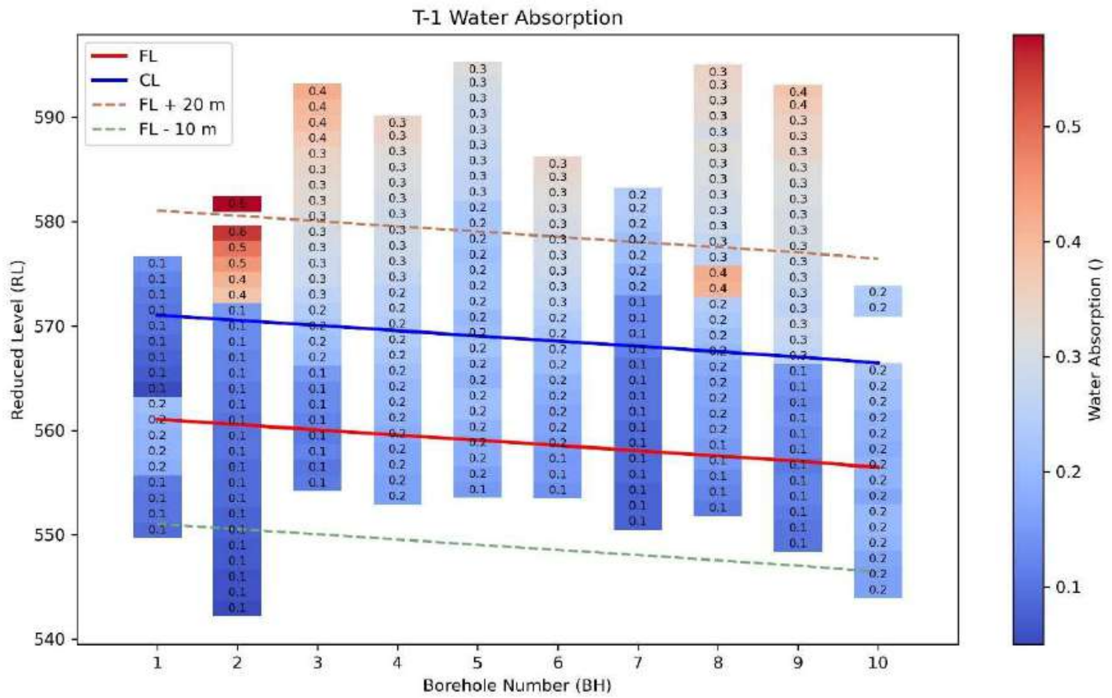
BH No.	Chainage	Mean	Std
BH01	41592	2.793077	0.02175
BH02	41642	2.801	0.008436
BH03	41692	2.744545	0.030451
BH04	41742	2.754545	0.010357
BH05	41792	2.782	0.007888
BH06	41842	2.767	0.011595

BH07	41892	2.820833	0.00793
BH08	41942	2.798182	0.009816
BH09	41992	2.806667	0.004924
BH10	42052	2.787692	0.004385

6.3.2.2 Recommended Density considering 1D zone of influence:

Chainage	Statistical / Reduction Method (d = μ - σ across boreholes within ±10 m of FL and CL)	Reference Standards / Guidelines
41592 to 42052	2.76	IS 13063:1991

6.3.3 Water absorption Test



6.3.3.1 Mean and Standard Deviation in Water Absorption Value considering 1D zone of influence:

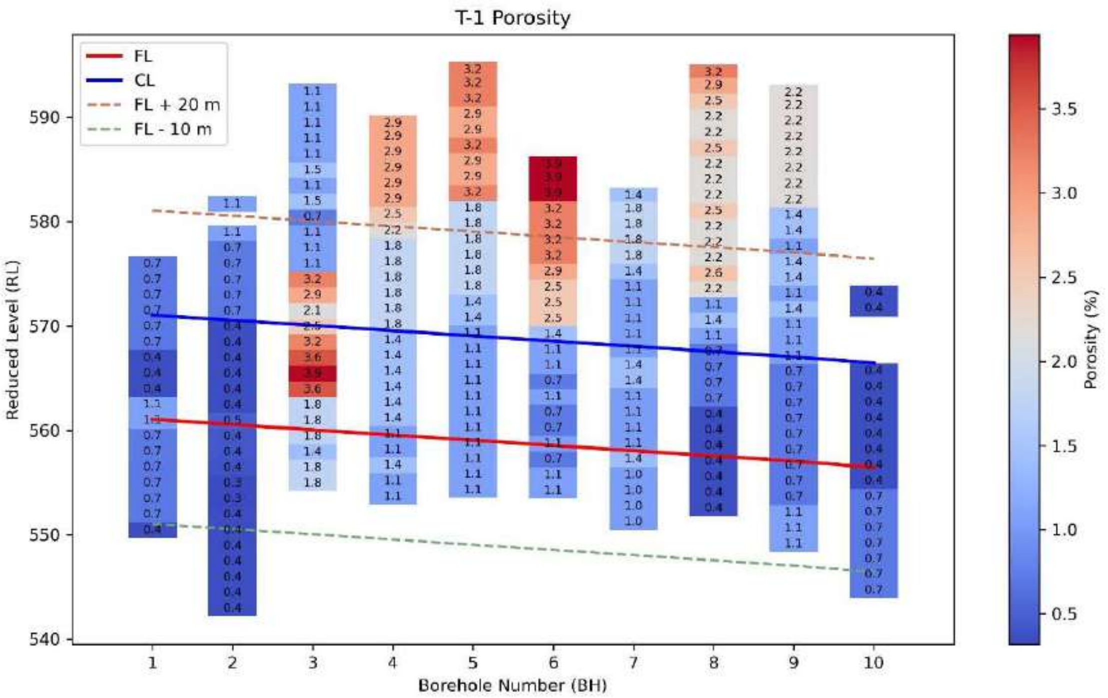
BH No.	Chainage	Mean	Std
BH01	41592	0.128462	0.056398
BH02	41642	0.105385	0.016132
BH03	41692	0.145455	0.043672
BH04	41742	0.191818	0.027136
BH05	41792	0.189	0.022336
BH06	41842	0.173	0.024518

BH No.	Chainage	Mean	Std
BH07	41892	0.093333	0.011547
BH08	41942	0.154545	0.026595
BH09	41992	0.123333	0.016143
BH10	42052	0.193846	0.016602

6.3.3.2 Recommended Water Absorption Value considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(A= μ + σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
41592 to 41842	0.19	IS 13063:1991; IS 2386 (Part 3):1963
41842 to 419992	0.18	
419992 to 42052 (P2)	0.21	

6.3.4 Porosity



6.3.4.1 Mean And Standard Deviation in Porosity considering 1D zone of influence in each Borehole:

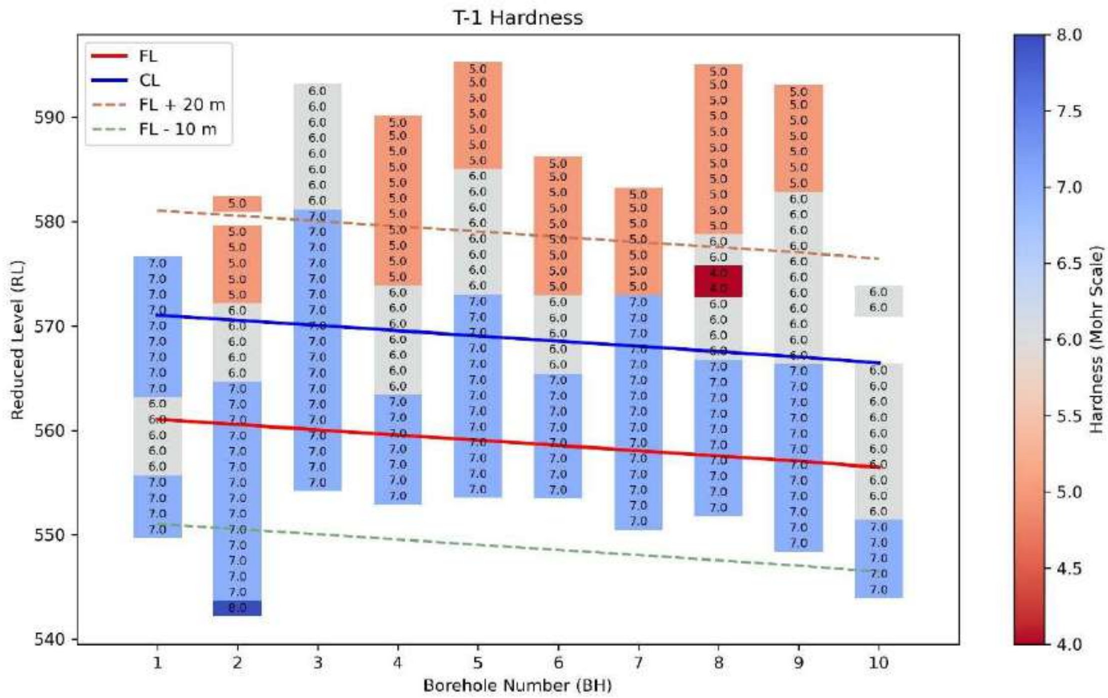
BH No.	Chainage	Mean	Std
BH01	41592	0.684495	0.230482
BH02	41642	0.374862	0.049733
BH03	41692	2.456807	0.922997
BH04	41742	1.30313	0.182201
BH05	41792	1.066864	0.002995
BH06	41842	0.93076	0.184151

BH07	41892	1.139024	0.158524
BH08	41942	0.485167	0.180142
BH09	41992	0.794955	0.157624
BH10	42052	0.493823	0.179317

6.3.4.2 Recommended Porosity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** (**n = μ ± σ across boreholes within ±10 m of FL and CL**)	**Reference Standards / Guidelines**
41592 to 41842	1.10±0.76	IS 1124:1974; ISRM (1981)
41842 to 419992	0.81±0.30	
419992 to 42052 (P2)	0.49±0.17	

6.3.5 Hardness



6.3.5.1 Mean And Standard Deviation in Hardness considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	41592	7	0.50637
BH02	41642	7	0.480384
BH03	41692	7	0
BH04	41742	7	0.504525
BH05	41792	7	0
BH06	41842	7	0.421637

BH No.	Chainage	Mean	Std
BH07	41892	7	0
BH08	41942	7	0.301511
BH09	41992	7	0
BH10	42052	6	0.438529

6.3.5.2 Recommended Hardness considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** (Hi = μ - σ across boreholes within ±10 m of FL and CL)	**Reference Standards / Guidelines**
41592 to 42052	6	IS 13311 (Part 2):1992; ISRM 1978

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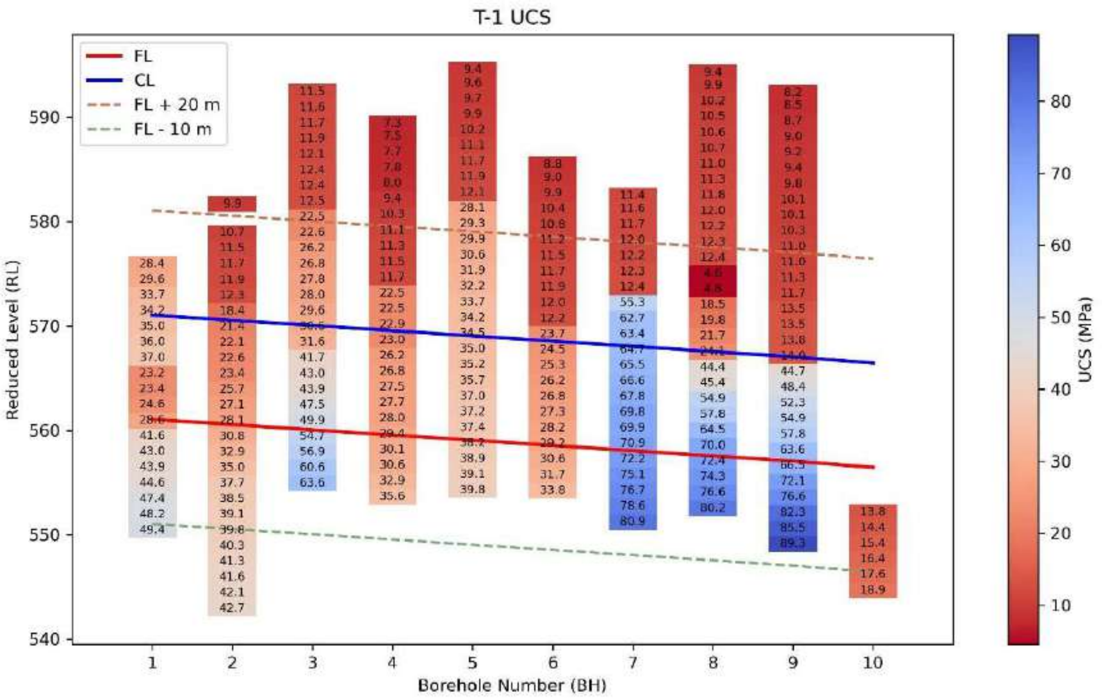
6.3.6 Compression Test



6.3.6.1 Mean And Standard Deviation in UCS considering 1D zone of influence in each Borehole:

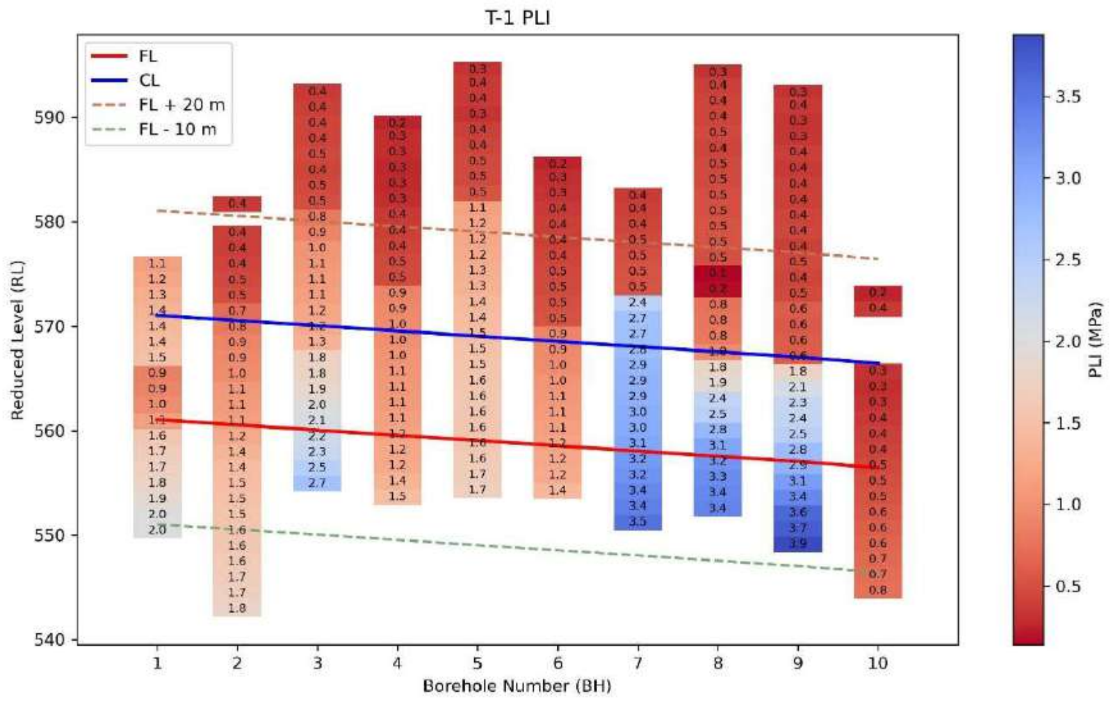
BH No.	Chainage	Mean	Std
BH01	41592	36.65086	9.130635
BH02	41642	29.57138	6.532359
BH03	41692	47.64305	10.87969
BH04	41742	28.89692	3.414377
BH05	41792	37.356	1.670325
BH06	41842	28.349	2.949272
BH07	41892	71.56678	5.256626

BH08	41942	60.41111	17.13712
BH09	41992	66.17499	15.02129
BH10	42052	14.99835	1.110398

6.3.6.2 Recommended UCS considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(UCS_k = μ - σ) across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(UCS_d = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)**	**Reference Standards / Guidelines**
41592 to 41842	25.24	7.5	IS 9143:1979; IS 13365 (Part 2):1998; Hoek & Brown (2002, 2019)
41842 to 419992	52.45	15.73	
419992 to 42052 (P2)	13.88	4.16	

6.3.7 Point Load Test



6.3.7.1 Mean And Standard Deviation in PLI considering 1D zone of influence in each Borehole:

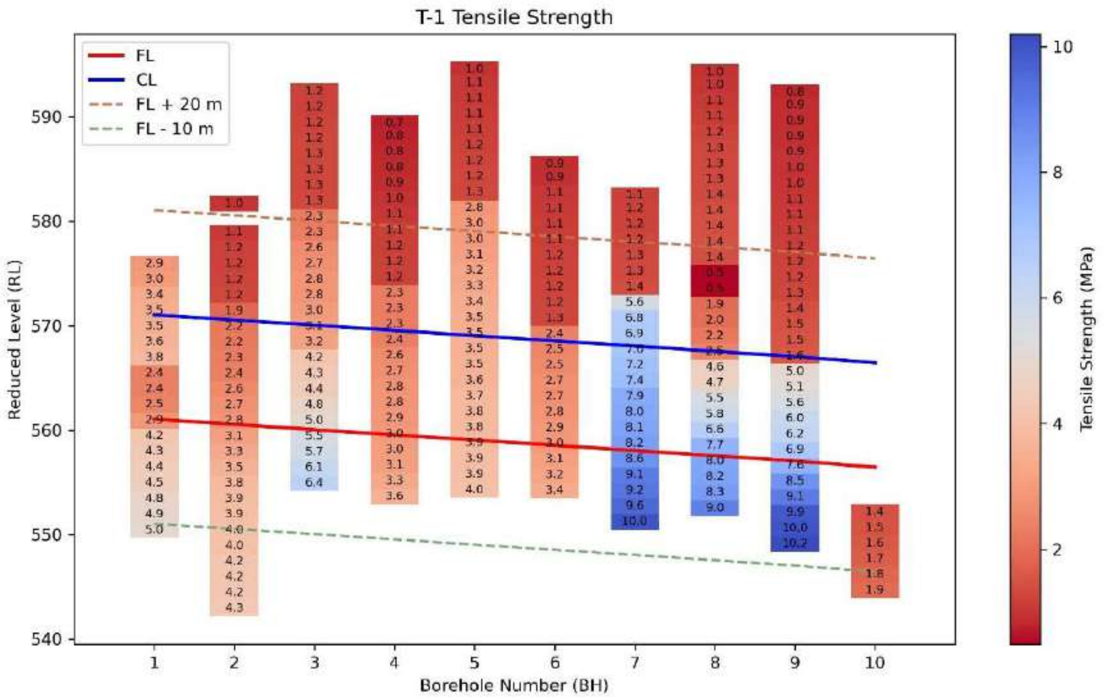
BH No.	Chainage	Mean	Std
BH01	41592	1.448462	0.38214
BH02	41642	1.186923	0.248944
BH03	41692	1.973636	0.449695
BH04	41742	1.165455	0.145008
BH05	41792	1.6	0.063944
BH06	41842	1.126	0.13914

BH07	41892	3.096667	0.242874
BH08	41942	2.601818	0.787843
BH09	41992	2.865833	0.685121
BH10	42052	0.469231	0.130605

6.3.7.2 Recommended PLI considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(PLI_k = μ - σ) across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(PLI_d = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)**	**Reference Standards / Guidelines**
41592 to 41842	1.01	0.3	IS 9143:1979; IS 13365 (Part 2):1998; Hoek & Brown (2002, 2019)
41842 to 419992	2.24	0.7	
419992 to 42052 (P2)	0.34	0.1	

6.3.8 Brazilian Test



6.3.8.1 Mean And Standard Deviation in Tensile Strength considering 1D zone of influence in each Borehole:

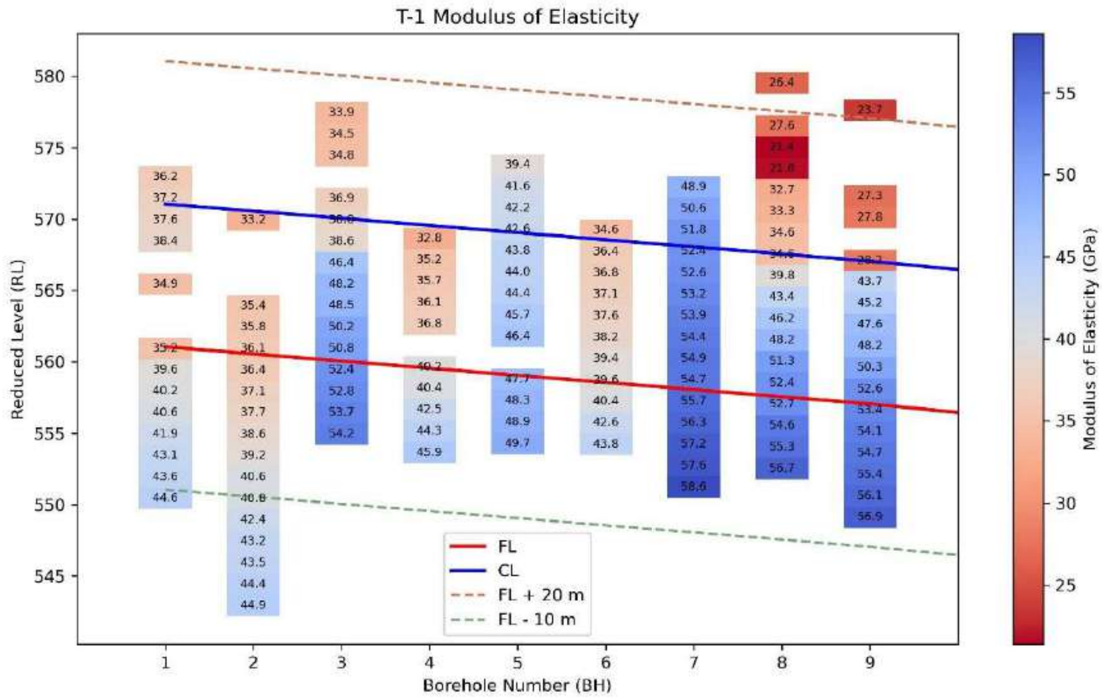
BH No.	Chainage	Mean	Std
BH01	41592	3.702308	0.918469
BH02	41642	2.987692	0.65136
BH03	41692	4.79	1.089927
BH04	41742	2.921818	0.33244
BH05	41792	3.768	0.177501
BH06	41842	2.877	0.293941

BH07	41892	8.360833	0.957349
BH08	41942	6.441818	1.998564
BH09	41992	7.519167	1.965454
BH10	42052	1.56	0.116333

6.3.8.2 Recommended Tensile Strength considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(mean value across boreholes within ±10 m of FL and CL× 0.8 (account for anisotropy))**	**Recommended Design Value** **(μ - σ)**	**Reference Standards / Guidelines**
41592 to 41842	3.5	2.5	IS 10082:1982; Hoek & Brown (1997); ISRM Suggested Methods

6.3.9 Modulus of elasticity test



6.3.9.1 Mean And Standard Deviation in Elasticity considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	41592	39.51	3.013101
BH02	41642	37.01	2.120508
BH03	41692	48.52727	5.605014
BH04	41742	38.99	4.325236
BH05	41792	46.54444	2.21535
BH06	41842	39.19	2.492856

BH07	41892	55.125	1.999602
BH08	41942	48.65455	7.010045
BH09	41992	51.51667	4.425306
BH10	42052	39.51	3.013101

6.3.9.2 Recommended Modulus of Elasticity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Ei = μ - σ across boreholes within ±10 m of FL and CL× 0.8 (account for anisotropy))**	**Recommended Design Value** **(Ed = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)**	**Reference Standards / Guidelines**
41592 to 41842	36.09	10.82	IS 13365 (Part 2):1998; Hoek & Diederichs (2006)
41842 to 419992	46.44	13.93	

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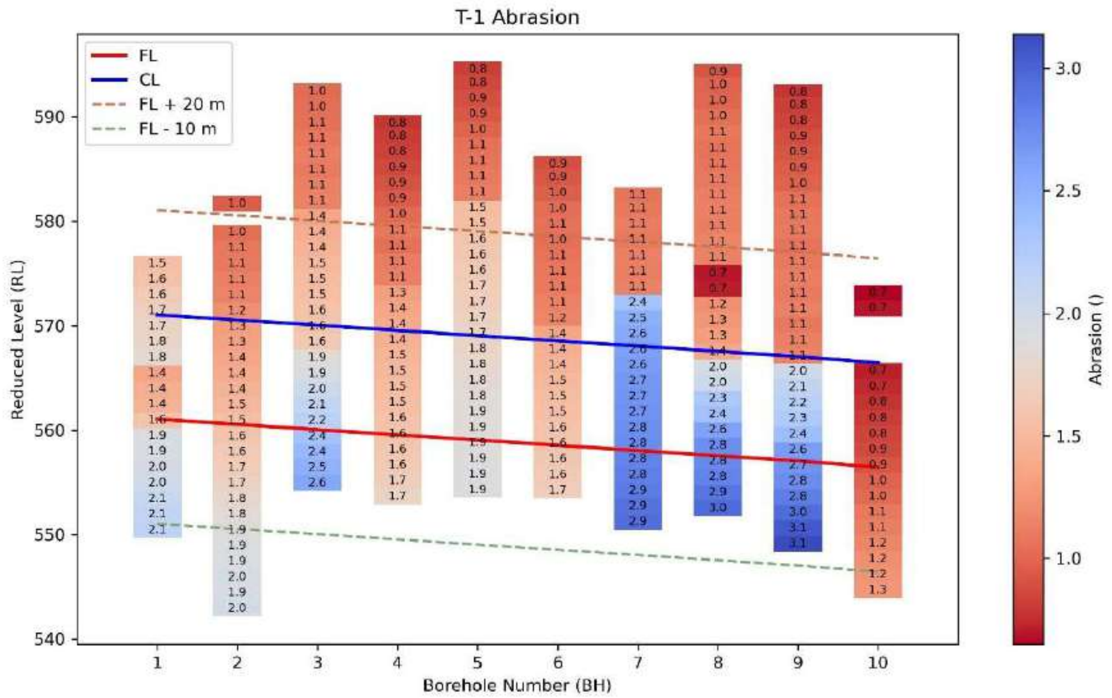
6.3.10 Abrasion test



6.3.10.1 Mean And Standard Deviation in Abrasion value considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	41592	1.783077	0.262055
BH02	41642	1.546154	0.177789
BH03	41692	2.111818	0.331597
BH04	41742	1.570909	0.097719
BH05	41792	1.852	0.049621
BH06	41842	1.553	0.087057

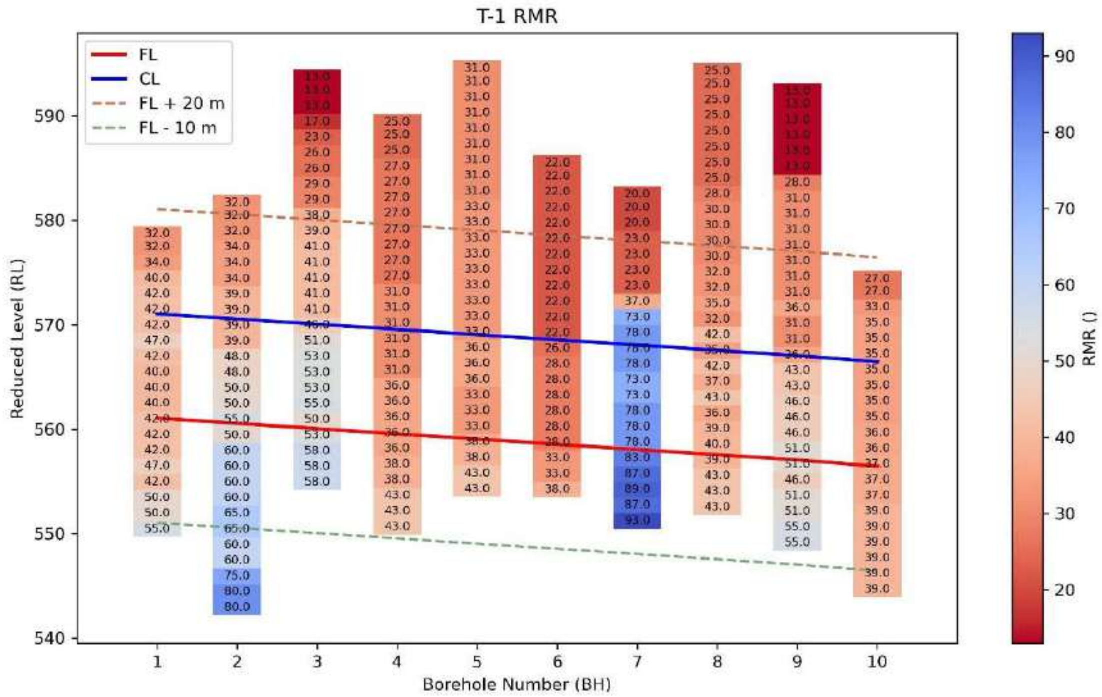
BH No.	Chainage	Mean	Std
BH07	41892	2.769167	0.103085
BH08	41942	2.439091	0.490397
BH09	41992	2.594167	0.379221
BH10	42052	0.941538	0.167871

6.3.10.2 Recommended Abrasion Value considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(A= μ + σ across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(A_v < 25%): Slightly abrasive; 25–35%: Moderately abrasive; >35%: Highly abrasive. (CAI = 0.5–6).**	**Reference Standards / Guidelines**
41592 to 41842	2.01	Moderately abrasive	IS 2386 (Part 4):1963; ISRM (2007); CERCHAR (1986)

6.4 Geological assessment:

6.4.1 RMR:



6.4.1.1 Mean And Standard Deviation in RMR considering 1D zone of influence in each Borehole:

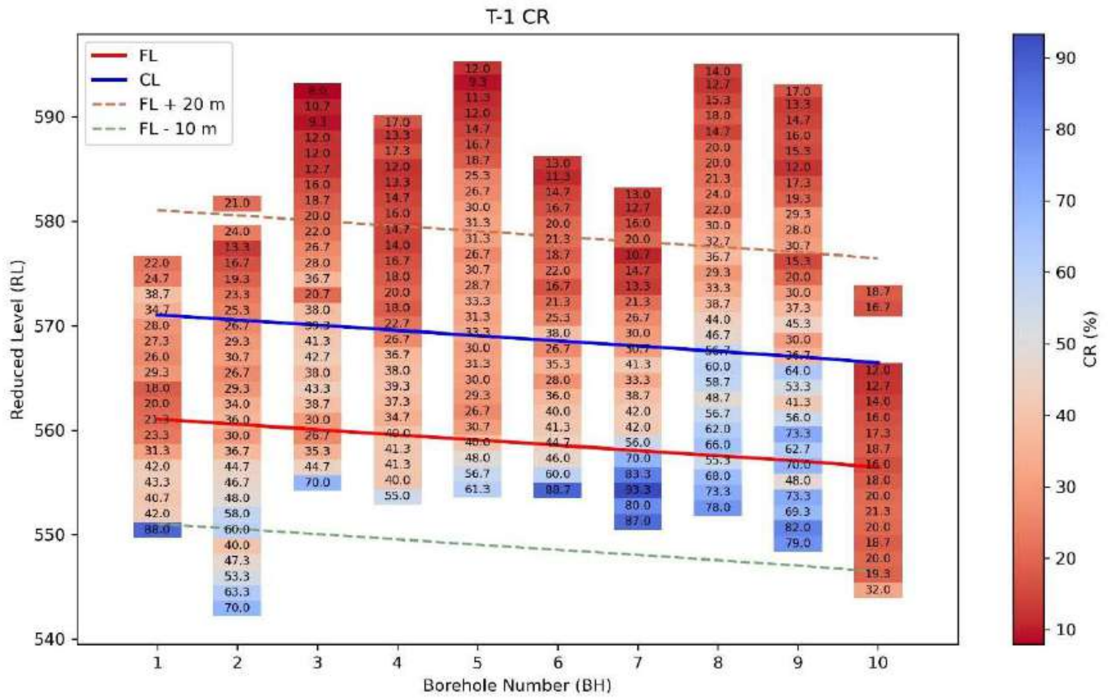
BH No.	Chainage	Mean	Std
BH01	41592	44	4
BH02	41642	53	8
BH03	41692	53	4
BH04	41742	37	4
BH05	41792	37	4

BH No.	Chainage	Mean	Std
BH06	41842	30	4
BH07	41892	81	6
BH08	41942	40	3
BH09	41992	49	4
BH10	42052	37	2

6.4.1.2 Recommended RMR considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Average across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(RMR_d = μ - σ)**	**Reference Standards / Guidelines**
41592 to 41842	43 (Class III)	33 (Class IV)	IS 13365 (Part 2): 1998, Cl. 5.1 + Note on “representative values”
41842 to 419992	57 (Class III)	38 (Class IV)	
419992 to 42052 (P2)	37 (Class IV)	35 (Class IV)	

6.4.2 Core Recovery:



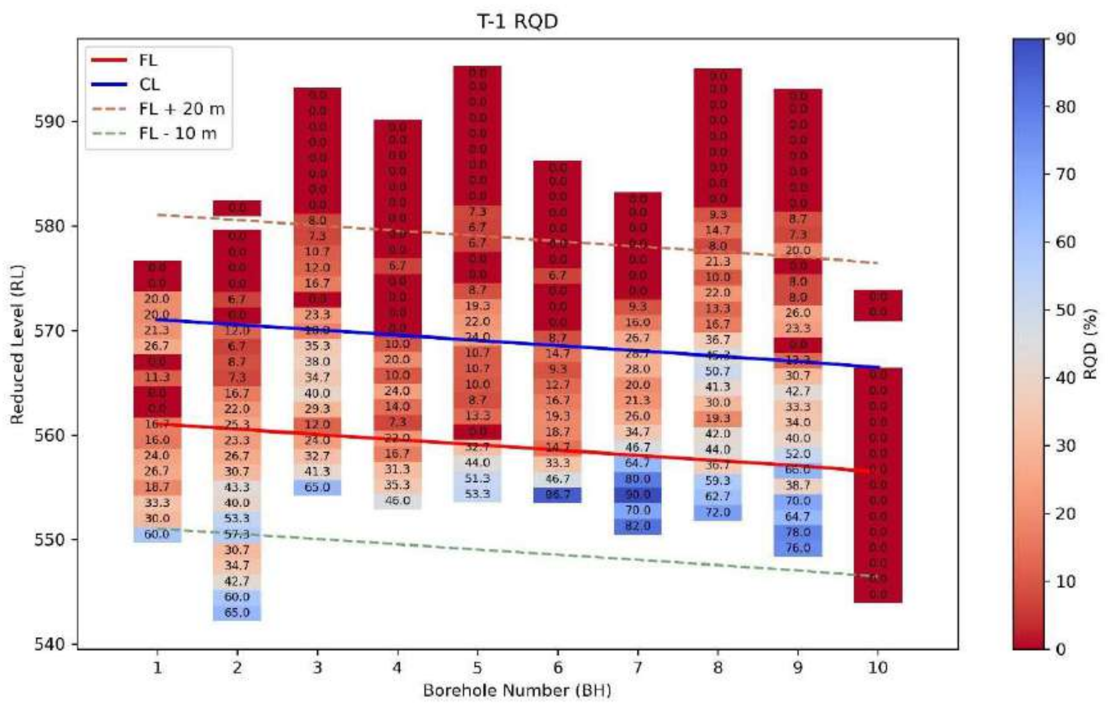
6.4.2.1 Mean And Standard Deviation in Core Recovery considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	41592	30.20308	8.999041
BH02	41642	36.66308	9.760958
BH03	41692	40.90545	11.12942
BH04	41742	39.11818	6.702155
BH05	41792	38.397	12.58322
BH06	41842	44.664	18.16442

BH07	41892	58.13667	23.12536
BH08	41942	62.11818	8.548956
BH09	41992	64.35917	12.56088
BH10	42052	17.28	2.97469

Overall Mean	Std
42.74155	18.78799

6.4.3 RQD:



6.4.3.1 Mean And Standard Deviation in RQD considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	41592	17.27923	11.5183
BH02	41642	24.30385	14.60052
BH03	41692	33.66455	13.89712
BH04	41742	21.51364	12.00775
BH05	41792	23.463	19.87842
BH06	41842	27.261	23.67121
BH07	41892	49.33083	26.31518
BH08	41942	45.75455	14.99693
BH09	41992	52.16417	17.75876
BH10	42052	0	0

6.4.3.2 Recommended RQD considering 1D zone of influence:

Chainage	Statistical / Reduction Method (Average across boreholes within ±10 m of FL and CL)	Recommended Design Value (RQD_d = μ - σ)	Reference Standards / Guidelines
41592 to 41842	24.35	8.10	IS 11315:1985; Deere (1963)
41842 to 419992	49.17	29.19	
419992 to 42052 (P2)	0	0	

6.5 Petrographic Assessments:

6.5.1 Description of rock masses

6.5.1.1 Khondalite

Typical khondalite assemblages in the EGMB are described as quartz–feldspar–garnet–sillimanite ± biotite gneisses, with quartz + feldspar together forming ~40–60 %, garnet ~20–30 %, and sillimanite + biotite ~10–20 % of the rock.

6.5.1.2 Migmatite

Migmatites of the Eastern Ghats consist of quartz- and feldspar-rich leucosome with garnet- and biotite-bearing melanosome/restite. Petrographic and modal descriptions are consistent with dominant quartz + feldspar (~60–70 %) and subordinate garnet and biotite.

6.5.1.3 Khondalite (Restite)

Restites represent refractory residues after partial melting of metapelites, enriched in garnet and ferromagnesian minerals and depleted in felsic phases. Modal characteristics inferred from petrographic descriptions of residual granulites and restitic layers in the EGMB.

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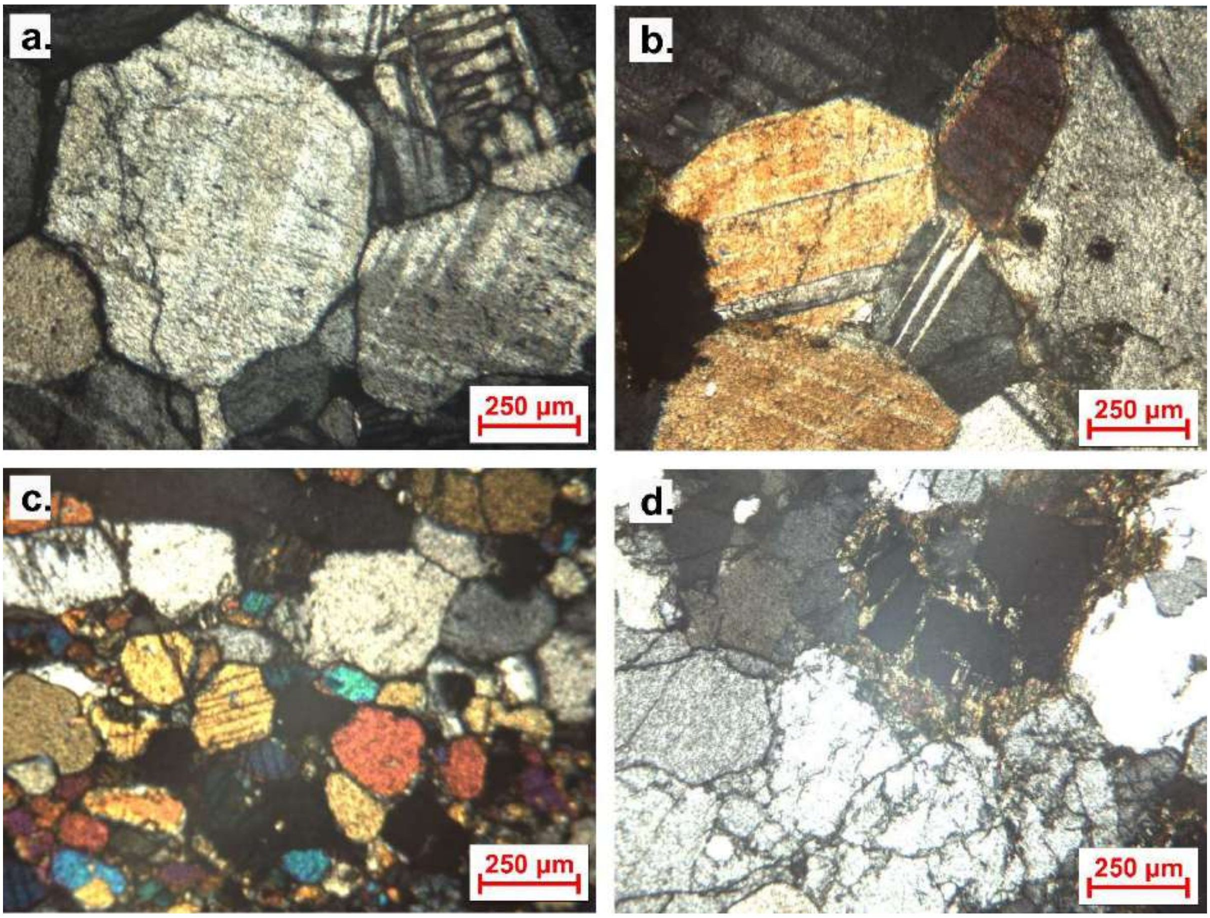
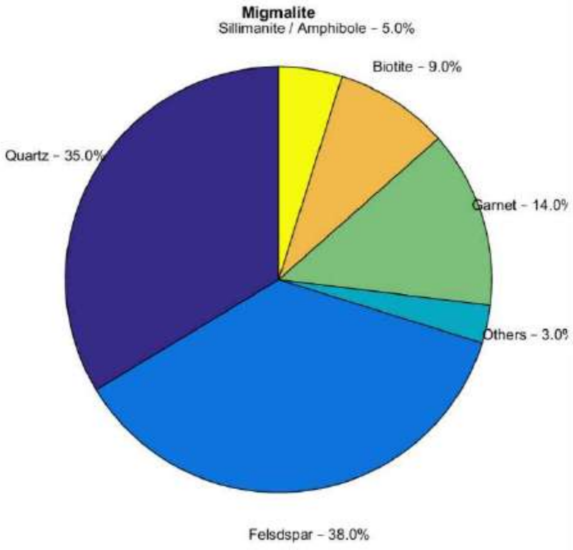
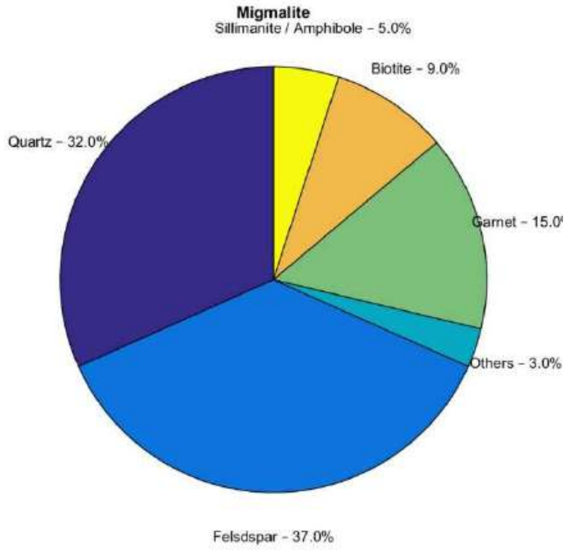
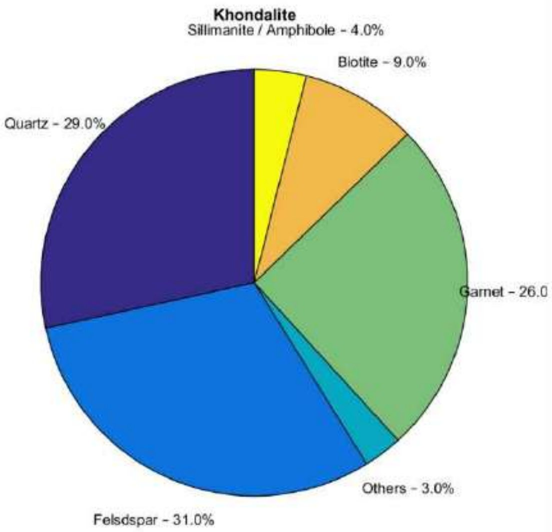
6.5.2 Micro Photographs

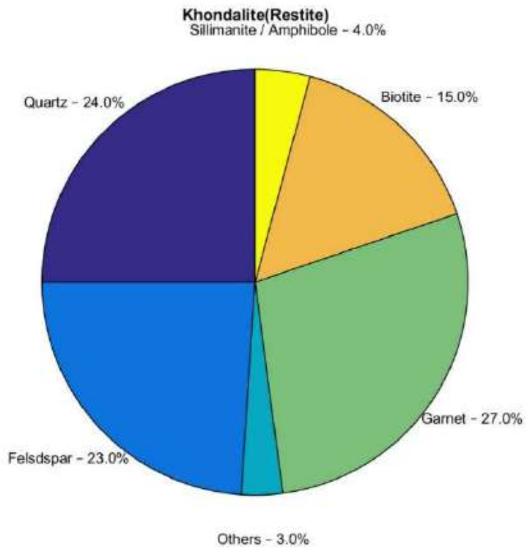


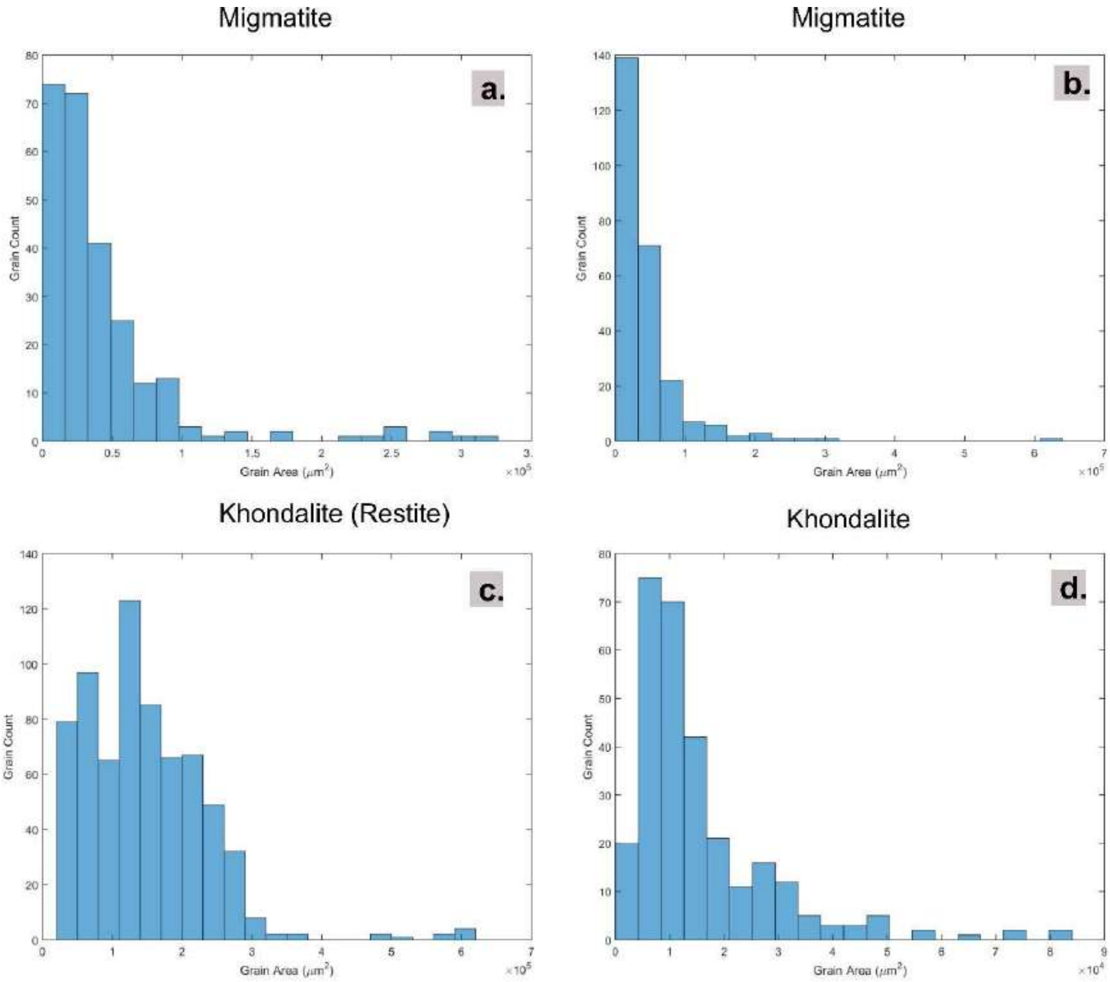
6.5.3 Mineral percentage and grain size distribution

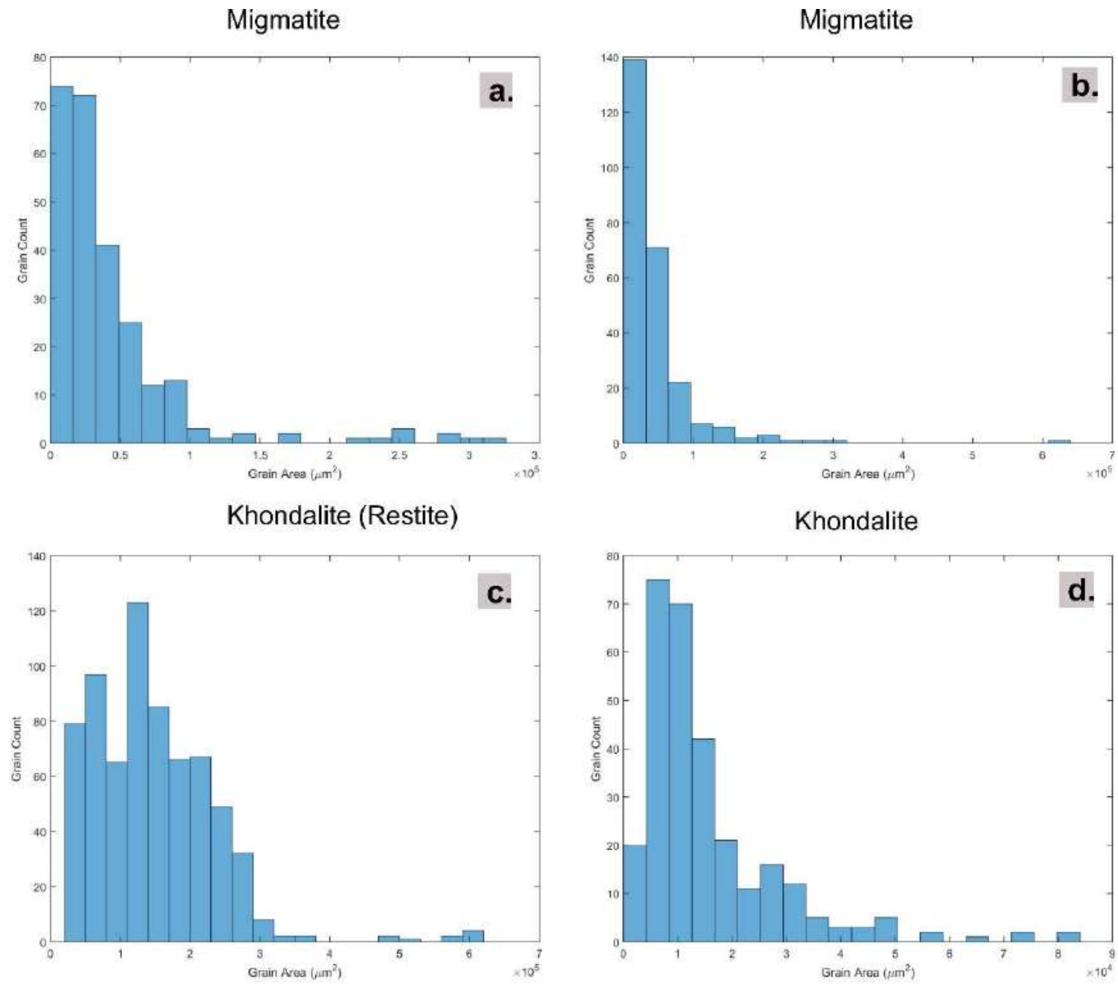




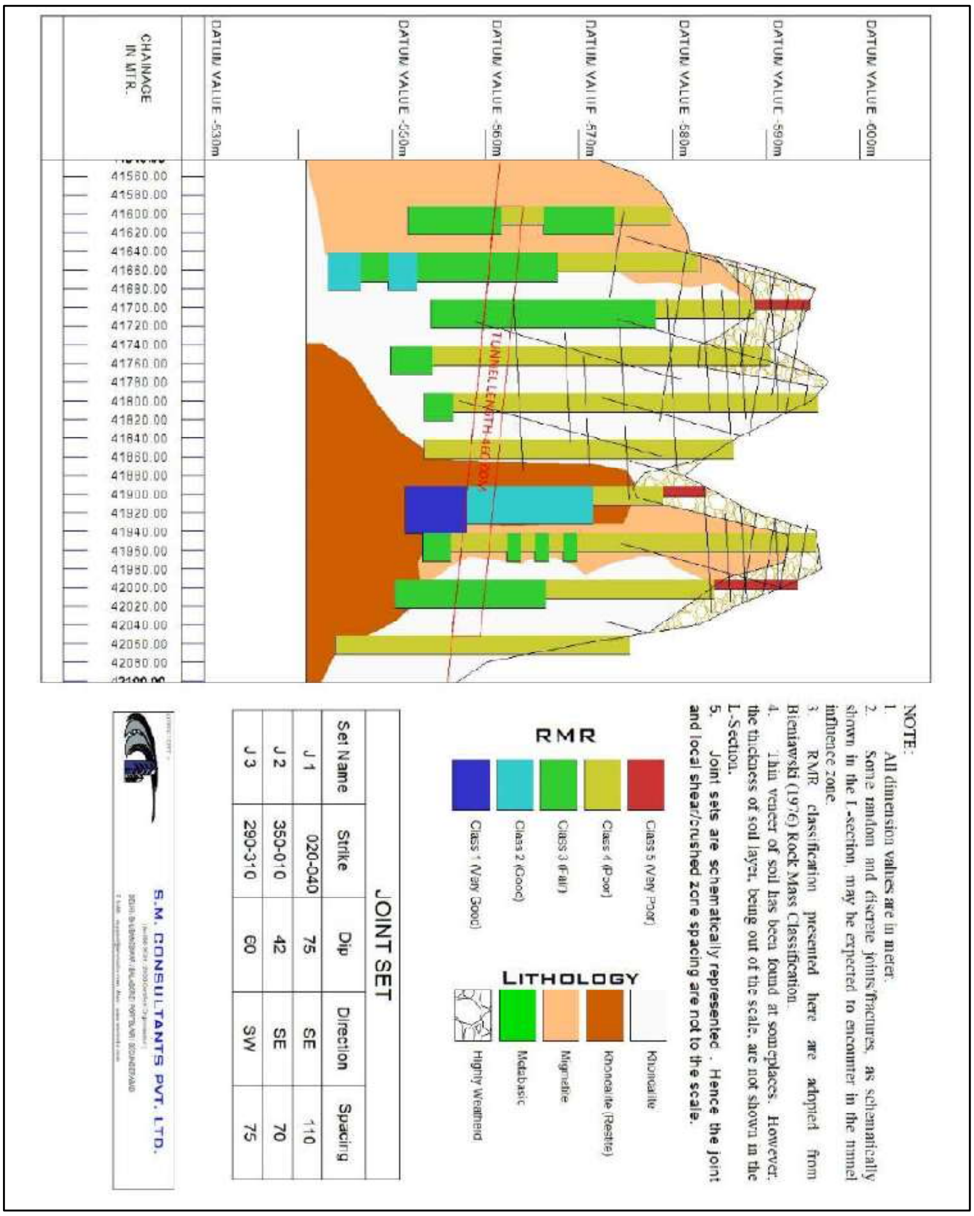





Migmatite
a.
Grain Count
Grain Area (μm²)
×10⁵
Migmatite
b.
Grain Count
Grain Area (μm²)
×10⁵
Khondalite (Restite)
c.
Grain Count
Grain Area (μm²)
×10⁵
Khondalite
d.
Grain Count
Grain Area (μm²)
×10⁴


Migmatite
a.
Grain Count
Grain Area (μm²)
×10⁵
Migmatite
b.
Grain Count
Grain Area (μm²)
×10⁵
Khondalite (Restite)
c.
Grain Count
Grain Area (μm²)
×10⁵
Khondalite
d.
Grain Count
Grain Area (μm²)
×10⁴

6.6 L section:


CHAINAGE IN MTR.
DATUM VALUE -530m
DATUM VALUE -550m
DATUM VALUE -560m
DATUM VALUE -570m
DATUM VALUE -580m
DATUM VALUE -590m
DATUM VALUE -600m
41560.00
41580.00
41600.00
41620.00
41640.00
41660.00
41680.00
41700.00
41720.00
41740.00
41760.00
41780.00
41800.00
41820.00
41840.00
41860.00
41880.00
41900.00
41920.00
41940.00
41960.00
41980.00
42000.00
42020.00
42040.00
42060.00
42080.00
TUNNEL LENGTH-460.00M
NOTE:
1. All dimension values are in meter.
2. Some random and discrete joints/fractures, as schematically shown in the L-section, may be expected to encounter in the tunnel influence zone.
3. RMR classification presented here are adopted from Bieniawski (1976) Rock Mass Classification.
4. Thin veneer of soil has been found at some places. However, the thickness of soil layer, being out of the scale, are not shown in the L-Section.
5. Joint sets are schematically represented. Hence the joint and local shear/crushed zone spacing are not to the scale.
RMR
Class 5 (Very Poor)
Class 4 (Poor)
Class 3 (Fair)
Class 2 (Good)
Class 1 (Very Good)
LITHOLOGY
Khondalite
Khondalite (Restite)
Migmatite
Metabasic
Highly Weathered
JOINT SET
Set Name | Strike | Dip | Direction | Spacing
J1 | 020-040 | 75 | SE | 110
J2 | 350-010 | 42 | SE | 70
J3 | 290-310 | 60 | SW | 75
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7 TUNNEL 2: GEOLOGICAL & GEOTECHNICAL ASSESMENT

7.1 Exploratory drillings

As per the requirement of scope of work outlined in the terms of reference, 16 bore holes were drilled with a cumulative length of 548m at different locations along the proposed alignment. Necessary care has been taken during drilling operations by deploying good quality diamond drill machines to obtain good core recovery to obtain RQD values. The locations of the boreholes were selected in such a way, so that these holes intersect the envisaged ground/ strata conditions at different depths. The location and details of boreholes drilled; total depth of drillings is shown in table below.

Chainage	BH Name	GL	FL	Depth
42870	BH-1	568.055	548.986	24.00
42920	BH-2	577.689	548.532	34.00
42970	BH-3	587.322	548.077	46.00
43020	BH-4	587.303	547.622	45.00
43070	BH-5	584.872	547.168	43.00
43120	BH-6	581.778	546.713	40.00
43170	BH-7	571.824	546.278	31.00
43220	BH-8	575.498	545.843	37.00
43270	BH-9	580.402	545.408	41.00
43320	BH-10	580.55	544.973	41.00
43370	BH-11	573.958	544.539	36.00
43420	BH-12	562.003	544.104	23.00
43470	BH-13	569.205	543.669	31.00
43520	BH-14	563.016	543.234	25.00
43570	BH-15	564.291	542.8	27.00
43620	BH-16	561.317	542.365	24.00

7.2 TUNNEL-2 SRT

7.2.1 Location:

Sr. No.	Chainage	Line	Spread	Location (T-02)		Length
				Start	End	In meter
1	42.870km to 43.620km	L1	S1 to S7	42.870km	43.620km	750

7.2.2 Seismic survey results and conclusion

Table 7.1: Summery of Tunnel-1 SRT test

Variation of maximum range of thicknesses below EGL (M)			Avg. V_p (m/sec)	Calculated V_s (m/sec)	Dynamic Young's Modulus (MPa)	Shear Modulus (MPa)
Layer	From	To				
Layer-I	0.50	3.50	700	327	466	172
Layer-II	3.50	7.50	2300	1183	8503	3221
Layer-III	7.50	25.00	3800	2101	29368	11472

Sample Calculation:

The Young's Modulus E is the uni-axial stress-strain ratio. Its dynamic value is expressed by the following equation:

$$E = \frac{\rho V_P^2 (1+\mu)(1-2\mu)}{1-\mu}$$

Where, E = Dynamic Young's Modulus in kN/m2

Vp = 700 m/sec

ρ = 1.6 gm/cc ≈ 1.60 kN.s2/m4 (mass density)

μ = 0.36

So, calculated E = 466480 kN/sqm ≈ 466 MPa

The Shear Modulus G is the stress-strain ratio for simple shear. Its dynamic value is obtained by the following:

$$G = \frac{E}{2(1+\mu)} = \rho V_S^2$$

So, Shear Modulus G comes out to be 171500 kN/sqm ≈ 172 MPa

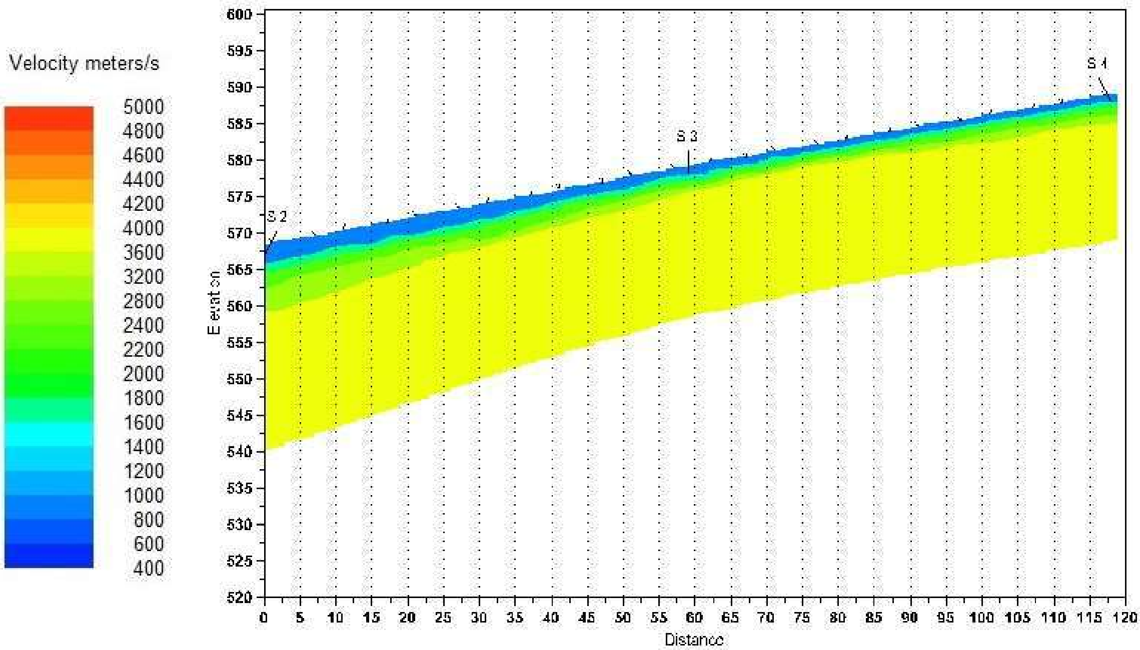
Again,

$G = \rho V_S^2$ giving Vs = √(G/ρ)

So, calculated Vs = 327.395 m/sec, say 327 m/sec

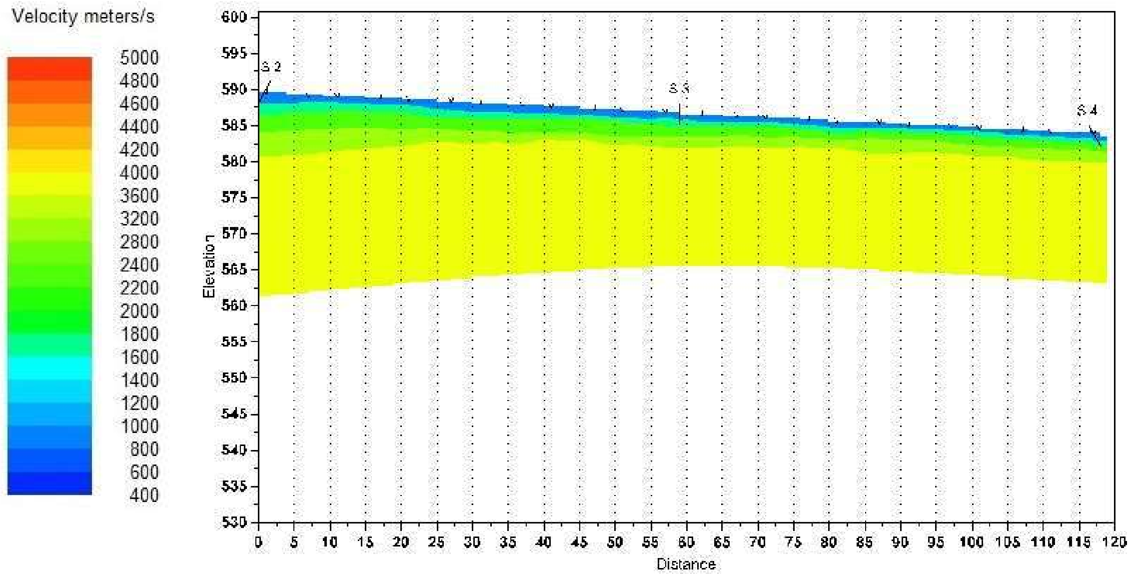
7.2.3 SEISMIC PROFILE(T02)

T2L1S1



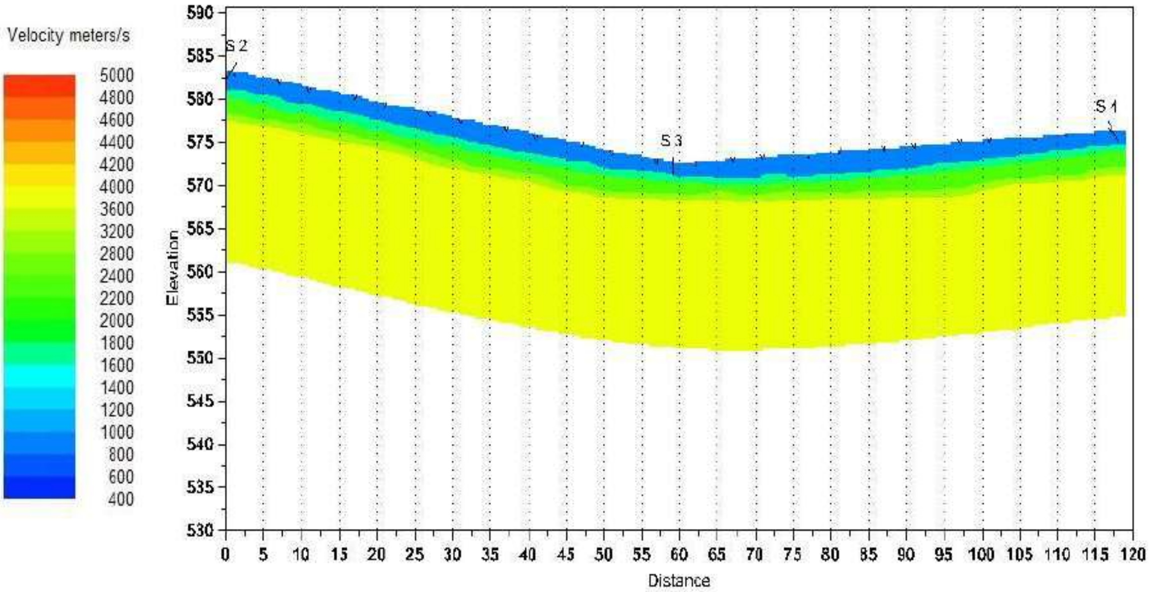
Shot point depth computation

T2L1S2



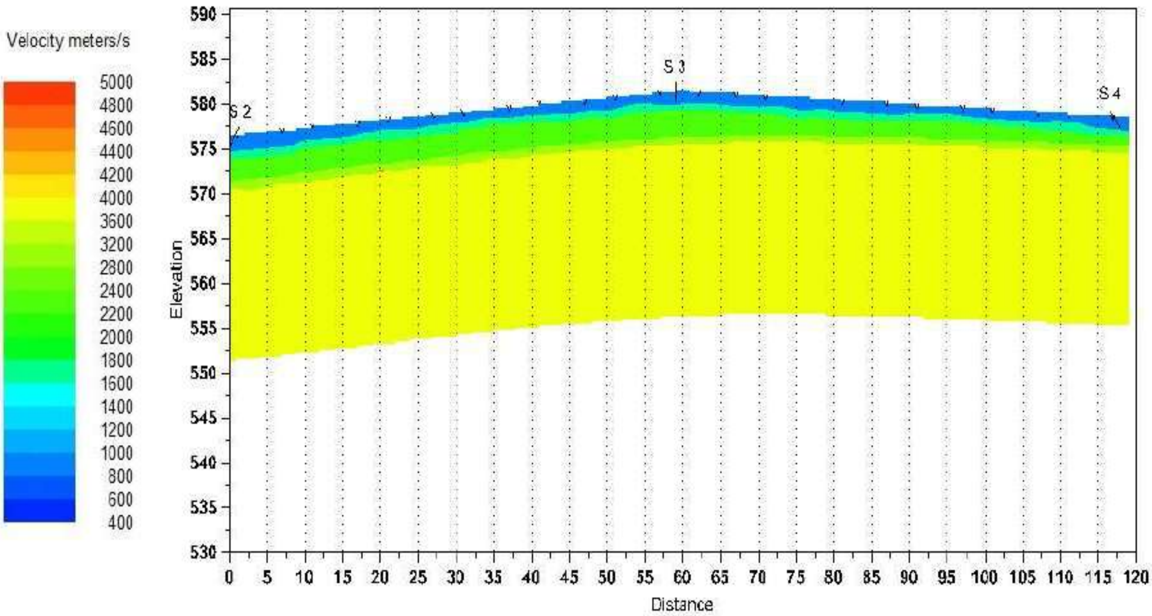
Shot point depth computation

T2L1S3



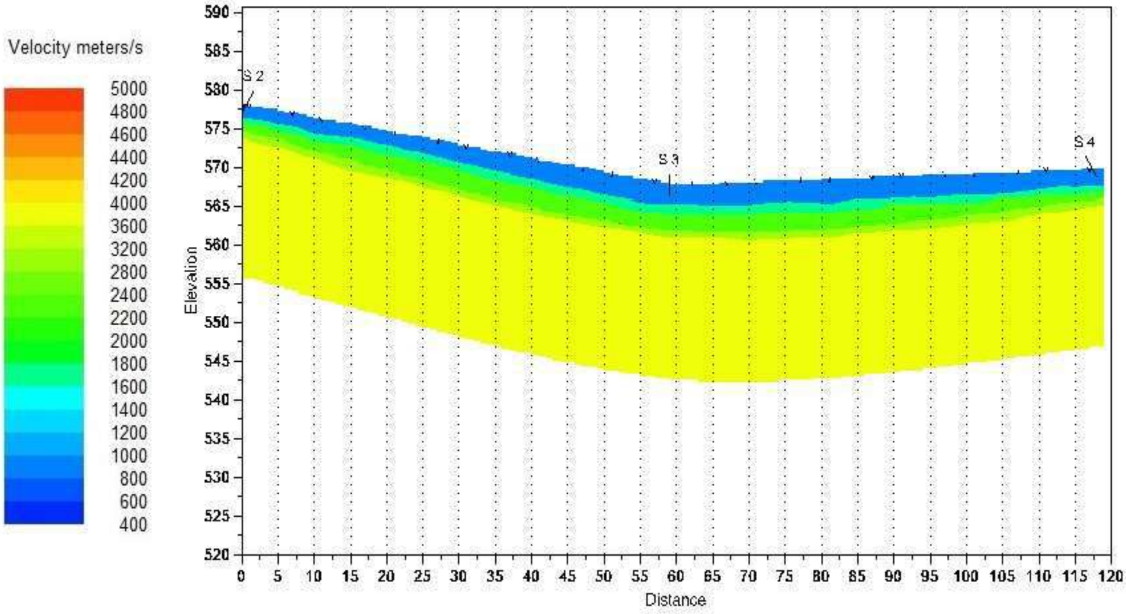
Shot point depth computation

T2L1S4



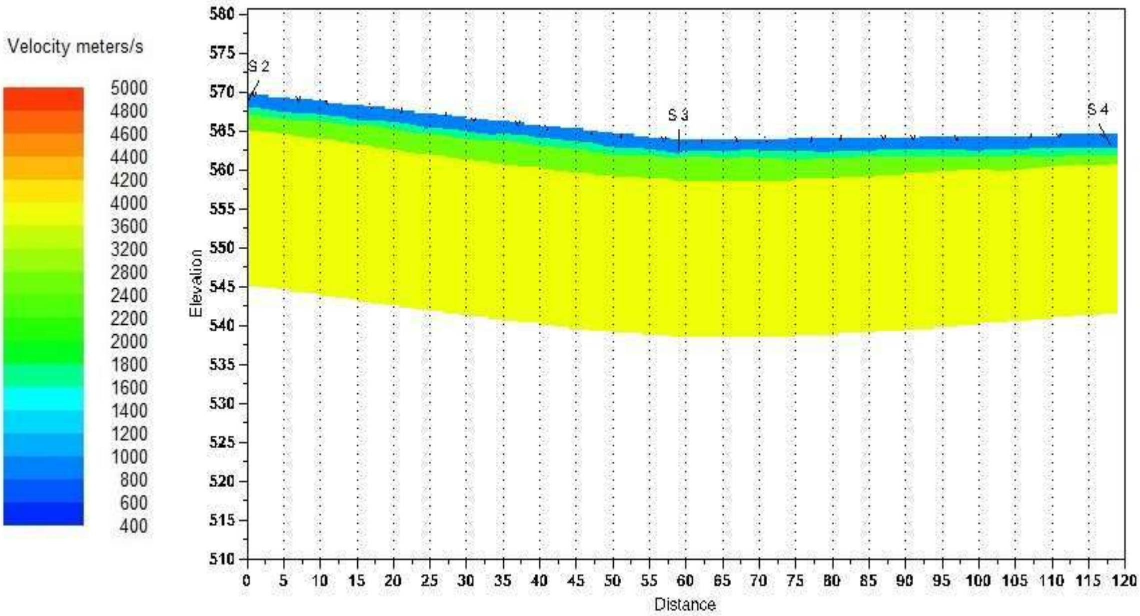
Shot point depth computation

T2L1S5



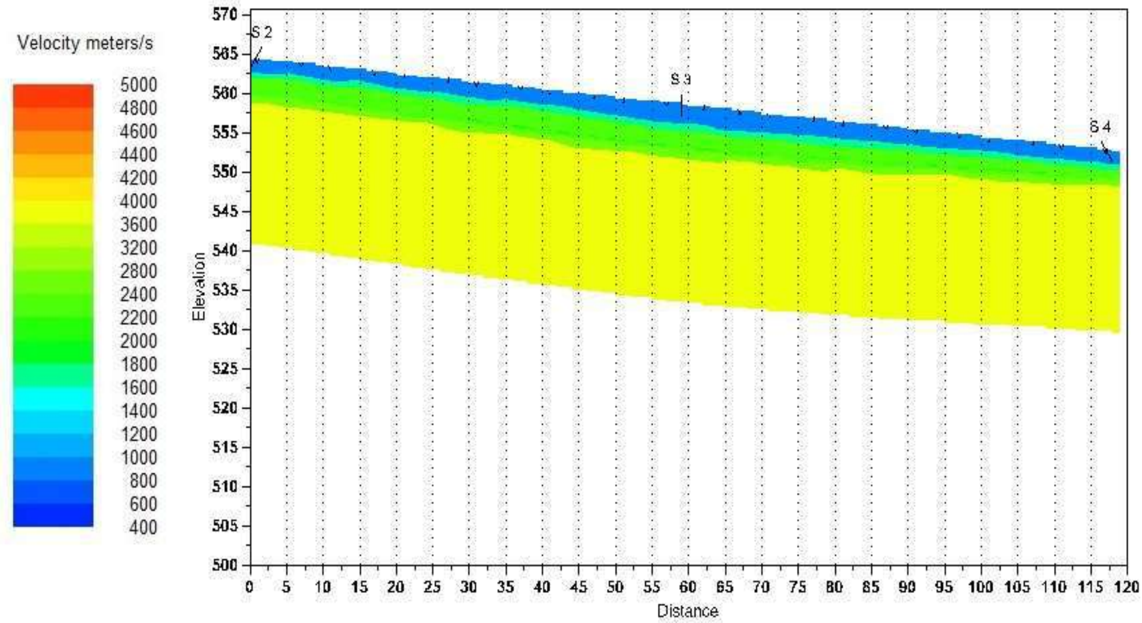
Shot point depth computation

T2L1S6



Shot point depth computation

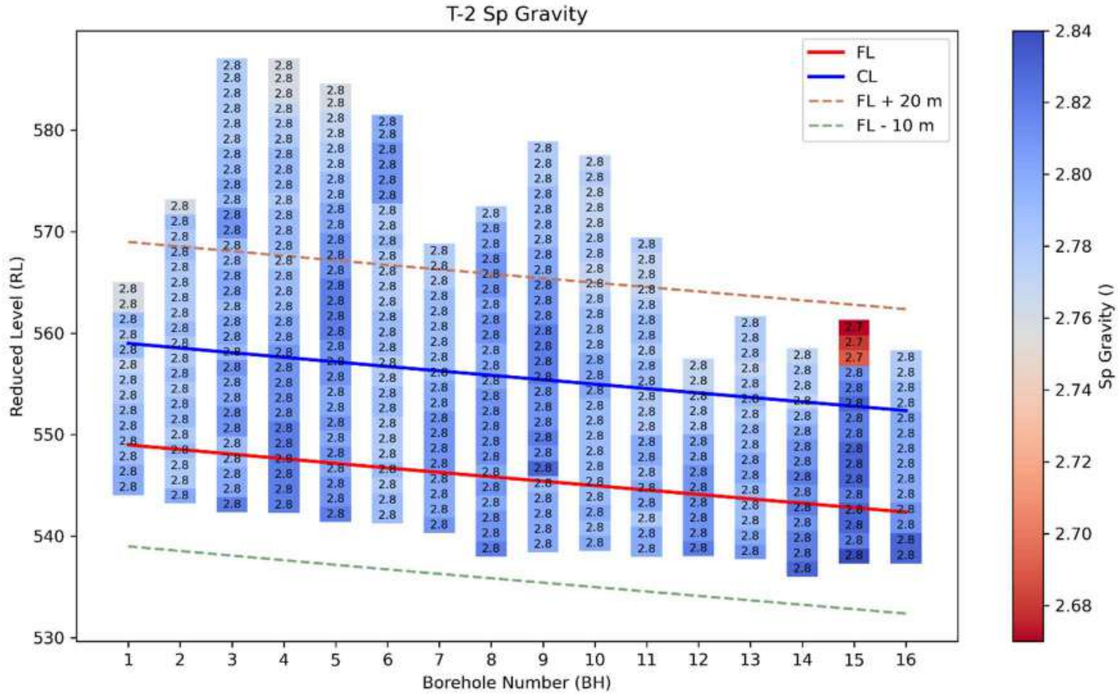
T2L1S7



Shot point depth computation

7.3 Assessment of the engineering properties of rock sample:

7.3.1 Specific Gravity



7.3.1.1 Mean and Standard deviation Specific Gravity considering 1D zone of influence of each borehole:

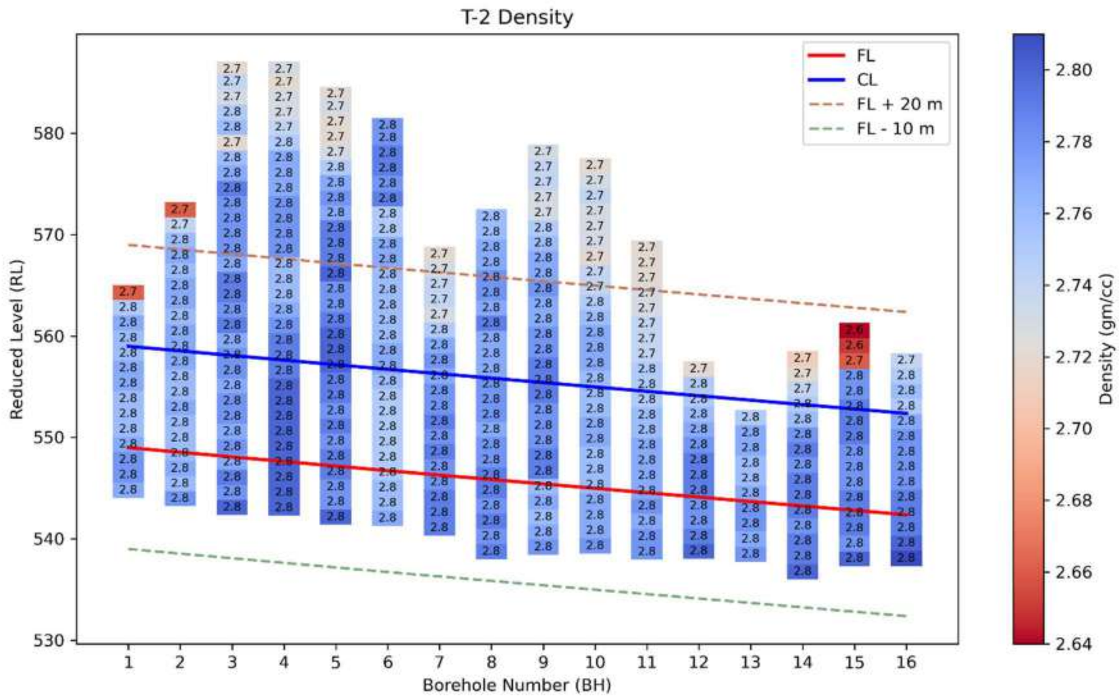
BH No.	Chainage	Mean	Std
BH01	42870	2.788	0.009189
BH02	42920	2.786	0.006992
BH03	42970	2.801818	0.009816
BH04	43020	2.816	0.005164
BH05	43070	2.807273	0.006467
BH06	43120	2.785	0.00527

BH No.	Chainage	Mean	Std
BH07	43170	2.803636	0.006742
BH08	43220	2.808333	0.005774
BH09	43270	2.807273	0.011037
BH10	43320	2.792727	0.004671
BH11	43370	2.791818	0.009816
BH12	43420	2.800909	0.01446
BH13	43470	2.793636	0.00809
BH14	43520	2.812727	0.011037
BH15	43570	2.827	0.006749
BH16	43620	2.808	0.012293

7.3.1.2 Recommended Specific Gravity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Sp = μ - σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
42870 to 43620	2.78	IS 13030:1991; IS 1124:1974

7.3.2 Dry Density



7.3.2.1 Mean and Standard Deviation in Density considering 1D zone of influence of each borehole:

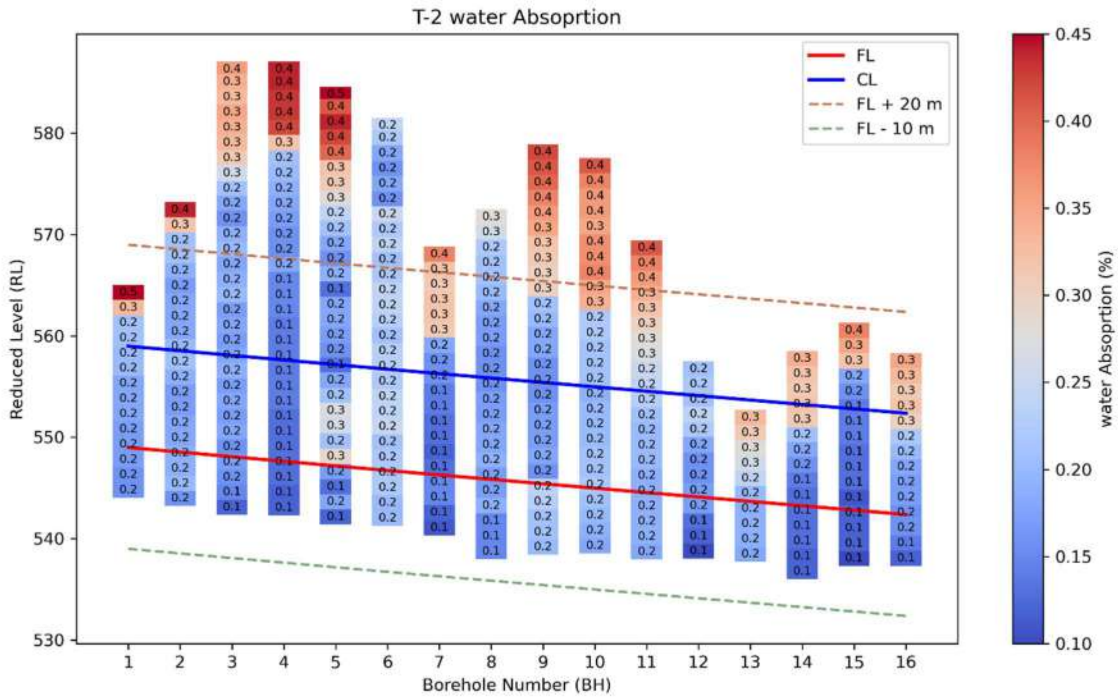
BH No.	Chainage	Mean	Std
BH01	42870	2.768	0.007888
BH02	42920	2.765	0.007071
BH03	42970	2.78	0.008944
BH04	43020	2.799	0.003162
BH05	43070	2.788182	0.007508
BH06	43120	2.755	0.007071

BH No.	Chainage	Mean	Std
BH07	43170	2.783636	0.006742
BH08	43220	2.7825	0.006216
BH09	43270	2.777273	0.011037
BH10	43320	2.766364	0.006742
BH11	43370	2.769091	0.009439
BH12	43420	2.78	0.014832
BH13	43470	2.769	0.009944
BH14	43520	2.785455	0.010357
BH15	43570	2.787	0.006749
BH16	43620	2.787	0.011595

7.3.2.2 Recommended Density considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(d = μ - σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
42870 to 43620	2.76	IS 13063:1991

7.3.3 Water absorption Test



7.3.3.1 Mean and Standard Deviation in Water Absorption Value considering 1D zone of influence:

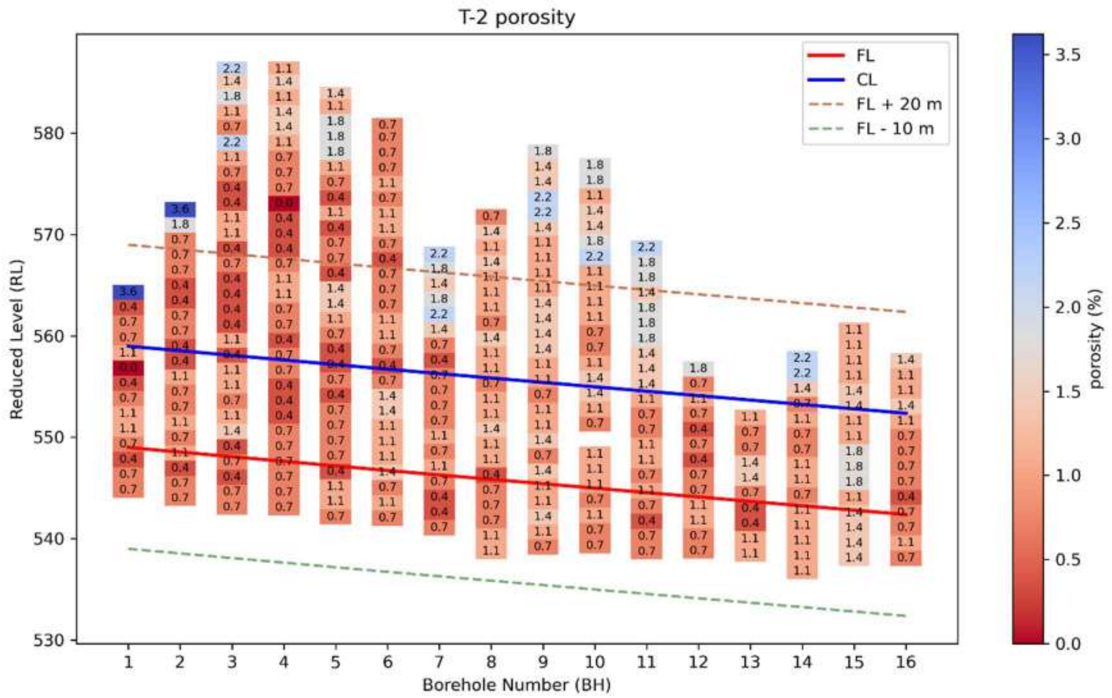
BH No.	Chainage	Mean	Std
BH01	42870	0.747599	0.201009
BH02	42920	0.177	0.014181
BH03	42970	0.184	0.015055
BH04	43020	0.166364	0.019117
BH05	43070	0.131	0.007379
BH06	43120	0.195455	0.058713
BH07	43170	0.222	0.016865

BH No.	Chainage	Mean	Std
BH08	43220	0.134545	0.015076
BH09	43270	0.169167	0.014434
BH10	43320	0.190909	0.028445
BH11	43370	0.200909	0.011362
BH12	43420	0.197273	0.013484
BH13	43470	0.169091	0.042768
BH14	43520	0.237	0.055388
BH15	43570	0.16	0.057096
BH16	43620	0.124	0.012649

7.3.3.2 Recommended Water Absorption Value considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(A= μ + σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
42870 to 43620	0.22	IS 13063:1991; IS 2386 (Part 3):1963

7.3.4 Porosity



7.3.4.1 Mean And Standard Deviation in Porosity considering 1D zone of influence in each Borehole:

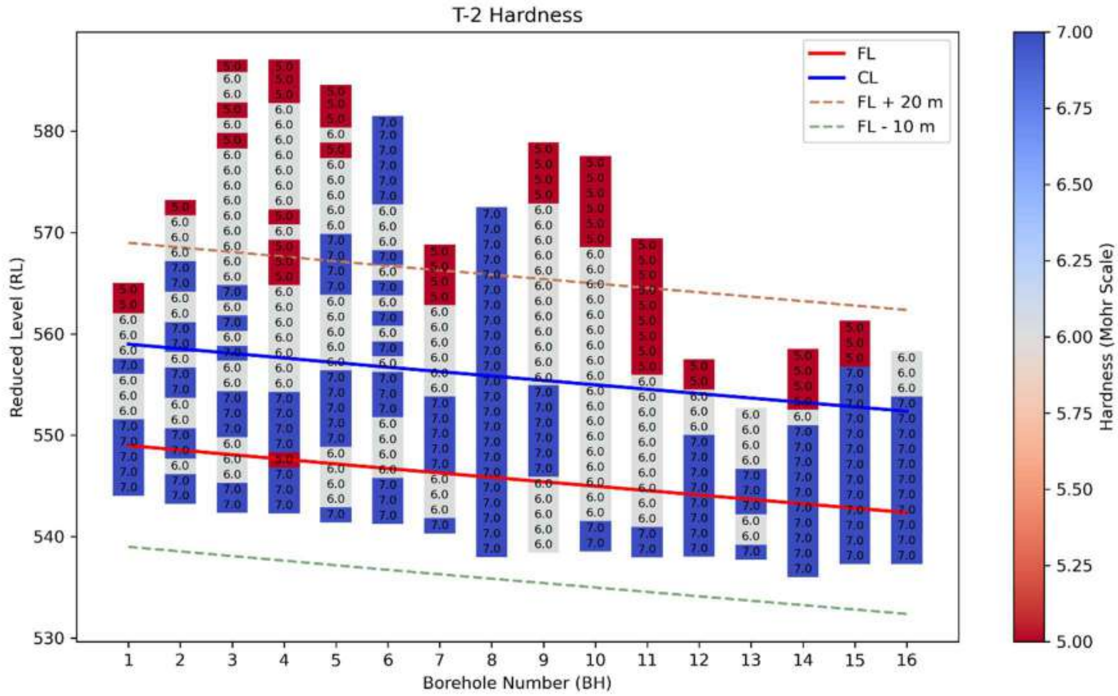
BH No.	Chainage	Mean	Std
BH01	42870	0.67945	0.356151
BH02	42920	0.753464	0.263967
BH03	42970	0.77807	0.348756
BH04	43020	0.603468	0.170859
BH05	43070	0.679856	0.249049
BH06	43120	1.076945	0.291954
BH07	43170	0.713132	0.224677

BH No.	Chainage	Mean	Std
BH08	43220	0.91955	0.281445
BH09	43270	1.068324	0.274074
BH10	43320	0.96659	0.240932
BH11	43370	0.81379	0.23079
BH12	43420	0.746308	0.249973
BH13	43470	0.894271	0.385179
BH14	43520	0.969424	0.164999
BH15	43570	1.414559	0.286991
BH16	43620	0.747599	0.201009

7.3.4.2 Recommended Porosity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(n = μ ± σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
42870 to 43620	0.86±0.32	IS 1124:1974; ISRM (1981)

7.3.5 Hardness



7.3.5.1 Mean And Standard Deviation in Hardness considering 1D zone of influence in each Borehole:

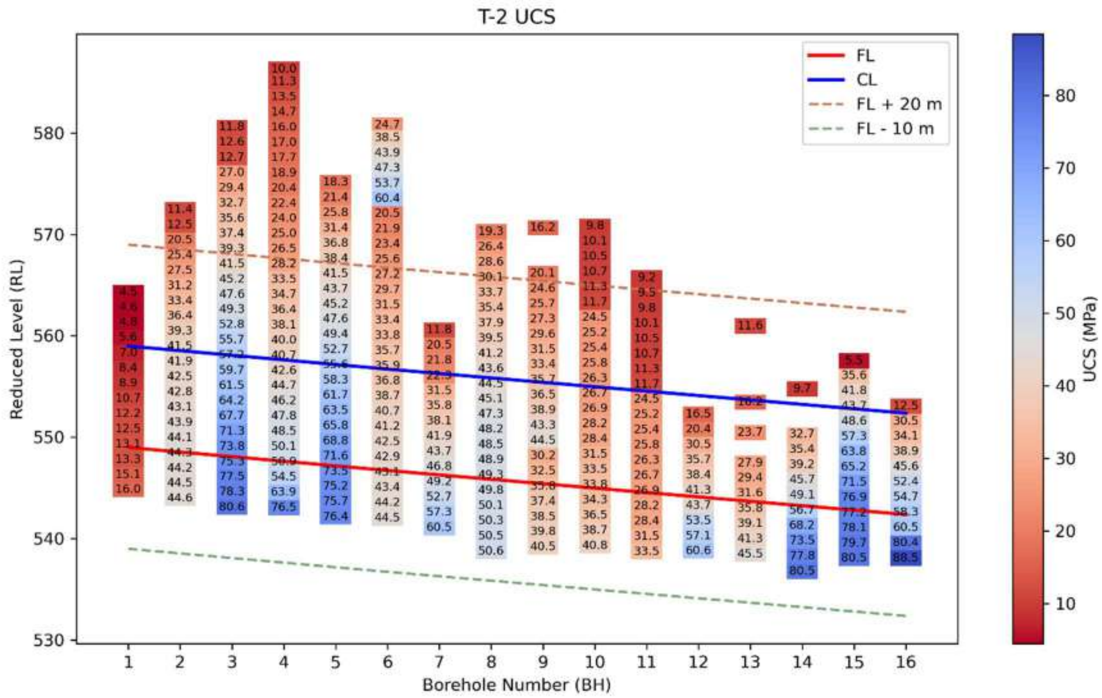
BH No.	Chainage	Mean	Std
BH01	42870	7	0.516398
BH02	42920	7	0.516398
BH03	42970	7	0.522233
BH04	43020	7	0.699206
BH05	43070	7	0.522233
BH06	43120	7	0.516398
BH07	43170	7	0.522233

BH No.	Chainage	Mean	Std
BH08	43220	7	0
BH09	43270	7	0.522233
BH10	43320	6	0.40452
BH11	43370	6	0.40452
BH12	43420	7	0.467099
BH13	43470	6	0.516398
BH14	43520	7	0.301511
BH15	43570	7	0
BH16	43620	7	0

7.3.5.2 Recommended Hardness considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Hi = μ - σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
42870 to 43620	6	IS 13311 (Part 2):1992; ISRM 1978

7.3.6 Compression Test



7.3.6.1 Mean And Standard Deviation in UCS considering 1D zone of influence in each Borehole:

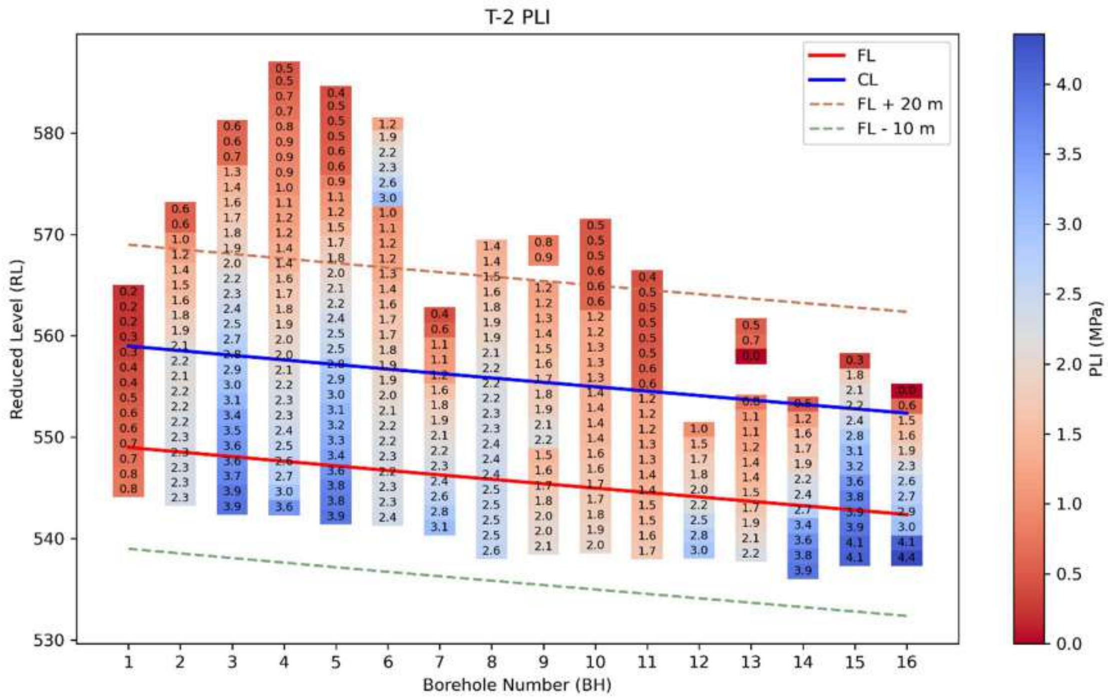
BH No.	Chainage	Mean	Std
BH01	42870	11.70549	2.9396
BH02	42920	43.58071	0.92861
BH03	42970	69.73627	8.136907
BH04	43020	52.56191	10.28889
BH05	43070	67.83281	7.340622
BH06	43120	41.80775	2.469843
BH07	43170	43.61002	11.35786

BH No.	Chainage	Mean	Std
BH08	43220	48.60021	2.033522
BH09	43270	37.97871	4.23343
BH10	43320	32.66828	4.780013
BH11	43370	27.51101	2.783045
BH12	43420	39.77777	14.77345
BH13	43470	32.26822	9.180901
BH14	43520	55.88474	18.06488
BH15	43570	69.87939	10.81292
BH16	43620	54.39665	18.84347

7.3.6.2 Recommended UCS considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(UCS_k = μ - σ) across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(UCS_d = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)**	**Reference Standards / Guidelines**
42870 to 43620	27	13.75	IS 9143:1979; IS 13365 (Part 2):1998; Hoek & Brown (2002, 2019)

7.3.7 Point Load Test



7.3.7.1 Mean And Standard Deviation in PLI considering 1D zone of influence in each Borehole:

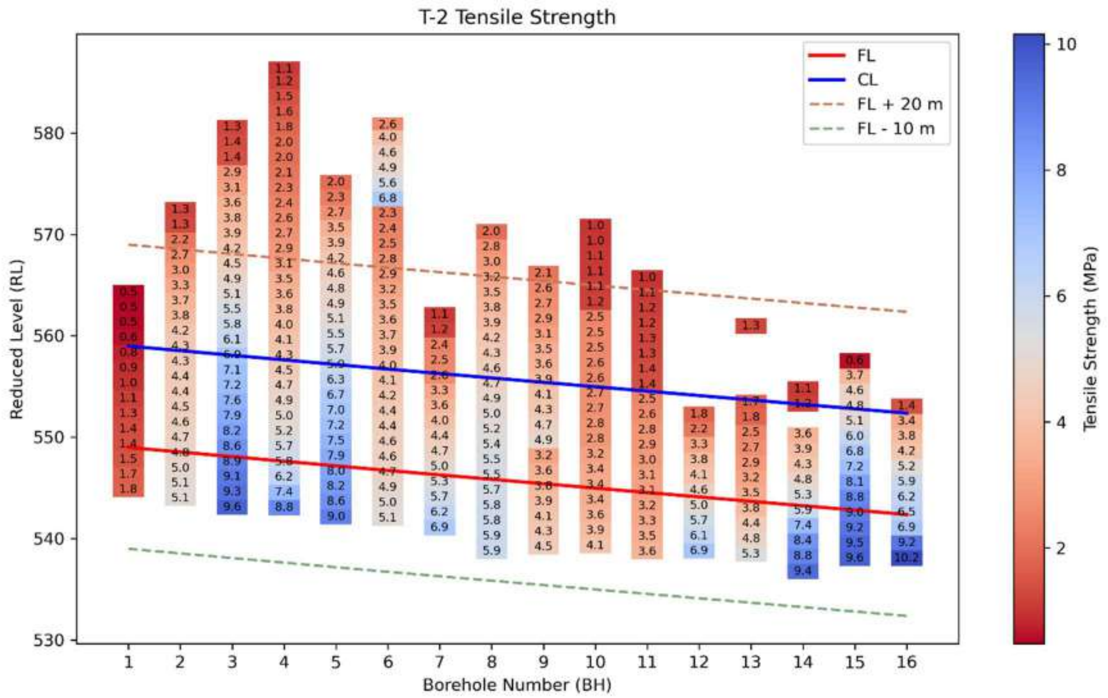
BH No.	Chainage	Mean	Std
BH01	42870	0.578	0.146954
BH02	42920	2.25	0.071492
BH03	42970	3.399091	0.410913
BH04	43020	2.56	0.435048
BH05	43070	3.346364	0.393326
BH06	43120	2.196	0.1393
BH07	43170	2.184545	0.550061

BH No.	Chainage	Mean	Std
BH08	43220	2.400833	0.114531
BH09	43270	1.870909	0.223269
BH10	43320	1.623636	0.214768
BH11	43370	1.397273	0.173614
BH12	43420	2.057778	0.640718
BH13	43470	1.500909	0.440124
BH14	43520	2.583636	0.968393
BH15	43570	3.483	0.577698
BH16	43620	2.708	0.962183

7.3.7.2 Recommended PLI considering 1D zone of influence:

Chainage	Statistical / Reduction Method (PLI_k = μ - σ) across boreholes within ±10 m of FL and CL)	Recommended Design Value (PLI_d = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)	Reference Standards / Guidelines
42870 to 43620	1.37	0.68	IS 9143:1979; IS 13365 (Part 2):1998; Hoek & Brown (2002, 2019)

7.3.8 Brazilian Test



7.3.8.1 Mean And Standard Deviation in Tensile Strength considering 1D zone of influence in each Borehole:

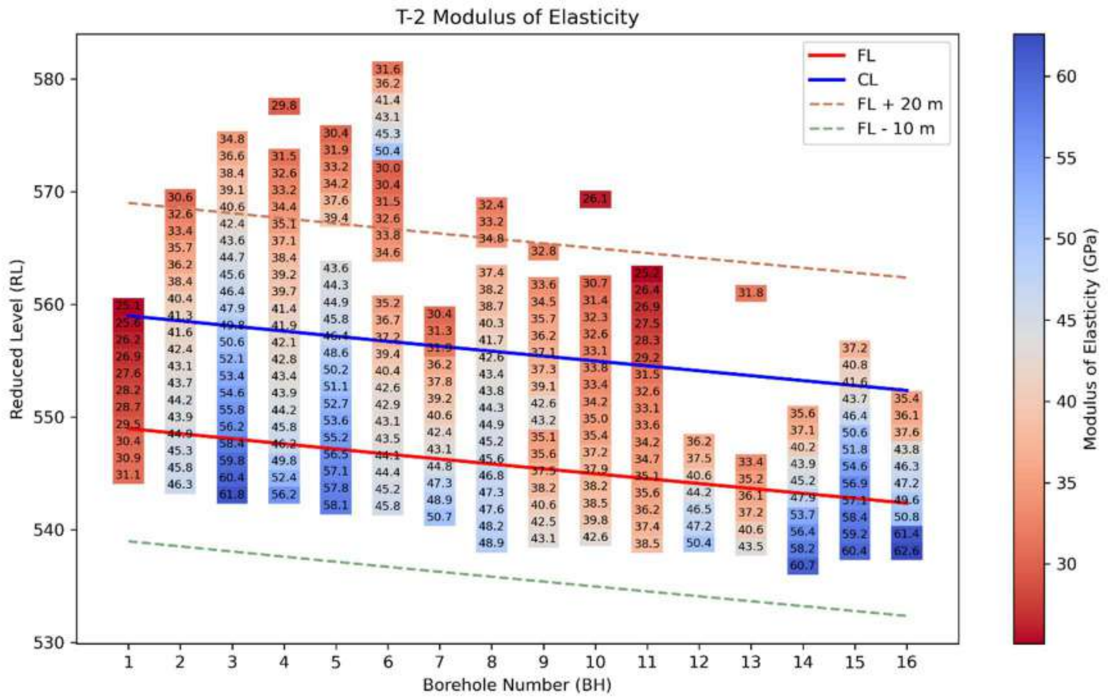
BH No.	Chainage	Mean	Std
BH01	42870	1.282	0.332024
BH02	42920	4.696	0.296843
BH03	42970	8.218182	0.944498
BH04	43020	5.807	1.338316
BH05	43070	7.483636	0.96386
BH06	43120	4.604	0.336822
BH07	43170	4.703636	1.288032

BH No.	Chainage	Mean	Std
BH08	43220	5.45	0.407029
BH09	43270	4.125455	0.497943
BH10	43320	3.266828	0.478001
BH11	43370	3.059091	0.348467
BH12	43420	4.363	1.656281
BH13	43470	3.310909	1.16034
BH14	43520	6.189	2.160979
BH15	43570	7.94	1.563834
BH16	43620	6.15	2.225069

Recommended Tensile Strength considering 1D zone of influence:

Chainage	Statistical / Reduction Method (mean value across boreholes within ±10 m of FL and CL× 0.8 (account for anisotropy))	Recommended Design Value (μ - σ)	Reference Standards / Guidelines
42870 to 43620	5.03	2.89	IS 10082:1982; Hoek & Brown (1997); ISRM Suggested Methods

7.3.9 Modulus of elasticity test



7.3.9.1 Mean And Standard Deviation in Elasticity considering 1D zone of influence in each Borehole:

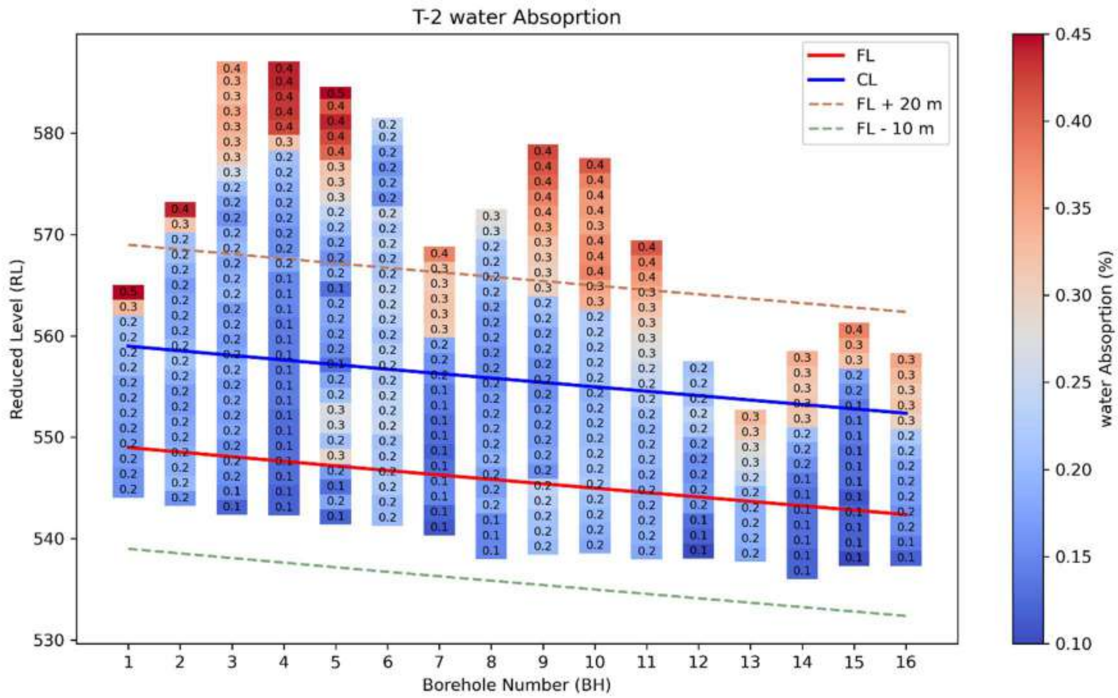
BH No.	Chainage	Mean	Std
BH01	42870	28.51	1.953032
BH02	42920	44.12	1.496514
BH03	42970	55.71818	4.048906
BH04	43020	46.68	4.651834
BH05	43070	53.39091	3.950811
BH06	43120	43.14	1.994548
BH07	43170	42.08182	5.688729

BH No.	Chainage	Mean	Std
BH08	43220	45.71667	2.027911
BH09	43270	39.52727	3.028561
BH10	43320	36.90909	2.850423
BH11	43370	34.77273	2.087147
BH12	43420	43.22857	5.291098
BH13	43470	37.66667	3.730773
BH14	43520	47.89	8.983497
BH15	43570	53.91	5.636281
BH16	43620	47.08	9.570533

7.3.9.2 Recommended Modulus of Elasticity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Ei = μ - σ across boreholes within ±10 m of FL and CL× 0.8 (account for anisotropy))**	**Recommended Design Value** **(Ed = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)**	**Reference Standards / Guidelines**
42870 to 43620	35.48	17.74	IS 13365 (Part 2):1998; Hoek & Diederichs (2006)

7.3.10 Abrasion test



7.3.10.1 Mean And Standard Deviation in Abrasion value considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	42870	1.075	0.146306
BH02	42920	1.89	0.095685
BH03	42970	2.727273	0.192255
BH04	43020	2.183	0.158958
BH05	43070	2.689091	0.174554
BH06	43120	1.98	0.10403
BH07	43170	1.882727	0.300835

BH No.	Chainage	Mean	Std
BH08	43220	2.261667	0.124742
BH09	43270	1.820909	0.081665
BH10	43320	1.75	0.13092
BH11	43370	1.562727	0.094772
BH12	43420	1.810909	0.460466
BH13	43470	1.626364	0.261812
BH14	43520	2.258182	0.527424
BH15	43570	2.748	0.274704
BH16	43620	2.184	0.536039

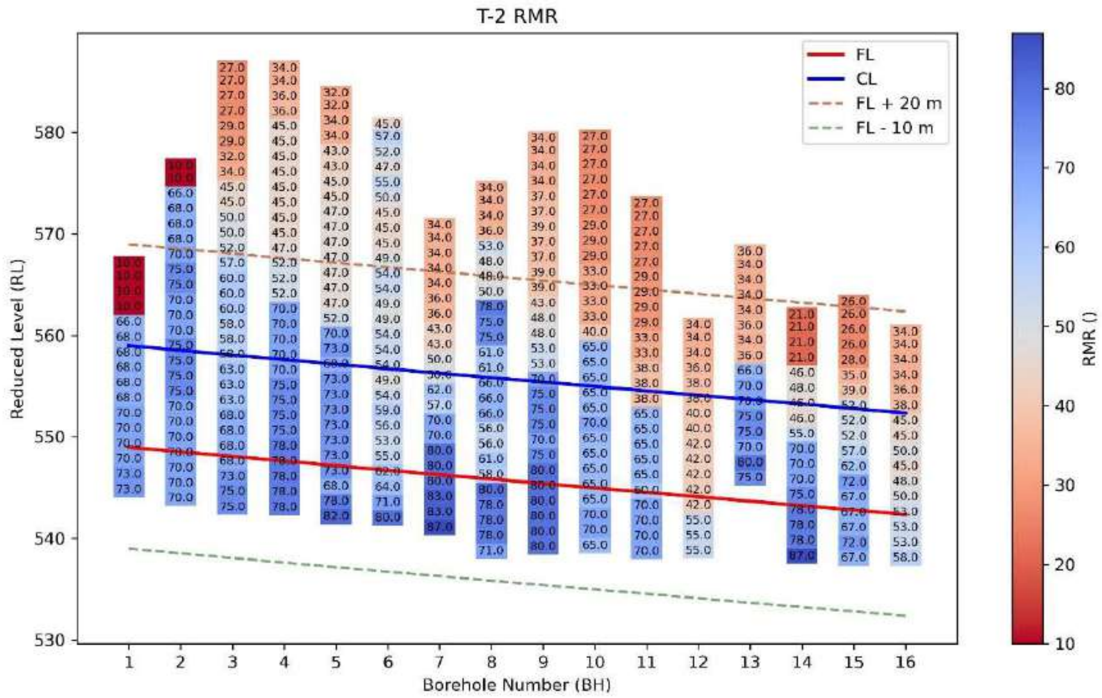
7.3.10.2 Recommended Abrasion Value considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(A= μ + σ across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(A_v < 25%): Slightly abrasive; 25–35%: Moderately abrasive; >35%: Highly abrasive. (CAI = 0.5–6).**	**Reference Standards / Guidelines**
42870 to 43620	2.53	Moderately abrasive	IS 2386 (Part 4):1963; ISRM (2007);

			CERCHAR (1986)

7.4 Geological assessment:

7.4.1 RMR:



7.4.1.1 Mean And Standard Deviation in RMR considering 1D zone of influence in each Borehole:

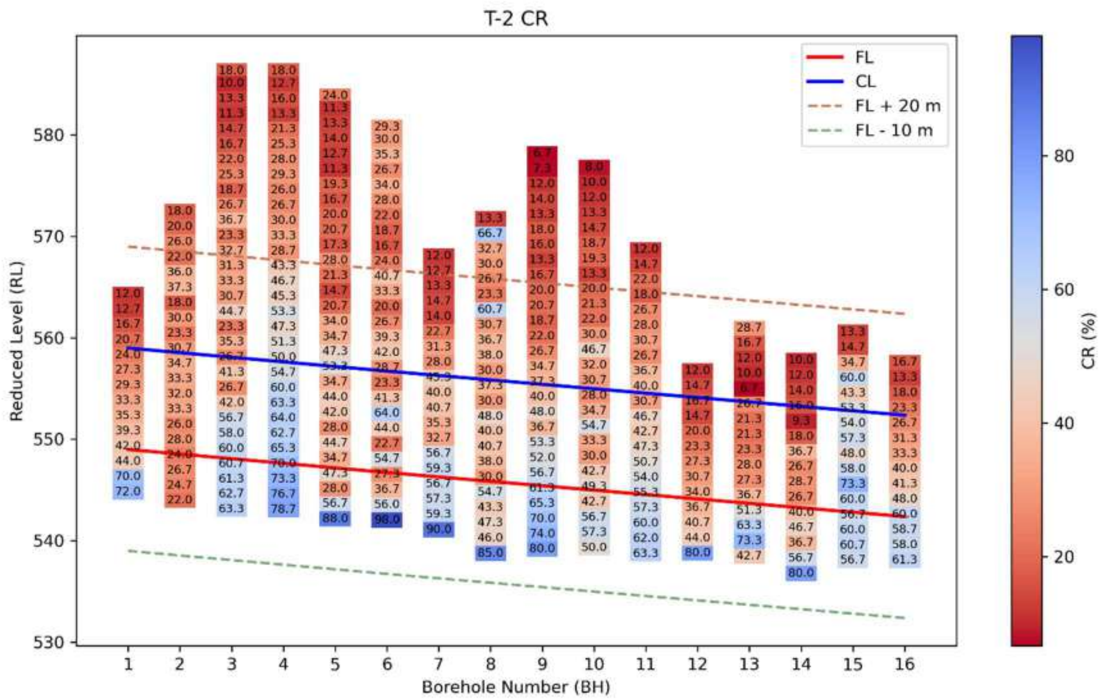
BH No.	Chainage	Mean	Std
BH01	42870	70	2
BH02	42920	72	2
BH03	42970	67	5
BH04	43020	76	3

BH No.	Chainage	Mean	Std
BH05	43070	73	4
BH06	43120	60	9
BH07	43170	73	12
BH08	43220	68	9
BH09	43270	77	3
BH10	43320	66	2
BH11	43370	64	9
BH12	43420	45	7
BH13	43470	74	4
BH14	43520	71	12
BH15	43570	64	7
BH16	43620	50	4

7.4.1.2 Recommended RMR considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Average across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(RMR_d = μ - σ)**	**Reference Standards / Guidelines**
42870 to 43620	67 (Class II)	56 (Class III)	IS 13365 (Part 2): 1998, Cl. 5.1 + Note on "representative values"

7.4.2 Core Recovery:



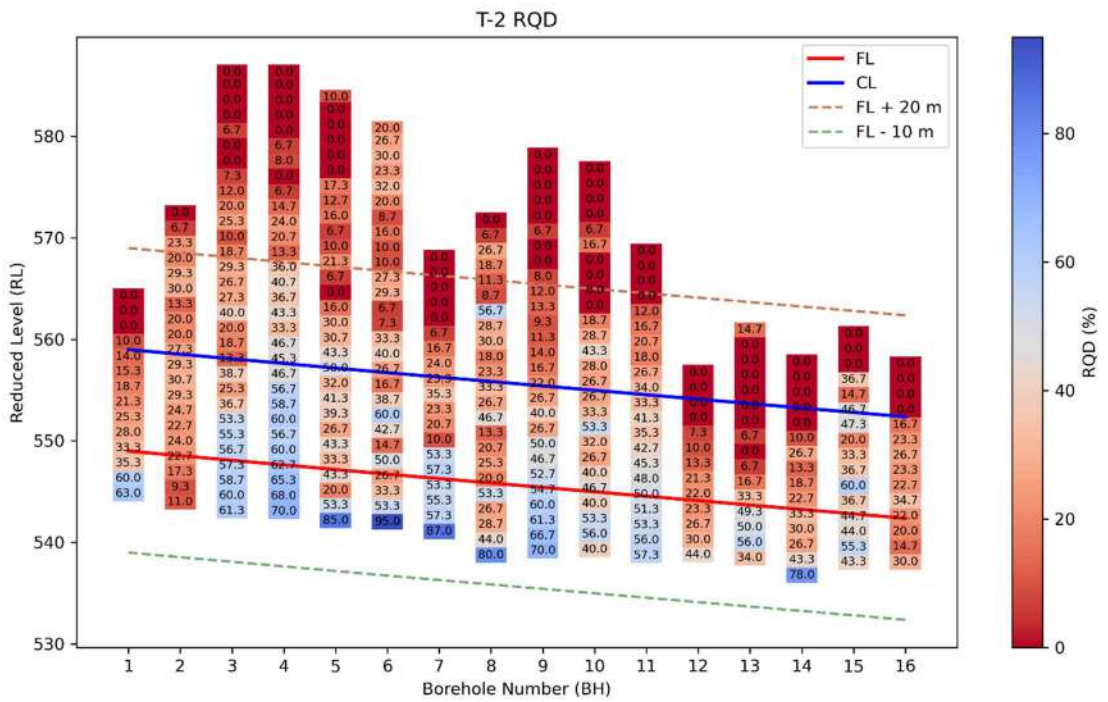
7.4.2.1 Mean And Standard Deviation in Core Recovery considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	42870	41.665	16.71128
BH02	42920	28.464	4.521205
BH03	42970	50.84455	14.17448
BH04	43020	66.863	7.626863
BH05	43070	45.57273	16.90595
BH06	43120	46.797	22.89333
BH07	43170	52.11727	16.10017
BH08	43220	45.02583	14.50138
BH09	43270	57.93727	13.73963
BH10	43320	43.57182	10.81974
BH11	43370	51.81455	9.644893
BH12	43420	33.45091	18.17779
BH13	43470	37.75364	17.89911
BH14	43520	36.90455	19.35969
BH15	43570	58.464	6.394196

BH No.	Chainage	Mean	Std
BH16	43620	45.864	13.10929

Overall Mean	**Std**
46.38082	16.99456

7.4.3 RQD:



7.4.3.1 Mean And Standard Deviation in RQD considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	42870	31.431	17.34872
BH02	42920	22.096	7.435992
BH03	42970	46.96545	16.12297
BH04	43020	60.463	6.661692
BH05	43070	42.51273	17.15831
BH06	43120	43.096	23.62913
BH07	43170	43.3	22.65365
BH08	43220	34.885	18.47262
BH09	43270	50.48091	14.60424
BH10	43320	40.72455	10.55632
BH11	43370	46.72455	7.953133
BH12	43420	17.99818	13.40952
BH13	43470	22.96727	22.13629
BH14	43520	30.263	19.36422
BH15	43570	42.13	11.3479

BH No.	Chainage	Mean	Std
BH16	43620	23.396	5.933592

7.4.3.2 Recommended RQD considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Average across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(RQD_d = μ - σ)**	**Reference Standards / Guidelines**
42870 to 43620	37.5	23.7	IS 11315:1985; Deere (1963)

7.5 Petrographic Assessments:

7.5.1 Description of rock Masses

7.5.1.1 Khondalite

Typical khondalite assemblages in the EGMB are described as quartz–feldspar–garnet–sillimanite ± biotite gneisses, with quartz + feldspar together forming ~40–60 %, garnet ~20–30 %, and sillimanite + biotite ~10–20 % of the rock.

7.5.1.2 Migmatite

Migmatites of the Eastern Ghats consist of quartz- and feldspar-rich leucosome with garnet- and biotite-bearing melanosome/restite. Petrographic and modal descriptions are consistent with dominant quartz + feldspar (~60–70 %) and subordinate garnet and biotite as.

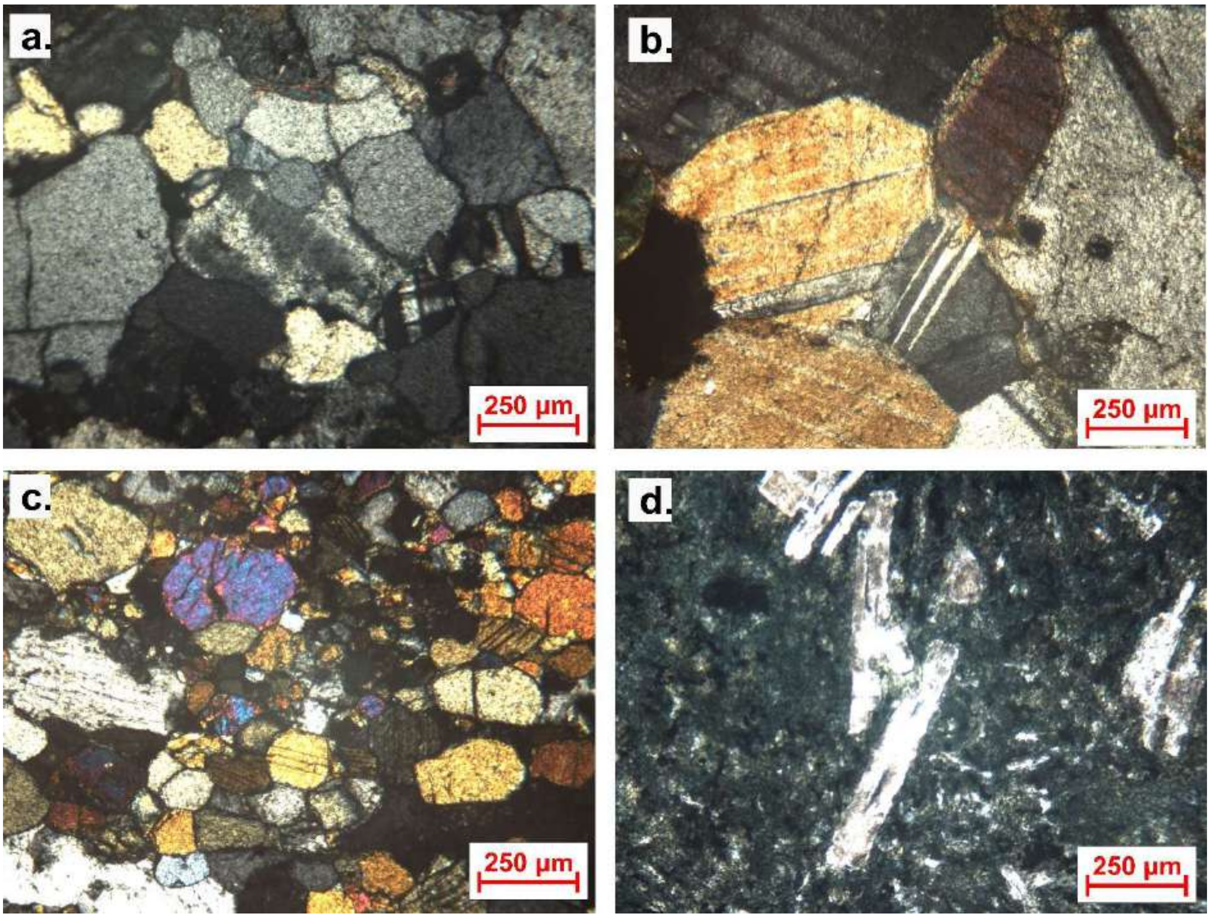
7.5.1.3 Khondalite (Restite)

Restites represent refractory residues after partial melting of metapelites, enriched in garnet and ferromagnesian minerals and depleted in felsic phases. Modal characteristics inferred from petrographic descriptions of residual granulites and restitic layers in the.

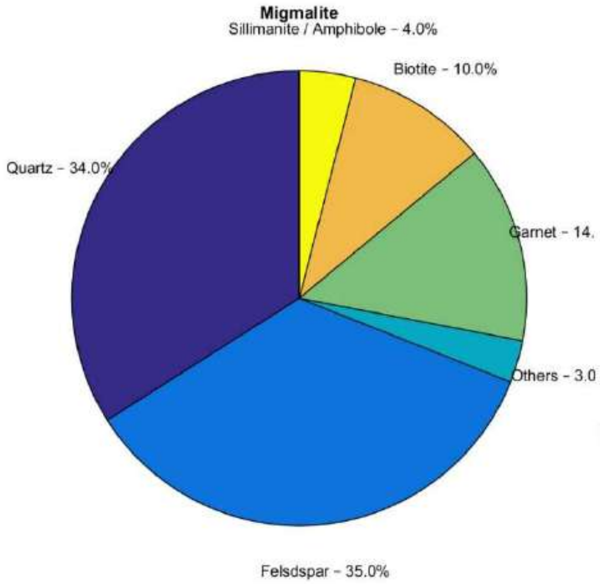
7.5.1.4 Meta basic rock

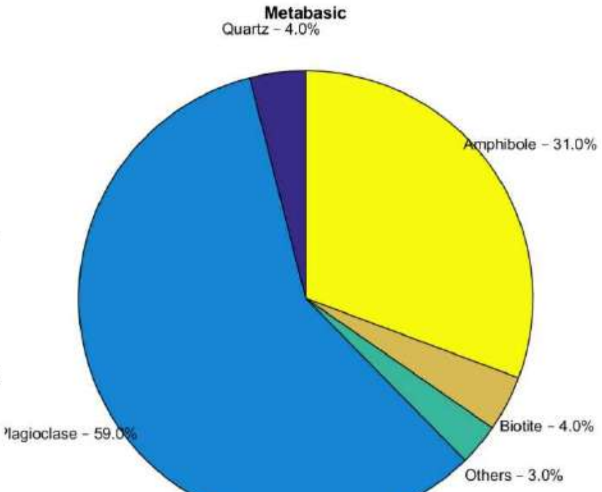
Metamorphosed doleritic rocks typically comprise plagioclase + amphibole as dominant phases, with minor quartz and accessory oxides. Modal proportions are consistent with amphibolite-facies metadolerites described from Indian Proterozoic terrains.

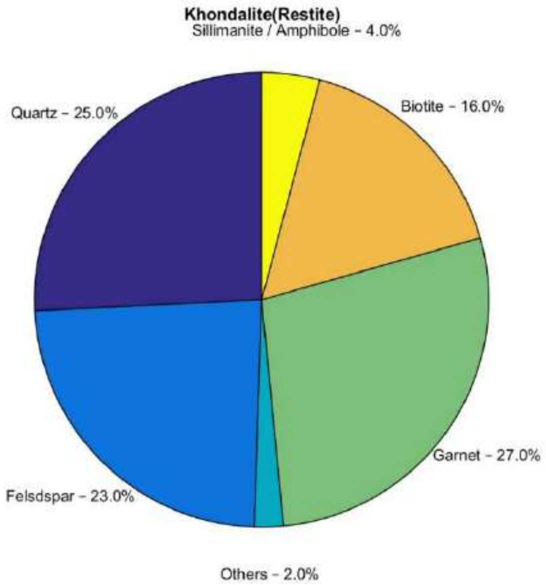
7.5.2 Micro Photographs

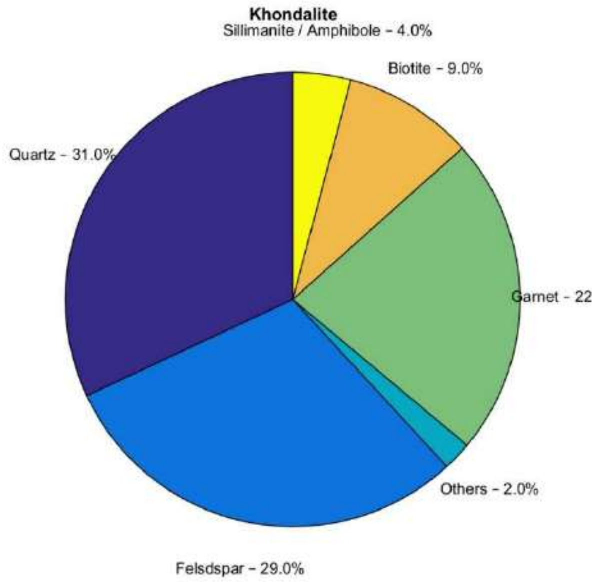


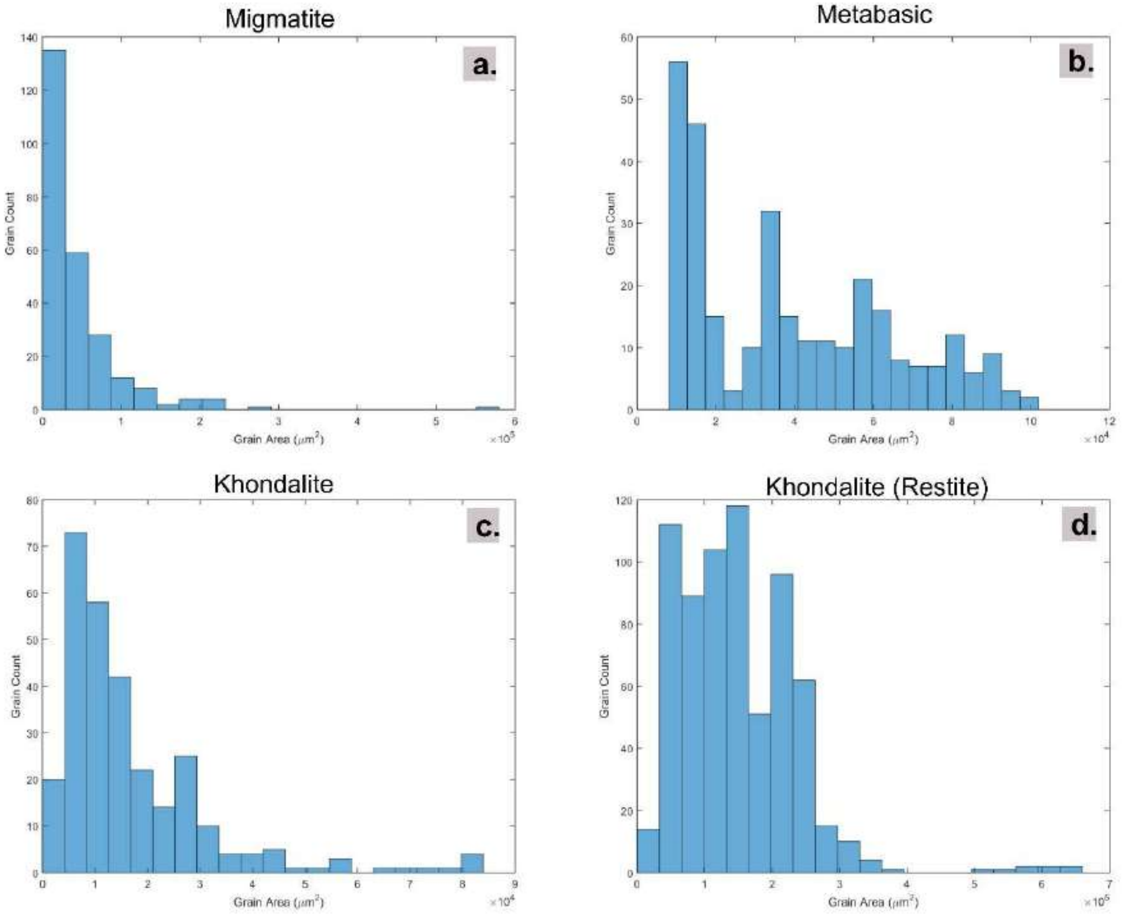
7.5.3 Mineral percentage and grain size distribution



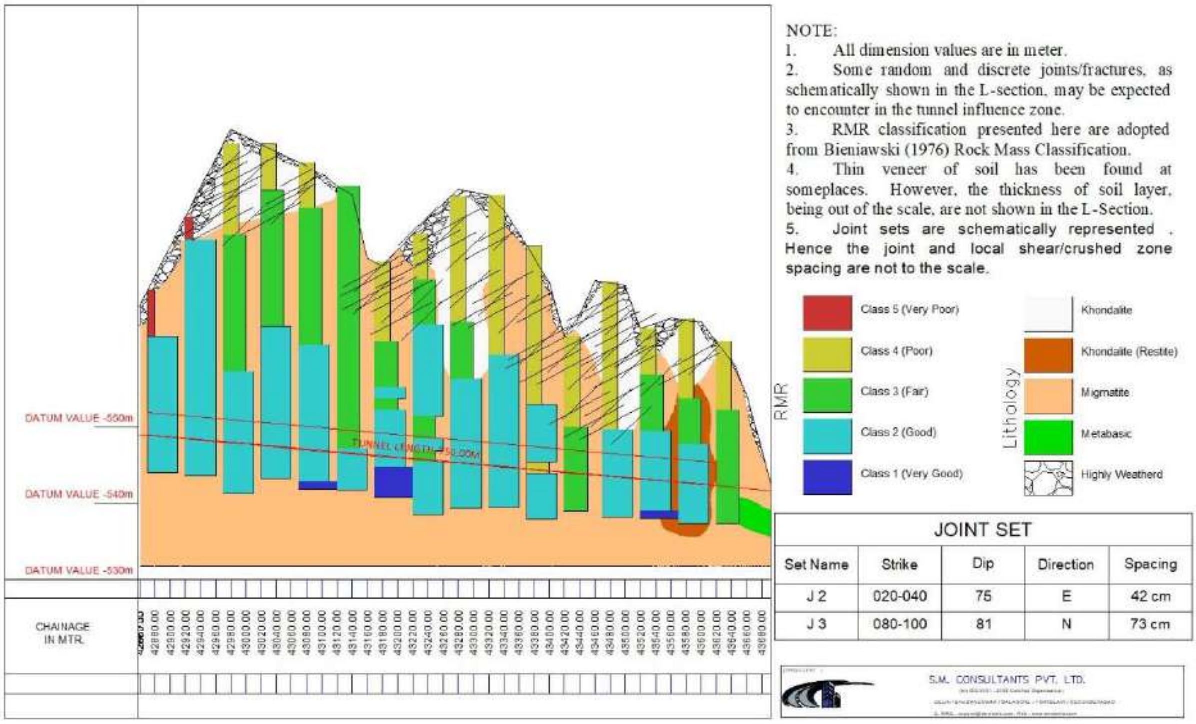







Migmatite
a.
Metabasic
b.
Khondalite
c.
Khondalite (Restite)
d.
Grain Count
Grain Area (μm²)
×10⁵
×10⁴

7.6 L section:



JOINT SET

Set Name	Strike	Dip	Direction	Spacing
J 2	020-040	75	E	42 cm
J 3	080-100	81	N	73 cm

8 TUNNEL 3: GEOLOGICAL & GEOTECHNICAL ASSESMENT

8.1 Exploratory drillings

As per the requirement of scope of work outlined in the terms of reference, bore holes were drilled with a cumulative length of 407m at different locations along the proposed alignment. Necessary care has been taken during drilling operations by deploying good quality diamond drill machines to obtain good core recovery to obtain RQD values. The locations of the boreholes were selected in such a way, so that these holes intersect the envisaged ground/ strata conditions at different depths. The location and details of boreholes drilled; total depth of drillings is shown in table below.

Chainage	Bh Name	GL	FL	Depth
44181	BH-1	556.387	537.412	25.00
44231	BH-2	559.918	536.931	29.00
44281	BH-3	564.439	536.45	33.00
44331	BH-4	567.197	535.969	37.00
44381	BH-5	562.017	535.489	32.00
44431	BH-6	561.797	535.008	34.00
44481	BH-7	568.657	534.527	39.00
44531	BH-8	568.512	534.046	40.00
44581	BH-9	559.968	533.565	31.50
44631	BH-10	556.998	533.085	29.00
44681	BH-11	551.227	532.604	23.50
44731	BH-12	557.45	532.123	31.00
44771	BH-13	550.35	531.739	23.50

8.2 TUNNEL-3 SRT

8.2.1 Location:

Sr. No.	Chainage	Line	Spread	Location (T-03)		Length
				Start	End	In meter
1	44.181km to 44.771km	L1	S1 to S5	44.181km	44.771km	570

8.2.2 Seismic survey results and conclusion

Table 8.1: Summery of Tunnel-1 SRT test

Variation of maximum range of thicknesses below EGL (M)			Avg. V_p (m/sec)	Calculated V_s (m/sec)	Dynamic Young's Modulus (MPa)	Shear Modulus (MPa)
Layer	From	To				
Layer-I	0.50	2.50	700	327	466	172
Layer-II	2.50	7.50	1700	875	4039	1530
Layer-III	7.50	25.00	3400	1879	21702	8477

<u>Sample Calculation:</u>

The Young's Modulus E is the uni-axial stress-strain ratio. Its dynamic value is expressed by the following equation:

$$E = \frac{\rho V_P^2 (1+\mu)(1-2\mu)}{1-\mu}$$

Where, E = Dynamic Young's Modulus in kN/m2

Vp = 700 m/sec

ρ = 1.6 gm/cc ≈ 1.60 kN.s2/m4 (mass density)

μ = 0.36

So, calculated E = 466480 kN/sqm ≈ 466 MPa

The Shear Modulus G is the stress-strain ratio for simple shear. Its dynamic value is obtained by the following:

$$G = \frac{E}{2(1+\mu)} = \rho V_s^2$$

So, Shear Modulus G comes out to be 171500 kN/sqm ≈ 172 MPa

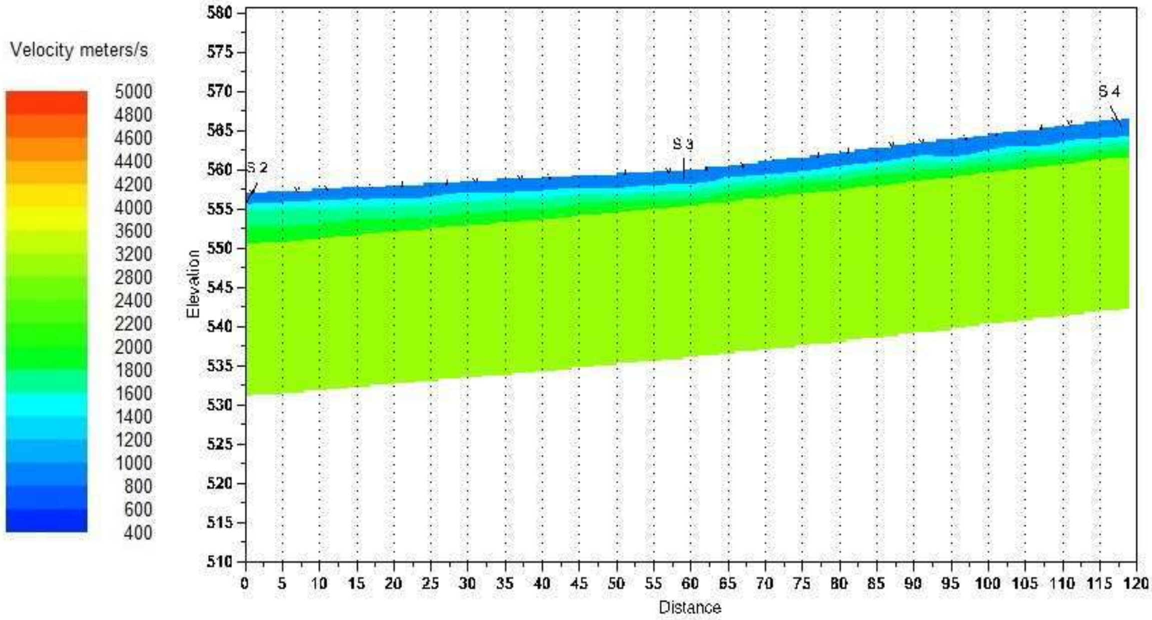
Again,

$G = \rho V_s^2$ giving Vs = √(G/ρ)

So, calculated Vs = 327.395 m/sec, say 327 m/sec

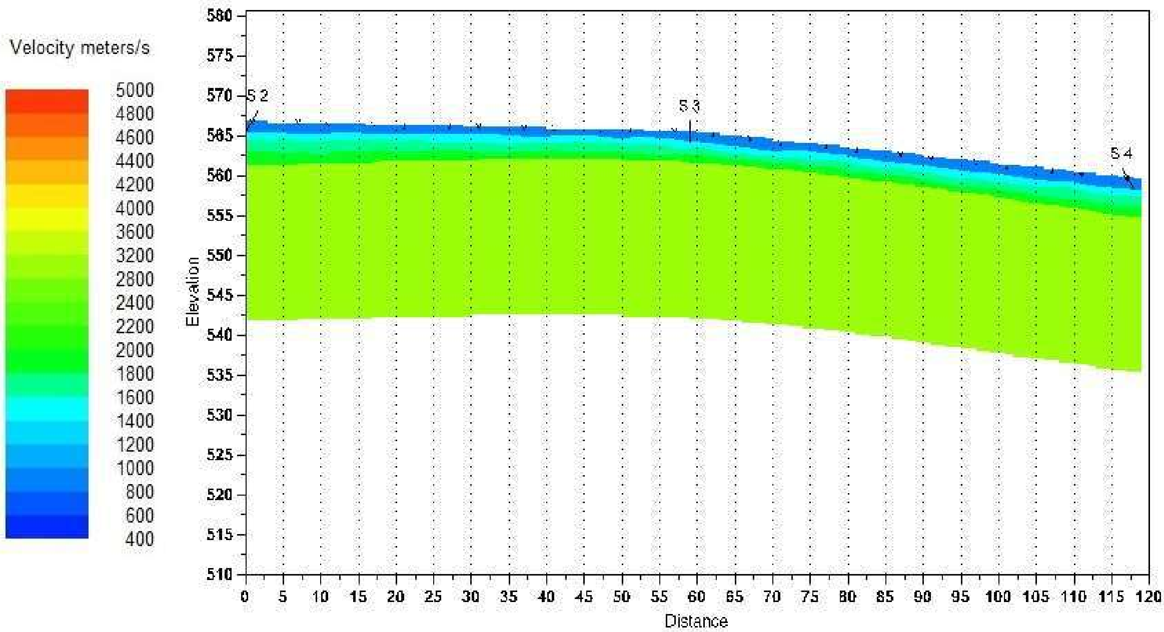
SEISMIC PROFILE(T04)

T3L1S1



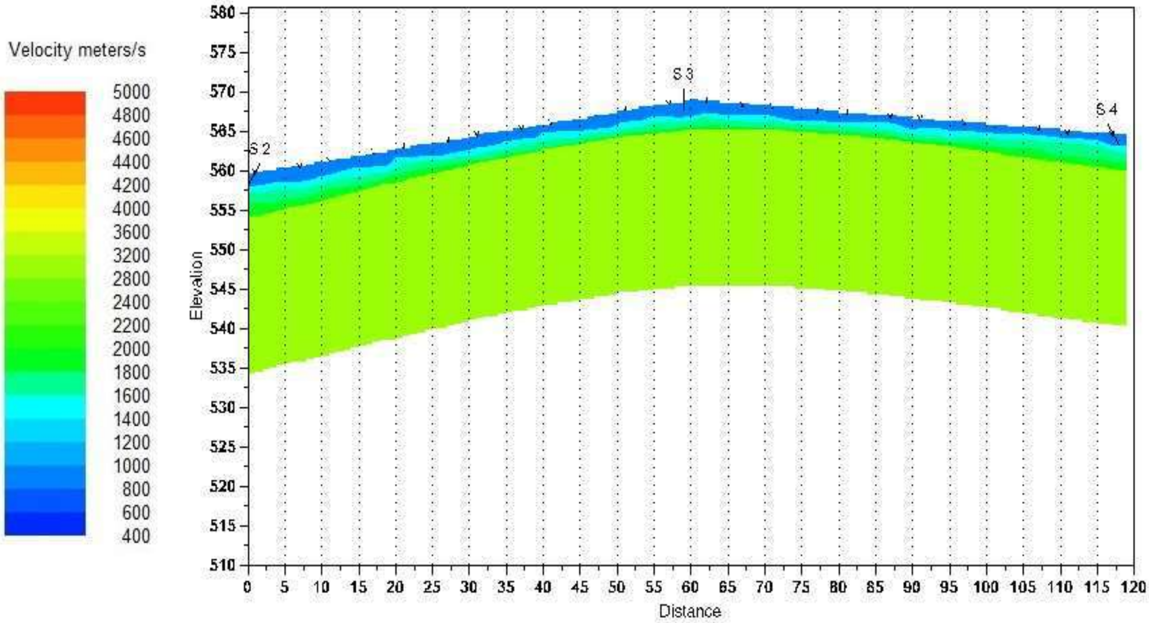
Shot point depth computation

T3L1S1



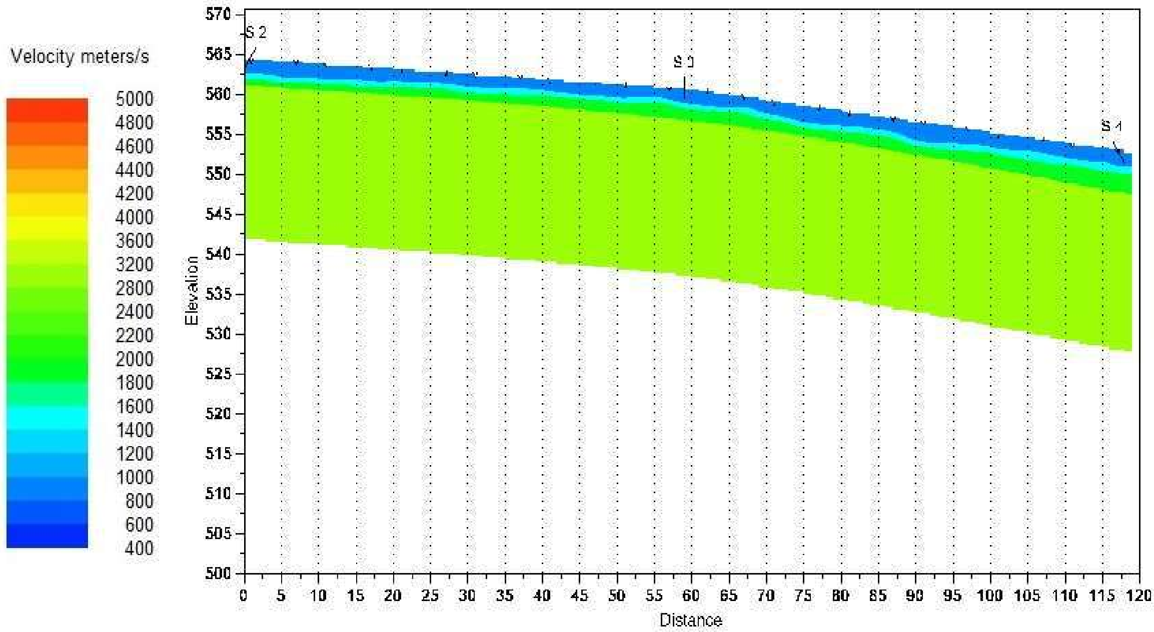
Shot point depth computation

T3L1S1



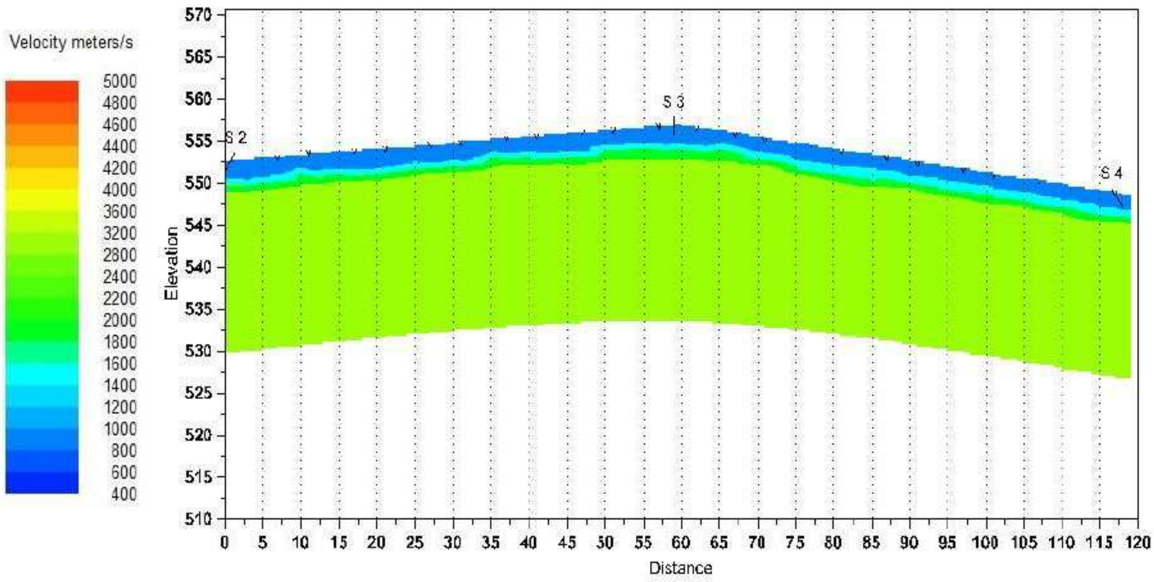
Shot point depth computation

T3L1S1



Shot point depth computation

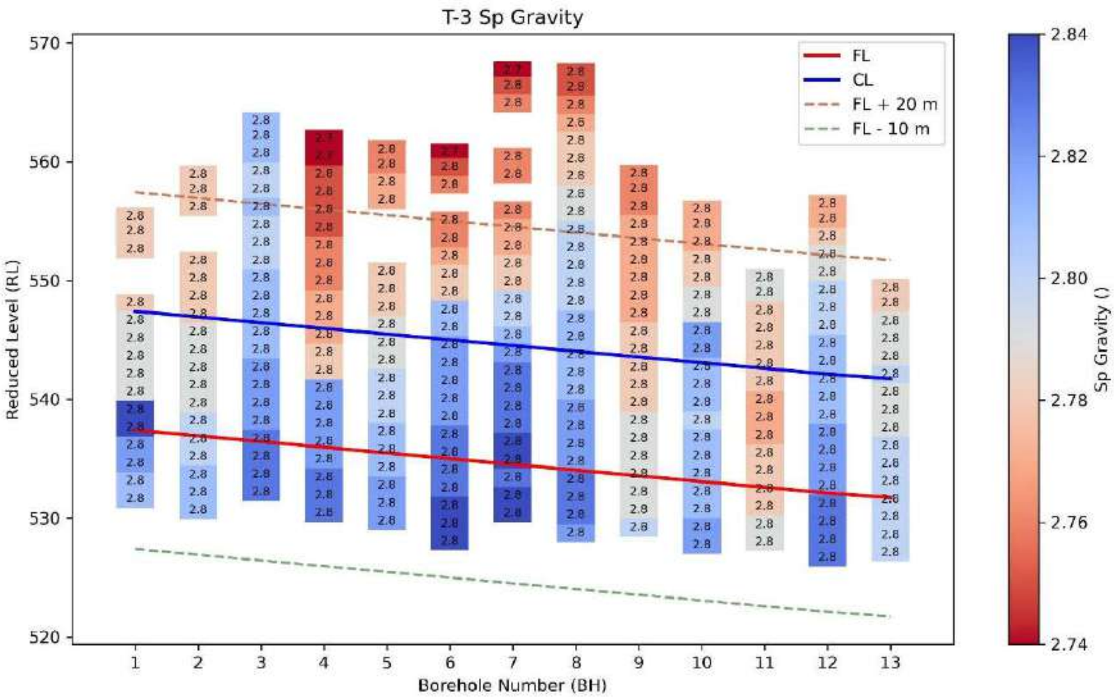
T3L1S1



Shot point depth computation

8.3 Assessment of the engineering properties of rock sample:

8.3.1 Specific Gravity



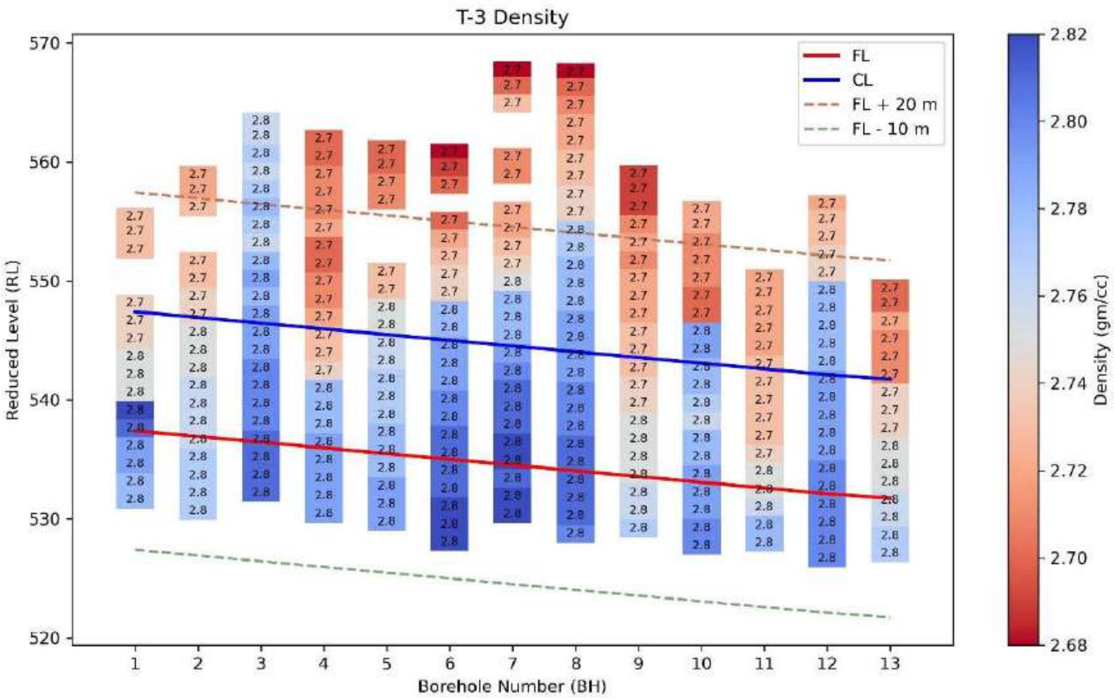
8.3.1.1 Mean and Standard deviation Specific Gravity considering 1D zone of influence of each borehole:

BH No.	Chainage	Mean	Std
BH01	44181	2.8	0.020
BH02	44231	2.8	0.009
BH03	44281	2.8	0.008
BH04	44331	2.8	0.023
BH05	44381	2.8	0.011
BH06	44431	2.8	0.008
BH07	44481	2.8	0.007
BH08	44531	2.8	0.008
BH09	44581	2.8	0.006
BH10	44631	2.8	0.005
BH11	44681	2.8	0.007
BH12	44731	2.8	0.008
BH13	44771	2.8	0.005

8.3.1.2 Recommended Specific Gravity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Sp = μ - σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
41592 to 41842	2.79	IS 13030:1991; IS 1124:1974
41842 to 419992	2.79	
419992 to 42052 (P2)	2.78	

8.3.2 Dry Density



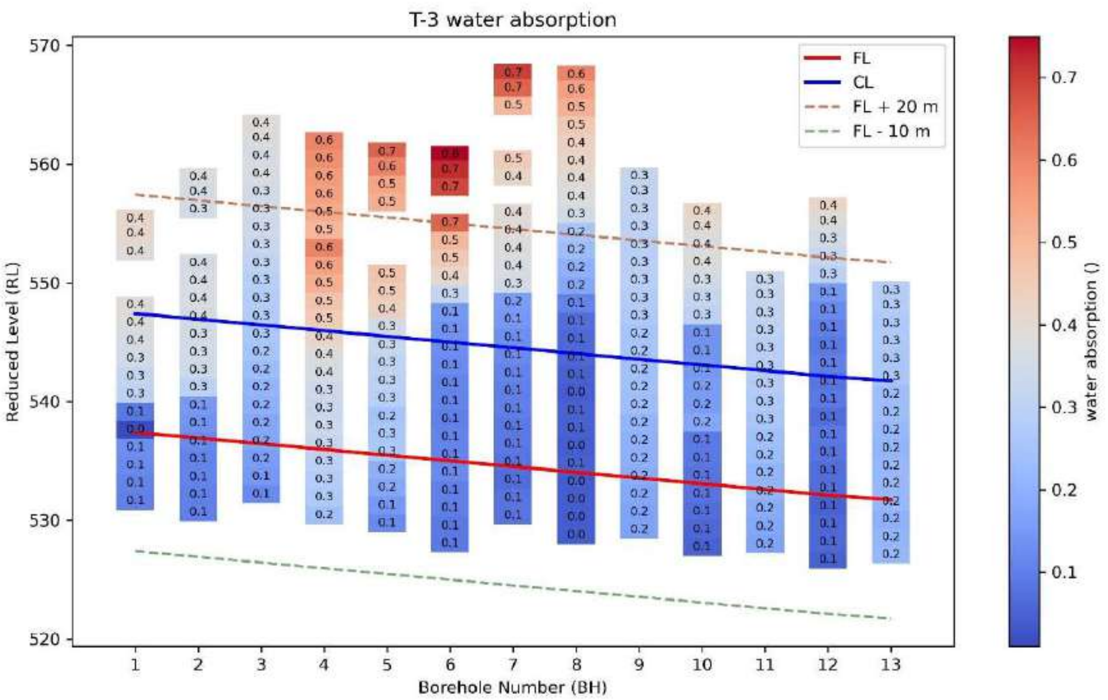
8.3.2.1 Mean and Standard Deviation in Density considering 1D zone of influence of each borehole:

BH No.	Chainage	Mean	Std
BH01	44181	2.77	0.028
BH02	44231	2.76	0.008
BH03	44281	2.80	0.008
BH04	44331	2.77	0.025
BH05	44381	2.78	0.011
BH06	44431	2.81	0.012
BH07	44481	2.81	0.010
BH08	44531	2.80	0.008
BH09	44581	2.75	0.012
BH10	44631	2.78	0.012
BH11	44681	2.75	0.020
BH12	44731	2.80	0.005
BH13	44771	2.75	0.012

8.3.2.2 Recommended Density considering 1D zone of influence:

Chainage	Statistical / Reduction Method (d = μ - σ across boreholes within ±10 m of FL and CL)	Reference Standards / Guidelines
41592 to 41842	2.75	IS 13063:1991
41842 to 419992	2.76	
419992 to 42052 (P2)	2.74	

8.3.3 Water absorption Test



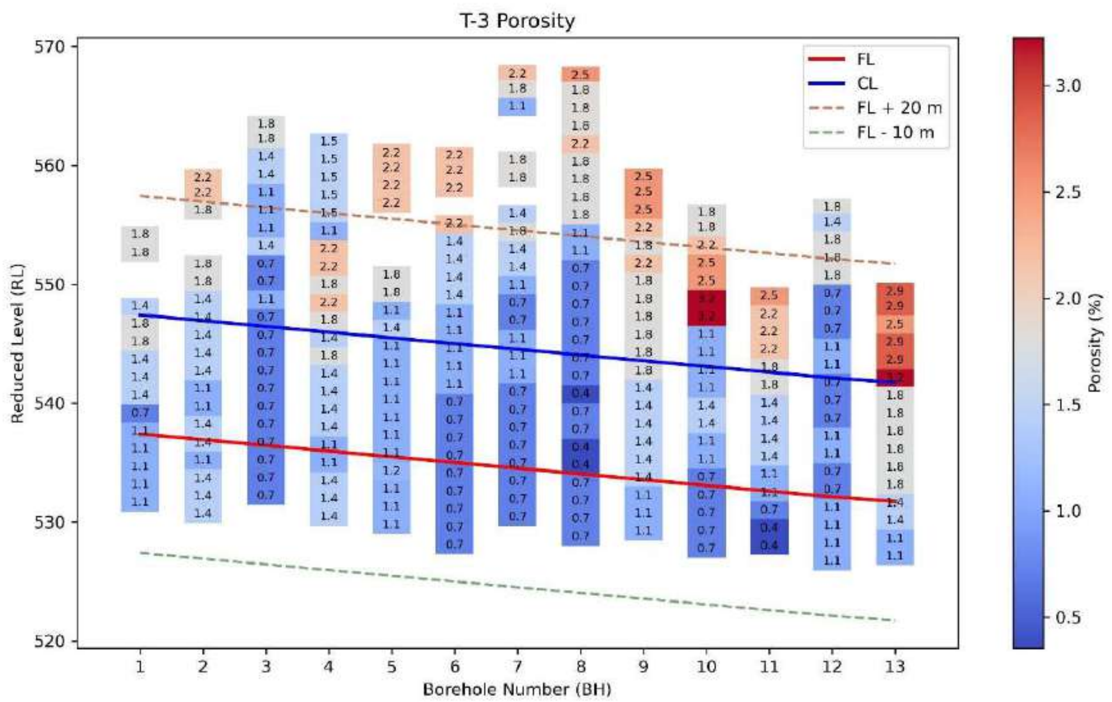
8.3.3.1 Mean and Standard Deviation in Water Absorption Value considering 1D zone of influence:

BH No.	Chainage	Mean	Std
BH01	44181	0.21	0.135
BH02	44231	0.20	0.111
BH03	44281	0.19	0.045
BH04	44331	0.33	0.052
BH05	44381	0.24	0.069
BH06	44431	0.11	0.012
BH07	44481	0.09	0.015
BH08	44531	0.05	0.007
BH09	44581	0.20	0.027
BH10	44631	0.11	0.047
BH11	44681	0.22	0.038
BH12	44731	0.07	0.014
BH13	44771	0.22	0.009

8.3.3.2 Recommended Water Absorption Value considering 1D zone of influence:

Chainage	Statistical / Reduction Method (A= μ + σ across boreholes within ±10 m of FL and CL)	Reference Standards / Guidelines
41592 to 41842	0.33	IS 13063:1991; IS 2386 (Part 3):1963
41842 to 419992	0.16	
419992 to 42052 (P2)	0.24	

8.3.4 Porosity



8.3.4.1 Mean And Standard Deviation in Porosity considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	1.26	0.340
BH02	44231	1.33	0.166
BH03	44281	0.71	0.002
BH04	44331	1.39	0.197
BH05	44381	1.08	0.026
BH06	44431	0.80	0.162
BH07	44481	0.78	0.150
BH08	44531	0.61	0.165
BH09	44581	1.36	0.229
BH10	44631	0.97	0.281
BH11	44681	1.12	0.495
BH12	44731	0.90	0.183
BH13	44771	1.573	0.303

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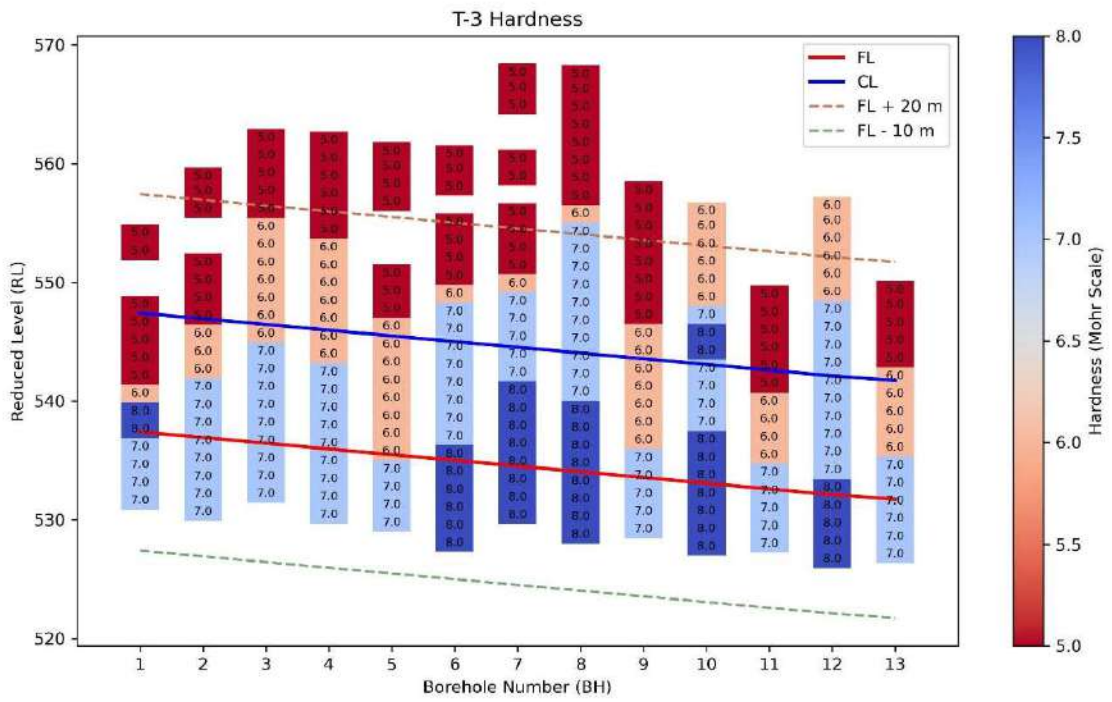
8.3.4.2 Recommended Porosity considering 1D zone of influence:

Chainage	Statistical / Reduction Method (n = μ ± σ across boreholes within ±10 m of FL and CL)	Reference Standards / Guidelines
41592 to 41842	1.16 ± 0.3	IS 1124:1974; ISRM (1981)
41842 to 419992	0.89 ± 0.3	
419992 to 42052 (P2)	1.18 ± 0.4	

8.3.5 Hardness



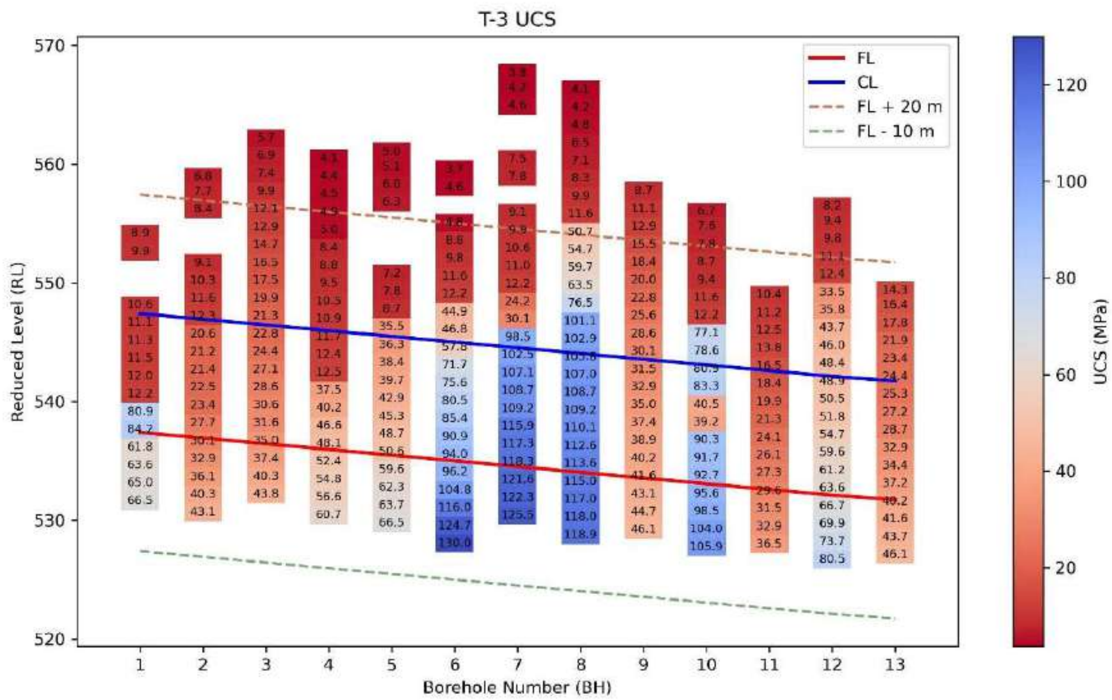
8.3.5.1 Mean And Standard Deviation in Hardness considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	6	1
BH02	44231	7	0
BH03	44281	7	0
BH04	44331	7	0
BH05	44381	6	1
BH06	44431	8	1
BH07	44481	8	0
BH08	44531	8	0
BH09	44581	7	1
BH10	44631	8	1
BH11	44681	6	1
BH12	44731	7	1
BH13	44771	7	1

8.3.5.2 Recommended Hardness considering 1D zone of influence:

Chainage	Statistical / Reduction Method (Hi = μ - σ across boreholes within ±10 m of FL and CL)	Reference Standards / Guidelines
41592 to 41842	6	IS 13311 (Part 2):1992; ISRM 1978
41842 to 419992	7	
419992 to 42052 (P2)	6	

8.3.6 Compression Test



8.3.6.1 Mean And Standard Deviation in UCS considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	43.63	31.42
BH02	44231	29.03	8.11
BH03	44281	32.15	6.88
BH04	44331	39.41	18.74
BH05	44381	50.37	10.97
BH06	44431	93.97	21.87
BH07	44481	114.83	7.59
BH08	44531	112.35	4.52
BH09	44581	39.14	4.94
BH10	44631	83.88	23.03
BH11	44681	26.76	5.94
BH12	44731	61.93	10.17
BH13	44771	35.75	7.23

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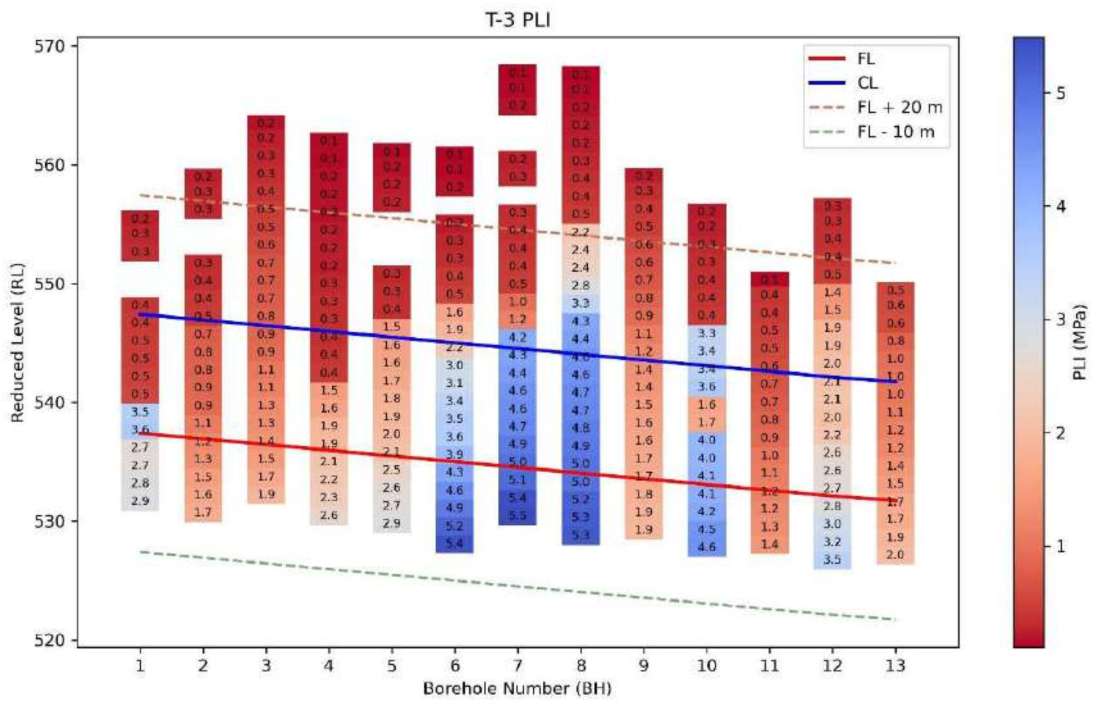
8.3.6.2 Recommended UCS considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(UCS_k = μ - σ) across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(UCS_d = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)**	**Reference Standards / Guidelines**
41592 to 41842	20.16	4.03	IS 9143:1979; IS 13365 (Part 2):1998; Hoek & Brown (2002, 2019)
41842 to 419992	58.77	29.38	
419992 to 42052 (P2)	24.91	7.47	

8.3.7 Point Load Test



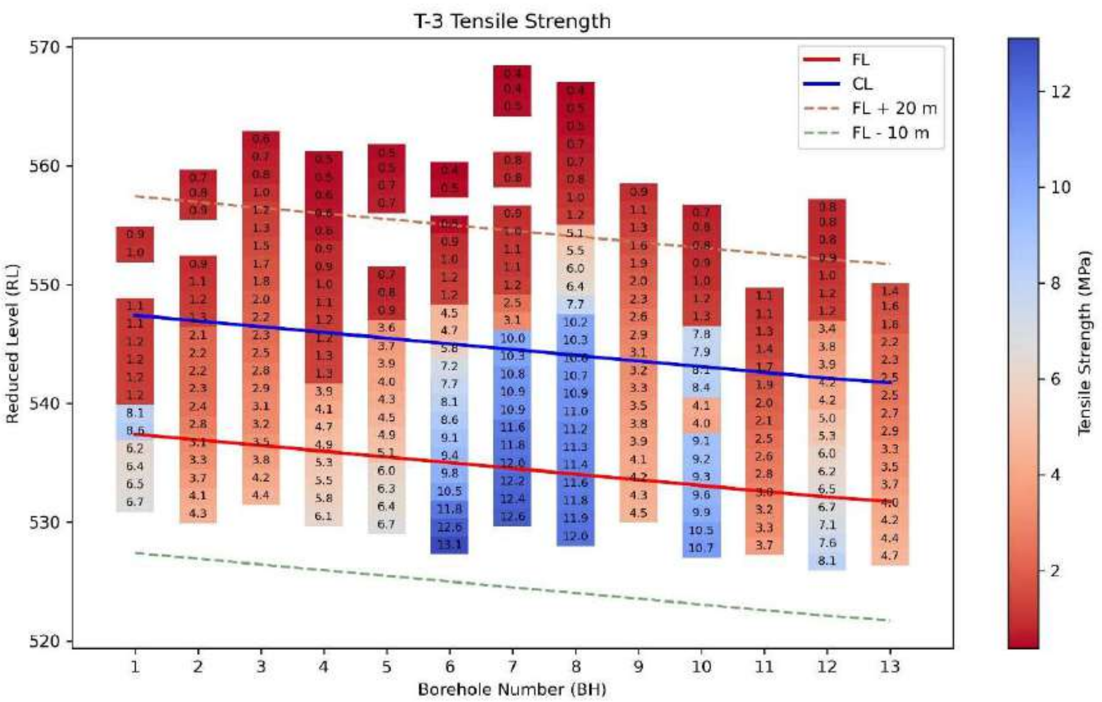
8.3.7.1 Mean And Standard Deviation in PLI considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	1.87	1.36
BH02	44231	1.15	0.36
BH03	44281	1.30	0.32
BH04	44331	1.57	0.82
BH05	44381	2.12	0.47
BH06	44431	3.93	0.97
BH07	44481	4.86	0.40
BH08	44531	4.92	0.26
BH09	44581	1.64	0.19
BH10	44631	3.63	1.03
BH11	44681	1.04	0.25
BH12	44731	2.61	0.48
BH13	44771	1.48	0.33

8.3.7.2 Recommended PLI considering 1D zone of influence:

Chainage	Statistical / Reduction Method (UCS_k = μ - σ) across boreholes within ±10 m of FL and CL)	Recommended Design Value (UCS_d = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)	Reference Standards / Guidelines
41592 to 41842	0.77	0.15	IS 9143:1979; IS 13365 (Part 2):1998; Hoek & Brown (2002, 2019)
41842 to 419992	2.4	1.23	
419992 to 42052 (P2)	0.97	0.29	

8.3.8 Brazilian Test



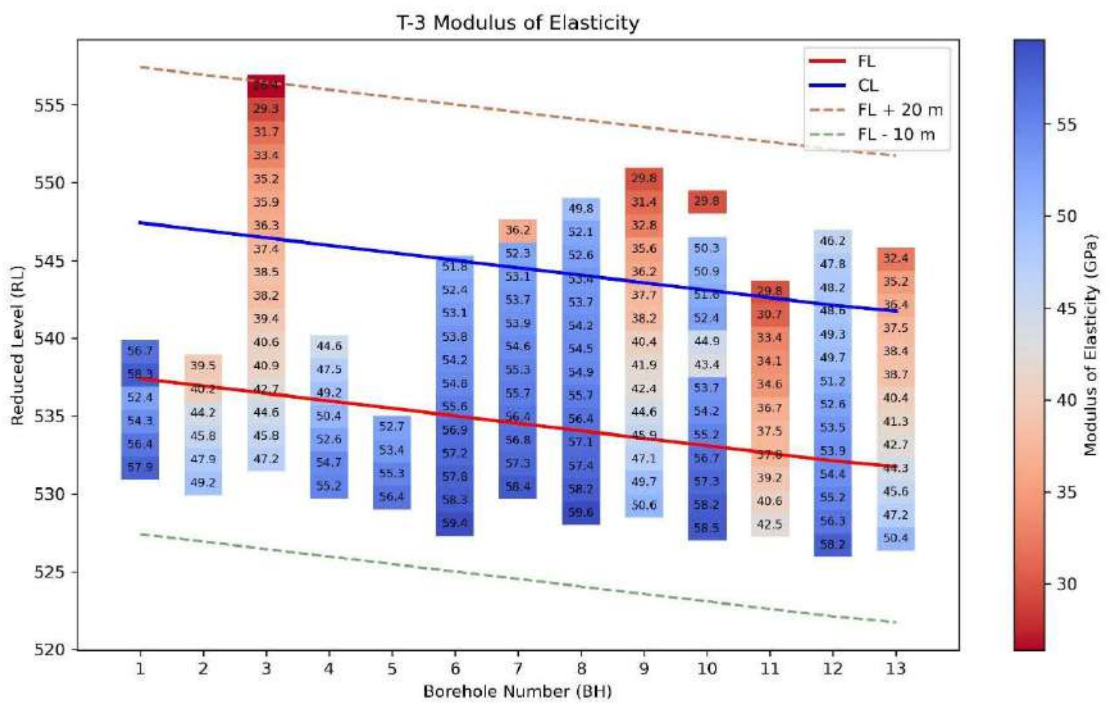
8.3.8.1 Mean And Standard Deviation in Tensile Strength considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	4.41	3.16
BH02	44231	2.96	0.81
BH03	44281	3.26	0.70
BH04	44331	4.01	1.87
BH05	44381	5.07	1.10
BH06	44431	9.48	2.21
BH07	44481	11.56	0.78
BH08	44531	11.32	0.45
BH09	44581	3.87	0.45
BH10	44631	8.46	2.30
BH11	44681	2.70	0.60
BH12	44731	6.10	1.29
BH13	44771	3.59	0.73

8.3.8.2 Recommended Tensile Strength considering 1D zone of influence:

Chainage	Statistical / Reduction Method (mean value across boreholes within ±10 m of FL and CL× 0.8 (account for anisotropy))	Recommended Design Value (μ - σ)	Reference Standards / Guidelines
41592 to 41842	3.95	2.06	IS 10082:1982; Hoek & Brown (1997); ISRM Suggested Methods
41842 to 419992	9.08	6.03	
419992 to 42052 (P2)	4.19	2.45	

8.3.9 Modulus of elasticity test



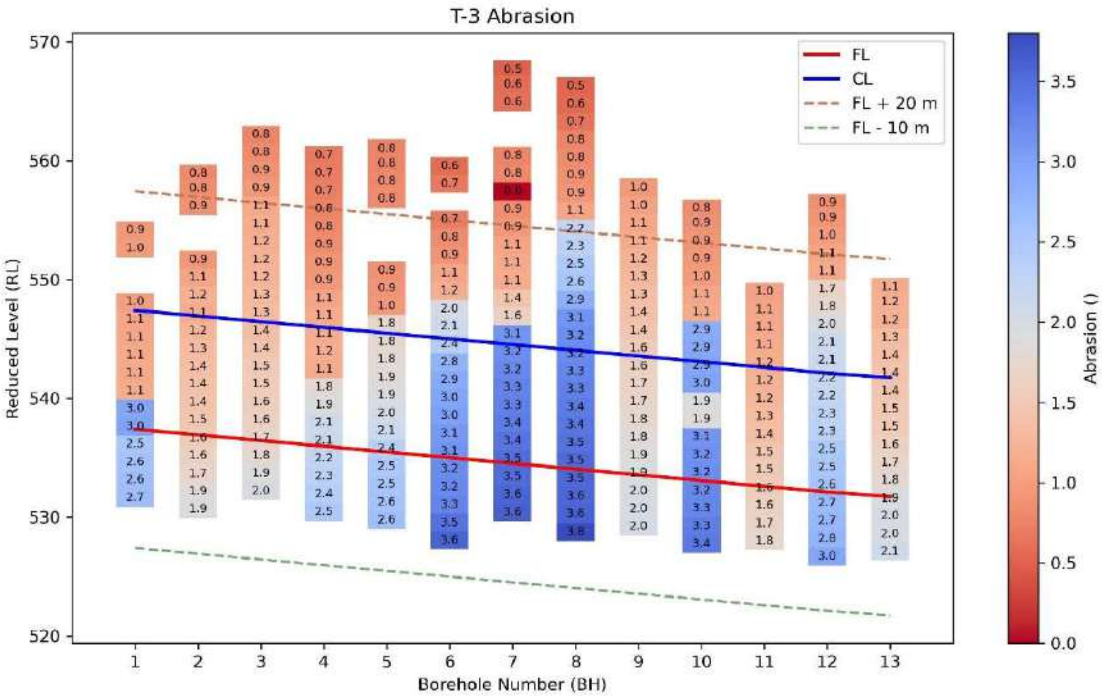
8.3.9.1 Mean And Standard Deviation in Elasticity considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	56.00	2.25
BH02	44231	44.47	3.97
BH03	44281	41.53	3.40
BH04	44331	50.60	3.86
BH05	44381	54.45	1.70
BH06	44431	55.44	2.47
BH07	44481	55.52	1.72
BH08	44531	55.92	2.00
BH09	44581	43.85	4.51
BH10	44631	53.28	5.05
BH11	44681	36.71	3.57
BH12	44731	52.99	3.05
BH13	44771	42.65	4.21

8.3.9.2 Recommended Modulus of Elasticity considering 1D zone of influence:

Chainage	Statistical / Reduction Method (Ei = μ - σ across boreholes within ±10 m of FL and CL× 0.8 (account for anisotropy))	Recommended Design Value (Ed = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)	Reference Standards / Guidelines
41592 to 41842	41.57	8.31	IS 13365 (Part 2):1998; Hoek & Diederichs (2006)
41842 to 419992	47.40	23.70	
419992 to 42052 (P2)	36.65	10.99	

8.3.10 Abrasion test



8.3.10.1 Mean And Standard Deviation in Abrasion value considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	1.99	0.860
BH02	44231	1.54	0.226
BH03	44281	1.63	0.196
BH04	44331	1.89	0.514
BH05	44381	2.19	0.329
BH06	44431	3.08	0.318
BH07	44481	3.40	0.143
BH08	44531	3.46	0.163
BH09	44581	1.84	0.139
BH10	44631	2.95	0.540
BH11	44681	1.48	0.185
BH12	44731	2.52	0.252
BH13	44771	1.76	0.227

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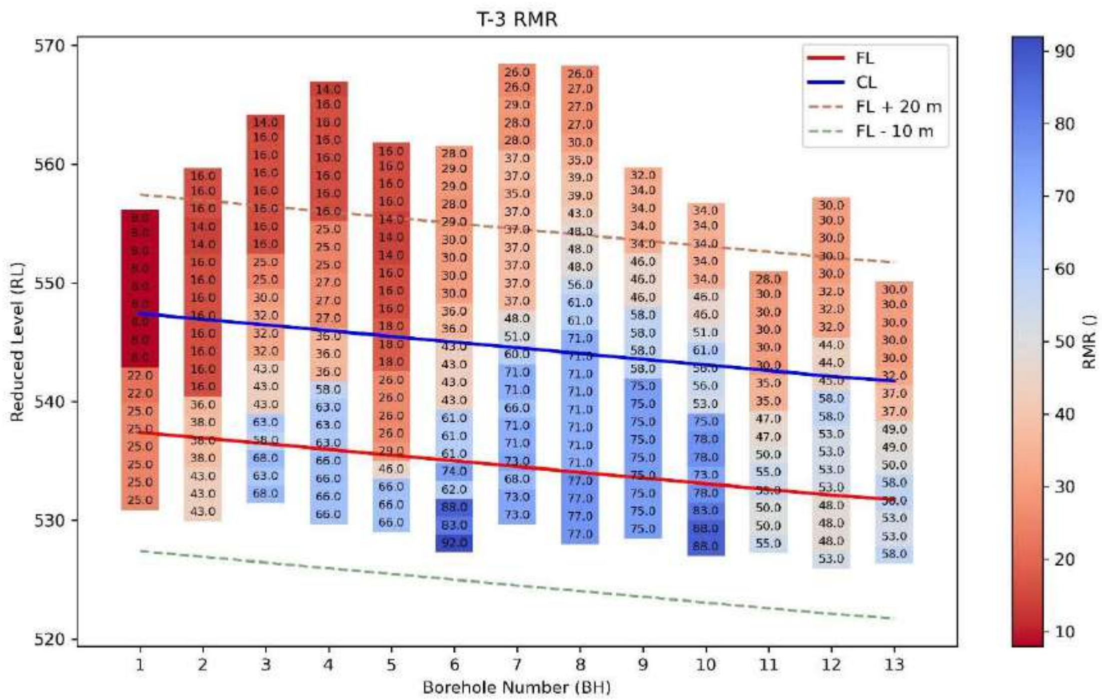
8.3.10.2 Recommended Abrasion Value considering 1D zone of influence:

Chainage	Statistical / Reduction Method (A= μ + σ across boreholes within ±10 m of FL and CL)	Recommended Design Value (A_v < 25%): Slightly abrasive; 25–35%: Moderately abrasive; >35%: Highly abrasive. (CAI = 0.5–6).	Reference Standards / Guidelines
41592 to 41842	2.38	Slightly abrasive	IS 2386 (Part 4):1963; ISRM (2007); CERCHAR (1986)
41842 to 419992	3.60	Moderately abrasive	
419992 to 42052 (P2)	2.44	Slightly abrasive	

8.4 Geological assessment:

8.4.1 RMR:



8.4.1.1 Mean And Standard Deviation in RMR considering 1D zone of influence in each Borehole:

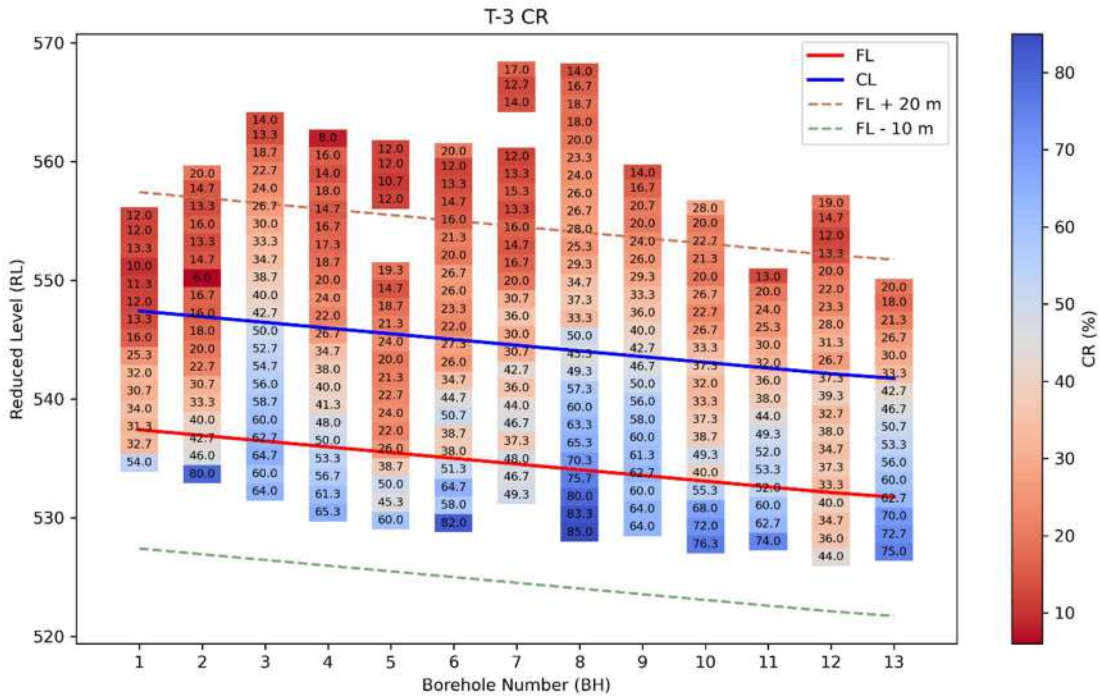
BH No.	Chainage	Mean	Std
BH01	44181	20	8
BH02	44231	31	12
BH03	44281	51	14
BH04	44331	56	13
BH05	44381	38	20
BH06	44431	63	18

BH07	44481	70	4
BH08	44531	73	3
BH09	44581	73	5
BH10	44631	73	13
BH11	44681	48	7
BH12	44731	52	4
BH13	44771	50	8

8.4.1.2 Recommended RMR considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Average across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(RMR_d = μ - σ)**	**Reference Standards / Guidelines**
44180 to 44400	39 (Class IV)	20 (Class IV)	IS 13365 (Part 2): 1998, Cl. 5.1 + Note on "representative values"
44400 to 44680	70 (Class II)	59 (Class III)	
44680 to 44800 (P2)	50 (Class III)	43 (Class III)	

8.4.2 Core Recovery:



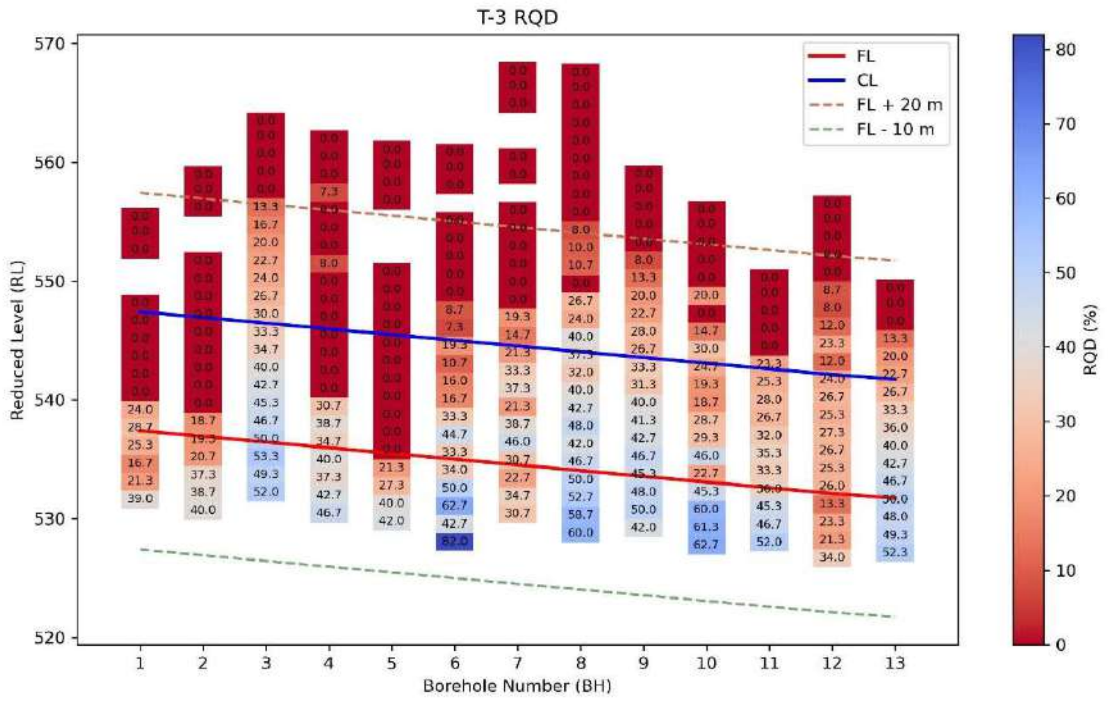
8.4.2.1 Mean And Standard Deviation in Core Recovery considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	26.60364	12.83222
BH02	44231	33.27	18.9262
BH03	44281	58.33	4.911313
BH04	44331	46.84545	11.88528
BH05	44381	32.18	13.92398
BH06	44431	44.83	17.56735
BH07	44481	41.13	7.152554

BH No.	Chainage	Mean	Std
BH08	44531	66.81545	13.36399
BH09	44581	58.265	5.8566
BH10	44631	49.05818	16.34598
BH11	44681	52.132	11.48119
BH12	44731	37.02727	3.299912
BH13	44771	58.963	11.09531

Overall Mean	**Std**
46.30187	16.79552

8.4.3 RQD:



8.4.3.1 Mean And Standard Deviation in RQD considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	14.09	14.52
BH02	44231	15.88	16.92
BH03	44281	44.73	6.97
BH04	44331	24.60	19.92
BH05	44381	11.88	17.36
BH06	44431	37.11	20.98
BH07	44481	31.66	8.13

BH No.	Chainage	Mean	Std
BH08	44531	46.36	8.68
BH09	44581	42.06	6.02
BH10	44631	38.06	17.47
BH11	44681	36.06	9.10
BH12	44731	24.85	4.96
BH13	44771	42.50	8.37

8.4.3.2 Recommended RQD considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Average across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(RQD_d = μ - σ)**	**Reference Standards / Guidelines**
44180 to 44400	21.81	2.46	IS 11315:1985; Deere (1963)
44400 to 44680	39.09	24.78	
44680 to 44800 (P2)	34.15	23.65	

8.5 Petrographic Assessments:

8.5.1 Description of rock Masses

8.5.1.1 Khondalite

Typical khondalite assemblages in the EGMB are described as quartz–feldspar–garnet–sillimanite ± biotite gneisses, with quartz + feldspar together forming ~40–60 %, garnet ~20–30 %, and sillimanite + biotite ~10–20 % of the rock.

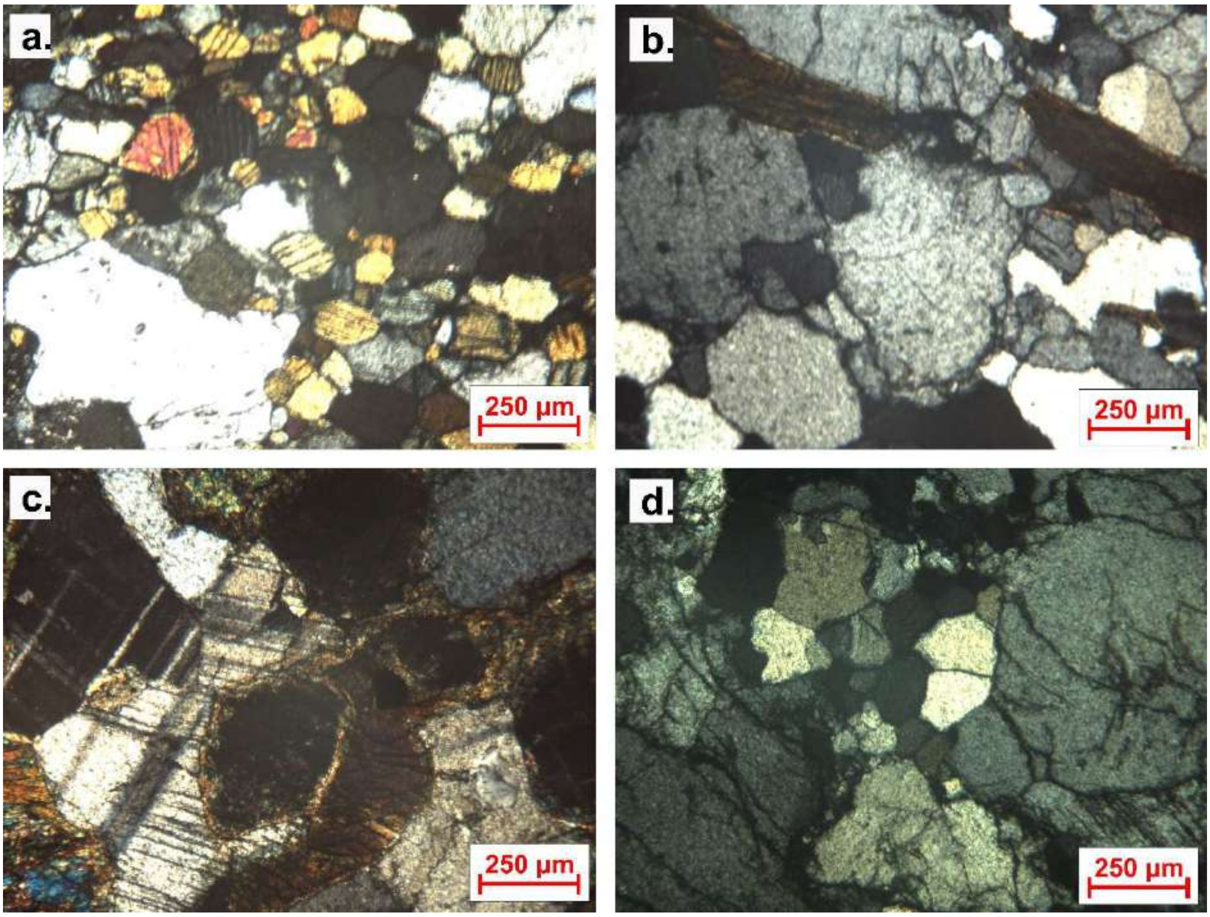
8.5.1.2 Migmatite

Migmatites of the Eastern Ghats consist of quartz- and feldspar-rich leucosome with garnet- and biotite-bearing melanosome/restite. Petrographic and modal descriptions are consistent with dominant quartz + feldspar (~60–70 %) and subordinate garnet and biotite.

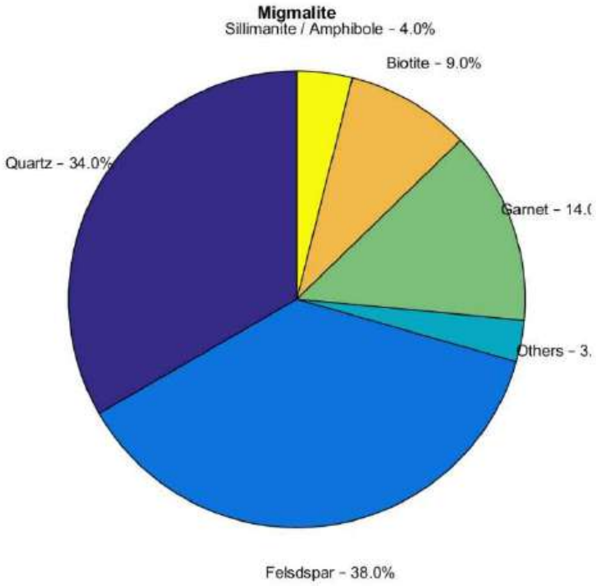
8.5.1.3 Khondalite (Restite)

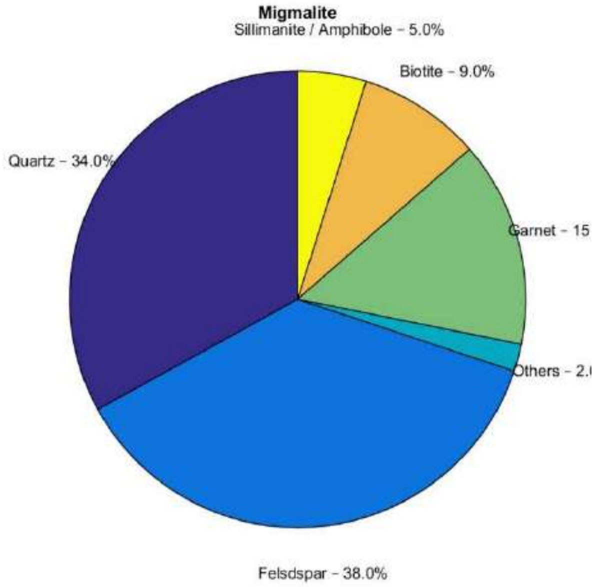
Restites represent refractory residues after partial melting of metapelites, enriched in garnet and ferromagnesian minerals and depleted in felsic phases. Modal characteristics inferred from petrographic descriptions of residual granulites and restitic layers in the EGMB.

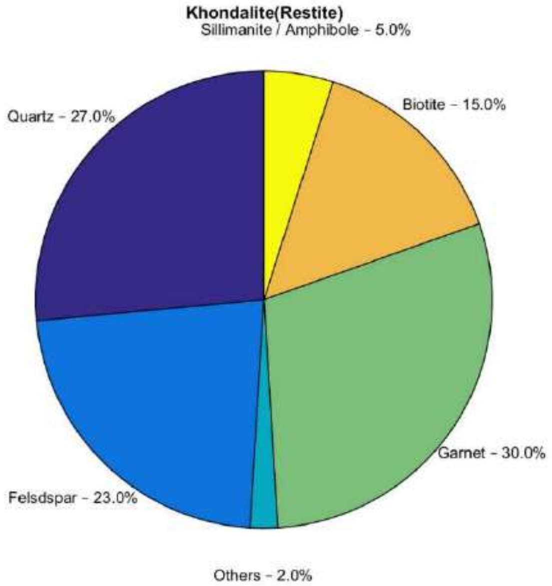
8.5.2 Micro Photographs

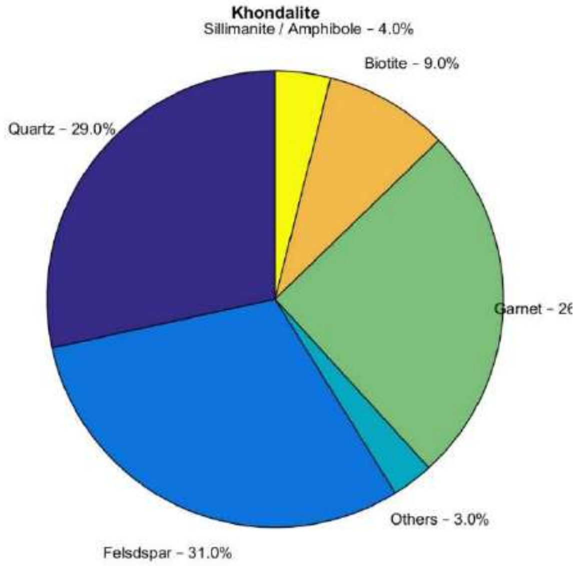


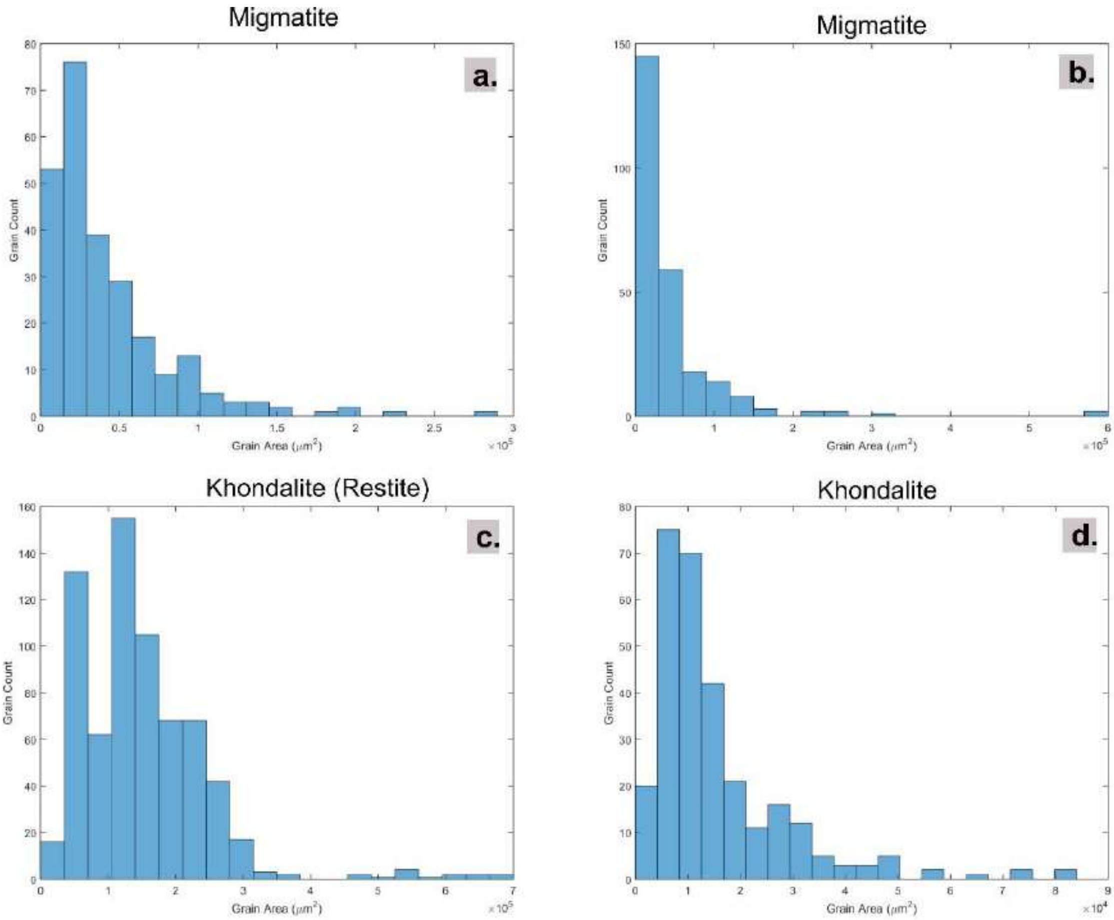
8.5.3 Mineral percentage and grain size distribution



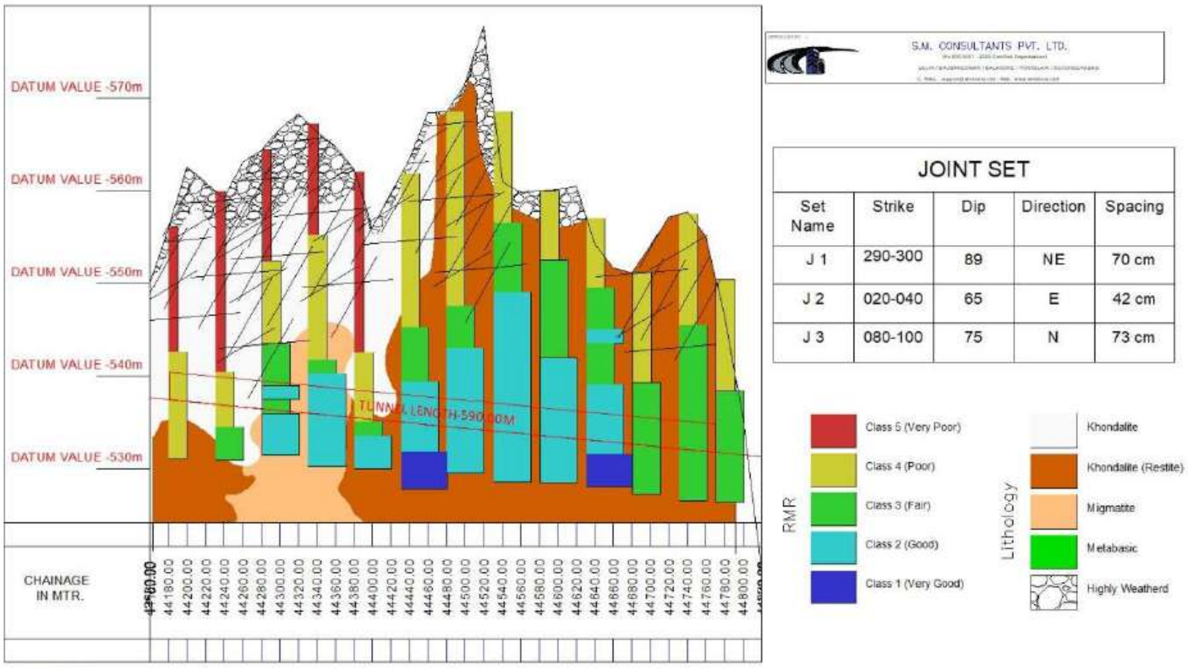







Migmatite
a.
Grain Count
Grain Area (μm^2)
$\times 10^5$
Migmatite
b.
Grain Count
Grain Area (μm^2)
$\times 10^5$
Khondalite (Restite)
c.
Grain Count
Grain Area (μm^2)
$\times 10^5$
Khondalite
d.
Grain Count
Grain Area (μm^2)
$\times 10^4$

8.6 L section:



JOINT SET

Set Name	Strike	Dip	Direction	Spacing
J 1	290-300	89	NE	70 cm
J 2	020-040	65	E	42 cm
J 3	080-100	75	N	73 cm

NOTE:

1. All dimension values are in meter.
2. Some random and discrete joints/fractures, as schematically shown in the L-section, may be expected to encounter in the tunnel influence zone.
3. RMR classification presented here are adopted from Bieniawski (1976) Rock Mass Classification.
4. Thin veneer of soil has been found at someplaces. However, the thickness of soil layer, being out of the scale, are not shown in the L-Section.
5. Joint sets are schematically represented . Hence the joint and local shear/crushed zone spacing are not to the scale.

9 TUNNEL 4: GEOLOGICAL & GEOTECHNICAL ASSESMENT

9.1 Exploratory drillings

As per the requirement of scope of work outlined in the terms of reference, 3 bore holes were drilled with a cumulative length of 84 m at different locations along the proposed alignment. Necessary care has been taken during drilling operations by deploying good quality diamond drill machines to obtain good core recovery to obtain RQD values. The locations of the boreholes were selected in such a way, so that these holes intersect the envisaged ground/ strata conditions at different depths. The location and details of boreholes drilled; total depth of drillings is shown in table below.

Chainage	BH	GL	FL	Depth
45460	BH-1	544.626	525.332	24.00
45510	BH-2	558.654	524.932	39.00
45560	BH-3	541.234	524.532	21.00

9.2 TUNNEL-4 SRT

9.2.1 Location:

Sr. No.	Chainage	Line	Spread	Location (T-04)		Length
				Start	End	In meter
1	45.460km to 45.560km	L1	S1	45.460km	45.560km	100

9.2.2 Seismic survey results and conclusion

Table 9.1: Summery of Tunnel-1 SRT test

Variation of maximum range of thicknesses below EGL (M)			Avg. V_p (m/sec)	Calculated V_s (m/sec)	Dynamic Young's Modulus (MPa)	Shear Modulus (MPa)
Layer	From	To				
Layer-I	0.50	1.50	700	327	466	172
Layer-II	1.50	4.50	2600	1338	11338	4295
Layer-III	4.50	25.00	3800	2101	29368	11472

<u>Sample Calculation:</u>

The Young's Modulus E is the uni-axial stress-strain ratio. Its dynamic value is expressed by the following equation:

$$E = \frac{\rho V_P^2 (1+\mu)(1-2\mu)}{1-\mu}$$

Where, E = Dynamic Young's Modulus in kN/m2

V_p = 700 m/sec

ρ = 1.6 gm/cc ≈ 1.60 kN.s2/m4 (mass density)

μ = 0.36

So, calculated E = 466480 kN/sqm ≈ 466 MPa

The Shear Modulus G is the stress-strain ratio for simple shear. Its dynamic value is obtained by the following:

$$G = \frac{E}{2(1+\mu)} = \rho V_s^2$$

So, Shear Modulus G comes out to be 171500 kN/sqm ≈ 172 MPa

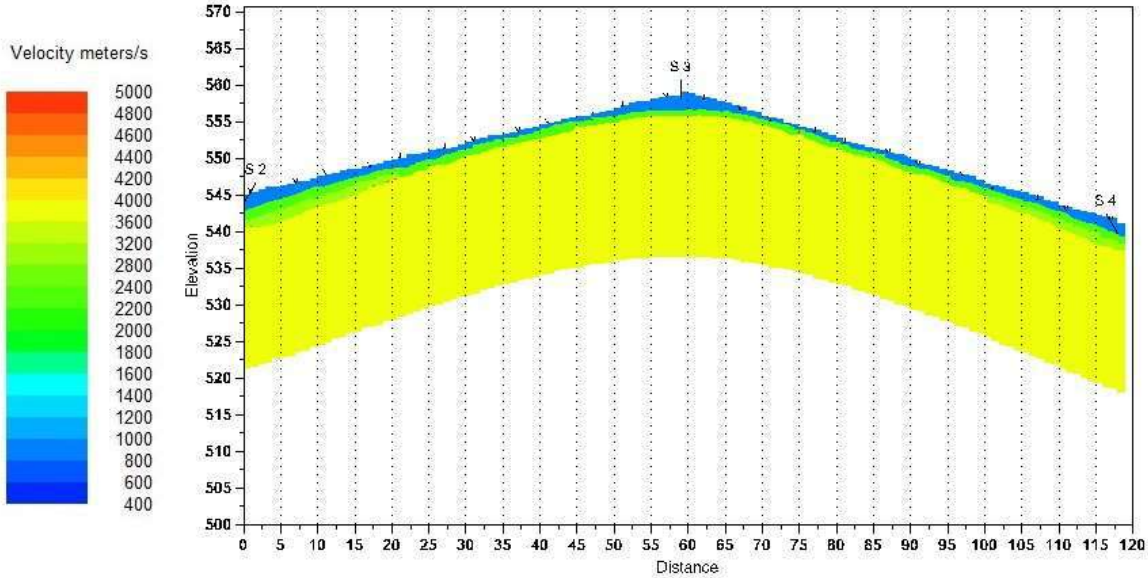
Again,

$G = \rho V_S^2$ giving Vs = √(G/ρ)

So, calculated Vs = 327.395 m/sec, say 327 m/sec

SEISMIC PROFILE(T04)

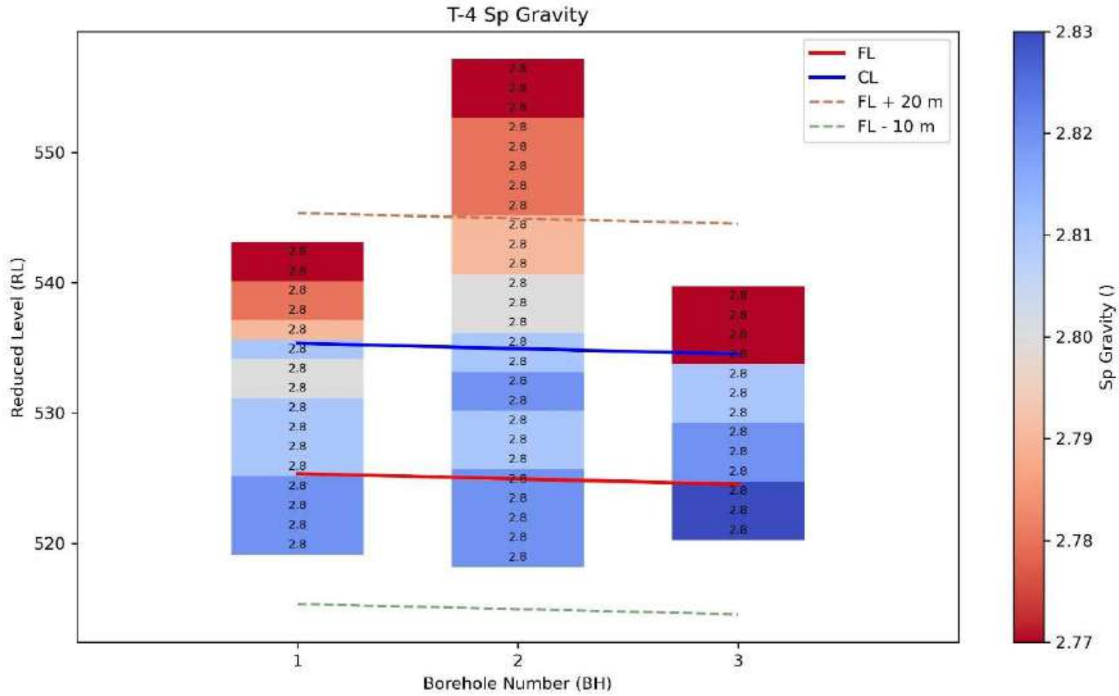
T4L1S1



Shot point depth computation

9.3 Assessment of the engineering properties of rock sample:

9.3.1 Specific Gravity



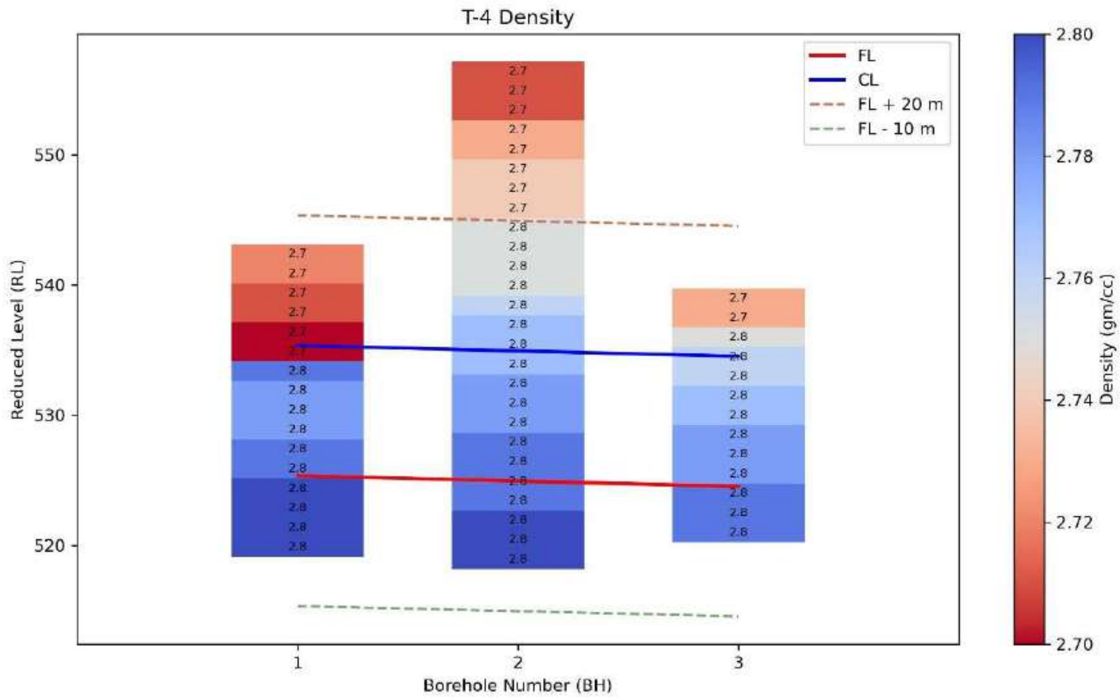
9.3.1.1 Mean and Standard deviation Specific Gravity considering 1D zone of influence of each borehole:

BH No.	Chainage	Mean	Std
BH01	44181	2.8	0.01
BH02	44231	2.8	0.01
BH03	44281	2.8	0.02

9.3.1.2 Recommended Specific Gravity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Sp = μ - σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
45460 to **45510**	2.8	IS 13030:1991; IS 1124:1974

9.3.2 Dry Density



9.3.2.1 Mean and Standard Deviation in Density considering 1D zone of influence of each borehole:

BH No.	Chainage	Mean	Std
BH01	44181	2.78	0.03
BH02	44231	2.79	0.01
BH03	44281	2.78	0.01

9.3.2.2 Recommended Density considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(d = μ - σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
45460 to **45510**	2.8	IS 13063:1991

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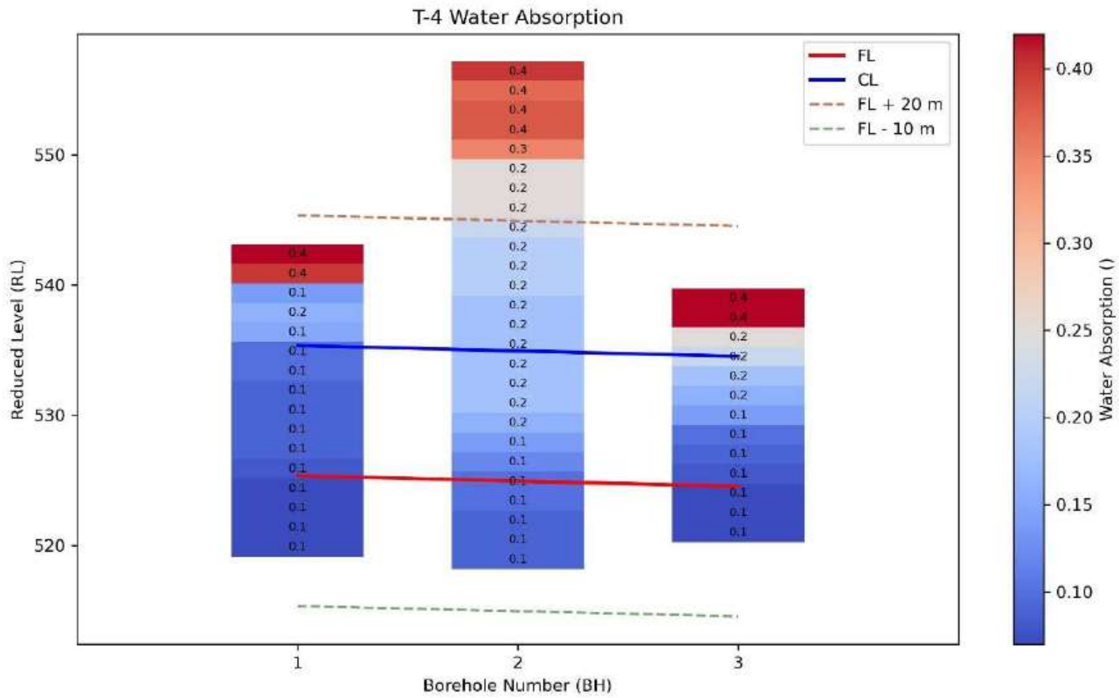
9.3.3 Water absorption Test



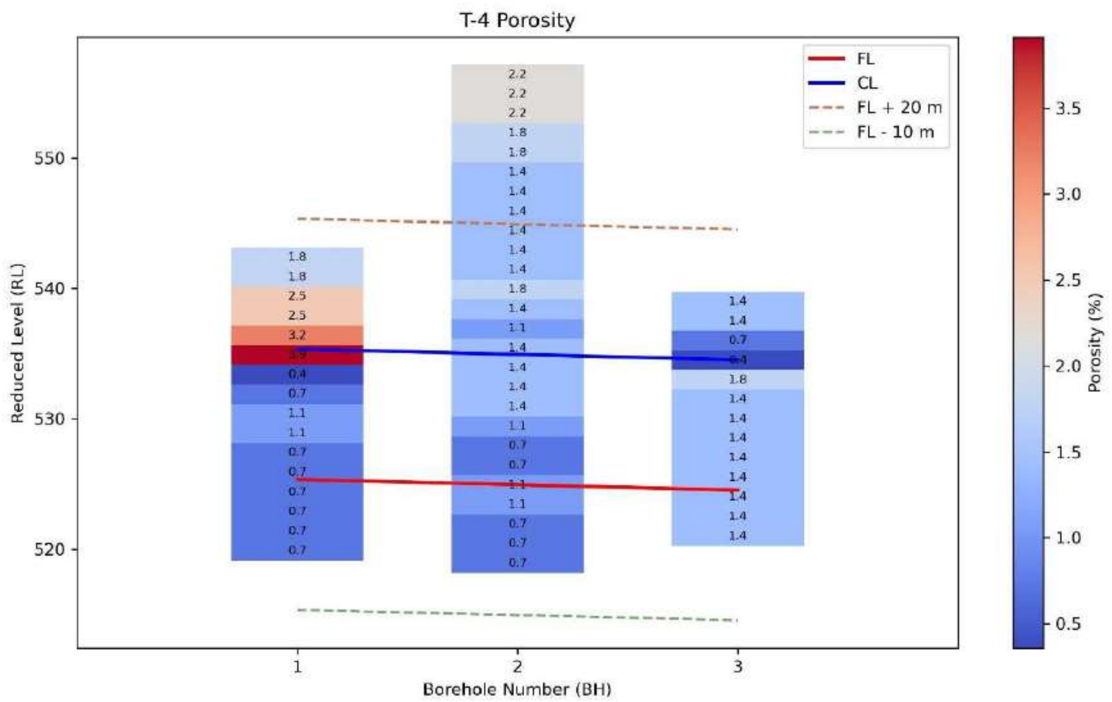
9.3.3.1 Mean and Standard Deviation in Water Absorption Value considering 1D zone of influence:

BH No.	Chainage	Mean	Std
BH01	44181	0.09	0.012
BH02	44231	0.13	0.039
BH03	44281	0.12	0.054

9.3.3.2 Recommended Water Absorption Value considering 1D zone of influence:

Chainage	Statistical / Reduction Method (A= μ + σ across boreholes within ±10 m of FL and CL)	Reference Standards / Guidelines
45460 to **45510**	0.15	IS 13063:1991; IS 2386 (Part 3):1963

9.3.4 Porosity



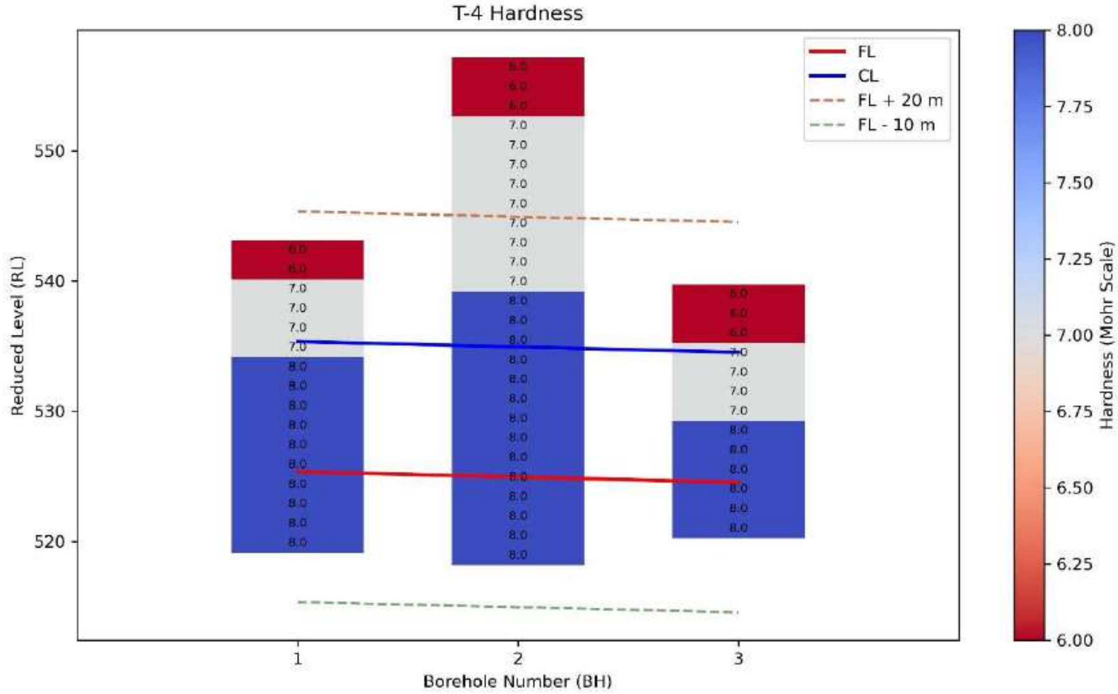
9.3.4.1 Mean And Standard Deviation in Porosity considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	1.067	1.021
BH02	44231	1.030	0.311
BH03	44281	1.348	0.365

9.3.4.2 Recommended Porosity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(n = μ ± σ across boreholes within ±10 m of FL and CL)**	**Reference Standards / Guidelines**
45460 to **45510**	1.14 ± 0.6	IS 1124:1974; ISRM (1981)

9.3.5 Hardness



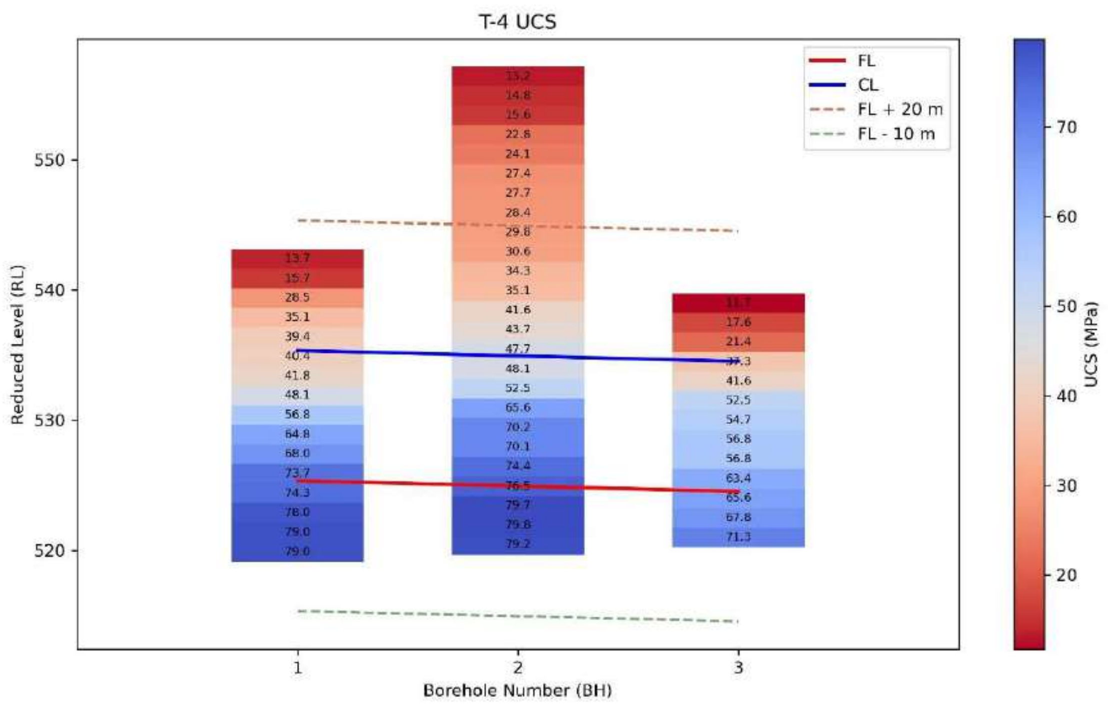
9.3.5.1 Mean And Standard Deviation in Hardness considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	8	0
BH02	44231	8	0
BH03	44281	8	1

9.3.5.2 Recommended Hardness considering 1D zone of influence:

Chainage	Statistical / Reduction Method (Hi = μ - σ across boreholes within ±10 m of FL and CL)	Reference Standards / Guidelines
45460 to **45510**	8	IS 13311 (Part 2):1992; ISRM 1978

9.3.6 Compression Test



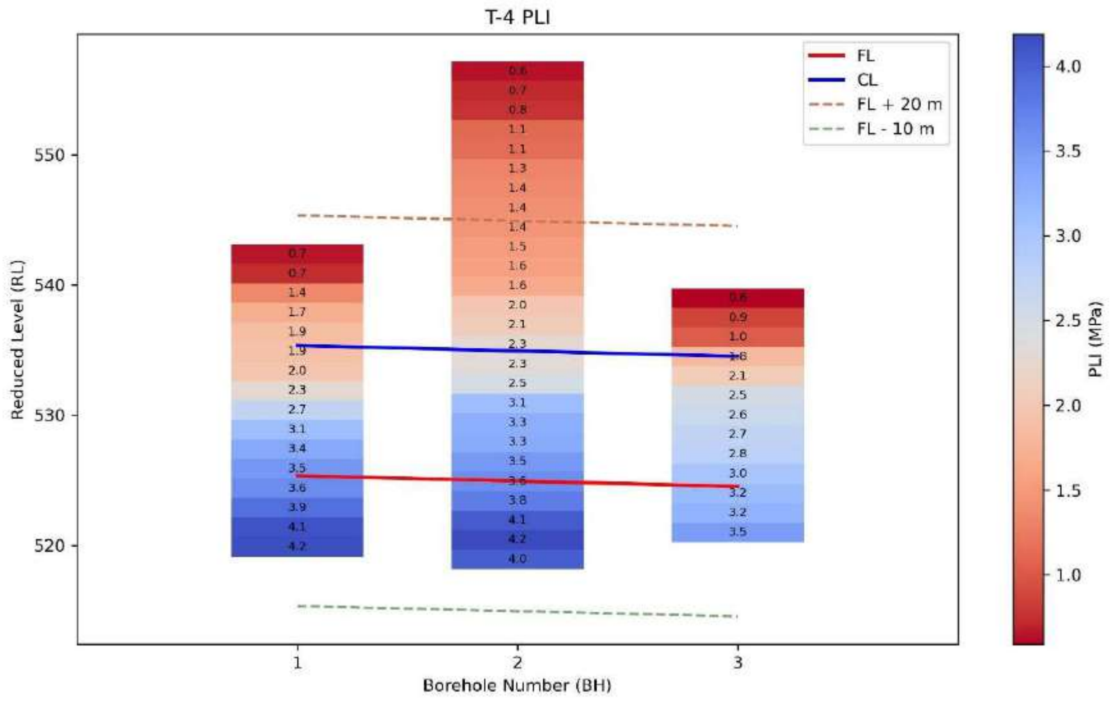
9.3.6.1 Mean And Standard Deviation in UCS considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	63	11
BH02	44231	61	7
BH03	44281	52	15

9.3.6.2 Recommended UCS considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(UCS_k = μ - σ) across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(UCS_d = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)**	**Reference Standards / Guidelines**
45460 to 45510	49.80	14.94	IS 9143:1979; IS 13365 (Part 2):1998; Hoek & Brown (2002, 2019)

9.3.7 Point Load Test



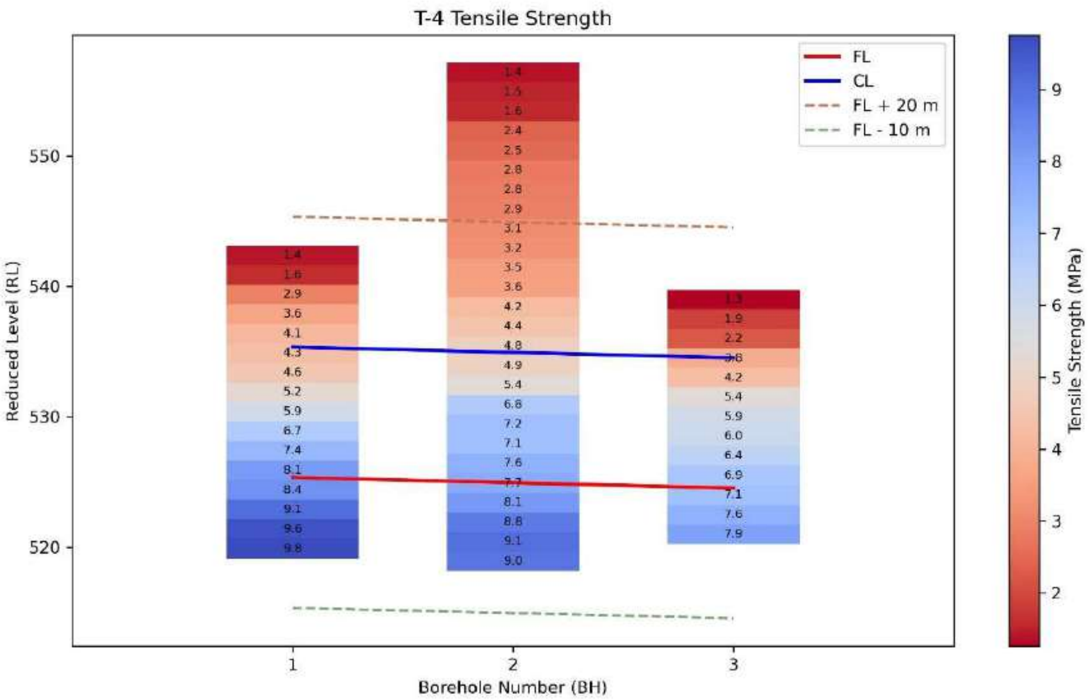
9.3.7.1 Mean And Standard Deviation in PLI considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	3.05	0.795
BH02	44231	3.37	0.611
BH03	44281	2.74	0.517

9.3.7.2 Recommended PLI considering 1D zone of influence:

Chainage	Statistical / Reduction Method (UCS_k = μ - σ) across boreholes within ±10 m of FL and CL)	Recommended Design Value (UCS_d = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)	Reference Standards / Guidelines
45460 to 45510	2.37	0.71	IS 9143:1979; IS 13365 (Part 2):1998; Hoek & Brown (2002, 2019)

9.3.8 Brazilian Test



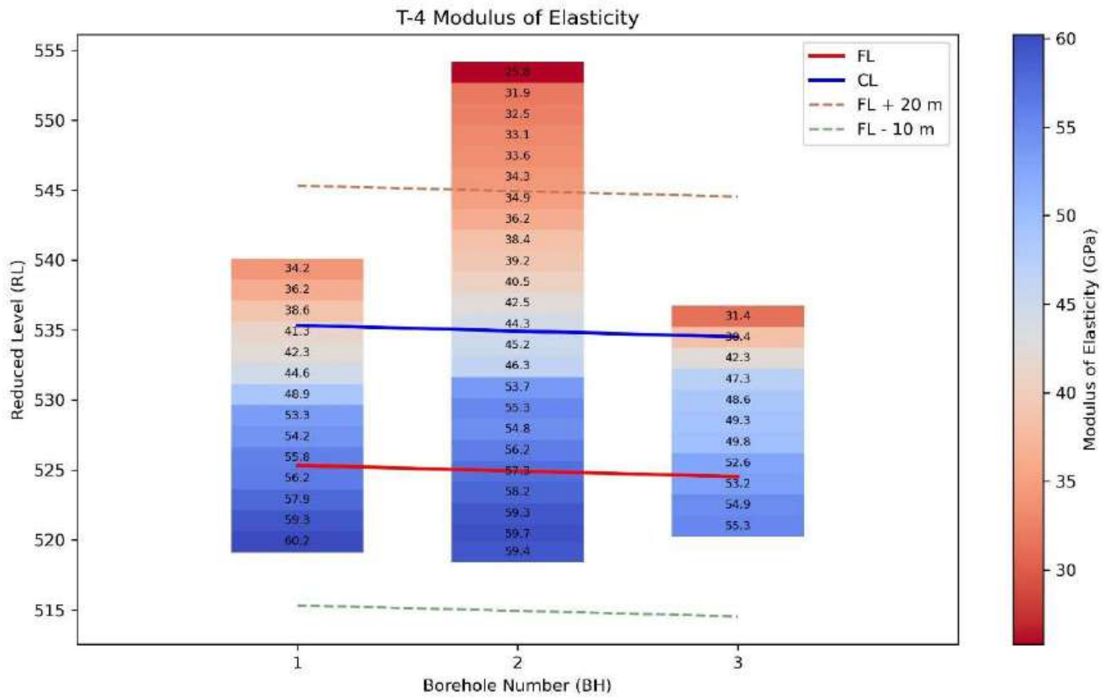
9.3.8.1 Mean And Standard Deviation in Tensile Strength considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	6.93	1.88
BH02	44231	7.27	1.34
BH03	44281	6.13	1.35

9.3.8.2 Recommended Tensile Strength considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(mean value across boreholes within ±10 m of FL and CL× 0.8 (account for anisotropy))**	**Recommended Design Value** **(μ - σ)**	**Reference Standards / Guidelines**
45460 to **45510**	6.77	5.21	IS 10082:1982; Hoek & Brown (1997); ISRM Suggested Methods

9.3.9 Modulus of elasticity test



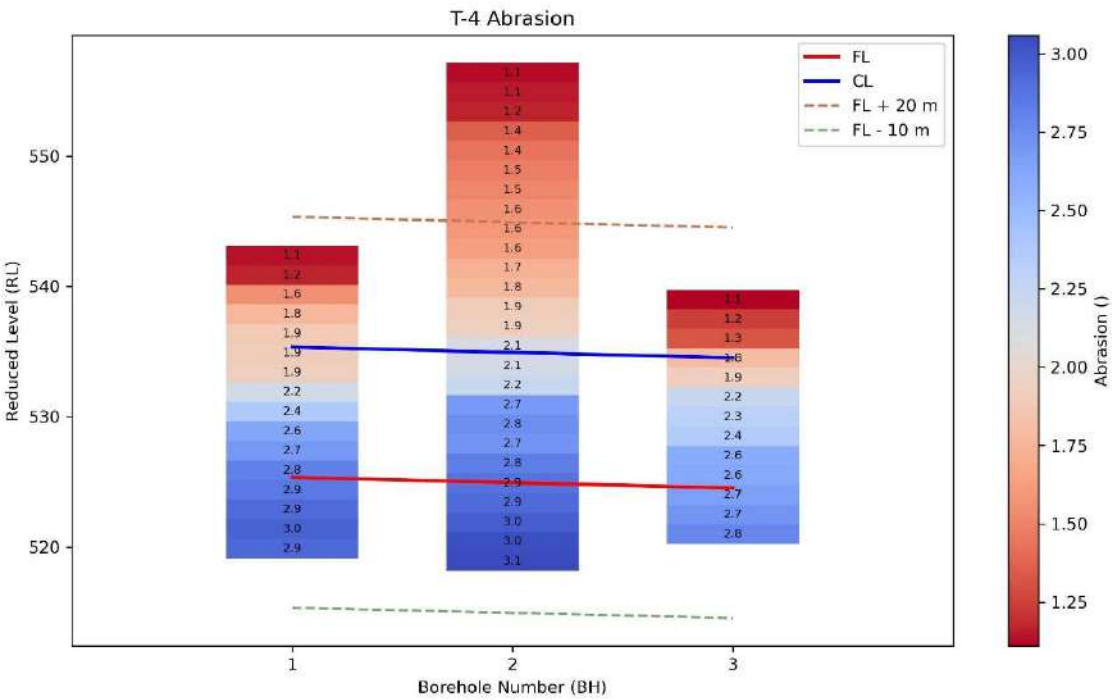
9.3.9.1 Mean And Standard Deviation in Elasticity considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	51.38	6.63
BH02	44231	54.60	5.05
BH03	44281	49.17	5.43

9.3.9.2 Recommended Modulus of Elasticity considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(Ei = μ - σ across boreholes within ±10 m of FL and CL× 0.8 (account for anisotropy))**	**Recommended Design Value** **(Ed = (μ - σ) × f(RMR)), where (f(RMR) = 0.1–0.7)**	**Reference Standards / Guidelines**
45460 to **45510**	45.72	13.71	IS 13365 (Part 2):1998; Hoek & Diederichs (2006)

9.3.10 Abrasion test



9.3.10.1 Mean And Standard Deviation in Abrasion value considering 1D zone of influence in each Borehole:

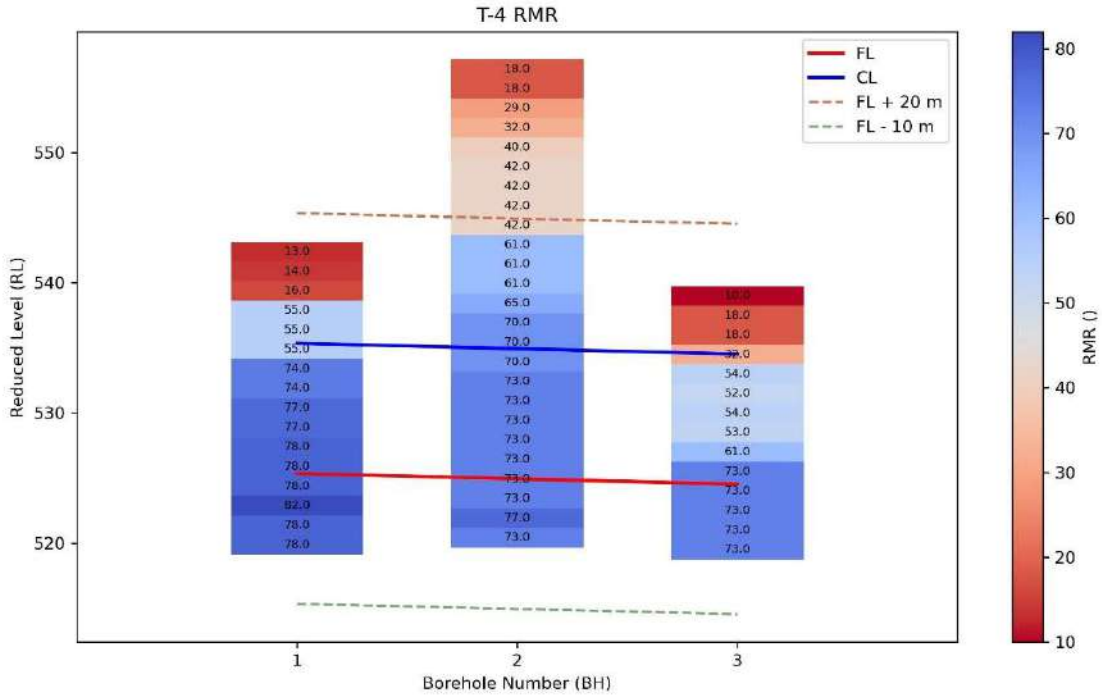
BH No.	Chainage	Mean	Std
BH01	44181	2.52	0.40
BH02	44231	2.72	0.30
BH03	44281	2.40	0.34

9.3.10.2 Recommended Abrasion Value considering 1D zone of influence:

Chainage	**Statistical / Reduction Method** **(A= μ + σ across boreholes within ±10 m of FL and CL)**	**Recommended Design Value** **(A_v < 25%): Slightly abrasive; 25–35%: Moderately abrasive; >35%: Highly abrasive. (CAI = 0.5–6).**	**Reference Standards / Guidelines**
45460 to **45510**	2.9	Moderately abrasive	IS 2386 (Part 4):1963; ISRM (2007); CERCHAR (1986)

9.4 Geological assessment:

9.4.1 RMR:



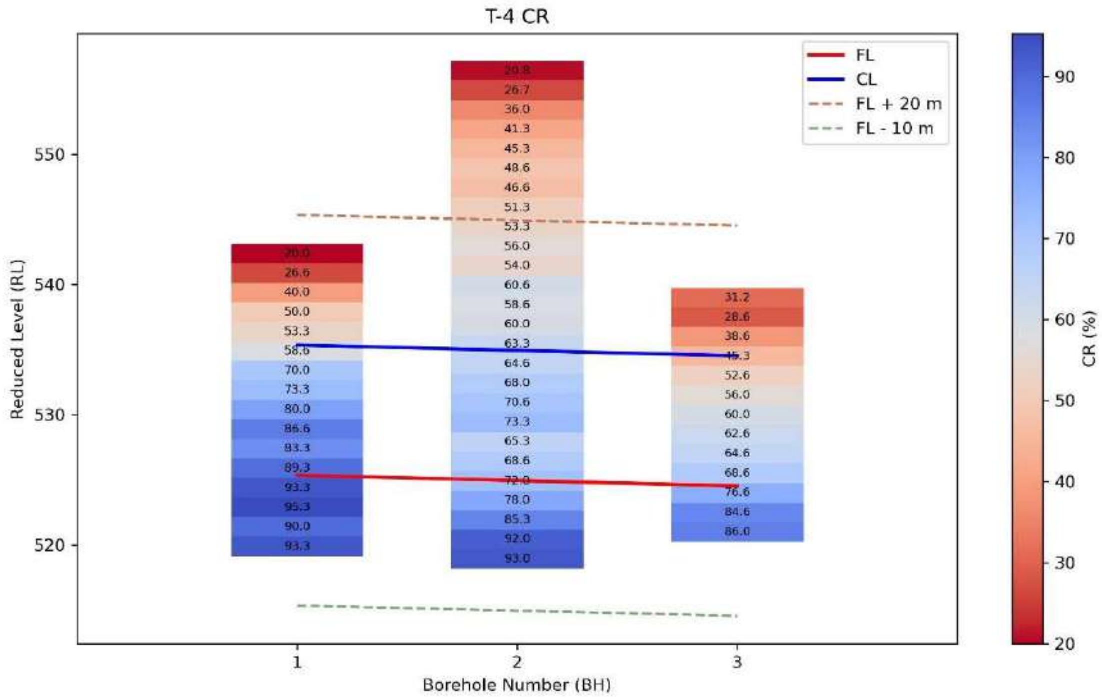
9.4.1.1 Mean And Standard Deviation in RMR considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	75	7
BH02	44231	73	2
BH03	44281	61	13

9.4.1.2 Recommended RQD considering 1D zone of influence:

Chainage	Statistical / Reduction Method (Average across boreholes within ±10 m of FL and CL)	Recommended Design Value (RMR_d = μ - σ)	Reference Standards / Guidelines
45460 to **45510**	70 (Class II)	59 (Class III)	IS 13365 (Part 2): 1998, Cl. 5.1 + Note on "representative values"

9.4.2 Core Recovery:

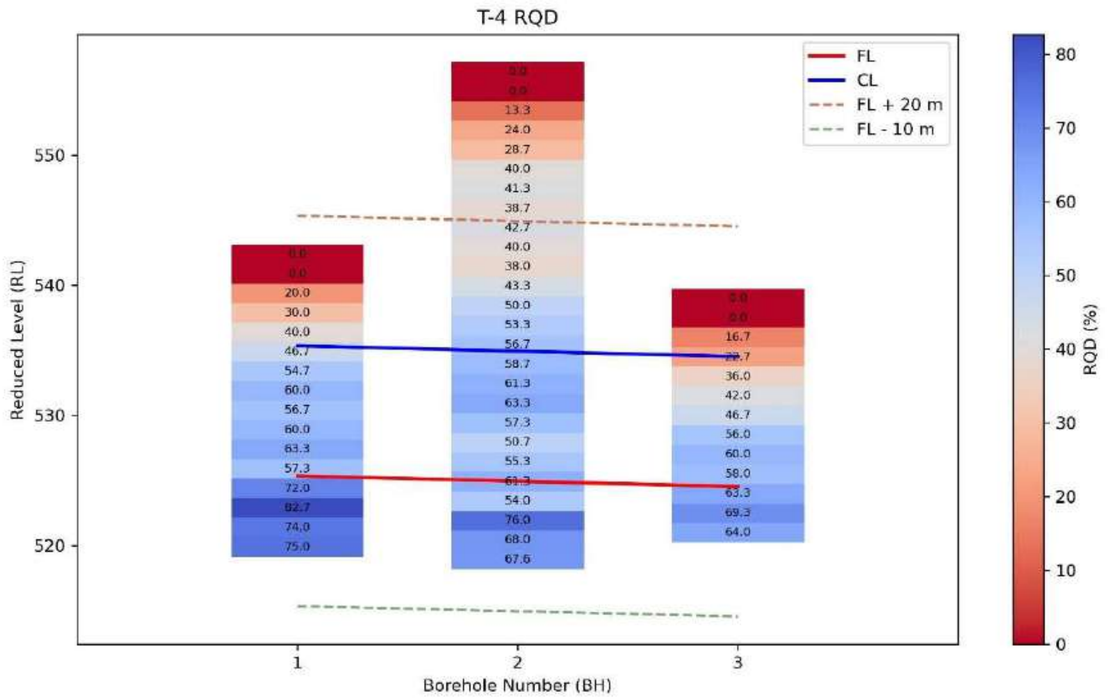


9.4.2.1 Mean And Standard Deviation in Core Recovery considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	81.97	11.62354
BH02	44231	73.77	8.897072
BH03	44281	65.69	13.4234

Overall Mean	**Std**
73.81	12.96585

9.4.3 RQD:



9.4.3.1 Mean And Standard Deviation in RQD considering 1D zone of influence in each Borehole:

BH No.	Chainage	Mean	Std
BH01	44181	62.73	10.62
BH02	44231	60.60	7.34
BH03	44281	51.80	14.66

9.4.3.2 Recommended RQD considering 1D zone of influence:

Chainage	Statistical / Reduction Method (Average across boreholes within ±10 m of FL and CL)	Recommended Design Value (RQD_d = μ - σ)	Reference Standards / Guidelines
45460 to 45510	58.37	46.48	IS 11315:1985; Deere (1963)

9.5 Petrographic Assessments:

9.5.1 Description of rock Masses

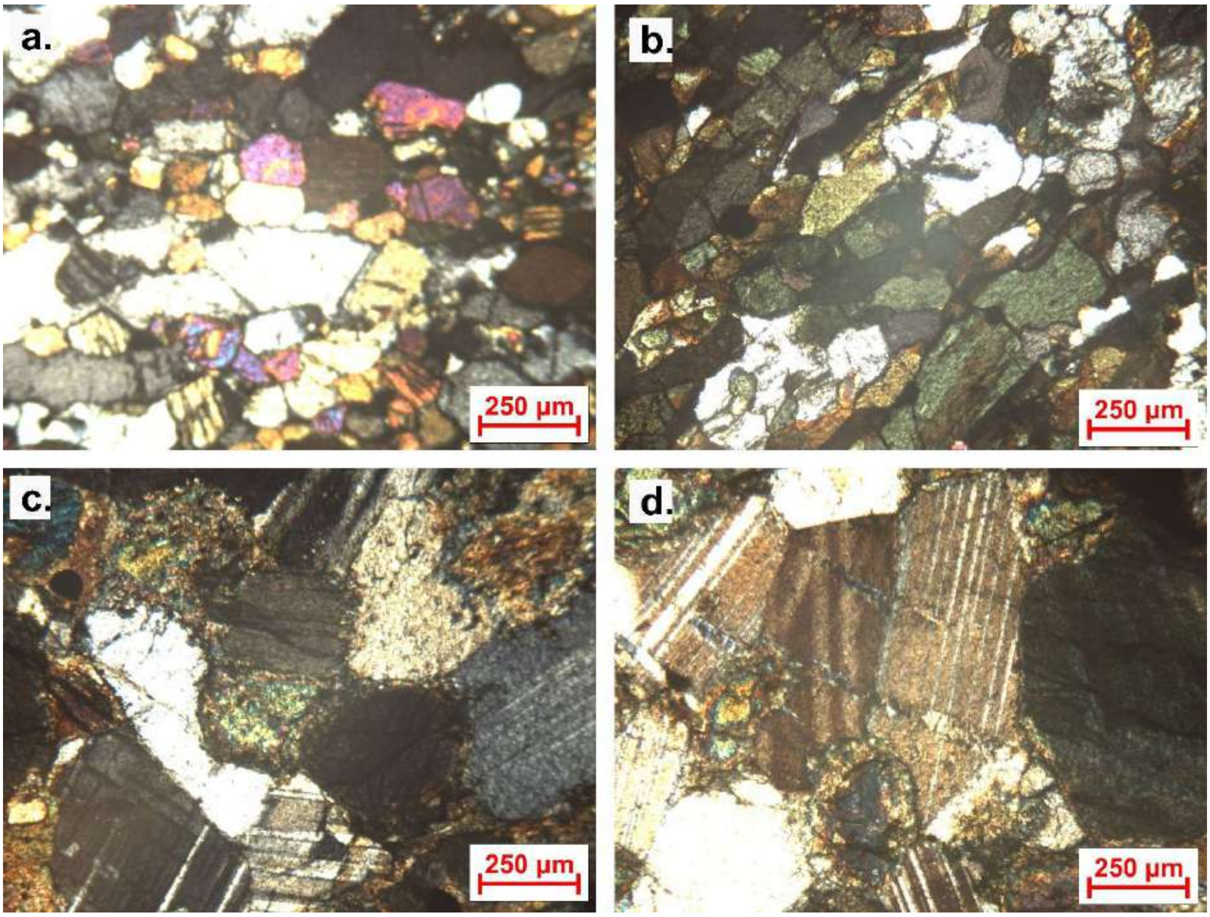
9.5.1.1 Khondalite

Typical khondalite assemblages in the EGMB are described as quartz–feldspar–garnet–sillimanite ± biotite gneisses, with quartz + feldspar together forming ~40–60 %, garnet ~20–30 %, and sillimanite + biotite ~10–20 % of the rock.

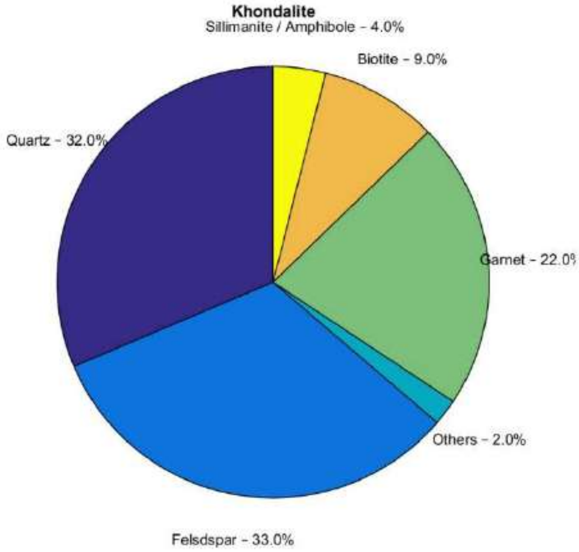
9.5.1.2 Khondalite (Restite)

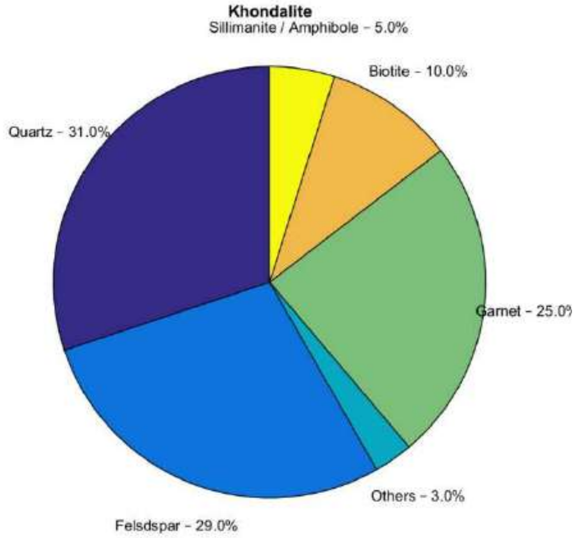
Restites represent refractory residues after partial melting of metapelites, enriched in garnet and ferromagnesian minerals and depleted in felsic phases. Modal characteristics inferred from petrographic descriptions of residual granulites and restitic layers in the EGMB.

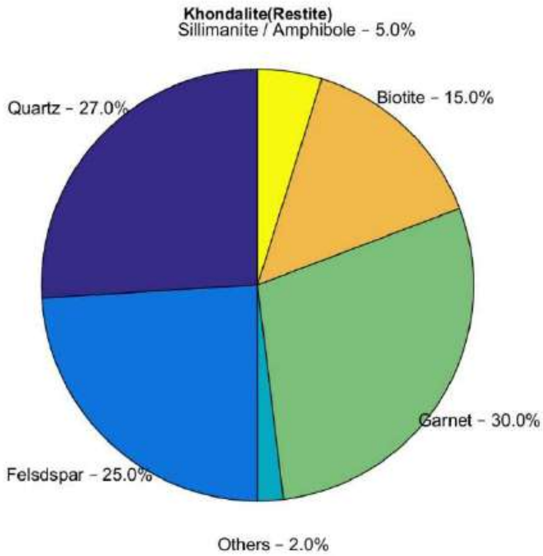
9.5.2 Micro Photographs

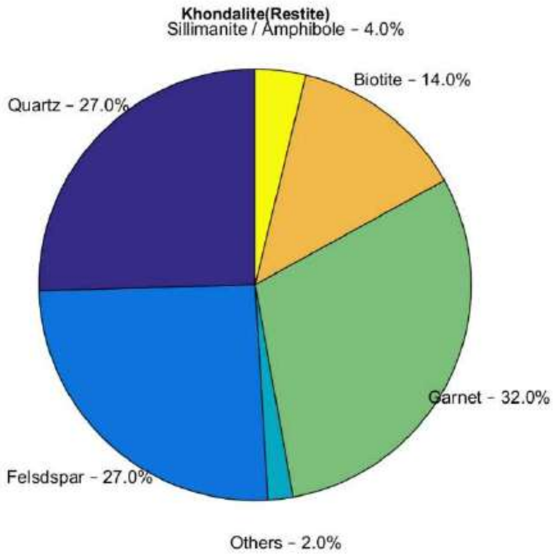


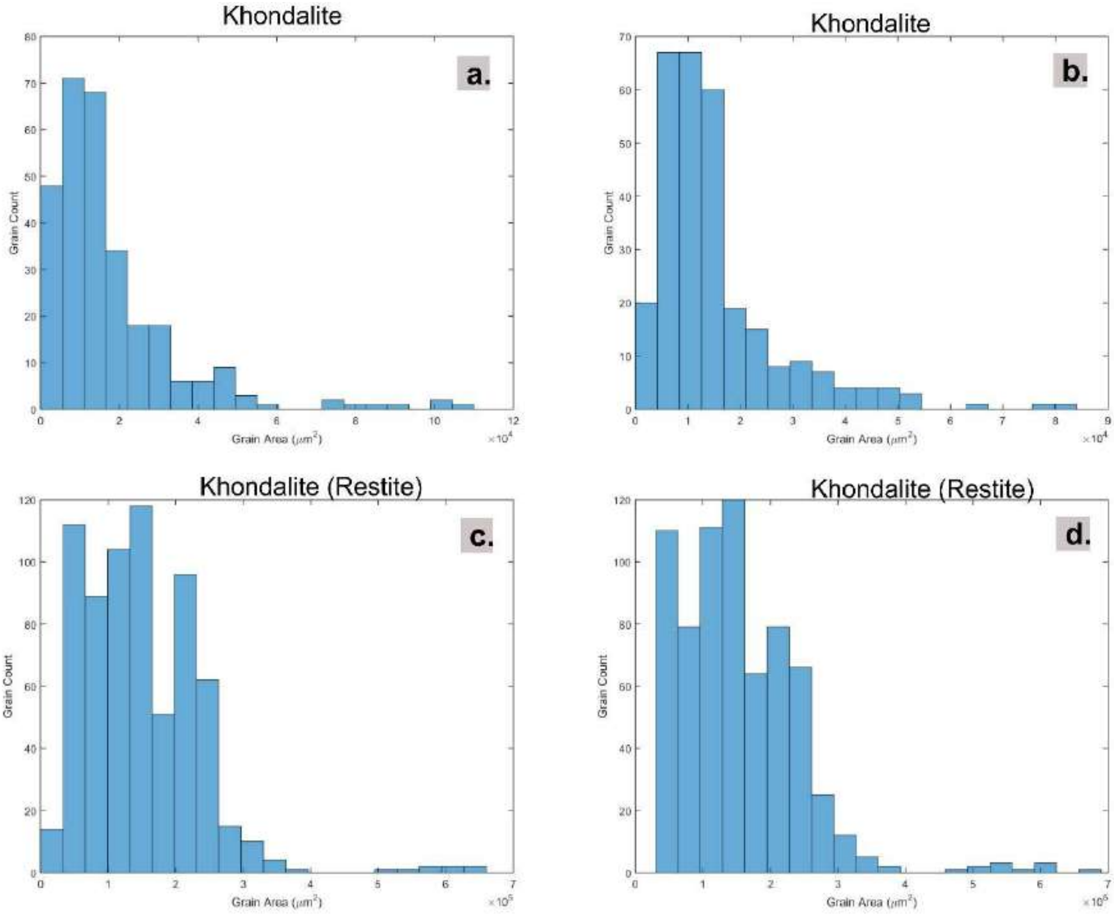
9.5.3 Mineral percentage and grain size distribution



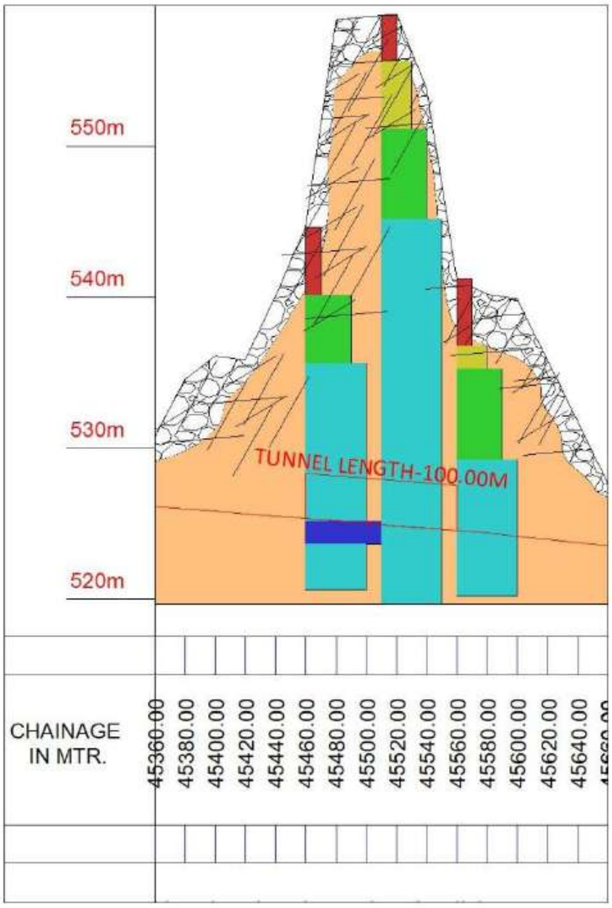







Khondalite
a.
Grain Count
Grain Area (μm²)
×10⁴
Khondalite
b.
Grain Count
Grain Area (μm²)
×10⁴
Khondalite (Restite)
c.
Grain Count
Grain Area (μm²)
×10⁵
Khondalite (Restite)
d.
Grain Count
Grain Area (μm²)
×10⁵

9.6 L section:



NOTE:

1. All dimension values are in meter.
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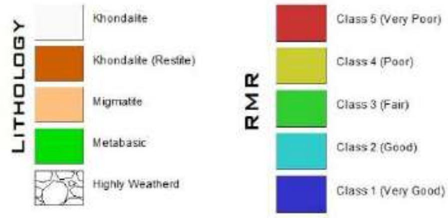
3. RMR classification presented here are adopted from Bieniawski (1976) Rock Mass Classification.
4. Thin veneer of soil has been found at someplaces. However, the thickness of soil layer, being out of the scale, are not shown in the L-Section.
5. Joint sets are schematically represented . Hence the joint and local shear/crushed zone spacing are not to the scale.



JOINT SET

Set Name	Strike	Dip	Direction	Spacing
J 1	290-300	89	NE	70 cm
J 2	020-040	65	E	42 cm
J 3	080-100	75	N	73 cm

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