ROCK MASS ENGINEERING OF THE PROPOSED BASARA DAM SITE, SULAIMANI, KURDISTAN REGION, NE-IRAQ

A THESIS
SUBMITTED TO THE COUNCIL OF THE COLLEGE OF SCIENCE, UNIVERSITY OF SULAIMANI / SULAIMANI – IRAQ, IN PARTIAL FULFILMENT OF THE REQUIREMENTS FOR THE DEGREE OF DOCTOR OF PHILOSOPHY IN GEOLOGY

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بسم الله الرحمن الرحيم

أَلَمْ تَرَ أَنَّ اللَّهَ أَنزَلَ مِنَ السَّمَاء مَاءً فَأَخْرَجْنَا بِـــهِ
ثَمَرَاتٍ مُّخْتَلِفًا أَلْوَانُهَا وَمِنَ الْجِبَالِ جُدَدٌ بِـــيضٌ
وَحُمْرٌ مُّخْتَلِفٌ أَلْوَانُهَا وَغَرَابِيبُ سُودٌ .

سورة فاطر

الاية {٢٧}

Seest thou not that Allah sends down rain from the sky? with it we then bring out produce of various colors and in the mountains are tracts white and red, of various shades of color, and black intense in hue. (27)

Supervisor's Certification

I certify that this thesis was prepared under my supervision at the ***University of Sulaimani, College of Science, Department of Geology*** as a partial requirement for the degree of Doctor of Philosophy in ***Geology (Engineering Geology).***

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Linguistic Evaluation Certification

I hereby certify that this thesis has been checked by me and after indicating all the grammatical and spelling mistakes; the thesis was given again to the candidate to make the adequate corrections. After the second reading, I found that the candidate corrected the indicated mistakes. Therefore, I certify that this thesis is free from mistakes.

Name: Bekhal Latif Muheedeen

Signature

Date: 3 / 10 / 2009

Sulaimani University/College of Languages/ English Department

Certification of the Examining Committee

We certify that we have read this thesis and as examining committee examined the student in its contents and whatever relevant to it and that in our opinion it is adequate thesis for the degree of Doctor of Philosophy in **Geology (Engineering Geology).**

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Dedicated to :

*The memory of my father and my mother ;

*My lovely wife;

*My darling Daughter ;

*Those who gave aid to me.

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Abstract

A rock mass engineering of the proposed Basara dam site, near Delaizha village – Sulaimani district – Kurdistan region – NE Iraq is accomplished, where Kolosh, Sinjar and Gercus Formations are exposed. Most parts of the dam reservoir are located within a synclinal structure (New Sola – Qazanqaya syncline).

A geological map of the study area is prepared for the first time on a scale of 1:20000.

This study consists of three parts: field, laboratory and office works. The field work included collecting data from three surface sections and three boreholes at Basara gorge, in which the rock masses were divided into 30 rock mass units (16 units in the surface sections and 14 units in the boreholes).

Laboratory tests show that the unconfined compressive strength ($б_{ci}$ of 50mm sample diameter) of carbonate rocks range between 40.14 - 92.26 MPa, these rocks are moderately strong – strong rocks present at the right side of Basara gorge and carbonate rocks in the boreholes at the left side of Basara gorge. The $б_{ci(50)}$ of clastic rocks range between 5 – 38.05MPa, these rocks are very weak – moderately strong rocks in the boreholes excavated at the left side of Basara gorge.

This study proposes a new Geological Strength Index chart, based on quantitative analysis of the rock mass structure (through volumetric joint count "Jv" or block volume "Vb") and surface conditions of discontinuities. The Geological Strength Index "GSI" values of strong carbonate rocks of dam foundation rocks were determined by this new chart which range between 55 - 81, the comparison of GSI value from this chart with $RMR_{(1976)}$ value, illustrated the high precision of the chart.

The GSI values of clastic Gercus and Kolosh Formations were determined by Molasse and Flysch charts respectively, in which of Molasse Gercus Formation is 33 and of Flysch Kolosh Formation range between 30 – 57.

The mechanical properties of all rock mass units were determined by Hoek – Brown failure criterion, using RocLab programme.

All rock mass units are evaluated by the DMR system for different aspects. This evaluation shows that the carbonate rocks at the right and those in the boreholes at the left side of Basara gorge: (1) have no problems resulted from E_c / E_m

(deformation modulus of the dam / deformation modulus of the foundation rocks), (2) they are desirable for excavation, (3) they need no or spot grouting and (4) they have no deformability problems (except one rock mass unit), while the rock mass units of clastic Kolosh Formation in the boreholes at the left side of Basara gorge: (1) have some important problems that can't be neglected especially if the dam type is Conventional Vibrated Concrete (CVC – Gravity), (2) they are less desirable for gravity dam, (3) they need systematic grouting especially in the case of gravity dam and (4) they have serious deformability problems especially in the case of gravity dam construction.

Rock mass classification systems and Hoek – Brown failure criterion results reveal that the limestone rock mass units of Sinjar Formation at the right side and of Kolosh Formation in the boreholes at the left side of Basara gorge are characterized by high values of RMR, DMR & GSI and better mechanical properties, but the rock mass units of clastic Kolosh Formation at the left side and Gercus Formation at the right side of Basara gorge are characterized by low – intermediate values of RMR, DMR and GSI and worse mechanical properties.

This study proposes a new model which is called rock mass – valley section model in this thesis and applied here for the three profiles (a-b, c-d & e-f profiles) which are drawn for the first time, in which the rock mass units are projected into each section.

Comparisons among these profiles for choosing the optimum one revealed that the e-f profile is better than c-d profile in most aspects and c-d profile is better than a-b profile, but the final comparison between e-f and c-d profiles revealed that the c-d profile is more suitable than e-f profile, due to the presence of a weak sheared zone in the right side of e-f profile which has a great negative effect on the stability of the dam after construction and filling the reservoir and causes instability which increases with time.

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Chapter One
Introduction

1-1 Preface

Dam is one of the most important major structures which supplies services in several aspects, such as irrigation, electric power, tourism, etc. It is also one of those projects that retends water and uses it for various purposes. Darband – Basara gorge is, therefore, selected for this purpose which lies on Basara stream near Delaizha village, Fig(1-1).

The selection of the mentioned location is due to suitability of the topography, absence of any big water project on the Basara stream and the catchment's area is within the Kurdistan region, this means that there is no any danger upon the project via neighboring countries in the future. It gives also a great benefit in irrigating thousands of hectares of agricultural area of Garmian – district which is characterized by the lack of water in the summer season.

In order to assess the suitability of the location of the proposed dam, three surface sections and three boreholes along Basara stream were selected, Fig(1-2) and the rock masses at each site were evaluated according to the most new and updated rock mass classification systems.

1-2 Location of the study area

The study area is located at Sulaimani district, Kurdistan region-NE Iraq, about 25 km to the southwest of Sulaimani city and lies between latitudes 35° 25´ 37˝ & 35° 31´ 02˝ N and longitudes 45° 09´ 10˝ & 45° 16´ 04˝ E, Fig(1-2).

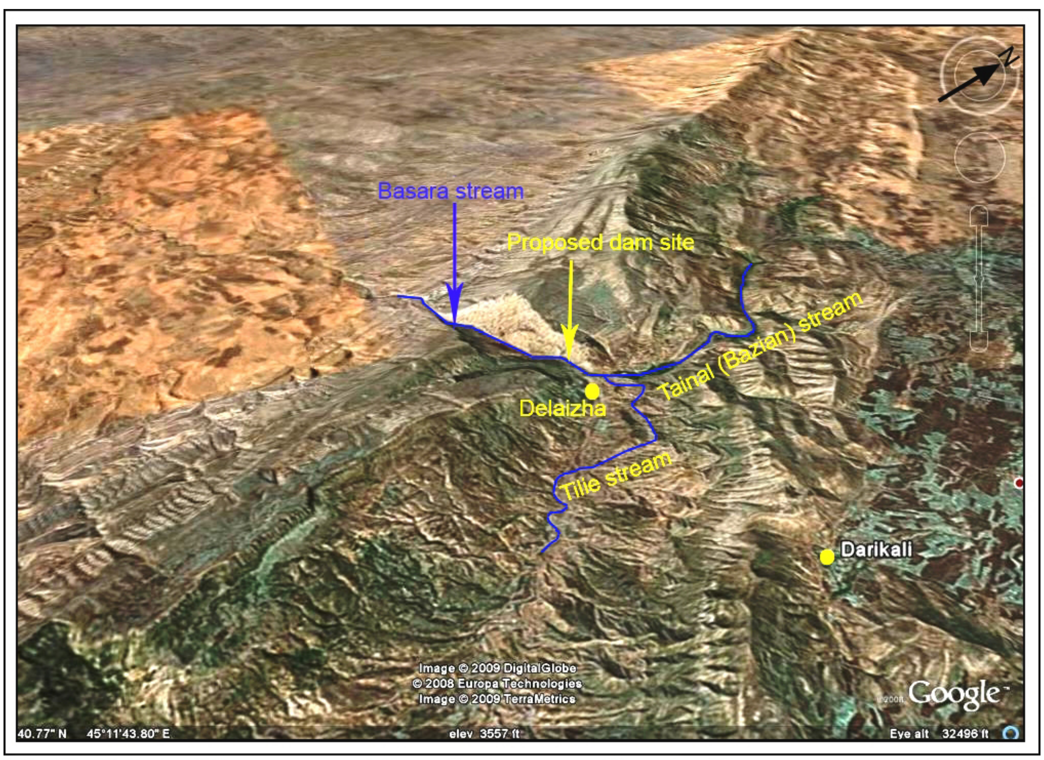
1-3 Previous studies

There are two engineering studies about the proposed Basara dam site, one of them by Agrocomplet consulting engineers company (Bullgarian company) in the year 1979(unavailable data).The second study was by "ITSC" - British company which proposed 46.5m to be the maximum height of the dam (the maximum water level with this height corresponds to an elevation of 716.5m above sea level) and concluded that the total water volume in the reservoir is equal to $59.7 \times 10^6 m^3$ with this height (ITSC, 2007). The ITSC report gives very brief geotechnical information about the Basara gorge area, including unspecified Rock Mass Rating (RMR) value. Most of their report is concentrated on the design of the dam.

Another related engineering study about dams in Iraq is by Ajjotheri (2003), who studied the effect of spacing, density, aperture and persistence of discontinuities on some geotechnical properties of rock mass, such as porosity, permeability and Rock Quality Designation (RQD) at Ejbail proposed dam site (Anah city-west of Iraq). He concluded that the discontinuities have a direct effect on these geotechnical properties.

There are numerous studies concerning the dam site investigations, e.g. Ghazifard et al. (2006) who used Rock Mass Rating(RMR) classification system and Hoek-Brown failure criterion in the evaluation of engineering geological characteristics for Kuhrang III dam site in Esfahan(Iran).

In addition to those studies concerning the dam investigation, there are some studies about or near to our study area, which can be summarized in table (1-1).



Fig(1-1) Satellite image of the study area showing the proposed dam site

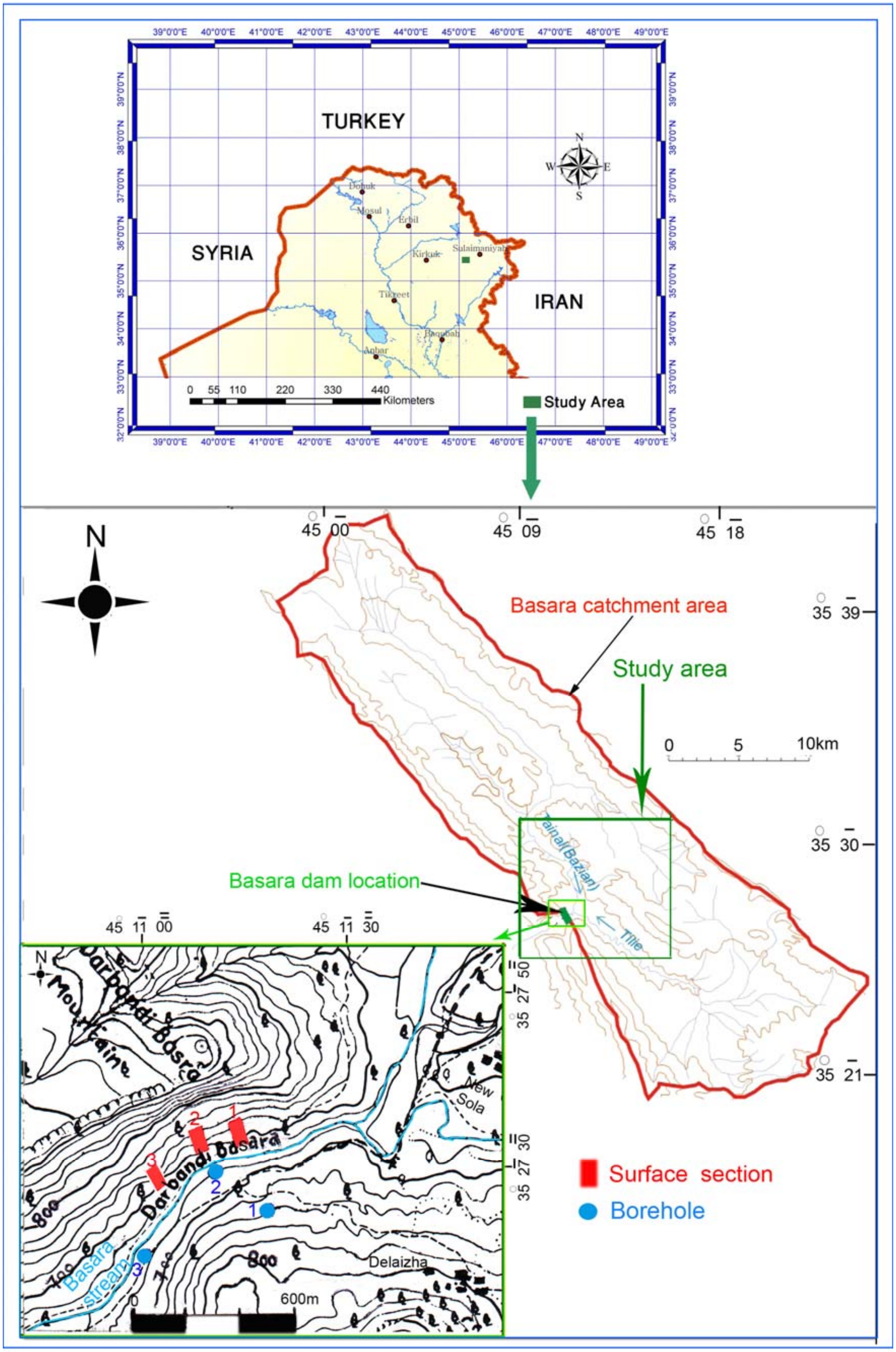


Fig (1-2) Location map of the study area

Table (1-1) Previous studies (except dam investigation) about or near the study area.

No.	Authors/Year	Subject	Area
1	Al-Etaby and Muhamad (1979)	Economic geology(exploration of the carbonate rocks for cement).	Tasluga area, Kuik village and Tainal quarry (Bazian area)
2	Surdashy(1988)	Microfacies and depositional environment (Sinjar Fn.).	Bazian, Baranan and Dokan areas.
3	Surdashy and Lawa (1993)	Facies analysis, Biostratigraphy and depositional environment.	Bazian (Hayassi village)
4	Stevanovic et al. (2003)	Climate,hydrology,geomorphology , geology and field document.	Northern Iraq (Kurdistan region)
5	Stevanovic et al. (2004)	General hydrogeology and aquifer system.	Northern Iraq (Kurdistan region)
6	Lawa(2004)	Sequence stratigraphy	Kani Gopala (Bazian), Bamo , Zhalla, Kashty, Sagrma, Darbandikhan & Dararash Sulaimani district
7	Aziz(2005)	Greophysics(geophysical investigation for hydrogeological purpose.	Bazian
8	Al-Samaraey(2007)	Physical Geography	Tainal, Bazian and Northwest Part of Qaradagh (Catchment area at Basara stream)

1-4- Aims of the study

This study aims at evaluating the suitability of the study area for dam construction by:

(**1**)- Estimating the mechanical properties of the rock masses and this involves two aspects:

a- Evaluating the rock masses at the proposed site according to various classification systems, such as the Rock Mass Rating (RMR), Dam Mass Rating (DMR) and Geological strength Index (GSI).

b- From the above mentioned classifications and rating systems in conjunction with the Hoek-Brown failure criterion, the mechanical properties of the rock masses (compressive strength, tensile strength, deformation modulus, shear strength parameters including the friction angle (Ø), and cohesion (c)) were then estimated.

(**2**)- Investigating the geology of the area and its influence on dam site selection by preparing a geological map (scale 1:20000) that shows the distribution of the different rock units (Formations) on the surface and the main geological structures in the area (folds, faults and attitudes of strata).

(**3**)- Investigating the topography of the area by constructing topographic profile at various sites to find the optimum profile for dam site selection.

(**4**)-Integrating all the various geotechnical, geological and topographic (slope) data and preparing rock mass – valley section models (for the first time) for different sites and comparing them to find the most suitable site for dam construction.

1-5 Research methods

1-5-1 Field work

The field work started in Feb.-2007 and extended to March-2008 for the period of 160 days (104 days for general survey of the study area and detailed study at and near the proposed dam site, 54 days for drilling three boreholes in the left side of the proposed dam site).

As there is no precise geological map about the area, a detailed geological survey of the area was conducted, using a topographic maps with a scale of 1:20000 (Maps No. 71/670 and 71/680 after the directorate general of surveys, Baghdad – Iraq) as a base map for preparing a geological map of the area.

The geological survey included a description of the existing formations, measuring the dip and strike of all contacts between formations at numerous locations in the area. The altitude, latitudes, longitudes of the mentioned locations and along the axes of anticlines and synclines were recorded using GPSMAP 60CSx. Finally, the obtained data were projected on the topographic map to draw the geological map of the area.

Detailed information at the proposed dam site was collected for rock mass evaluation. To achieve this task, three surface stratigraphic sections and three boreholes were selected, Fig (1-2).

Each section or borehole was divided into a number of rock mass units depending on the lithological (change in rock type) and structural properties (such as changes in discontinuity spacing and characteristics or the presence of a fault zone).

For each surface section unit, some procedures were carried out:

(A) The following data were obtained:

1- Type of the rock

2- Dip direction and angle of bedding plane

3- Conducting a detailed study of discontinuities, this includes:

3-1- Dip direction and angle

3-2- Discontinuities spacing

3-3- Discontinuities frequency

3-4- Discontinuities persistence

3-5- Discontinuities roughness

3-6-Weathering and alteration of discontinuity surface

3-7-Aperture and infilling material

4-Geological information for “GSI” determination (such as blockiness “structure” and surface condition of discontinuities).

(B) Rock samples for laboratory studies were collected.

(C) Determining the altitude and fixing the location of each surface section by GPS MAP 60CSx.

Three boreholes (two on Basara stream left bank and the third one on the valley side) were drilled by directory of water well in Sulaimani using Dando (Rig-5/24 ton.) drilling machine which drilled with double-tube core barrel of 3.5 inch diameter. Whereby at each borehole the following points were recorded:

1- The depth to the bed rock which indicates the thickness of weathered or drifts material that is necessary to be excavated when the dam is constructed.
2- Preservation of the rock samples in wood boxes.
3- Calculation of the Rock Quality Designation (RQD).

4- Determining water table level.
5- Transporting the core samples to the laboratory for detailed study.
6- Determining the altitude and fixing the location of the boreholes by GPSMAP 60CSx.

1-5-2- Laboratory work

Laboratory work included petrographic inspection, determining the strength properties of the rocks (from surface sections and boreholes), furthermore other descriptive and numerical studies were done on boreholes.

Petrographic inspection included making thin sections for study under the microscope, then classifying the carbonate rocks in the dam site according to Folk's (1962) classification which is suitable for the Hoek-Brown failure criterion elements, such as material constant (mi) and modulus ratio (MR) of intact rock.

The strength properties included conducting the point load and the unconfined (uniaxial) compression tests to find the unconfined compressive strength of the intact rock.

The descriptive and numerical studies on boreholes comprised information about GSI and counting the number of discontinuities and measuring their dip angle with respect to the borehole axis in each rock mass unit.

1-5-3- Office work

The office work included representation and analysis of the field and laboratory information and measurements, wherein some rock mass classification parameters which were obtained descriptively were converted into rated parameters, such as GSI, discontinuities roughness, weathering of the discontinuity surfaces, infilling material, discontinuity persistence, water condition and discontinuities orientation.

Other parameters that were obtained numerically were also rated according to the weight of each one in the classification system, such as uniaxial compressive strength, RQD and discontinuity spacing. Furthermore some parameters were adjusted in order to suit the requirements of the classification system, such as unconfined compressive strength and RQD.

In addition to that, some parameters were adjusted in order to suit the requirements of the classification system, such as water condition and discontinuity orientation in the Dam Mass Rating (DMR) classification system.

After all the parameters in the classification system are rated, they were arranged in a table for determining the value of each classification system and subsequently this value was used in an equation to estimate the mechanical properties of the rock mass.

A “RocLab” programme was also used in performing the Hoek-Brown failure criterion for estimating the mechanical properties of the rock mass, in which the “2002 edition” of the above criterion and empirical rock mass modulus equations of “Hoek and Diederichs, 2006”were used in the programme.

Moreover, attitudes of the bedding planes were represented on the Schmidt equal area net to find the average dip of the bedding planes.

The last stage of office work involved the integrating of all the geotechnical, geological and topographic (slope) data to find the optimum site for dam construction and finally writing the thesis.

The summary of all the work from first until choosing the optimum site for the dam is illustrated in Fig (1-3) as flow chart.

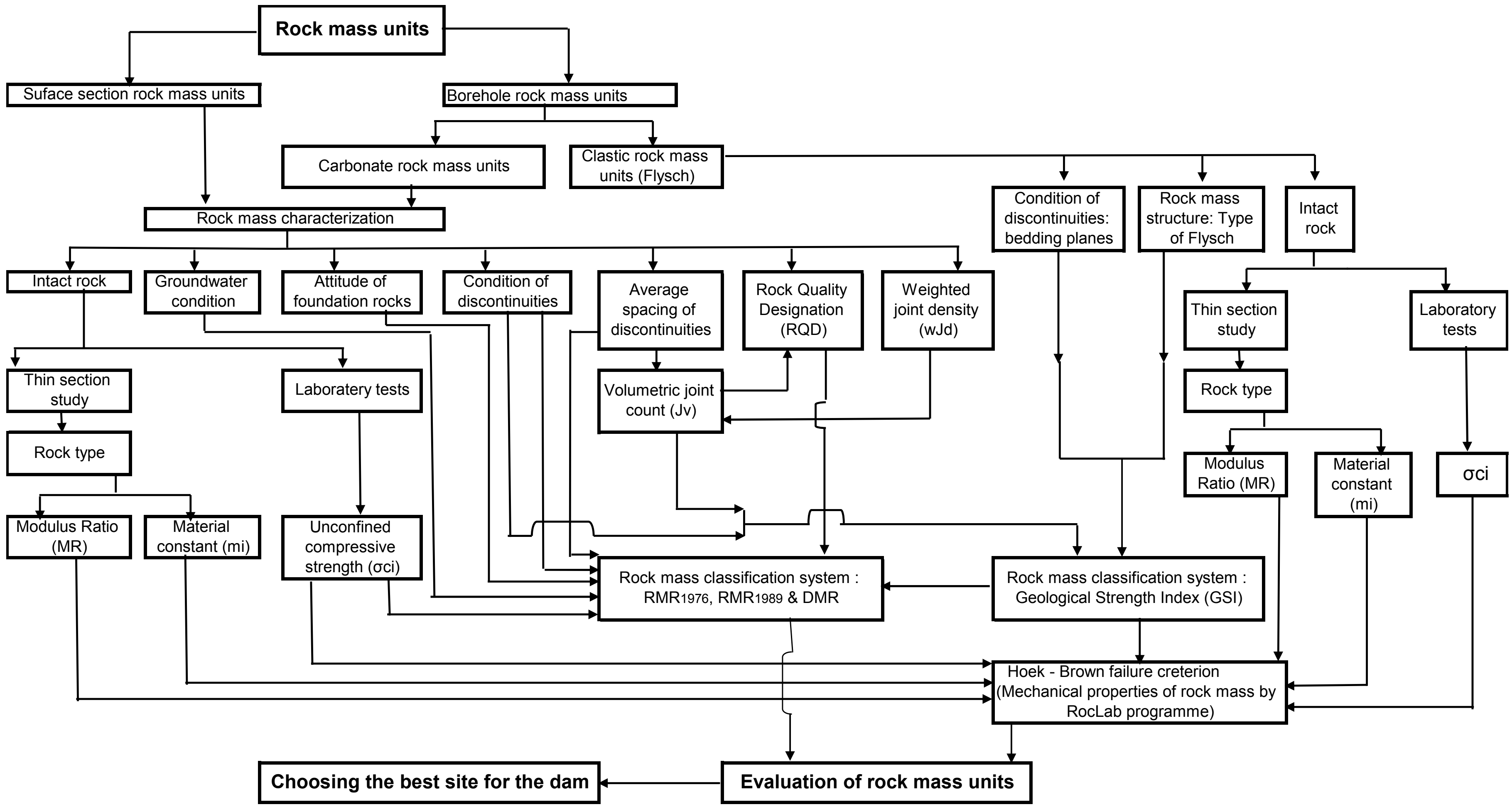


Fig (1- 3) : Flow chart showing the procedure followed in the dam site evaluation

Chapter Two
Geology of the study area

2-1- Preface

Tectonically, the area is located near the SW boundary of the High Folded Zone, Fig (2-1), which is characterized by high mountains and intense folding resulted from Alpine orogeny .

Field survey revealed that the area lies between two mountain series representing an anticlinal structural feature; these are Bakhshi – kalawe mountain series running along northeast and Hanjira - Darband Basara - Sagrma mountain series running along southwest. The later mountain series represent the boundary between the low and high folded zone in the area. This boundary runs along the southwest limbs of the first high anticlines (Buday and Jassim,1987; Jassim and Goff,2006).

The area between the mentioned mountain series in general represents broad synclinal depression(valley).

From the seismological point of view, Iraq is located in a relatively active seismic zone at the northeastern boundaries of the Arabian plate. The north and northeastern zones of Iraq depict the highest seismic activity (Alsinawi and Al-Qasrani, 2003).

Earthquake data for the period 1900 -1988 was utilized for the seismicity studies. Most earthquakes clustered on the edges of Zagros – Taurus subduction zones between the Arabian subcontinent plate and the Iranian and Anatolian plates; in addition to few intraplate types on the tectonically stable zone to the west. It was found that 95.5% of the events have magnitudes range of 4.0 – 5.4 mb (Alsinawi and Al-Qasrani, 2003).

For the area of Sulaimani and according to the methods used by some authors for determining the surface of 51 events, it was found that local magnitude is at the range of 2.3 – 4.4 (Basil et al., 1989)

The seismic zoning map of Iraq which is based on MM scale (Alsinawi and Al-Qasrani, 2003) showing that the study area is located at the interface of minor and moderate damage zones.

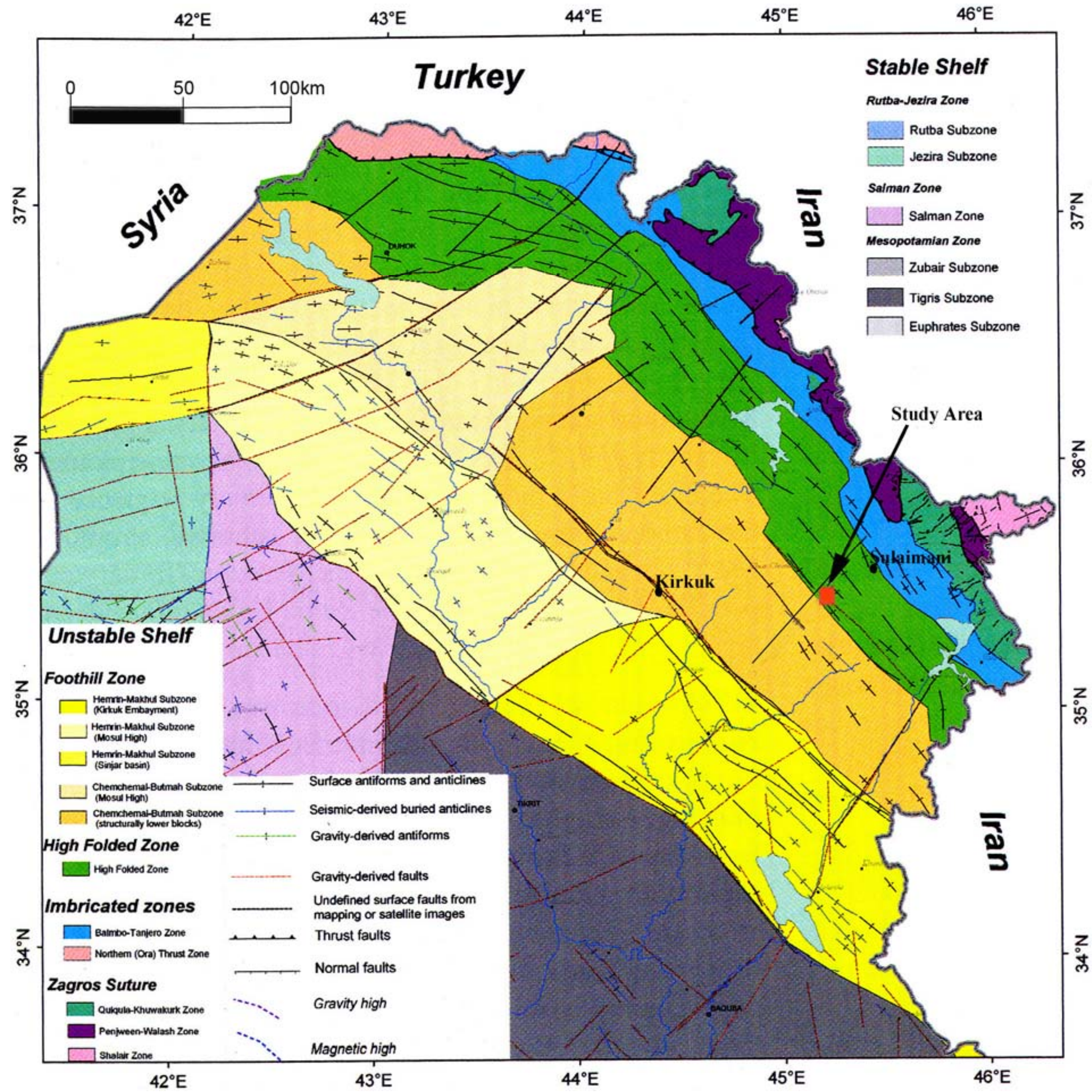


Fig (2-1) Tectonic map of Iraq (Jassim and Goff, 2006) showing location of the study area

2-2-Stratigraphy

All the existing Formations in the study area were determined as shown in Enclosure (an envelope, enclosed with this thesis) and a brief description of them from oldest to youngest as follows:

2-2-1 Kolosh Formation

The age of the Formation is Paleocene - Lower Eocene (Bellen, et al., 1959), but according to Lawa (2004) the formation is of Paleocene age. Lithologically, its upper part exposed in the area consists of gray to dark gray calcareous shale alternating with siltstone and silty marlstone, greenish gray calcareous sandstone, pebbly sandstone and sandy limestone(rich in fossils).

The lower contact is not exposed. The upper contact which was studied by Surdashy(1988), Surdashy and Lawa (1993) is regarded as gradational conformable contact with Sinjar Formation, but in another study by Lawa (2004) it is regarded as an unconformable contact with Sinjar Formation in the sections (Sagrma and Kanigopala) adjacent to our study sites and he referred to the presence of minor gap of about (0.5) million year.

2-2-2 Sinjar Formation

Its age is Paleocene – Lower Eocene (Bellen, et al., 1959) but Lawa (2004) determined its age as Early Eocene (Ypresian). It consists of thick to massive beds of yellowish gray Limestone, argillaceous limestone, sandy limestone and conglomeratic limestone.

The upper contact is unconformable with Gercus formation which is indicated by colour and lithological variation into red clastic with weak conglomerate bed at this boundary.

2-2-3 Gercus Formation

Its age is Middle Eocene (Bellen, et al., 1959). Lithologically, it consists of red clastic sequence of pinkish red to purple siltstone and claystone alternating with green marl, gray to reddish brown coarse-grained sandstone with conglomerate bed at the bottom of sandstone beds.

The upper contact is unconformable with Pila Spi Formation represented by conglomerate of about 2.5-6m thickness, Fig (2-2).

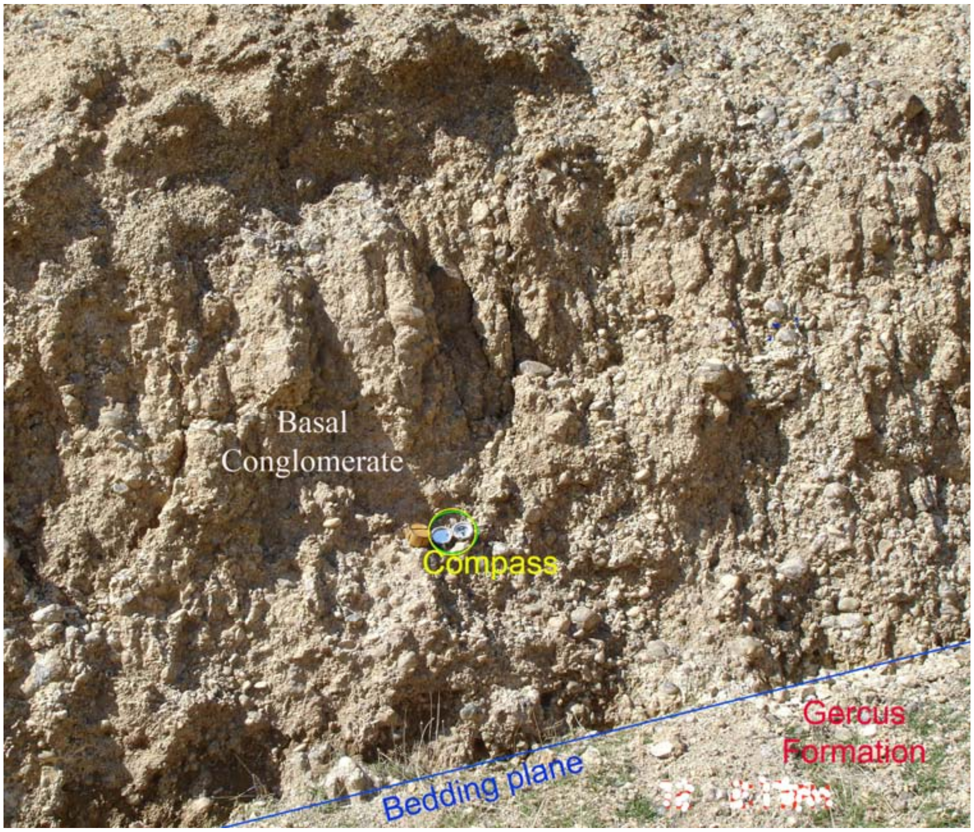


Fig (2-2) Basal conglomerate represents unconformity between Gercus and Pila Spi Formations (Compass is a scale)

2-2-4 Pila Spi Formation

Its age is Middle – Upper Eocene (Bellen et al., 1959).Lithologically, it consists of well bedded, highly fractured limestone, dolomitic limestone, dolomite and chalky limestone. The upper contact of the Pila Spi Formation with the Fat'ha (previously Lower Fars) Formation is unconformable which is indicated by the Basal conglomerate, Fig (2-3).

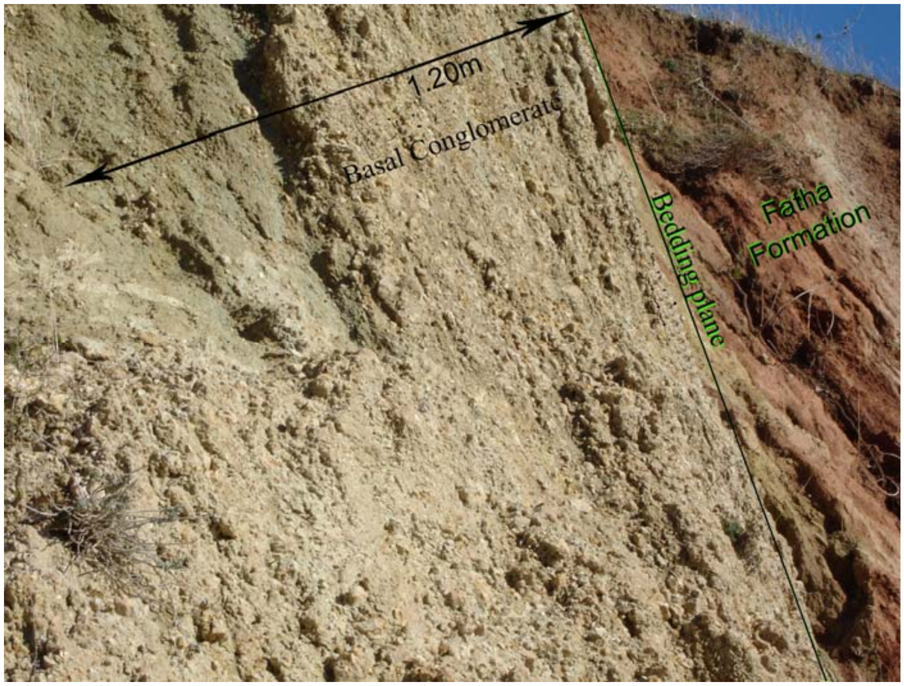


Fig (2-3) Basal conglomerate represents unconformity between Pila Spi and Fatha Formation, at a distance 800m to N10E of Darband Sutaw village

2-2-5 Fat'ha (previously Lower Fars) Formation

Its age is Middle Miocene (Bellen et al., 1959). Lithologically, it consists of alternating sedimentary cycles of yellowish gray fossiliferous limestone, green marlstone, claystone, siltstone and sandstone without gypsum at Darband Sutaw village and surrounding areas. While, it also contains gypsum, exactly at Sola village and towards southeast. The upper contact is gradational and conformable with Injana Formation.

2-2-6 Injana (Previously Upper Fars) Formation

Its age is Upper Miocene (Bellen et al., 1959). Lithologically, it consists of reddish brown- gray colored claystone, silty marlstone, siltstone and brownish grey sandstone. The upper contact is conformable with Mukdadiya Formation, it is marked in the field by the first conglomerate bed, having yellowish colour,Fig(2-4).This conglomeratic bed is regarded as the best marker of the limit between Injana and Muqdadiya Formations(Buday,1980).

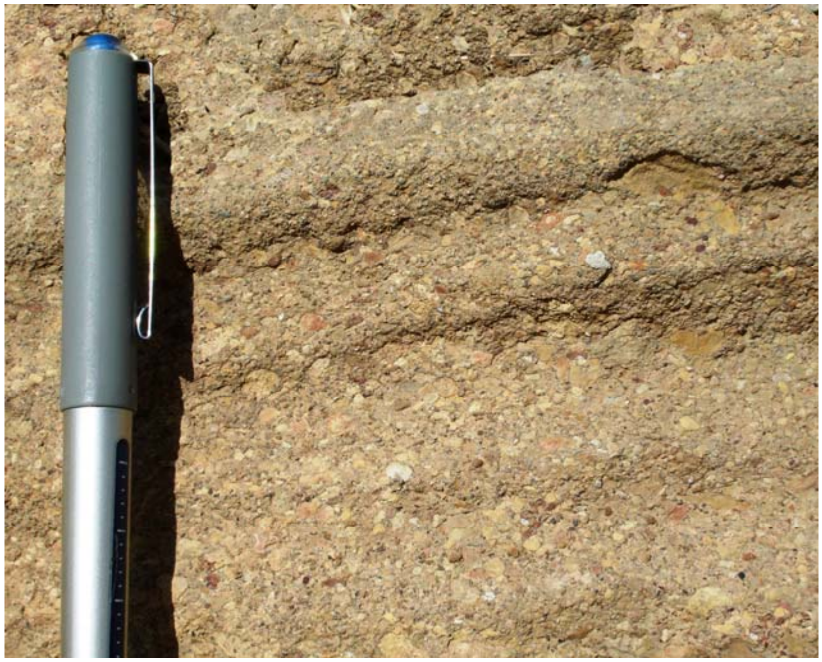


Fig (2-4) Conglomerate represents the contact between Injana and Mukdadiya Formations

2-2-7 Mukdadiya (previously Lower Bakhtiari) Formation

Its age is Pliocene (Bellen et al., 1959). Lithologically, it consists of red clay, sandstone and pebbly sandstone, the size of grains increasing upwards until reaching the massive conglomerate beds of Bai Hassan Formation which is the boundary between them, and it is mostly conformable.

2-2-8- Bai Hassan (previously Upper Bakhtiari) Formation

It is of Pliocene age (Bellen etal.,1959).It mainly consists of coarse and thick fluviatile conglomerate, sandstone and claystone.

2-2-9- Alluvial deposits

They include deposits of flood plains, alluvial fans and river terraces.

Flood plain deposits consist of a mixture of coarse grains soil(gravel and sand) plus fine grains soil(clay and silt) on the banks of the two permanent streams(Tainal and Tilie streams) wherein the grain size becomes smaller when it moves away from the stream channel . These deposits are formed during periods of flooding (Thornbury, 1969; Ritter, 1986).

Alluvial fan deposits are found in the lower part of mountain valleys, Fig (2-5), where change in slope occurs, and they are excellent examples of water–spreading wash slopes (Bloom,2002), which consist of large amount of coarse grains plus some fine grains soil.

River terrace deposits are observed at higher level from the present base level of erosion of Tainal stream, near New Sola village, Fig(2-6). They represent valley floors abandoned by the rivers as they start to cut down to the new and lower base level (sparks, 1972).

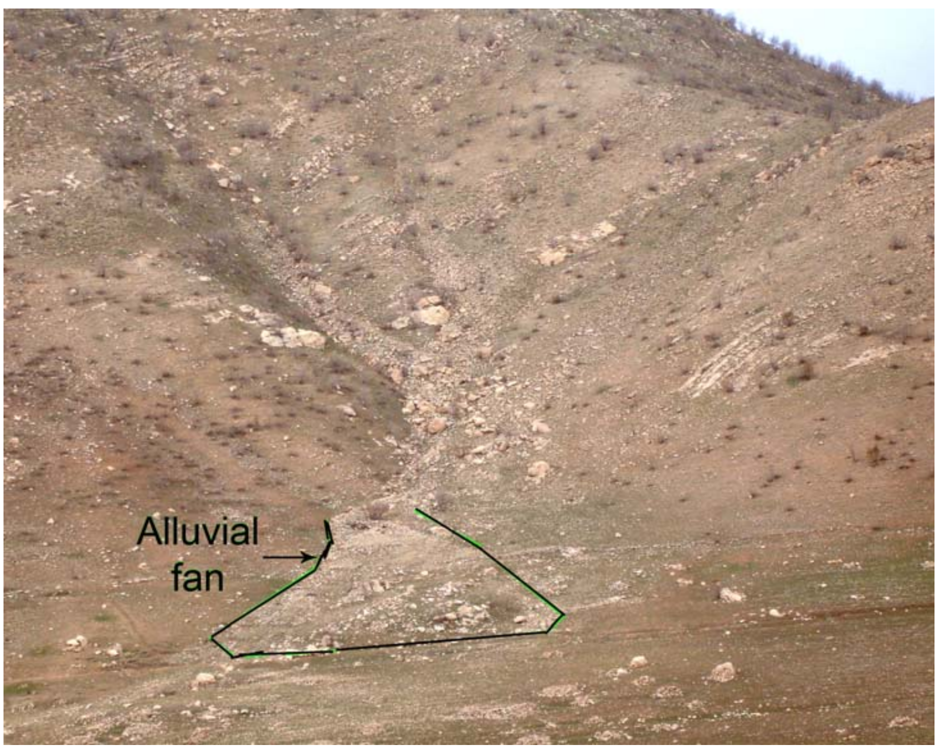


Fig (2- 5) Alluvial fan (cone) deposits, at a distance 700m to the southwest of Darband Sutaw village (the trees are 1. 2 – 3.5m high)

Fig (2-6) River terrace deposit of Tainal stream, near New Sola village (the trees are 3 – 4m high)

2-3 Microscopic study

For precise description of the rocks, thin sections of various rocks of Sinjar and Kolosh Formations (which are prevalent at the proposed dam site) were made and studied under polarized microscope.

In general, the microscopic inspection shows that the Sinjar Formation is composed of limestone rich in fossils and some pebbles, while the Kolosh Formation is composed mainly of clastic rocks with some limestone beds.

The carbonate rocks were classified according to Folk(1962) and the clastic rocks according to grain size of rock components. These classifications are suitable in giving value to the material constant (mi) element of Hoek-Brown failure criterion which is used in the determination of the mechanical properties of the rock mass.

A precise microscopic study of thin sections revealed the following types, as in table (2-1).

Table (2-1) Microscopic study of rock types for Sinjar and Kolosh Formations at the proposed Basara dam site.

Geologic Unit	Rock Type	Figure Number
Sinjar Formation	Micrite	Fig (2-7)
	Biomicrite	Fig (2-8)
	Intramicrite	Fig (2-9)
	Intrabiomirite	Fig (2-10)
Unconformity	Intrapelbiosparite	Fig (2-11)
Kolosh Formation	Sandstone	Fig (2-12)
	Pebbly Sandstone	Fig (2-13)
	Siltstone	Fig (2-14)
	Shale	Fig (2-15)
	Silty Shale	Fig (2-16)
	Intrabiomicrite	Fig (2-17)
	Biomicrite	Fig (2-18)

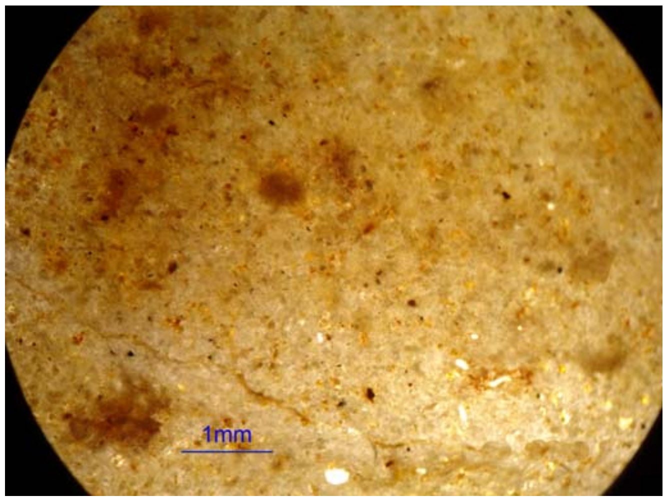


Fig (2- 7) Micritic limestone of Sinjar Formation

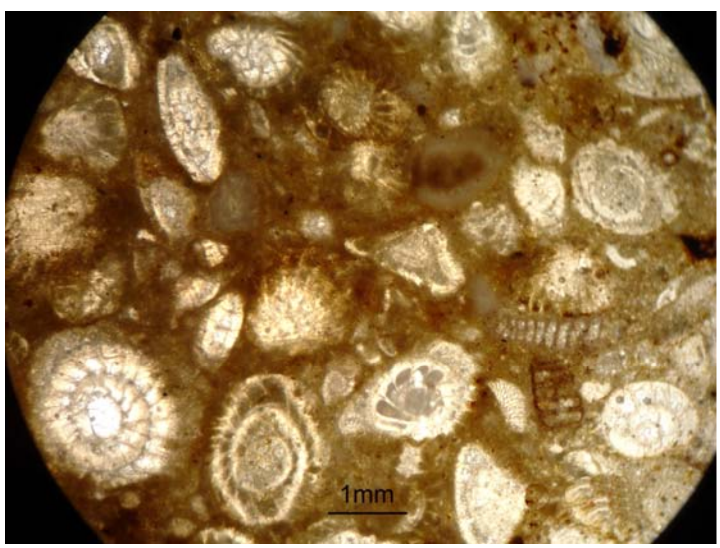


Fig (2- 8) Biomicritic limestone of Sinjar Formation

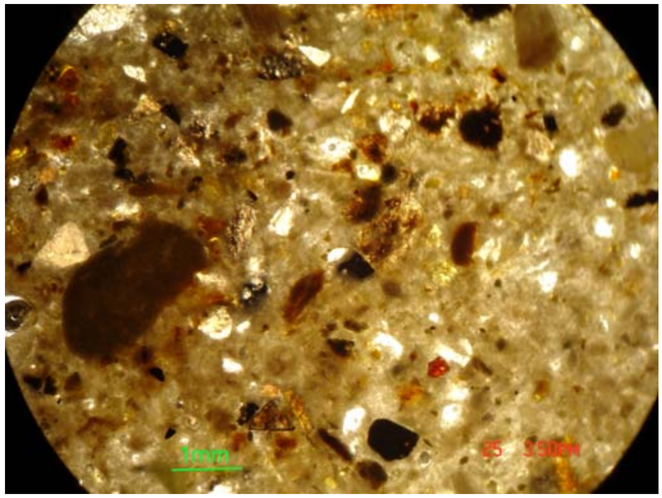


Fig (2- 9) Intramicritic limestone of Sinjar Formation

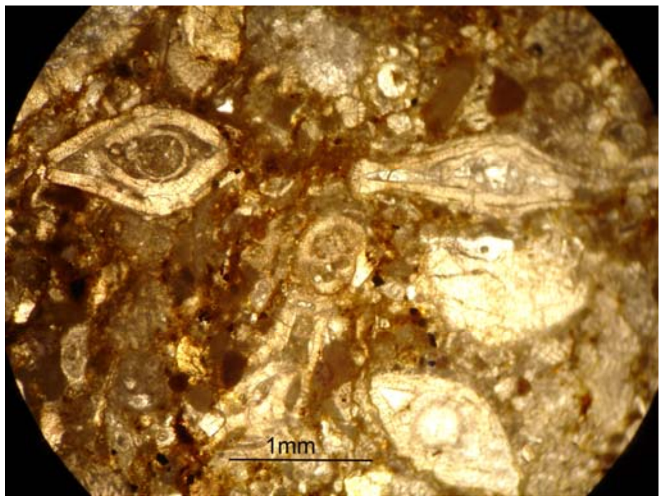


Fig (2-10) Intrabiomicritic limestone of Sinjar Formation

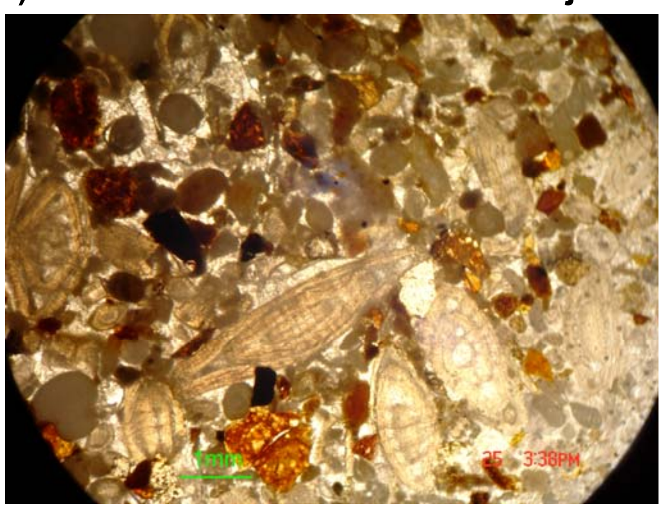


Fig (2-11) Intrapelbiosparitic limestone represents the unconformity between Sinjar and Kolosh Formations

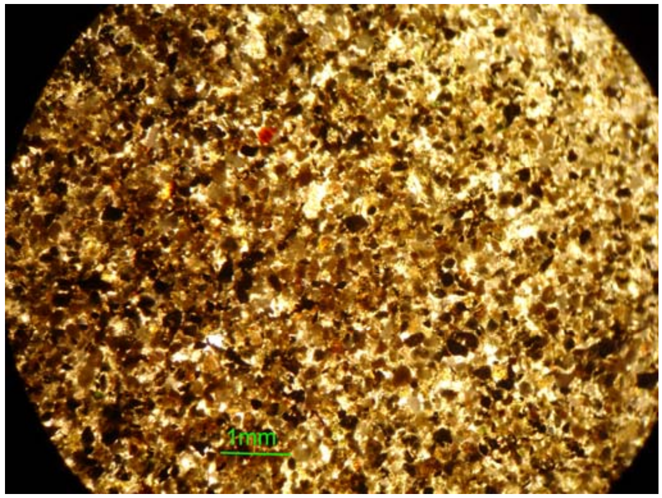


Fig (2-12) Sandstone of Kolosh Formation

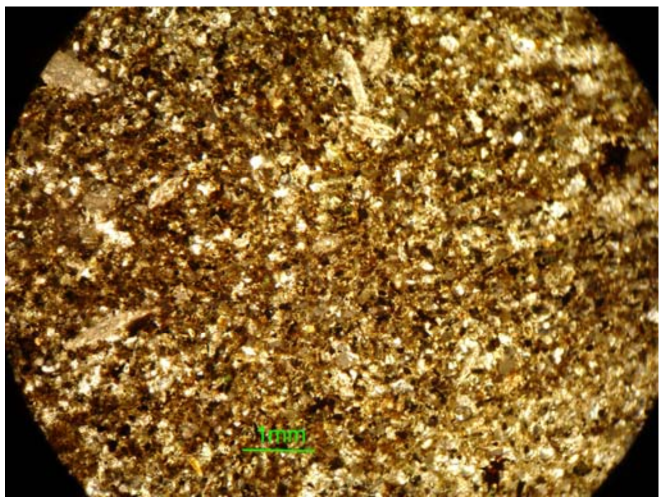


Fig (2-13) Pebbly Sandstone of Kolosh Formation

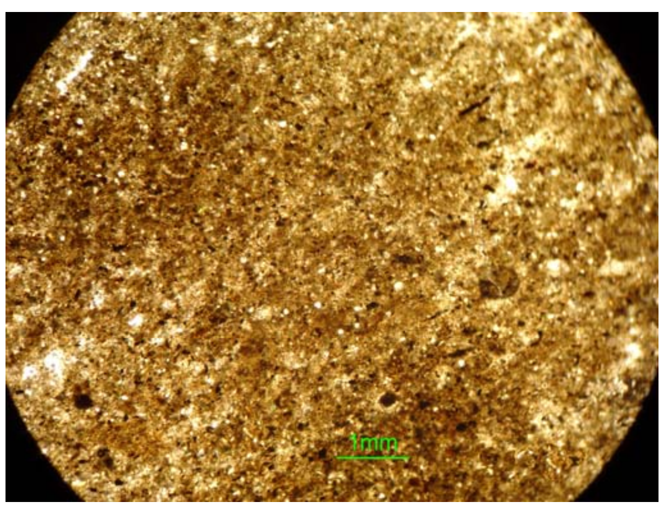


Fig (2-14) Siltstone of Kolosh Formation

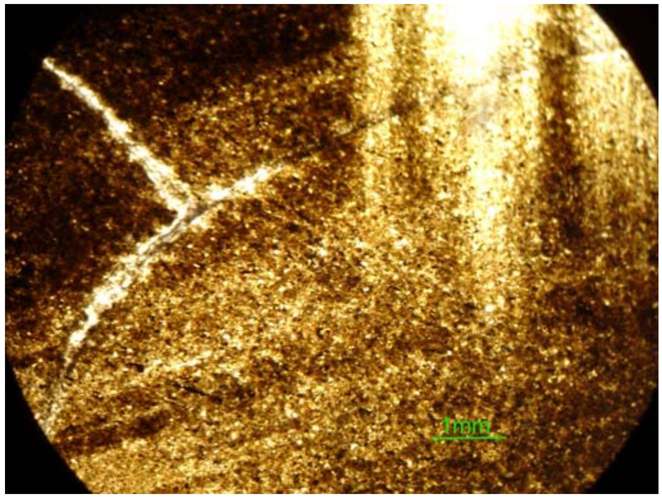


Fig (2-15) Shale of Kolosh Formation

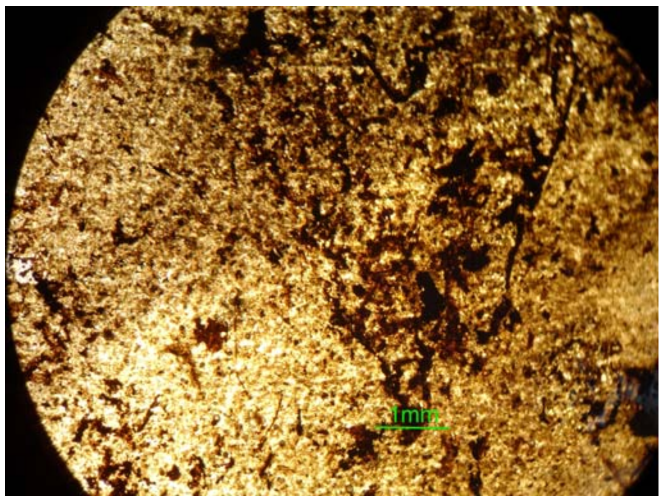


Fig (2-16) Silty Shale of Kolosh Formation

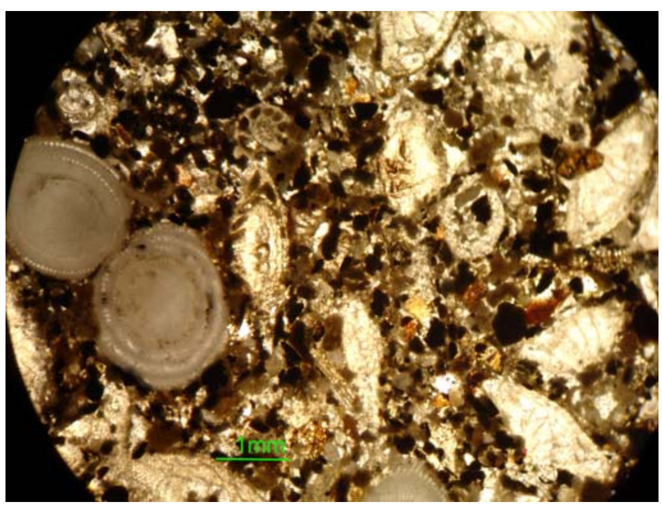


Fig (2-17) Intrabiomicritic limestone of carbonate unit of Kolosh Formation

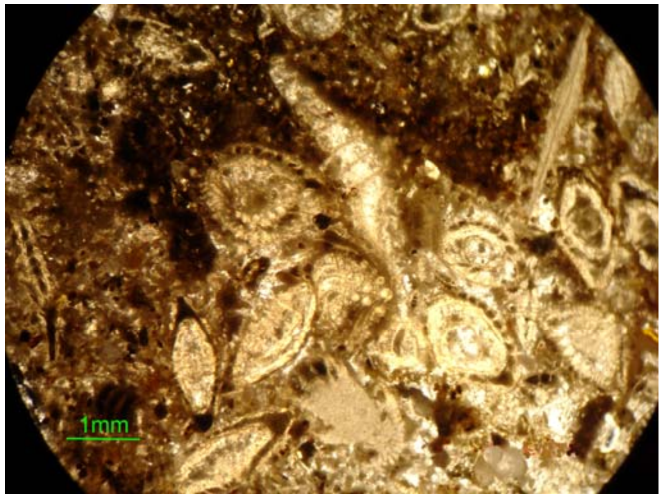


Fig (2-18) Biomicritic limestone of carbonate unit of Kolosh Formation

2-4 Geological map of the study area

The geological map (look at the enclosure) shows the existing Formations, structural features (anticlines, synclines, faults) and the strike and dip symbols.

From the geological map it appears that the area was subjected to intense folding which resulted in the formation of complex structural features.

In general, the anticlines are asymmetric, some has the northeast vergency (Darband Basara anticline) which is due to the existing of ductile rocks at the core of anticlines (Ibrahim, 2009). This is opposite to that of most anticlines of the high folded zone which are characterized by southwest vergency (Buday and Jassim,1987; Jassim and Goff,2006).In some places, the northeast limb of major anticline(Darband Basara anticline) was dragged forming secondary folds and are overturned.

The major anticlines (e.g.Darband Basara) have broad crests (box-like shape) and all folds are trending NW-SE.

The northwest part of the area was intensively folded which is leading to the creating of a number of smaller anticlines and synclines between the major anticlines, while this phenomenon was not observed in the southeast area which is characterized by a broad syncline(New Sola – Qazanqaya Syncline) as in the Enclosure. Kolosh Formation crops out at the core of the major anticlines (Darband Basara anticline in the SW part and Bazian anticline in the NE part of the area) and Gercus Formation at some cores of the smaller one.

There are some mapable faults in the area, some of them cutting the folds transversally, two of them (F1) & (F2) are oblique slip faults having a displacement with a strike and dip components of the fault, and the other (F3) is a strike slip fault having a displacement parallel to the strike of the fault.

Other large fault is Delaizha reverse fault (F4), which cuts the northeast limb of Darband Basara anticline, passing near Delaizha village and parallel to the general trend of the fold, as shown in the Enclosure. This reverse fault is formed (created) after the deposition of Mukdadiya and Bai Hassan Formations, in which the southwest hanging wall block (Pila Spi Formation) moved over Fatha, Injana and Muqdadiya Formation and preserved them from erosion at relatively high altitude.

The mentioned reverse fault subsequently created unstable situation in the hanging wall and eventually sliding and rolling of the Pila Spi Formation over Mukdadiya, Injana and Fatha Formation and resting over them.

Some small faults are present in the area; one of them is located at the proposed dam site which is of reverse fault type, having 2.5m displacement, Fig (2-19).The other reverse fault is located somewhat near the dam site, in the limestone of Pila Spi Formation on the right side of Darband Basara inlet, Fig (2-20).

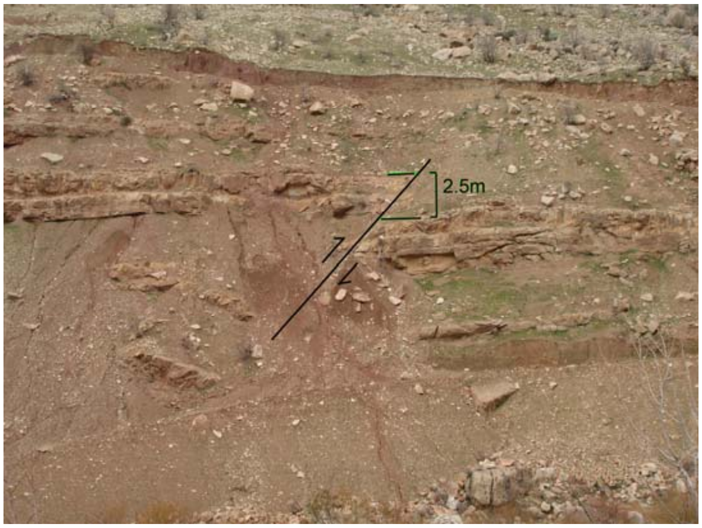


Fig (2-19) Reverse fault cutting the limestone beds of Sinjar Formation at the right side of proposed dam site

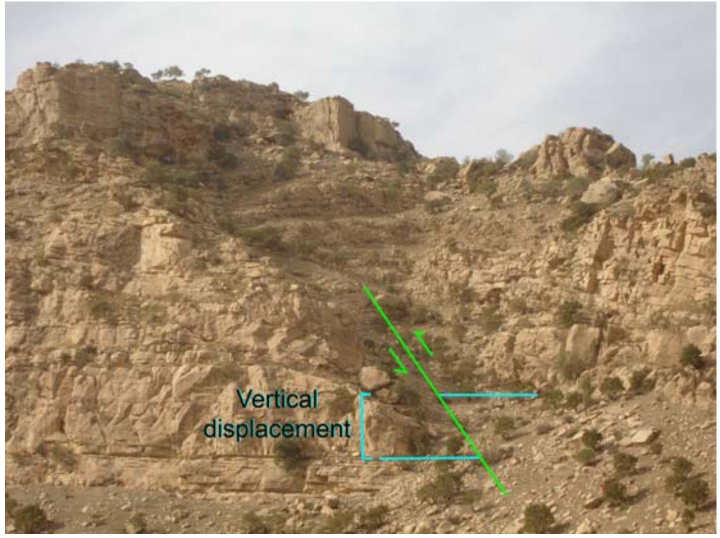


Fig (2-20) Reverse fault near the right side of proposed dam site (the trees are 2.5-4m high)

2-5-Geomorphology of the area

There is a close relationship between topography and structure of some major folds, such as Darband Basara-Sagrma mountain series, which structurally represents an anticline, and also Tilie valley structurally represents a syncline (look at the enclosure). This phenomenon reflects the fact that the geologic structure is a dominant control factor in the evolution of landforms and is reflected in them (Thornbury, 1969).

There are other landforms which bear an inverse relationship with the structure that they were developed upon them, so an inversion of topography is resulted, such as Chapa Chnara valley which is running with the fold axis of Darband Basara anticline. This is obvious from the geological map. Another example is the presence of synclinal ridge that is located 2km to the north of Darband Sutaw village, Fig (2-21).

Cliffs or escarpments of resistant limestone surrounding Darband Basara gorge are due to undergoing the exposure of Darband Basara anticline to the dissection process at its crest. The outcropping edge of resistant limestone of Pila Spi and Sinjar Formations are also formed escarpments at various locations in the area.

Fig (2-21) Synclinal ridge in the limestone of Pila Spi Formation
(the trees are 2- 3.5m high)

Homoclinal ridges are wide spreading in the area where there are alternations of resistant and weak rocks, especially in the Fat'ha, Injana and Mukdadiya Formations, that are extending as strike ridges for long distance. They have the form of asymmetric Cuesta, Fig (2-22) where the resistant bed dips gently, then a steep escarpment and gentle dip slope have resulted (Bloom,2002). Other ridges which are approximately steep symmetric (dip slope and escarpment have approximately equal angle) have the form of Hog back, Fig (2-23) such as those ridges extending from Sola village towards Kuna Kuter village in the southeast.

Beside the mentioned structural and erosional landforms, there are also depositional landforms in the study area, such as flood plains on the banks of Tainal and Tilie streams, alluvial fans in the toe of mountain vallies and river terraces as previously mentioned.

Some features which can hardly be classed as landforms are geologic features, adding varities to topographic surface are presented in the area. They are Stone Lattice(Elephant Skin),Fig(2-24) and Stratification Ribbons, Fig(2-25). Stone Lattice is resulted from differential weathering along joint planes, while Stratification Ribbons are due to differential weathering along bedding planes

(Thornbury, 1969). The solution landforms, especially caves and lapies are also found in the rocks of Sinjar and Pila Spi Formations.

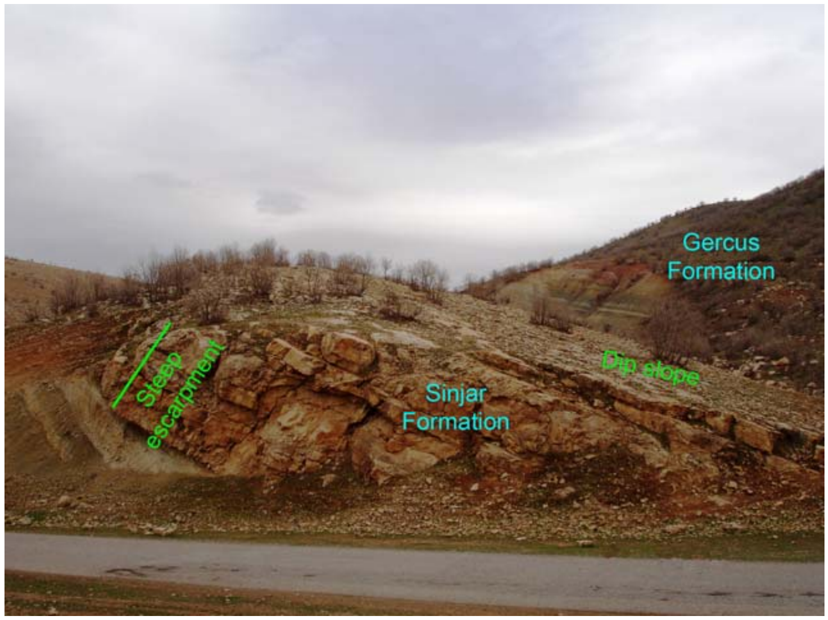


Fig (2- 22) Cuesta in the limestone of Sinjar Formation, at a distance 700m to the southeast of Khewata village (the trees are 2- 3.5m high)

Fig (2- 23) Hog back in the limestone of Fatha Formation, at the east of Sola village (the trees are 2.5 - 4m high)

Fig (2- 24) Stone lattice (Elephant skin) in the limestone of Fatha Formation


Fig (2- 25) Stratification ribbons in the limestone of Pila Spi Formation (all strata dip to the left and the trees are 2.5- 4m high)

2-6 Climate, hydrology and hydrogeology of the study area

The area represents a semi- humid climate (Al-Samaraey, 2007). Due to the lack of meteorological data in the Bazian and Qaradagh stations and because the study area is adjacent to Sulaimani city and has approximately the same climatic properties, the meteorological data of Sulaimani station are used.

2-6-1 Temperature

The monthly average temperature for the years 1980-2006 is shown in table (2-2).

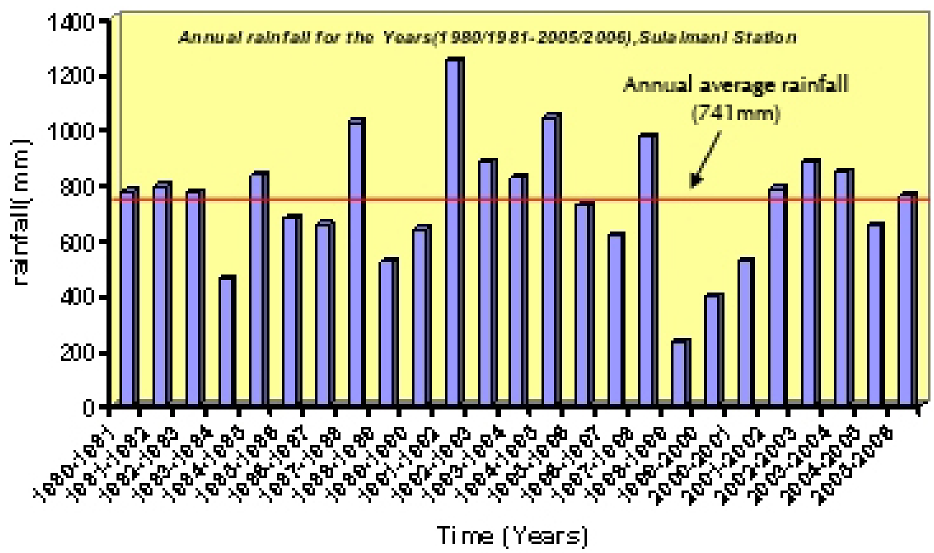
Table (2-2) maximum, minimum and average monthly temperature (C°) in Sulaimani metrological station for the years 1980-2006 (Sulaimani Meteorological station)

Month	Maximum Temp.(C°)	Minimum Temp.(C°)	Average Temp.(C°)
Jan	9.7	1.45	6
Feb	11.2	2.1	7
Mar	14.7	2.45	11
Apr.	24.85	14.8	17.5
May	28.7	19.6	23.3
Jun	34.9	24.9	29
Jul	37.5	30.8	33.6
Aug.	37.5	28.6	32.7
Sept.	31.8	26.6	29
Oct.	28.3	17.7	22.3
Nov.	20.3	9.3	13.5
Dec.	14.6	2.2	8

From table (2-2), it appeared that the maximum temperature in Jan. was 9.7C° and in Jul. and Aug. was 37.5C°, while the minimum temperature in Jan. was 1.45C°and of Jul. was 30.8C°.Moreover, there were several days in Jan. and Feb. in which the temperature was below 0C°, and also there were several days in Jul. and Aug. in which the temperature was above 40C°.

2-6-2 Precipitation

The annual average rainfall in Sulaimami station for the years 1980-2006 is 741mm, Fig(2-26), the maximum, minimum and average monthly rainfall for the years 1980-2006 are also shown in table(2-3).



Fig(2-26) Annual average rainfall in Sulaimani station for the years 1980-2006 (After Ali, 2007)

Table (2-3) Maximum, minimum and average monthly rainfall (mm) in Sulaimani station for the years 1980-2006 (Sulaimani meteorological station)

Months	Rainfall (mm)		
	Average	Max.	Min.
Oct.	31.8	146.2	0.0
Nov,	104.5	264.4	0.0
Dec.	119.5	354	3.8
Jan.	131.2	273.6	17.2
Feb.	115.7	309.6	41
Mar.	107.0	191.5	2.6
Apr.	86.4	223	1.1
May	42.2	89.9	0.0
Jun.	1.4	18.6	0.0
Jul.	0.0	0.0	0.0
Aug.	0.0	0.0	0.0
Sep.	1.4	12.1	0.0
Annual average rainfall (mm)	741		

2-6-3 Evaporation

The maximum, minimum and average monthly evaporation from class-A pan in Sulaimani station for the years 1980-2006 are shown in table (2-4).

Table (2-4) Maximum, minimum and average monthly evaporation from class-A pan in Sulaimani station for the years 1980-2006(Sulaimani meteorological station)

Months	Evaporation (mm)		
	Average	Max.	Min.
Oct.	166	233.5	111.6
Nov,	93	278	42.57
Dec.	52	92.5	34.1
Jan.	52	79.9	38.1
Feb.	59	93.4	37.7
Mar.	97	152.2	34.6
Apr.	139	255.3	110.2
May	230	416.5	176.6
Jun.	341	405.7	283.2
Jul.	414	498.2	330.4
Aug.	286	474.3	306.7
Sep.	283	382	193
Annual average(mm)	2312		

From table (2-4), it appears that the evaporation is at maximum in Jul. and is at minimum in Dec. and Jan. The annual average evaporation from class-A pan is 2312mm; this means that evaporation during one year from any water body in the area is that vertical height of water which is equal to 2312 mm. If one knows the surface area of the water body, so the total volume of the evaporated water can be calculated.

2-6-4 The Discharge of Basara Stream

Two permanent streams (Tainal “Bazian” and Tilie streams) continuously recharge the Basara stream because they receive water from hundreds of springs, some of them have a discharge more than 200 L/sec as mentioned by Aziz (2005) for springs in Bazian basin (main part of Basara catchment area).

The total amount of discharge of Basara stream which results from converging of Tainal (Bazian) and Tilie streams was 131.123 x 10^6 m^3/year in 2002, with the average discharge equals to 4.157883 m^3/sec, table (2-5).

The total amount of discharge (131.123 x 10^6 m^3/year) comes from surface runoff and ground water discharge. That part of rainfall which becomes a surface runoff can be calculated roughly according to Danderkar and Sharm (1989) equation as below:

R= (P-17.8) x P/254

Where: R=Surface runoff in cm.

P=Annual rainfall in cm.

R= (74.1 cm-17.8 cm) x 74.1cm/254 cm

R =16.42 cm=164.2mm=0.1642m

The catchment area of the Basara stream at the study area equals 572.8 km^2 (=572.8 x $10^6 m^2$), Fig (2-27), so the total amount of water from surface runoff equals 94.053 x 106m^3/ year (=0.1642m x 572.8 x $10^6 m^2$).

From the above calculation, it appears that the amount of water which comes from groundwater discharge of Tainal (Bazian) and Tilie streams is equal to 37.07x10^6 m^3/year (=131.123 x $10^6 m^3$–94.053 x $10^6 m^3$), which is due to the existence of highly permeable rocks of Sinjar, Pila Spi, Injana (its sandstone beds), Muqdadiya(its sandstone and pebbly sandstone beds) and Bai Hassan Formations in most parts of the catchment area except some parts covered by Kolosh Formation. The reservoir area of the proposed dam is composed mostly of Fat'ha and Injana (its claystone beds) Formations have a good ability of the retention of water in the reservoir area.

Reservoir area (Ar) and volume (Vr) curves are presented in Fig (2-28) and (2-29) respectively. Reservoir volume curve shows that the total amount of discharge (131.123*106m3/year) corresponds to an elevation of 730m above sea level, and at this elevation, the area impounded by the reservoir is 6.25km^2 (see Fig“2-28”).

Sediment amount that would enter and be settled in reservoir is estimated to be 612 m^3/km^2/year (ITSC, 2007), which is equal to 350553.6m^3/year comes from the total catchment area and approximately equals to 20 x $10^6 m^3$ during 57 years. This volume (20 x $10^6 m^3$) corresponds to an elevation of 702m above sea level.

Table (2- 5) Monthly discharge of Basara stream in the year 2002 (Stevanovic, et al., 2003)

Months	No. of days	Discharge L/sec	Discharge m^3/sec	Discharge $m^3/month$
Jan.	31	7810	7.810	20918304
Feb.	28	6326	6.326	15303859
Mar.	31	6510	6.510	17436384
Apr.	30	6842	6.842	17734464
May	31	4401	4.401	11787638
Jun.	30	1485	1.485	3849912
Jul.	31	957	0.957	2563228
Aug.	31	854	0.854	2287353
Sep.	30	1072	1.072	2778624
Oct.	31	1580	1.580	4231872
Nov.	30	4004	4.004	10378368
Dec.	31	8159	8.159	21853065
Total discharge ($m^3/year$)				$131.123*10^6$
Average discharge (m^3/sec)				4.157883

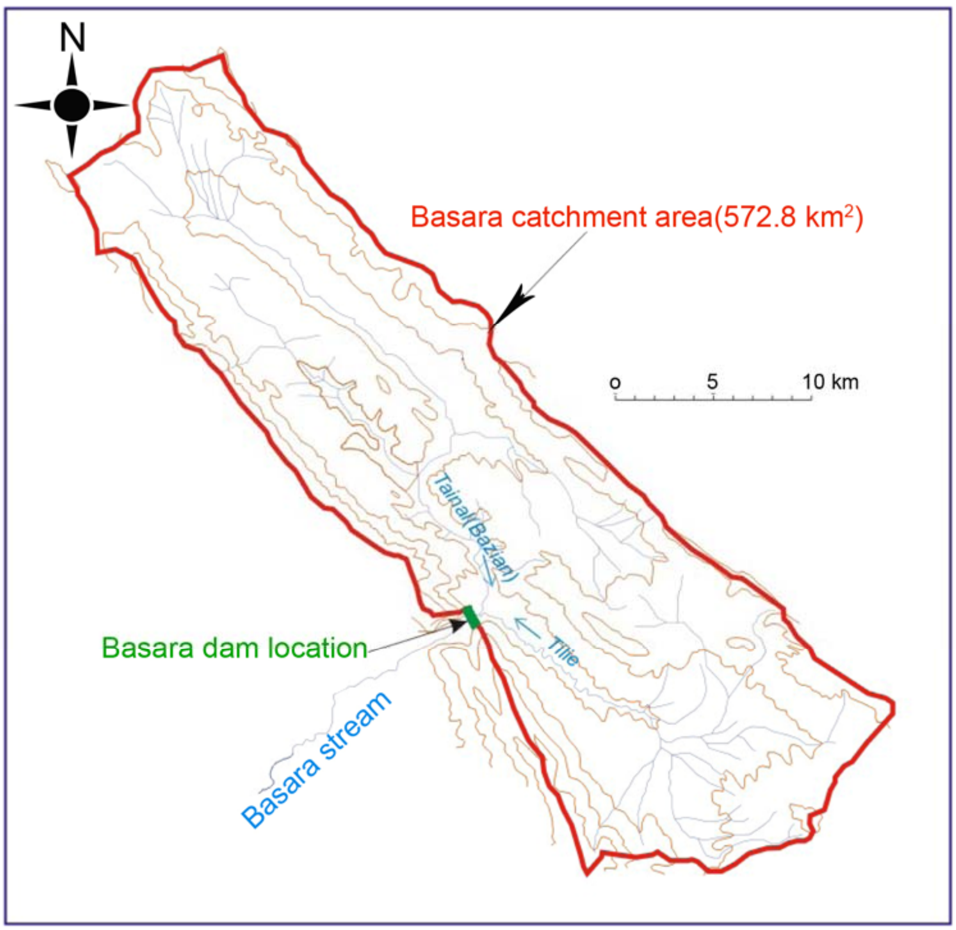
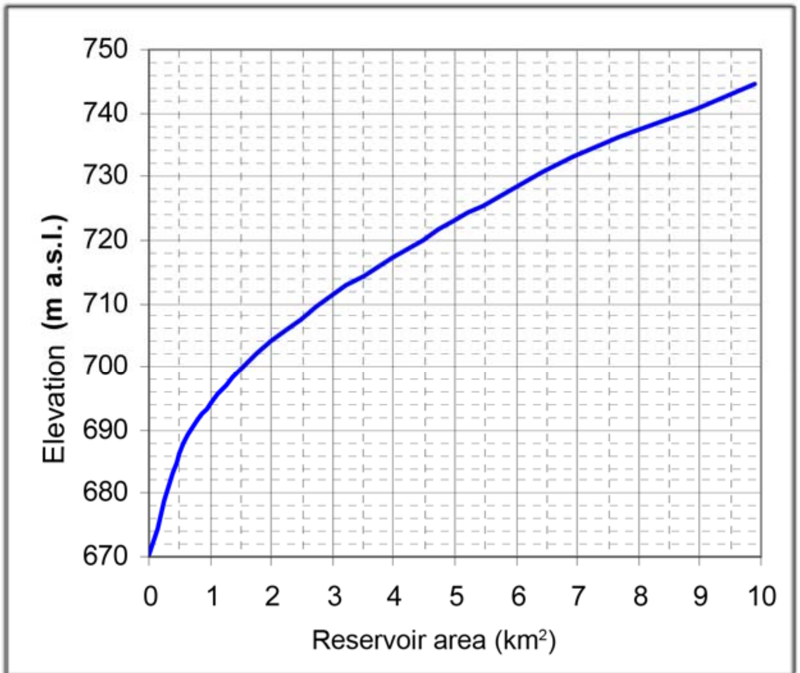
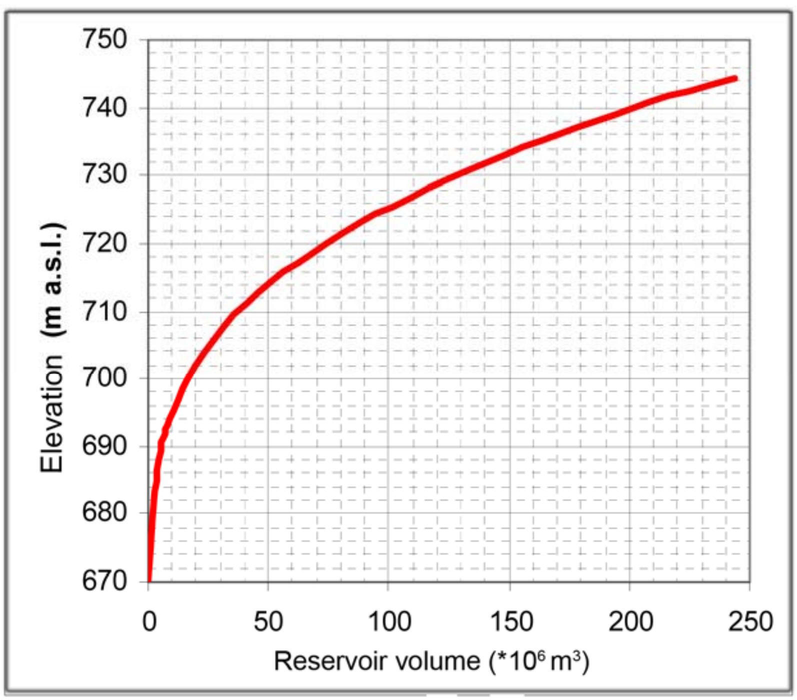


Fig (2-27) Basara catchment area



Fig(2-28) Reservoir area curve (After ITSC, 2007)



Fig(2-29) Reservoir volume curve (After ITSC, 2007)

Chapter Three
Laboratory tests

3-1- Preface:

Laboratory test is an important part in any engineering project. Two types of tests were done on the foundation rocks of the proposed dam site, they are unconfined (uniaxial) compression tjest and point load test. Both tests aim at calculating (finding) the unconfined compressive strength (UCS) of the intact rock which is an important parameter for most of the rock mass classification systems (Hack and Huisman,2002) , such as all versions of RMR system (Bieniawski, 1973, 1974, 1975, 1976 and 1989), Hoek–Brown failure criterion (Hoek and Brown,1980 and 1997), Rock Mass index (RMi) (Palmstrom,1995) and even it was introduced as a parameter in Q-System in 1995 (after 21 years since it was coined in 1974)(Barton,1995).

The importance of UCS is very clear from the above short review of UCS, because one of the main aims of this thesis is an evaluation of rock masses by rock mass classification systems or criterion. Two types of tests were done as follows:

3-2-Unconfined compression test:

The unconfined compression test was conducted in the laboratories of engineering college / Sulaimani University on the intact rocks of Sinjar and Kolosh Formations according to the procedure of ASTM 1979 and 1986 under the D2938 code.

The test is done on core specimens of 54mm diameter (surface sections core specimens) and 90mm diameter (boreholes core specimens).

The UCS has been determined by subjecting each rock specimen to loading at a nearly constant rate in the unconfined testing machine until the rock specimen is failed, Fig (3-1). The UCS of the test specimen is calculated by the following formula:

$$UCS = F/A \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots(3\text{-}1)$$

Where: ‘F’ is the force (load) at failure.

‘A’ is the cross sectional area of the specimen.

Due to the effect of shape (length / diameter) on the value of unconfined compressive strength, the shape correction for the unconfined compressive strength was done through an equation proposed by Thuro et al. (2001) as in Fig (3-2), which shows the shape correction for L/D ratio from 1-3 in the calculation of UCS as follows:

UCS^{*}= UCS (0.925+0.036*L/D) ……….(3-2)

Where: UCS^{*}=Calculated unconfined compressive strength of an equivalent 2:1 Length/Diameter specimen.

UCS=Measured unconfined compressive strength of the specimen tested

L=Test core length (height).

D=Test core diameter.

Also due to the effect of size (core diameter) on the uniaxial compressive strength value, the below equation (Hoek and Brown,1980) is used:

$\sigma_{ci\,(d)} = \sigma_{ci\,(50)}\,(50/d)^{0.18}$ ……………….(3-3)

Where: $\sigma_{ci\,(d)}$ = Unconfined compressive strength of specimen of diameter d.

$\sigma_{ci\,(50)}$ = Unconfined compressive strength of 50mm diameter specimen.

From the above equation, the unconfined compressive strength of 50mm diameter specimen can be calculated, which is the acceptable size in the rock mass classification systems.

The results of the unconfined compression test for core specimens of both surface sections and boreholes are shown in tables (3-1) and (3-2) respectively.

3-3-Point load test

An extensive investigation by Broch and Franklin (1972) proved that this test has a great importance in rock mechanics and engineering geology as an indirect measure of the UCS, and it was regarded as a standard test by the international society of rock mechanics in 1973 (Bieniawski, 1975).

This test is widely used in practice due to easiness of the test, simplicity of specimen preparation and possible field application (Gunsallus and Kulhawy, 1984).

The test is performed according to the procedure of Brook (1985) and ISRM (1985).The point load test involves the compressing of a rock sample between conical steel platens until failure occurs, Fig (3-3), in which the point load test allows the determination of the uncorrected point load strength index (Is),which can be derived from Fig.(3-4) as follows:

$Is=F / (D_e)^2 = \pi F / 4A = \pi F / 4*D*W$ …………..(3-4)

Where: Is = Uncorrected point load strength index

F = Force at failure (breaking load)

D_e=Equivalent core diameter which is given by:

(1) D_e=D for the diametral test, Fig (3-4)

(2) $D_e= \sqrt{(4A) / \pi}$ for the axial, block or irregular lump tests, Fig (3-4), where A=D*W ; A is minimum cross sectional area of a plane through the platen contact points, D is the thickness of specimen and W is width of specimen.

The Is must be corrected to the standard equivalent diameter of 50mm, as follows:

$\left.\begin{array}{l} Is_{(50)} = f*(F/De^2) = f*Is \\ f = (D_e/50)^{0.45} \end{array}\right\}$ ....(3-5) (Brook (1985) and ISRM (1985)

Where: $Is_{(50)}$ = Point load strength index of a specimen of 50mm diameter.

f= Size correction factor.

Early studies by Broch and Franklin (1972) and Bieniawski (1975) were conducted on hard, strong rock and found that the relationship between UCS and the strength index (Is) could be expressed as:

$UCS = K*Is_{(50)} = 24*Is_{(50)}$

Where K is the index - to – strength conversion factor

Other authors (Rusnak and Mark, 2000) found that a conversion factor K=21 worked well for a variety of rock types, and they suggested the below equation for calculation the unconfined compressive strength:

$UCS= 21* Is_{(50)}$.

The results of the point load test for both surface sections and boreholes core samples are shown in table (3-3) and table (3-4).

Note: The UCS of intact rock in some rock mass units were estimated with the aid of table (3-5), due to their low strength, as they are shown in the table (3-6).

Fig (3 – 1) Failure of the core specimen in the unconfined testing machine

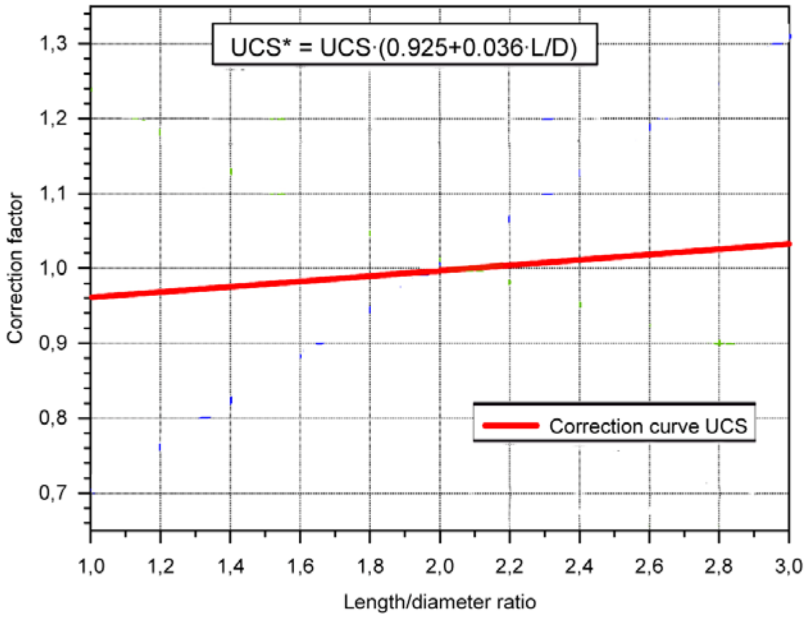


Fig (3-2) Shape correction curve for unconfined compressive strength (After Thuro et al. 2001)

Table (3-1) Results of unconfined compression test for surface sections core specimens of 54mm diameter.

Geologic unit	Sections	Units	Rock type	Core Length (L)(mm)	Core Diameter (D)(mm)	Radius (r) (cm)	r^2 (cm^2)	L/D	Area(A) =($r^2\pi$) (cm^2)	Force at failure (F)(KN)	UCS (MPa)	UCS* $б_{ci\ (d)}$ (MPa)	UCS(50) $б_{ci\ (50)}$ (MPa)
Gercus Fn	1	1	Siltstone	108	54	2.7	7.29	2	22.9	41	17.910	17.856	18.105
		1	Sandstone	102	54	2.7	7.29	1.888	22.9	59	25.764	25.582	25.939
Sinjar Fn.	1	2	Micrite	91.0	54	2.7	7.29	1.685	22.9	92	40.17	39.59	40.14
		2	Biomicrite	113	54	2.7	7.29	2.092	22.9	170	74.235	74.25	75.28
		3	Intramicrite	115.1	54	2.7	7.29	2.131	22.9	120	52.40	52.49	53.22
		5	Biomicrite	118.2	54	2.7	7.29	2.188	22.9	170	74.235	74.50	75.54
	2	1	Intramicrite	62.2	54	2.7	7.29	1.151	22.9	150	65.50	63.32	64.20
		2	Micrite	119	54	2.7	7.29	2.203	22.9	115	50.20	50.41	51.11
		2	Intramicrite	55	54	2.7	7.29	1.018	22.9	105	45.85	44.09	44.70
		3	Intramicrite	115.3	54	2.7	7.29	2.135	22.9	140	61.13	61.24	62.09
		3	Intramicrite	105.8	54	2.7	7.29	1.959	22.9	125	54.58	54.33	55.09
		3	Intramicrite	81	54	2.7	7.29	1.500	22.9	110	48.03	47.02	47.67
		4	Biomicrite	85	54	2.7	7.29	1.574	22.9	95	41.48	40.71	41.27
	3	2	Intramicrite	101	54	2.7	7.29	1.870	22.9	210	91.70	90.99	92.26
		2	Intramicrite	95.1	54	2.7	7.29	1.761	22.9	130	56.76	56.10	56.88
		3	Biomicrite	68.6	54	2.7	7.29	1.270	22.9	180	78.60	67.29	77.35
		4	Biomicrite	109.9	54	2.7	7.29	2.035	22.9	195	85.15	85	86.18
		4	Intrabiomicrite	64.3	54	2.7	7.29	1.190	22.9	125	54.58	52.82	53.55
Uncon-formity	3	5	Pelintrabio-sparite	93.1	54	2.7	7.29	1.724	22.9	155	76.68	66.80	67.73

Where: ($A=(r^2\pi)$)=Cross sectional area of the specimen tested

UCS=Measured unconfined compressive strength of the specimen tested

UCS*=Calculated unconfined compressive strength of an equivalent 2:1 Length / Diameter specimen

$б_{ci\ (d)}$=Unconfined compressive strength of specimen of diameter d, with the L / D = 2 / 1

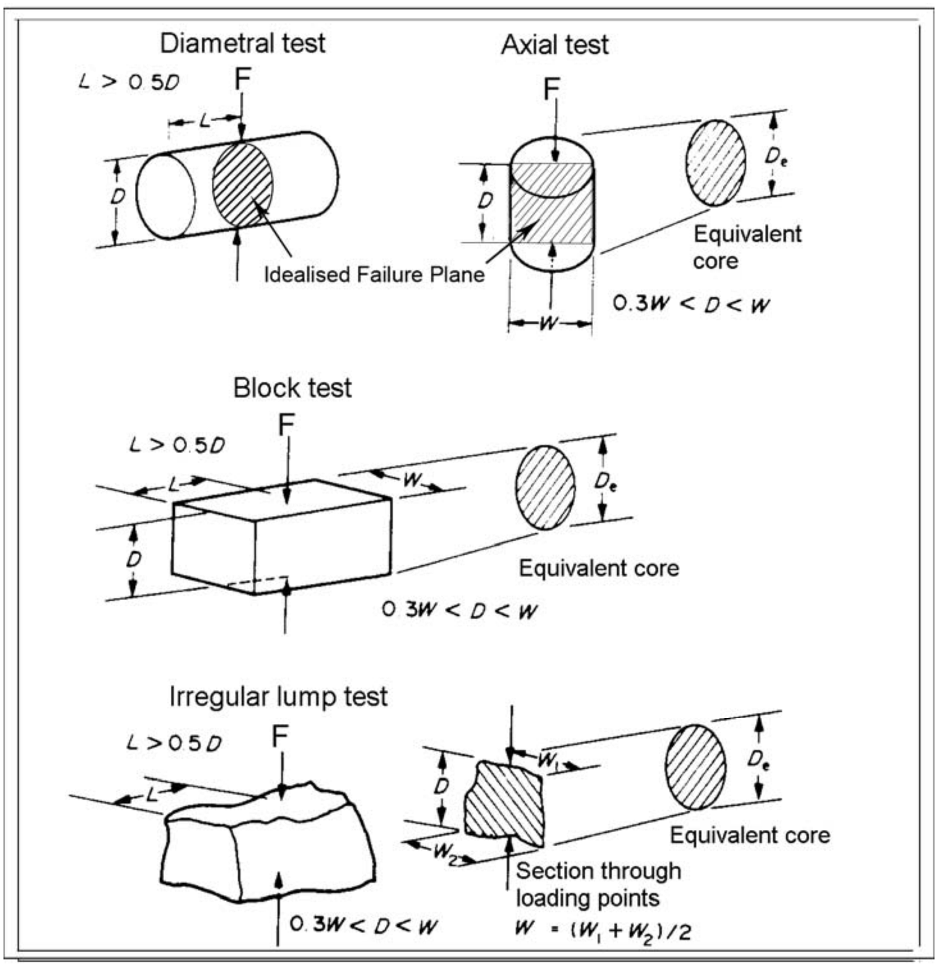
$б_{ci\ (50)}$=Unconfined compressive strength of 50mm diameter specimen

Table (3-2) Results of unconfined compression test for boreholes core specimens of 90mm diameter.

Geologic unit	Bore holes	Units	Depth below sur.(m)	Rock type	Core Length (L)(mm	Core Diameter (D)(mm)	Radius (r) (cm)	r^2 (cm^2)	L/D	Area(A) =($r^2\pi$) (cm^2)	Force at Failure (F)(KN)	UCS (MPa)	UCS* $б_{ci\ (d)}$ (MPa)	UCS(50) $б_{ci\ (50)}$ (MPa)
Kolosh Fn.	1	2	43.40	Biomicrite	201.1	90	4.5	20.25	2.234	63.62	242	38.038	38.244	42.512
		3	48.50	Sandstone	207	90	4.5	20.25	2.300	63.62	177	27.821	28.038	31.167
			50.50	Pebbly Sandstone	198	90	4.5	20.25	2.263	63.62	120	18.862	18.941	21.054
		4	55.20	Pebbly Sandstone	194.7	90	4.5	20.25	2.263	63.62	100	15.710	15.750	17.500
			57.35	Sandstone	198.4	90	4.5	20.25	2.204	63.62	150	23.570	23.670	26.31
	2	1	20.00	Siltstone	189.5	90	4.5	20.25	2.105	63.62	100	15.718	15.730	17.485
			21.10	Sandstone	148	90	4.5	20.25	1.644	63.62	152	23.891	23.513	26.137
		2	22.80	Sandstone	192	90	4.5	20.25	2.133	63.62	163	25.620	25.665	28.529
			25.00	Siltstone	111.6	90	4.5	20.25	1.240	63.62	140	22.005	21.336	23.717
		5	34.60	Siltstone	161.7	90	4.5	20.25	1.796	63.62	170	26.721	26.444	29.395
			35.20	Sandstone	191.4	90	4.5	20.25	2.126	63.62	195	30.650	30.697	34.122
		6	36.45	Biomcrite	186.8	90	4.5	20.25	2.075	63.62	335	52.656	52.640	58.514
		7	40.20	Siltstone	165.2	90	4.5	20.25	1.835	63.62	200	31.436	31.154	34.630
			41.30	Sandstone	107.4	90	4.5	20.25	1.193	63.62	225	35.366	34.232	38.052
	3	2	18.20	Biomicrite	172	90	4.5	20.25	1.911	63.62	298	46.840	46.549	51.744
			22.40	Biomcrite	194.3	90	4.5	20.25	2.158	63.62	315	49.512	49.645	55.185
			26.20	Intrabiomicrite	186.7	90	4.5	20.25	2.074	63.62	305	47.940	47.923	53.271
		3	34.70	Sandstone	155	90	4.5	20.25	1.722	63.62	210	33.008	32.578	36.213
			36.60	Siltyshale	165.5	90	4.5	20.25	1.٨٣٨	63.62	130	20.433	20.252	22.512
			38.30	Siltstone	160	90	4.5	20.25	1.777	63.62	168	26.406	26.114	29.028

Where: (A=($r^2\pi$))=Cross sectional area of the specimen tested
UCS=Measured unconfined compressive strength of the specimen tested
UCS*=Calculated unconfined compressive strength of an equivalent 2:1 Length / Diameter specimen
$б_{ci\ (d)}$=Unconfined compressive strength of specimen of diameter d, with the L / D = 2 / 1
$б_{ci\ (50)}$=Unconfined compressive strength of 50mm diameter specimen

Fig (3 – 3) Failure of the specimen in the point load tester



Fig(3-4) Specimen shape requirements for different test types (After Brook,1985 ; ISRM,1985)

Table (3-3) Results of the point load test for surface sections core specimens:

Geologic unit	Section	Units	Rock type	D (mm)	W (mm)	D*W Cm^2	f (size correc –tion factor)	F (KN)	Is (MPa)	$Is_{(50)}$ (MPa)	UCS(MPa) $UCS=21*Is_{(50)}$
Sinjar Fn.	1	3	Biomicrite	55	54	29.70	1.04	9.8	2.59	2.69	56.49
	2	5	Biomicrite	68	54	36.72	1.15	16.8	3.59	4.12	86.52
			Intrabiomicrite	45	٥٤	24.3	0.95	9.65	3.12	2.96	62.16
	3	1	Micrite	64	54	34.56	1.12	10	2.27	2.54	53.34
			Biomicrite	66	54	35.64	1.13	14	3.08	3.48	73.08
Unconformity	3	6	Pelintrabio - micrite	53	54	28.62	1.02	11.7	3.21	3.27	68.67

Table (3-4) Results of the point load test for boreholes core specimens:

Geologic unit	Bore holes	Units	Depth below surface (m)	Rock type	D (mm)	W (mm)	D*W cm^2	f (size correc –tion factor)	F (KN)	Is MPa	$Is_{(50)}$ MPa	UCS(MPa) $UCS=21.Is_{(50)}$
Kolosh Fn.	2	3	26.20	Sandstone	75	90	67.5	1.20	11.9	1.38	1.65	34.65
			27.90	Siltstone	63	90	56.7	1.11	8.7	1.20	1.33	27.93
		4	29.35	Siltstone	72	90	64.8	1.18	8.7	1.05	1.24	26.04
			31.90	sandstone	80	90	72	1.23	11.4	1.24	1.52	31.92
		6	37	Biomicrite	50	90	45	1	11.5	2	2	44
	3	1	13.95	Sandstone	73	90	65.7	1.18	9.3	1.11	1.32	27.72
			15.10	Siltstone	70	90	63	1.16	7.7	0.960	1.11	23.31

Where: D=Thickness of the specimen (distance between the two loaded points)
W=Width of the specimen (as it is illustrated in Fig (3-4))
D*W=A (Area of idealized failure plane)
F=Force at failure
Is=Point load strength index

Table(3-5) Field estimates of uniaxial compressive strength of intact rock. (Modified from Marinos and Hoek, 2000)

Grade*	Term	Uniaxial Comp. Strength (MPa)	Point Load Index (MPa)	Field estimate of strength	Examples
R6	Extremely Strong	> 250	>10	Specimen can only be chipped with a geological hammer	Fresh basalt, chert, diabase, gneiss, granite, quartzite
R5	Very strong	100 - 250	4 - 10	Specimen requires many blows of a geological hammer to fracture it	Amphibolite, sandstone, basalt, gabbro, gneiss, granodiorite, peridotite , rhyolite, tuff
R4	Strong	50 - 100	2 - 4	Specimen requires more than one blow of a geological hammer to fracture it	Limestone, marble, sandstone, schist
R3	Moderately** strong	25 - 50	1 - 2	Cannot be scraped or peeled with a pocket knife, specimen can be fractured with a single blow from a geological hammer	Concrete, phyllite, schist, siltstone
R2	Weak	5 - 25	***	Can be peeled with a pocket knife with difficulty, shallow indentation made by firm blow with point of a geological hammer	Chalk, claystone, potash, marl, siltstone, shale, rocksalt,
R1	Very weak	1 - 5	***	Crumbles under firm blows with point of a geological hammer, can be peeled by a pocket knife	Highly weathered or altered rock, shale
R0	Extremely Weak	0.25 - 1	***	Indented by thumbnail	Stiff fault gouge

* Grade according to Brown (1981). **Modified from medium.
*** Point load tests on rocks with a uniaxial compressive strength below 25 MPa are likely to yield highly ambiguous results.

Note that this table contains a few changes in the column of examples from previously published versions.

Table (3-6) The unconfined (uniaxial) compressive strength (UCS) for low strength rocks from field estimation

Geologic unit	Surface section	Bore hole	Unit	Rock type	Field estimate of strength	UCS (MPa)
Gercus Fn.	1	-	1	Silty shale	Can be peeled with a pocket knife with difficulty, shallow indentation made by firm blow with point of a geological hammer	7
Sinjar Fn.	1	-	4	Sandy marlstone		10
Kolosh Fn.	-	1	1	Sandstone & Siltstone	Crumbles under firm blows with point of a geological hammer, can be peeled by a pocket knife	5

Chapter Four

Theoretical background

4-1 Preface:

Rock mass classification is a means of assigning a numeric rating to the quality and likely performance of a rock mass, based on easily measurable parameters (Goodman, 1989).

During the feasibility and preliminary design stages of a project when very little detailed information on the rock mass is available, the use of a rock mass classification scheme can be of considerable benefit. This may involve using the classification scheme as a check-list to ensure that all relevant information has been considered. One or more rock mass classification systems can be used to build up a picture of the composition and characteristics of a rock mass to provide initial estimates of support requirements, and to provide estimates of the strength and deformation properties of the rock mass(Hoek et al.,1995).

All the classification schemes consider a few of the key rock mass parameters, and assign numerical values to the classes within which these parameters lie for a given rock type. As it will be shown, the schemes provide a shortcut to the rock mass properties that are more difficult to assess (e.g. the prediction of rock mass deformability and compressive strength) (Hudson and Harrison, 1997).

The Hoek-Brown failure criterion is also a good scheme in estimating the mechanical properties of the rock mass.

This chapter provides a review of the rock mass classification systems and criterion which are used in the evaluation of the rock masses at the proposed dam site.

4-2 Rock Mass Rating (RMR) system

It is also known as geomechanics classification. It was originally proposed by Bieniawski in the year 1973 , for use in tunnels, slopes and foundations. Over the years, this system has been successively evolved and refined, due to a better understanding of the importance of the different parameters and increased experience as more cases have been examined and the reader should be aware that Bieniawski has made significant changes in the rating assigned to different parameters (Hoek et al.,1995). Table (4-1) summarizes the evolution of RMR ratings as well as the modification to the weights assigned to each factor.

Table (4-1) Evolution of RMR ratings (Modified from Milne et al., 1998)

RMR	1973	1974	1975	1976	1989
Compressive strength	10	10	10	15	15
RQD	16	20	20	20	20
Discontinuity spacing	30	30	30	30	20
Ground water	10	10	10	10	15
Conditions of joints	34	30	30	25	30
Discontinuity strike and dip orientation in tunnels	-(3-15)	-(0-15)	-(0-12)	-(0-12)	-(0-12)

By 1989, around 350 case histories (covering 15 years) had been the basis of the system development. As it was recognized by Bieniawski (1989), the system benefited from extensions and modifications by various researchers, and such developments allowed the system to adopt to various engineering applications (Vardakos, 2004).The six parameters that are used to classify a rock mass using the RMR system are as follows:

1-Unconfined (uniaxial) compressive strength of intact rock (rock material)

2-Rock Quality Designation (RQD)

3-Spacing of discontinuities

4-Condition of discontinuities

5-Ground water conditions

6-Orientation of discontinuities

In applying this classification system, the rock mass is divided into a number of geotechnical units (Bieniawski, 1989) and each unit is classified separately.

Table (4-2) provides the most recent version of the RMR system; it gives the rating values for each of the six parameters.

In section A of table (4-2) with the first five parameters of the classification and their rating, each parameter covers a range of values appropriate to that parameter. When assessing a given rock mass, one establishes the rating value of each parameter, and then sums the resulting numerical ratings for the five parameters.

The orientation of the discontinuities becomes progressively more important from tunnels and mines, through foundations, to slopes, sections B and F.

In section E of table (4-2), there are ratings for discontinuity characteristics (conditions).

In sections C and D of the table, the rock mass classes are given a description from ‘very good rock’ through to ‘very poor rock’ with estimates for tunnel stand-up time and the Mohr- Coulomb strength parameters of cohesion and friction angle for the rock mass.

A set of guidelines is given for the selection of support in tunnels in rock for which the value of RMR has been determined. These guidelines are reproduced in table (4-3).

Figures (4-1) and (4-2) present the variation of the ratings for the intact rock strength and RQD, as a continuous function of the physical parameter value, these figures remove the impression that abrupt changes in ratings occur between categories.

Because the Geological Strength Index (GSI) is based on RMR_{76} (Hoek et. al., 1995). Therefore, the rating values for each of the six parameters of the RMR_{76} are shown in table (4-4).

Table(4-2) Rock Mass Rating(RMR) system (After Bieniawski, 1989)

A. CLASSIFICATION –PARAMETERS AND THEIR RATINGS

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

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)

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

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

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

D. MEANING OF ROCK CLASSES

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

E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions

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

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

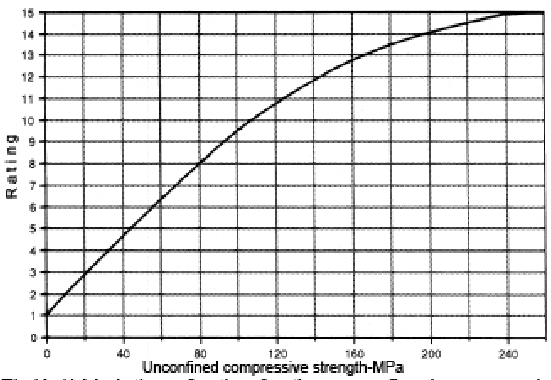
<table>
<tr><th colspan="2">Strike perpendicular to tunnel axis</th><th colspan="2">Strike parallel to tunnel axis</th></tr>
<tr><td>Drive with dip - Dip 45 - 90°</td><td>Drive with dip - Dip 20 - 45°</td><td>Dip 45 - 90°</td><td>Dip 20 - 45°</td></tr>
<tr><td>Very favourable</td><td>Favourable</td><td>Very favourable</td><td>Fair</td></tr>
<tr><td>Drive against –dip - Dip 45-90°</td><td>Drive against –dip - –Dip 20-45°</td><td colspan="2">–Dip 0-20 - Irrespective of strike°</td></tr>
<tr><td>Fair</td><td>Unfavourable</td><td colspan="2">Fair</td></tr>
</table>

* Some conditions are mutually exclusive . –For example, if infilling is present, the roughness of the surface will be overshadowed by the influence of the gouge. –In such cases use A.4 directly.

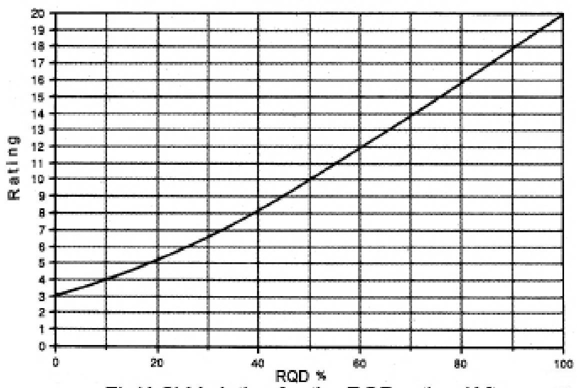
** Modified after Wickham et al (1972).

Table (4-3) Guidelines for excavation and support of 10m span rock tunnels in accordance with the RMR system (After Bieniawski, 1989)

Rock mass class	Excavation	Rock bolts (20 mm diameter, fully grouted)	Shotcrete	Steel sets
I - Very good rock *RMR*: 81-100	Full face, 3 m advance.	Generally no support required except spot bolting.		
II - Good rock *RMR*: 61-80	Full face , 1-1.5 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.
III - Fair rock *RMR*: 41-60	Top heading and bench 1.5-3 m advance in top heading. Commence support after each blast. Complete 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.
IV - 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 to medium ribs spaced 1.5 m where required.
V – Very poor rock *RMR*: < 20	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 forepoling if required. Close invert.



Fig(4-1) Variation of rating for the unconfined compressive strength (After Bieniawski, 1989)



Fig(4-2) Variation for the RQD rating (After Bieniawski, 1989

Table(4-4) Rock Mass Rating (RMR) system (After Bieniawski, 1976)

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	Parameter		Range of values				
1	Strength of intact rock material	Point-load strength index	> 8 MPa	4 - 8 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred
		Uniaxial comp. strength	> 200MPa	100 - 200 MPa	50 - 100 MPa	25 - 50 MPa	10 - 25 MPa / 3-10 MPa / 1-3 MPa
	Rating		15	12	7	4	2 / 1 / 0
2	Drill core Quality *RQD*		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%
	Rating		20	17	13	8	3
3	Spacing of joints		> 3 m	1-3 m	0.3-1 m	50-300 mm	< 50 mm
	Rating		30	25	20	10	5
4	Condition of joints		Very rough surfaces Not continuous No separation Hard joint wall contact	Slightly rough surfaces Separation < 1 mm Hard joint wall contact	Slightly rough surfaces Separation < 1 mm Soft joint wall contact	Slickensided surfaces or Gouge < 5 mm thick or Joints open 1-5mm Continuous joints	Soft gouge >5 mm thick or Joints open >5 mm Continuous joints
	Rating		25	20	12	6	0
5	Ground water	Inflow per 10 m tunnel length (l/m)	None	< 25	25 - 125	> 125	
		(Joint water press)/(Major principal σ)	0	0.0 - 0.2	0.2 - 0.5	> 0.5	
		General conditions	Completely dry	Moist only	Water under mod. pressure	Severe water problems	
	Rating		10	7	4	0	

B. RATING ADJUSTMENT FOR JOINTS ORIENTATIONS

Strike and dip orientations		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
Ratings	Tunnels	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	- 60

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

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

D. MEANING OF ROCK CLASSES

Class number	I	II	III	IV	V
Average stand-up time	10 yrs for 5m span	6 yrs for 4m span	1 week for 3m span	5 hrs for 1.5m span	10 min for 0.5m span
Cohesion of rock mass (kPa)	> 300	200 - 300	150 - 200	100 - 150	< 100
Friction angle of rock mass (deg)	> 45	40 - 45	35 - 40	30 - 35	< 30

The six parameters which are used in the RMR system can be evaluated as follows:

1-Unconfined (uniaxial) compressive strength of the intact rock

It can be evaluated indirectly by means of the point load test and by correlations with the Schmidt hammer rebound value, or directly by unconfined compression test, or estimated from table (3-5) when laboratory tests are not possible (in this thesis, values from unconfined compression test, point load test and from table (3-6) were reported in chapter three).

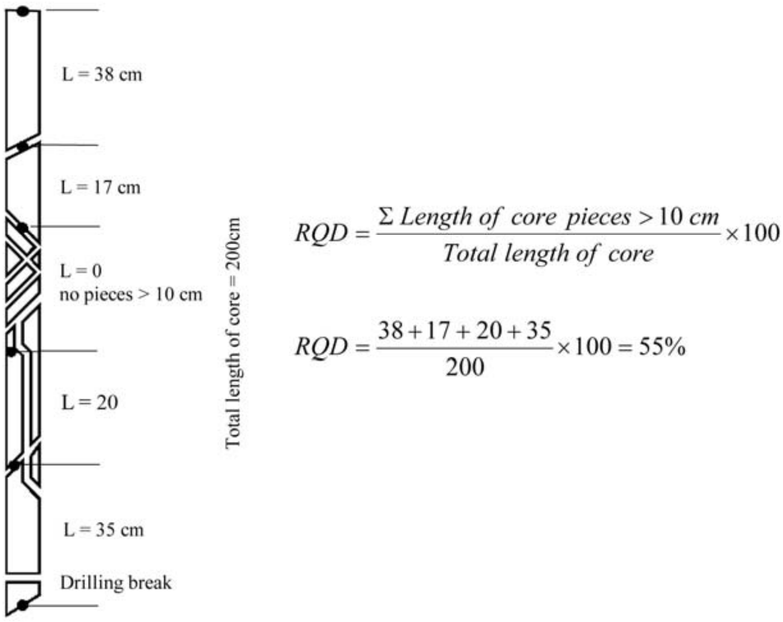
2- Rock Quality Designation (RQD)

RQD was developed by Deere et al. (1967) to provide a quantitative estimate of rock mass quality from drill core logs. RQD is defined as the percentage of intact core pieces longer than 100mm in the total length of core (Deere, 1989):

RQD= (∑ Length of core pieces > 10 cm)/(Total length of core)* 100 -----(4-1)

The core should be at least of diameter 54.7mm or larger and should be drilled with double-tube core barrel (Deere, 1989; Milne et al., 1998; Palmstrom, 2005).

The correct procedures for the measurement of the length of core pieces and the calculation of RQD are summarized in Fig (4-3).



Fig(4-3) Procedure for the measurement and the calculation of RQD (After Deere, 1989)

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

RQD=115 - 3.3Jv ……… (4-2)

In another attempt, a new relationship between RQD and Jv was suggested by Palmstrom (2005) as follows:

RQD=110 - 2.5 Jv …….(4-3)

Where Jv is known as the volumetric joint count and is defined as the number of joints intersecting a volume of one m^3. Where the jointing occurs mainly as joint sets the following equation can be used:

$$Jv=1/S1+1/S2+1/S3+............1/Sn..........(4-4)$$

Where S1, S2 and S3 are the average spacing in meters for the joint sets.

Random joints are not included in a particular joint set. As they may present a significant part of the total number of discontinuities (Palmstrom, 1982; 1996a and 2005).

Neglecting the random joints would lead to erroneous quantification of the discontinuity nature of rock mass (Grenon and Hadjigeorgiou, 2003).Therefore Palmstorm has presented an approximate rule of thumb correction for this with a spacing of 5m for each random joint (Palmstrom, 2005):

$$Jv=1/S1+1/S2+1/S3+..........1/Sn+Nr/5..... (4-5)$$

Where Nr is the number of random joints.

2-1 Calculation of the Jv from wJd (weighted joint density)

In addition to surface observations, the Jv can be measured from borehole (drill core) (Palmstrom 1995, 1996b & 2005). This measurement which is called weighted joint density (wJd) applies an adjustment value for the orientation of the joints relative to the surface or the drill core.

The weighted joint density method offers a relatively quick and simple way to measure the joint density. It reduces the inaccuracy caused by the attitude of joints and thus leads to a better characterization of the rock mass. This may in turn lead to a reduction in the number of boreholes required for investigations(Palmstrom 1995, 1996b & 2005).

In principle the weighted joint density method is based on measuring the angle (δ) between each joint and the horizontal surface or the borehole axis, as shown in Fig. (4-4).

The same angle intervals and rating of *fi* were selected for both the surface and the borehole registrations. Various rating of *fi* (1/sinδ) and various limits of the angle intervals has been given in table (4-5), each joint is given a rating *fi* depending on the actual angle interval. The definition of the wJd is then:

-For two-dimensional measurement in rock surfaces:

wJd=1/√A Σ1/sinδ =1/√A Σ n x *fi* =(1/√A) Nw ……(4-6)

-For one-dimensional measurements along boreholes:

wJd=1/L Σ1/sinδ =1/L Σ n x *fi* =(1/L) Nw ………..(4-7)

Where: A=Size of the observation area. L=Length of section measured in the borehole. n=Number of joints within each interval. Nw=Number of weighted joints.

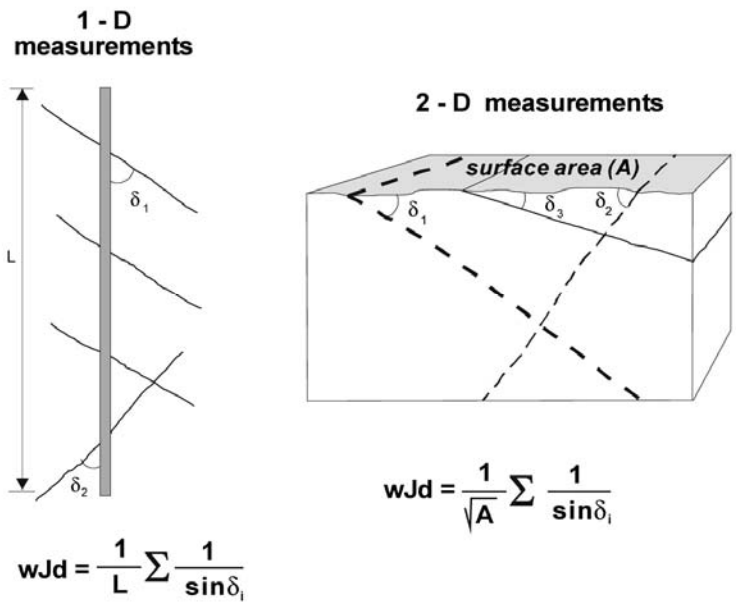


Fig (4-4) The intersection between joints and a drill core hole (left) and a surface (right) (After Palmstrom, 1995)

Table 4-5 Angle intervals and ratings of the factor *fi* (After Palmstrom, 1996b)

Angle interval (between joint and borehole axis or horizontal surface)	Rating of the factor *fi*
>60°	1
31°-60°	1.5
16°-30°	3.5
< 16°	6

An example from core logging is shown in Fig. (4-5). The 5m long part of the core has been divided into the following three sections with similar density of joints: 50- 52.17m, 52.17- 53.15m and 53.15- 55m. For each section the number of joints within each angle interval has been counted (Palmstrom, 1996b) and the results are shown in table (4-6).

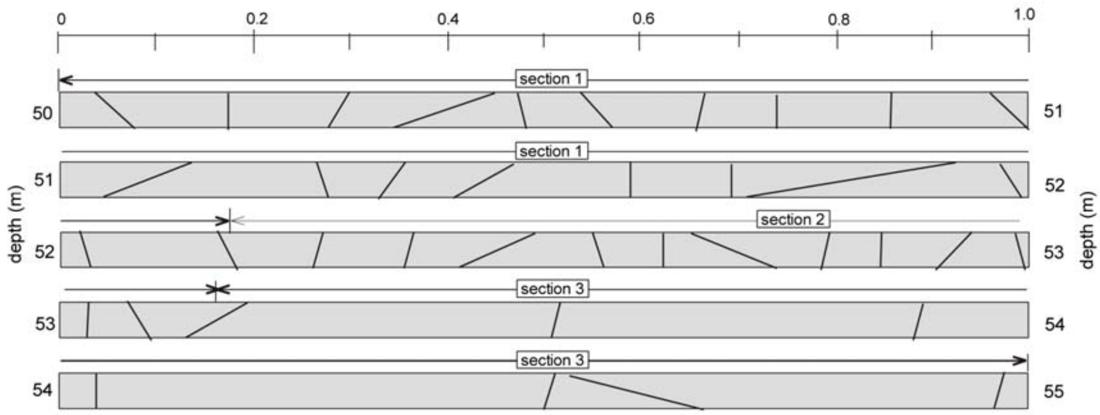


Fig (4-5) Example of jointing along part of a borehole (After Palmstrom, 1996b)

Table (4-6)The calculation of the weighted joint density from registration of jointing in the borehole in Fig.(4-5) (After Palmstrom,1996b)

Depth	Length L	Number of joints(n) with each interval				Total number of joints	Number of weighted joints	wJd=(1/L)Nw
m	m	>60°	31-60	16-30	<16°	From Fig(4-5)	Nw=Σ n* *fi*	
50-52.17	2.17	11	6	2	1	20	33	15
52.17-53.15	0.98	9	3	2	0	14	20.5	20.9
53.15- 55	1.85	5	0	1	0	6	8.5	4.6
Rating of *fi* =		1	1.5	3.5	6			

Simulation has been used to select appropriate ratings based on a comparison with the volumetric joint count (Jv). Therefore, where accurate measurements have been performed, wJd should be similar to the volumetric joint count (wJd ≈ Jv).

3- Spacing of discontinuities

It is the perpendicular distance between adjacent discontinuities (ISRM, 1978), and it is evaluated from scan line data in surface exposure or drill core. The rock mass rating for discontinuities spacing increases as the spacing of joints increase.

It is widely accepted that spacing of joints is of great importance in appraising a rock mass structure. The very presence of joints reduces the strength of a rock mass and their spacing governs the degree of such a reduction (Bieniawski,1973)

When one distinct joint set occurs, it is easy to measure the spacing, but when more than one joint set occur, or more complicated jointing patterns exist, Beiniawski (1973) did not indicate how to calculate the spacing in such cases. According to Edelbro (2003) the lowest rating should be considered if there is more than one joint set and the spacing of joints varies.

4- Condition of discontinuities

Discontinuity condition is examined with respect to the discontinuity sets most likely to influence the rock. In general, the descriptions of discontinuity surface roughness and coating materials are weighed towards the smoothest weakest discontinuity set.

5- Groundwater condition

Groundwater can strongly influence rock mass behavior. The groundwater rating varies according to the conditions encountered (dry, damp, wet, dripping or flowing), with a higher rating for a drier rock mass (Bieniawski, 1989).

6- Orientation of discontinuities

The orientation of discontinuities relative to an excavated face can have an influence on the behavior of the rock mass. For this reason, Bieniawski recommends adjusting the sum of the first five rating numbers to account for favorable or unfavorable orientations. The final RMR value is determined as the sum of the ratings from the six categories (Bieniawski,1989).

4-2-1 Link between RMR system and rock mass properties

Since the rock mass properties, e.g. strength and deformability, are functions of the intact rock properties and the discontinuity properties, it follows that one may be able to use the classification system value (RMR value) to estimate the strength and modulus of rock masses.

These rock mass properties are necessary to be estimated approximately in the preliminary stages of rock engineering design.

Several authors have published empirical estimates of the rock mass strength, based on RMR system as follows:

$б_{cm}$ / $б_{ci}$ = $\sqrt{}$(exp ((RMR-100)/9)) ………..(4-8) (Hoek and Brown, 1988)

$б_{cm}$ / $б_{ci}$ = exp((RMR-100)/24)…..(4-9)(Kalamaras and Bieniawski,1995)

Where: $б_{cm}$ =Unconfined compressive strength of the rock mass.

$б_{ci}$ = Unconfined compressive strength of the intact rock.

Based on extensive experimental results in uniaxial compression on jointed rocks and rock like materials, compressive strength in the unconfined case is given by:

$б_{cm}$ / $б_{ci}$ = exp(-0.008*Jf*)…………..(4-10)(Ramamurthy, 2001&2003)

Where: *Jf* =Joint factor (values of *Jf* varying from 0 to beyond 500, i.e. from Intact "0" to heavily fractured rock "500").

The joint factor considers joint frequency, inclination and strength of critical joint. The unconfined compressive strength calculated from equation (4-10) agree closely with the values from equation (4-9).This is mainly because of the following relation, considering *Jf* =500 as a maximum value for practical purpose.

Jf / 5=100-RMR……………..(4-11)

Since *Jf* =0 and RMR =100 for intact rock. By inserting RMR in place of *Jf*, equation (4-10) will result:

$б_{cm}$ / $б_{ci}$=exp ((RMR-100)/25)……..(4-12)(Ramamurthy, 2001&2003)

This is close to equation (4-9).

These empirical equations for estimation of the rock mass strength were derived from the Hoek-Brown failure criterion, whose material constants (m_b and s) can be estimated from the 1976 version of Bieniawski's Rock Mass Rating (RMR), assuming completely dry conditions and a very favorable joint orientation. An explanation about these relationships is in the Hoek-Brown failure criterion section.

All in situ measurements of the modulus of deformation used today are time – consuming, expensive, imply operational difficulties, and the reliability of the results of these tests are sometimes questionable (Palmstrom and Singh, 2001; Hoek and Diederichs, 2006). Because of this, the deformation modulus is often estimated indirectly from classification systems (Palmstrom and Singh,2001).

There are empirical equations for indirect estimation of the deformation modulus of the rock mass (Em) as follows:

Em=(2RMR-100)GPa (for RMR>50)….(4-13)(Bieniawski,1978)

Em=$10^{(RMR-10)/40}$GPa (for RMR<50)…(4-14)(Serafim and Pererira,1983)

Fig (4-6) shows both Bieniawski and Serafim & Pereira correlations.In practice, most of engineers follow procedures similar to the guidelines of USA Federal Energy Regulatory commission (Romana, 2003a):

“for RMR>58 use Bieniawski formula; for RMR<58 use Serafim-Pereira one”. The RMR=58 value appears to have been selected because it is the abscissa of the lower intersection between both curves.

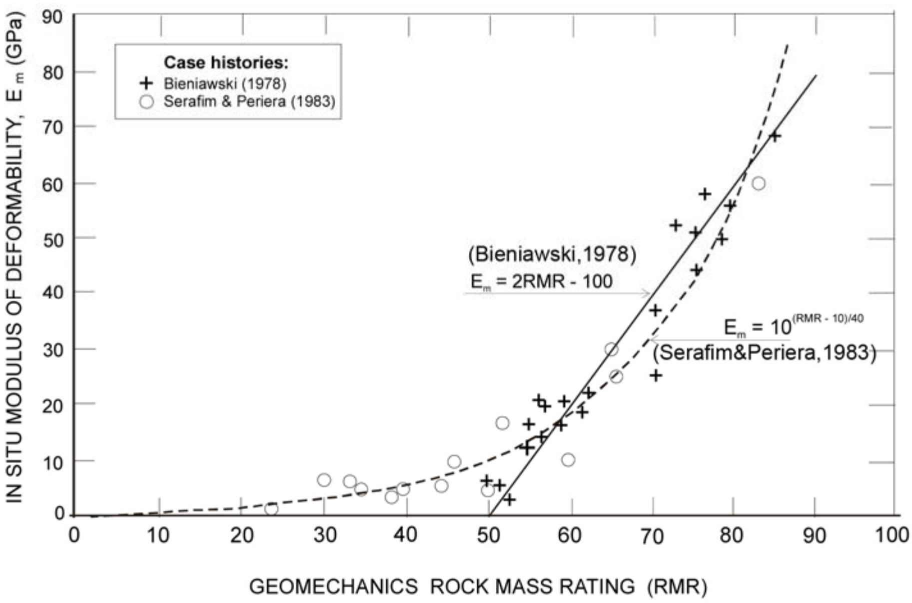


Fig (4-6) Correlation between the in situ modulus of deformation and the RMR system (After Serafim and Pereira, 1983)

Another equation for indirect estimation of deformation modulus of the rock mass was proposed by Ramamurthy (2003) as follows:

Em/ Ei = exp ((RMR-100)/17.4)……… (4-15)

Where: Ei=deformation modulus (modulus of elasticity) of intact rock.

And: Ei=MR σ_{ci}…………..(4-16)

Where: MR=modulus ratio of intact rock.

This relationship is useful when no direct values of the intact modulus (Ei) are available.

The average values of modulus ratio given in table (4-7) can be used for calculating the intact rock modulus (Ei). In general, measured values of Ei are seldom available and even when they are; their reliability is suspected because of specimen damage (micro-cracking) due to stress relaxation and blasting, even for invisibly intact rock. This specimen damage has a greater impact on modulus than on strength and, hence, the intact rock strength when available, can usually be considered more reliable (Hoek and Diederichs, 2006).

Table (4-7) Guidelines for the selection of modulus ratio(MR) values, based on Deere,1968 and Palmstrom and Singh, 2001 (After Hoek and Diederichs, 2006)

Rock type	Class	Group	Texture			
			Coarse	Medium	Fine	Very fine
Sedimentary	Clastic		Conglomerates 300–400 Breccias 230–350	Sandstones 200–350	Siltstones 350–400 Greywackes 350	Claystones 200–300 Shales 150–250[a] Marls 150–200
	Non-clastic	Carbonates	Crystalline limestones 400–600	Sparitic limestones 600–800	Micritic Limestones 800–1000	Dolomites 350–500
		Evaporites		Gypsum (350)[b]	Anhydrite (350)[b]	
		Organic				Chalk 1000+
Metamorphic	Non-foliated		Marble 700–1000	Hornfels 400–700 Metasandstone 200–300	Quartzites 300–450	
	Slightly foliated		Migmatite 350–400	Amphibolites 400–500	Gneiss 300–750[a]	
	Foliated[a]			Schists 250–1100[a]	Phyllites/Mica Schist 300–800[a]	Slates 400–600[a]
Igneous	Plutonic	Light	Granite[c] 300–550 Granodiorite[c] 400–450	Diorite[c] 300–350		
		Dark	Gabbro 400–500 Norite 350–400	Dolerite 300–400		
	Hypabyssal		Porphyries (400)[b]		Diabase 300–350	Peridotite 250–300
	Volcanic	Lava		Rhyolite 300–500 Andesite 300–500	Dacite 350–450 Basalt 250–450	
		Pyroclastic	Agglomerate 400–600	Volcanic breccia (500)[b]	Tuff 200–400	

[a]Highly anisotropic rocks: the value of MR will be significantly different if normal strain and/or loading occurs parallel (high MR) or perpendicular (low MR) to a weakness plane. Uniaxial test loading direction should be equivalent to field application.
[b]No data available, estimated on the basis of geological logic.
[c]Felsic Granitoids: coarse grained or altered (high MR), fined grained (low MR).

4-3 Dam Mass Rating (DMR) system:

Dam Mass Rating is a new geomechanics classification. It was originally proposed by Romana for use in dam foundation, as an adaptation of RMR, due to the difficult effective use of RMR for dam foundation (Romana, 2003a). (It is worth mentioning that most topics of section 4-3 are quoted from Romana '2003a, 2003b and 2004' except some topics of other authors which are referred to).

Needs of terrain strength and deformability quantification are quite different for each type of dam: arch, gravity (CVC, RCC or hardfill concrete), rockfill (CFRD, AFRD), earthfill. As a rule of thumb concrete dams (and the face of CFRD / AFRD) require rock foundations whereas fill dams can be found in soil.

Difficulties in RMR use for dam foundations arise from following points: (1)consideration of the water pressure is very doubtful (the pore pressure ratio varies along the dam foundation), (2)there are no good rules for quantifying the adjusting factor for the joint orientation.

4-3-1 Influence of water on basic RMR

It is common to define a "basic" RMR_B as the addition of the first five RMR parameters without any adjusting factor for joint orientation. The fifth parameter, water rating (WR), is related to water, with a weight on RMR_B up to 15 points (15٪ of the maximal total)(Romana, 2003a ; 2003b & 2004).

The best method to determine the effect of water on the water rating parameter is the use of the water pressure ratio (r_u):

$(r_u) = u/σ_v$ ……….(4-17)

Where: u is the water pressure and $σ_v$ the total vertical stress

The water rating can be approximated by the following formula:

WR = 10 *log* (1/ r_u) - 1.5 (valid for 0.02< r_u<0.7)…….(4-18)

This parameter (WR) can be applied in RMR determination, according to table (4-8).

Table (4-8) Relationship between WR and r_u (After Romana, 2003a)

WR	15	10	7	4	0
r_u(Bieniawski)	0	0-0.1	0.1-0.2	0.2-0.5	> 0.5
r_u (Formula 5-18)	0-0.02	0.07	0.14	0.28	0.7

Around the dam r_u changes in every point depending on the valley geometry, the water level, and the efficiency of the grouting curtains (if exist). Anyway, $r_u > 0.4$ for almost all the upstream points, so the WR parameter would get values of less than 2.5.

Furthermore, the compressive strength of the rock will diminish when saturated. So a very crude way of taking account of the water effect on Em would be to subtract around 15-20 points of the dry values of RMR. If the serafim- Pereira formula [Em (GPa) = $10^{(RMR-10)/40}$] is accepted for determination of Em from RMR, the value of Em(dry) would be approximately three times the value of Em (saturated), for 10 < RMR < 70 . This result is not consistent with published data, which allow for a reduction on the order of 40 % for Em when saturated. Therefore, a rule of thumb could be to subtract 10 points of RMR (dry) (RMR dry obtained with the maximal rating of the water parameter) to obtain Em (saturated). It is interesting to note that this is congruent with prior versions of RMR(before the 1989 version which actually has become the standard one), and it is also the preferred method in Hoek's GSI index practice.

Anyway, it seems that the water consideration is a serious handicap not only for the accurate determination of Em by correlations with RMR, but also for the use of RMR itself in dams.

Then, it will be defined a "Basic dry RMR": RMR_{BD} as the addition of the first four parameters (compressive strength of intact rock, RQD of the rock mass, spacing and condition of the significant governing discontinuity) of RMR plus 15.

4-3-2 Stability of dams against sliding

Table (4-9) shows tentative adjusting factors for the effect of the main discontinuities orientation in horizontal stability. The numerical rating values proposed originally by Bieniawski have been retained.

When the dip direction of the significant joint is not almost parallel to the downstream- upstream axis of the dam, the danger of sliding diminishes due to the geometrical difficulties to slide. It is possible to take account of this effect multiplying the rating of the adjusting factor for dam stability R_{STA} , by a geometric correction factor CF:

Table (4-9) Adjusting factors for the dam stability R_{STA} , according to joints orientation (After Romana, 2003a)

Type of dam	VF Very favorable	F Favorable	FA Fair	U Unfavorable	VU Very unfavorable
Fill	Others	10-30 DS	0-10 A	-	-
Gravity	10-60 DS	30-60 US 60-90 A	10-30 US	0-10 A	-
Arch	30-60 DS	10-30 DS	30-60 US 60-90 A	10-30 US	0-10 A
R_{STA}	0	-2	-7	-15	-25

DS dip downstream/ US dip upstream/ A any dip direction, Gravity dams include CVC (Conventional vibrated concrete) and RCC (Roller compacted concrete), and hardfill concrete dams.

$$CF = (1- \sin|\alpha_d - \alpha_j|)^2 \quad \text{.................(4-19)}$$

Where α_d is the direction upstream – downstream of the dam axis and α_j is the dip direction of the significant governing joint. The value of DMR_{STA} (related to the dam stability against sliding) is:

$$DMR_{STA} = RMR_{BD} + CF * R_{STA} \quad \text{.........(4-20)}$$

Where RMR_{BD} (“basic dry RMR”) is the addition of the first four parameters of RMR plus a water rating of 15 and R_{STA} is the adjusting factor for dam stability as in Table (4-9).

Actually, there are no data allowing to establish a correlation between the value of DMR_{STA} and the degree of safety of the dam against sliding. As a rule of thumb, it can suggest:

$DMR_{STA} > 60$ No primary concern

$60 > DMR_{STA} > 30$ Concern

$DMR_{STA} < 30$ Serious concern

These can not be taken, at all, as numerical statements, but only as danger signals for the designer.

4-3-3 Guidelines for excavation and consolidation grouting of dam foundations

The most usual requirement for the quality of the rock foundation for a concrete dam was something as "good quality, sound rock, fresh, not weathered", Sharma(1998)(In Romana, 2003a, 2003b, 2004) is more specifically demanding that "*the entire (foundation) area should be stripped to firm rock capable of withstanding the loads*. *Any layer of weak or soft material has to be excavated and replaced with concrete*". He prescribes dental concrete treatment filling with concrete any open (or filled with soft fill) joint.

In most cases, the foundation is excavated until class II rock in the central part of the valley (where the dam is higher) and until class II-III rock in the abutments. Spillways are founded, if possible, in class I rock.

It is desirable to gather data on the RMR value of dam foundations. Actually, some simple guidelines can be tentatively proposed, as in table (4-10), for the depth of foundation excavation and for the required consolidation grouting of some few meters deep below the surface of foundation excavation.

Table (4-10) Tentative guidelines for dam foundation excavation and consolidation grouting (Romana, 2003a)

Type of dam	Excavate to $RMR_{BD(+)}$	Consolidation Grouting According to RMR_{BD}		
		Systematic	Spot	None
Earth	-	-	?	-
Rockfill	>20(>30)	20-30	30-50	> 50
Gravity	>40(>60)	40-50	50-60	> 60
Arch	>50(>70)	50-60	60-70	> 70

(+) minimum (desirable)

- gravity dams include CVC, RCC and hard fill concrete.

- rockfill dams included are the ones sensible to settlement (with concrete –CFRD- or asphaltic -AFRD - face upstream).

4-3-4 Influence of the foundation deformability on dam behavior

There is a fact that two cases are dangerous for the normal behavior of a concrete dam: if E_m varies widely across dam foundation, or if E_c/ E_m reaches certain values (E_c being the deformation modulus of concrete). Rocha (1964) (In Romana, 2003a) established the most followed rule for arch dams, table (4-11) in a paper which has become a "classic" reference for dam designers.

Table (4-11) Effect of E_c/E_m on arch dam behavior

(Modified from Rocha,1964)(After Romana, 2003a)

E_c/E_m	Influence on dam	Problems
< 1	Negligible	None
1-4	Low importance	None
4-8	Important	Some
8-16	Very important	Serious
> 16	Special measures	Very dangerous

(Note: The third column had been added by "Romana, 2003a")

$E_c/E_m < 4$ allows for an easy behavior. The minimal sure (but with problems) value of E_m for an arch dam would be around 5GPa. The reported cases of arch dams founded in rock masses with $E_m < 5$GPa show serious problems (cracking included) because of the low value of E_m .

Rocha (1975 and 1976)(In Romana,2003a,2003b and 2004) extended his work to gravity dams. $E_c/E_m < 8$ would be safe and $E_c > 16$ would get to moderate to big problems as in table (4-12). The existence of joints in concrete dams helps to cope with relative deformability problems. This may be the main reason in the changes in the design of RCC concrete dams, from the first dam with almost no joints to the actual standards. Nevertheless, RCC concrete gravity dams are less prone to problems than CVC concrete dams due to the lesser value of $E_{c.}$

Table (4-12) Effect of E_c/E_m on gravity dam behavior

(Modified from Rocha,1975 & 1976)(After Romana, 2003a)

E_c/E_m	Influence on dam	Problems
< 1	Negligible	None
1-4	Negligible	None
4-8	Low important	None
8-16	Important	Some
> 16	Very important	Moderate - Big

4-3-5 Guidelines for DMR_{DEF}

Zeballos and Soriano (1993) (In Romana, 2003a, 2003b and 2004) have published the results of Zeballos Ph.D. thesis, an extensive and intensive study on the effects E_c/ E_m value on gravity and arch dams. Table (4-13) (gathered using their data and others) shows the different ranges of DMR_{DEF} related to the different ranges of possible problems in the dam due to the differences of deformability between the dam and its foundation.

DMR_{DEF}(RMR related to deformability by the serafim and Pereira formula) depends on E_m (when the rock mass is saturated) and can be estimated with WR=5 (a mean value which corresponds to a nominal mean value of r_u= 0.25).

Table (4-13) deformability problems in concrete dams according to the value of DMR_{DEF} (modified from Romana 2003a)(Romana , 2004)

DAM E_C (GPa)	Height (m)	Normal	Problems	Serious Problems
Arch 36 GPa	< 100 100-150 150-200	>50 > 65 >75	40-50 50-65 60-75	<40 <50 <60
Gravity CVC 30GPa	< 50 50-100 100-150	> 40 > 50 >60	25-40 40-50 50-60	<25 <40 <50
Gravity RCC 20 GPa	< 50 50-100 > 100	>35 >45 >55	20-35 35-45 45-55	<20 <35 <45
Hardfill 10 GPa	< 50 50-100	>30 >40	15-30 30-40	<15 <30

4-4 Geological Strength Index (GSI)

The strength of a jointed rock mass depends on the properties of the intact rock pieces and also upon the freedom of these pieces to slide and rotate under different stress conditions. This freedom is controlled by the geometrical shape of the intact rock pieces as well as the condition of the surfaces separating the pieces. Angular rock pieces with clean, rough discontinuity surfaces will result in a much stronger rock mass than one which contains rounded particles surrounded by weathered and altered material (Hoek and Brown, 1997; Vulcanhammer. Net, 1997).

4-4-1 History of the GSI

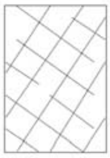
The Geological strength Index (GSI), introduced by Hoek (1994), Hoek et al. (1995) and Hoek and Brown (1997) to overcome the deficiencies in Bieniawski's RMR for very poor quality rock masses. This system provides a system for estimating the reduction in rock mass strength for different geological conditions as identified by field observations. The rock mass characterization is based upon the visual impression of the rock structure, in terms of blockiness, and the surface condition of the discontinuities indicated by joint roughness and alteration.

This system is presented in tables (4-14) and (4-15). Experience has shown that table (4-14) is sufficient for field observation since it is only necessary to note the letter code (resulted from the combination of rock structure and joint conditions) which identifies each rock mass category. These codes can then be used to estimate the GSI value from the contours given in table (4-15).

This system (GSI) was expanded as experience was gained on its application to practical rock engineering problems, such as its use to account for foliated, laminated or sheared weak rocks (highly heterogeneous rock masses) in the lower range of its applicability as shown in table(4-16) (Hoek et al., 1998). Then a new row was added to GSI chart to include an intact or massive rock mass structure in the upper range of its applicability as shown in table (4-16) (Hoek, 1999; Marinos and Hoek, 2000).

Extension of GSI was proceeded to include its application for heterogeneous rock masses such as Flysch (Marinos and Hoek, 2000 & 2001), as shown in table (4-17), and also applied to Molasses sedimentary rocks (Hoek et al.,2005) as in tables (4-18).

Table(4-14) Characterization of rock masses on the basis of interlocking and joint surface conditions (Adjusted from Hoek,1994; Hoek and Brown, 1997)

ROCK MASS CHARACTERISTICS FOR STRENGTH ESTIMATES Based upon the appearance of the rock, choose the category that you think gives the best description of the 'average' undisturbed in situ conditions. Note that exposed rock faces that have been created by blasting may give a misleading impression of the quality of the underlying rock. Some adjustment for blast damage may be necessary and examination of diamond drill core or of faces created by pre-split or smooth blasting may be helpful in making these adjustments. It is also important to recognize that the Hoek-Brown criterion should only be applied to rock masses where the size of individual blocks is small compared with the size of the excavation under consideration. STRUCTURE	SURFACE CONDITIONS	VERY GOOD Very rough,fresh unweathered surfaces	GOOD Rough, slightly weathered, iron stained surfaces	FAIR Smooth, moderately weathered or altered surfaces	POOR Slickensided, highly weathered surfaces with compact coatings or fillings of angular fragments	VERY POOR Slickensided, highly weathered surfaces with soft clay coatings or fillings
		DECREASING SURFACE QUALITY ⇨				
 BLOCKY - very well interlocked undisturbed rock mass consisting of cubical blocks formed by three orthogonal discontinuity sets	DECREASING INTERLOCKING OF ROCK PIECES ⇩	B/VG	B/G	B/F	B/P	B/VP
 VERY BLOCKY - interlocked, partially disturbed rock mass with multifaceted angular blocks formed by four or more discontinuity sets		VB/VG	VB/G	VB/F	VB/P	VB/VP
 BLOCKY/DISTURBED- folded and/or faulted with angular blocks formed by many intersecting discontinuity sets		BD/VG	BD/G	BD/F	BD/P	BD/VP
 DISINTEGRATED - poorly inter-locked, heavily broken rock mass with a mixture or angular and rounded rock pieces		D/VG	D/G	D/F	D/P	D/VP

[1] In earlier versions of this table the terms BLOCKY/SEAMY and CRUSHED were used, following the terminology used by Terzaghi [9]. However, these terms proved to be misleading and they have been replaced, in this table by BLOCKY/DISTURBED, which more accurately reflects the increased mobility of a rock mass which has undergone some folding and/or faulting, and DISINTEGRATED which encompasses a wider range of particle shapes.

Table(4-15) Estimation of Geological Strength Index (GSI) on the basis of interlocking and joint surface conditions (After Hoek and Brown, 1997)

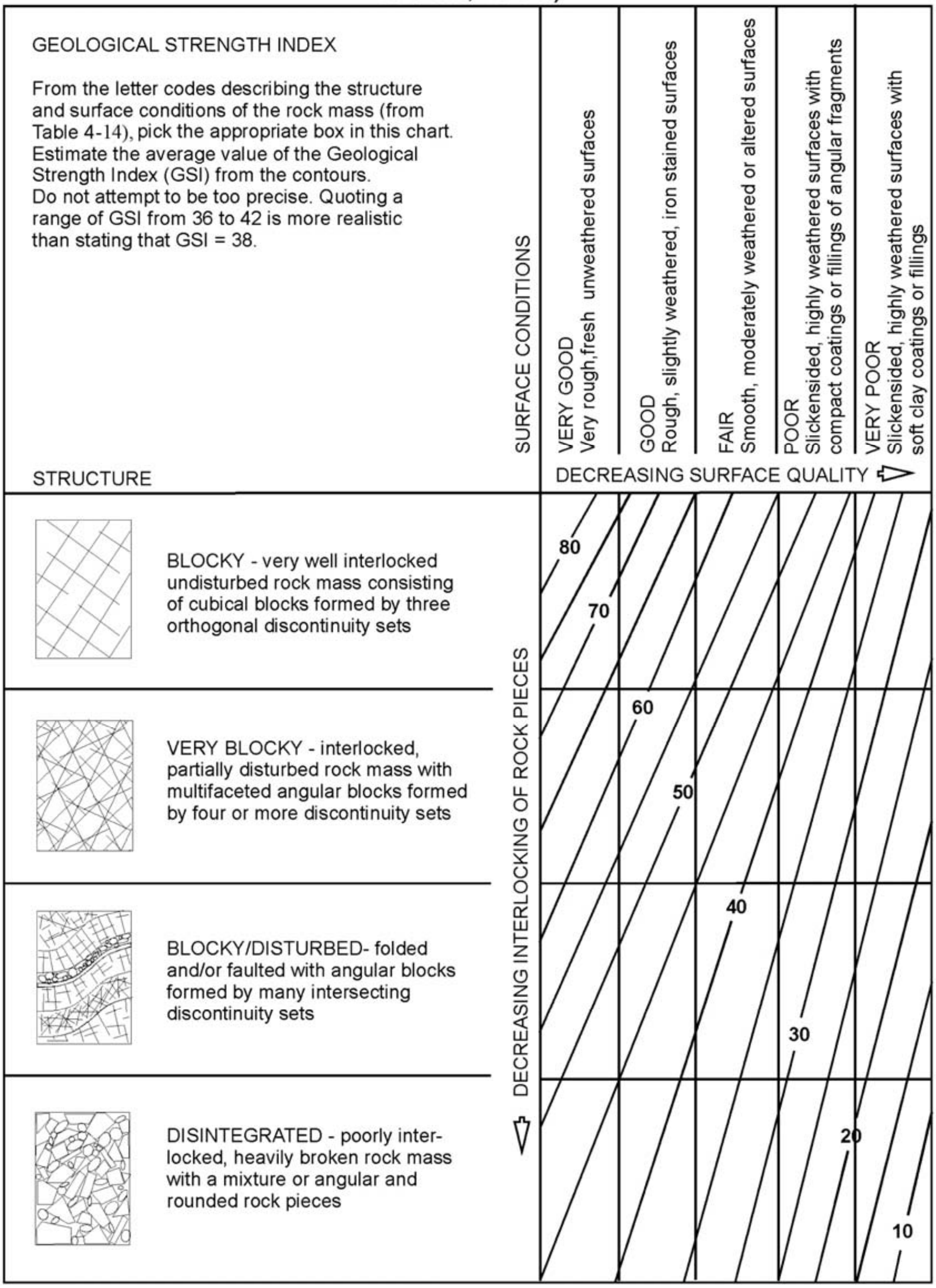


Table (4 -16) Estimate of Geological Strength Index(GSI), including Foliated / Laminated / Sheared, intact and massive rock masses (Hoek, 1999 ; Marinos & Hoek, 2000)

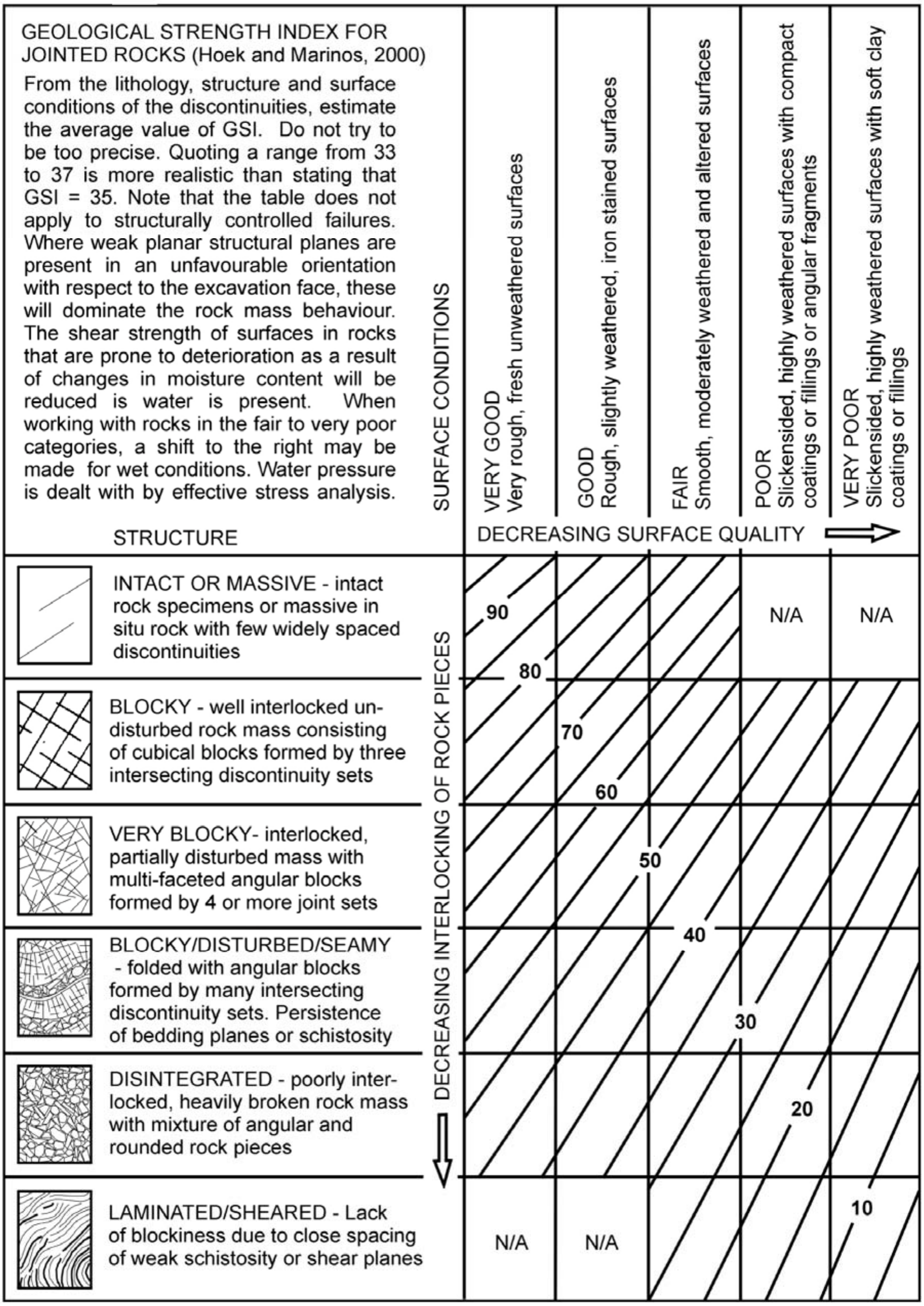


Table (4-17) Estimate of Geological Strength Index GSI for heterogeneous rock masses such as Flysch(After Marinos and Hoek, 2000)

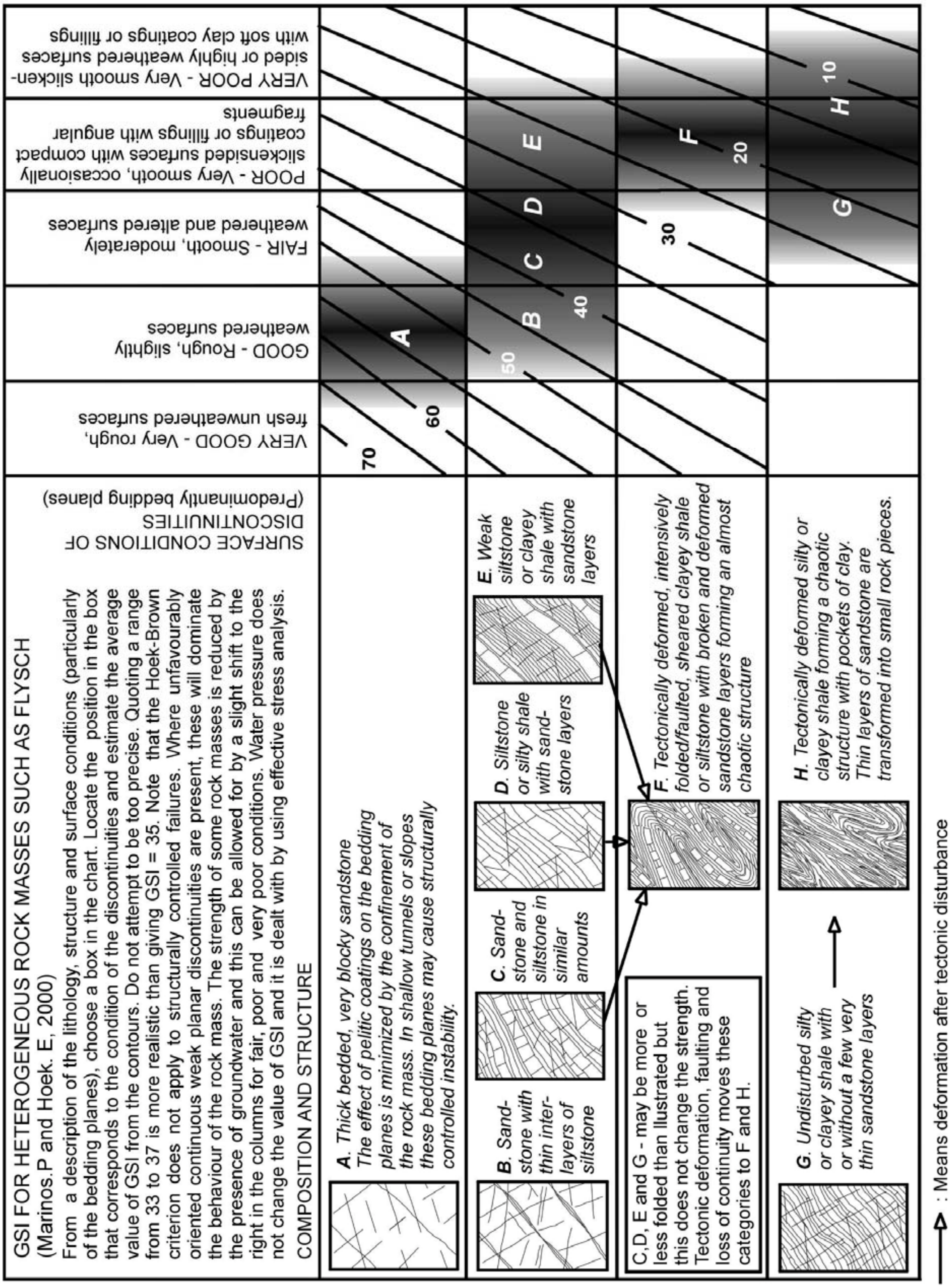
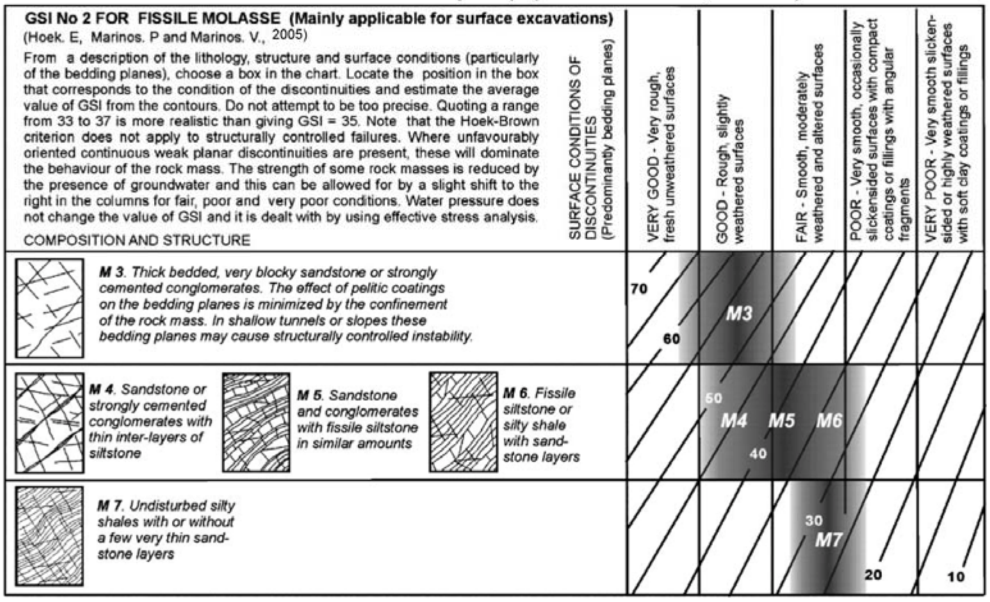


Table (4-18) Geological Strength Index for fissile Molasse (surface excavation and slopes) (Hoek et. al., 2005)



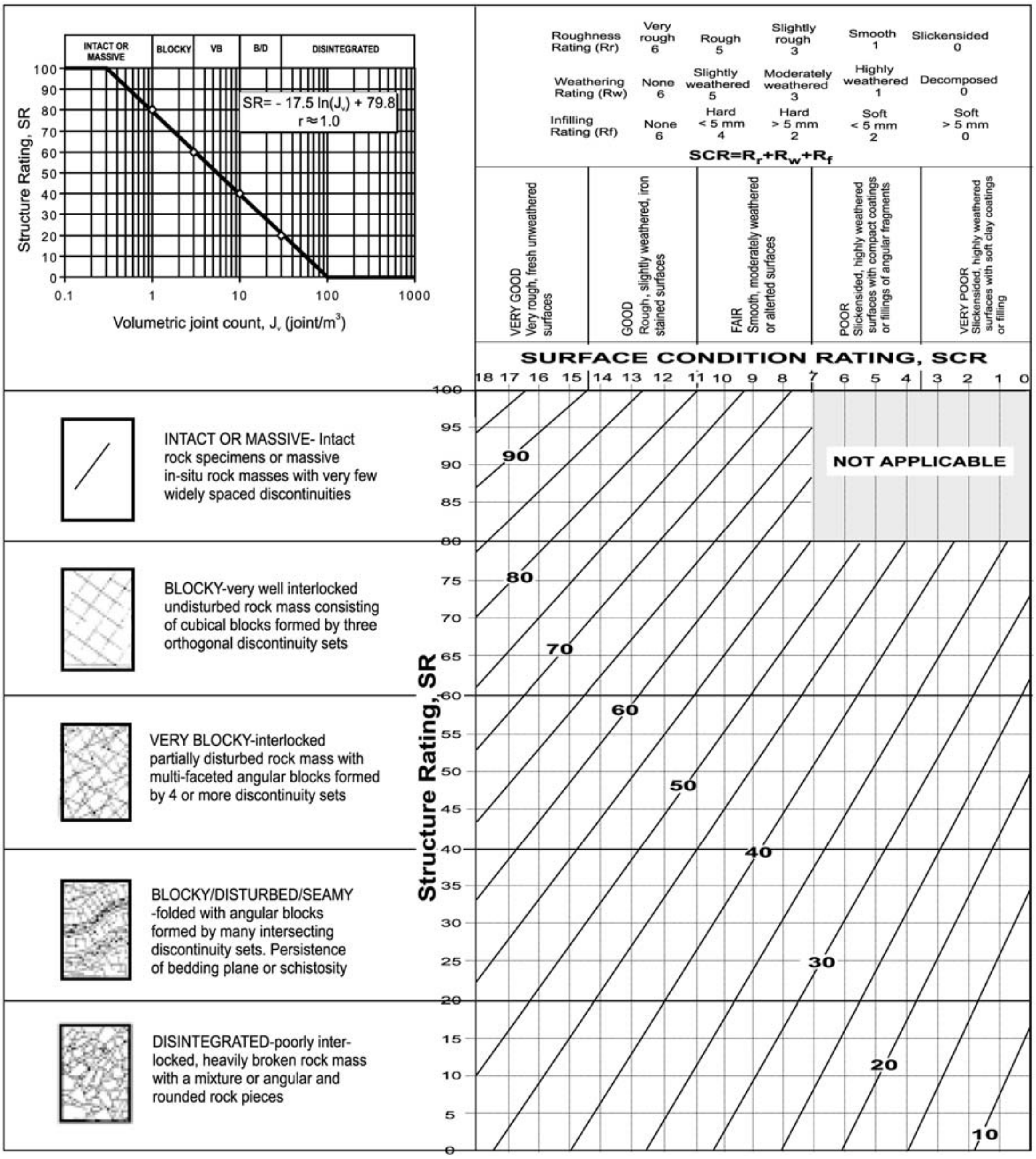
4-4-2 Quantitative modified GSI chart by Sonmez and Ulusay

The input parameter of the GSI is qualitative, and the GSI value is obtained by combining the two fundamental parameters, they are the blockiness of the rock mass and conditions of discontinuities.

Different authors have proposed a quantification of the input parameters for the determination of the GSI, for example, Sonmez and Ulusay (1999), Cai et al. (2004) and Russo (2008).

Sonmez and Ulusay (1999) saw that due to lack of measurable and more representative parameters, and related interval limits or ratings for describing the surface conditions of the discontinuities, value of the GSI for each rock mass category appearing in table (4-14) represents a range of values. For example, for a blocky rock with very good surface condition of discontinuity (B/VG), GSI values varying between 63 and 85 are obtained from table (4-15). This consideration placed focus on the question “how can a more precise GSI value be obtained from the existing chart for design?”. Therefore, they suggested two terms namely, structure rating (SR) based on volumetric joint count (Jv) and surface condition rating (SCR), estimated from sum of three parameters, they are roughness, weathering and infilling materials (they have the same ratings as existed in the RMR_{89}), see table (4-19). A new rock mass category to accommodate thinly foliated or laminated, folded and predominantly sheared weak rock of non-blocky structure proposed by Hoek et al.(1998) has not been included into Sonmez-Ulusay quantitative modified GSI system.

Table (4-19) The modified quantitative GSI system (After Sonmez and Ulusay, 2002)



Based on the intervals of Jv and corresponding descriptions for the blockiness rating, structure rating (SR) was assigned to each category according to the following relationship:

SR=79.8 -17.5 *In* (Jv) ………(4-21) (Sonmez and Ulusay, 2002)

The SR limits between five rock mass groups are selected as 80, 60, 40 and 20 respectively. The relationship between these SR limits and corresponding Jv values (1, 3, 10 and 30 joints/m^3) are obtained. For the upper and lower limits of SR (100 and 0 respectively), the corresponding Jv values are 0.3 and 100 joints/m^3 respectively.

Other two quantitative parameters were proposed, they are block volume (Vb) and joint condition factor (Jc) of Palmstrom instead of rock mass structure (blockiness) and conditions of discontinuities respectively, for estimating the GSI value (Cai et al., 2004 ; Russo, 2008).

4-4-3 The proposed quantitative chart for GSI determination in this study:

The alternative proposed chart is based on the use of structure rating (SR) (based on Jv or block volume (Vb)) and surface condition rating (SCR) (estimated from roughness, weathering and infilling) as shown in table (4-20).

The GSI chart is the same as proposed by Hoek (1999) and Marinos and Hoek (2000) without including foliated / laminated / sheared rock mass category.

The Jv limits between the five rock mass groups, which were selected by Sonmez & Ulusay are changed. The new Jv limits are selected as 3 (massive – blocky), 10 (blocky – very blocky), 30 (very blocky – blocky/disturbed) and 100 joints/m^3 (blocky/disturbed – disintegrated).

On the basis of Jv interval and corresponding descriptions for the blockiness rating, structure rating (SR) was assigned to each category according to the following relationship:

SR=100-17.5322 *In* (Jv) (for Jv ≤ 1 SR =100; for Jv ≥300 SR=0)….(4-22)

The SR limits between five rock mass groups 80, 60, 40 and 20 respectively are obtained from equation (4-22), which are corresponding to Jv values 3, 10, 30 and 100 joints/m^3 respectively. For the upper and lower limits of SR, 100 and 0 respectively, the corresponding Jv values are 1 and 300 joints/m^3 respectively.

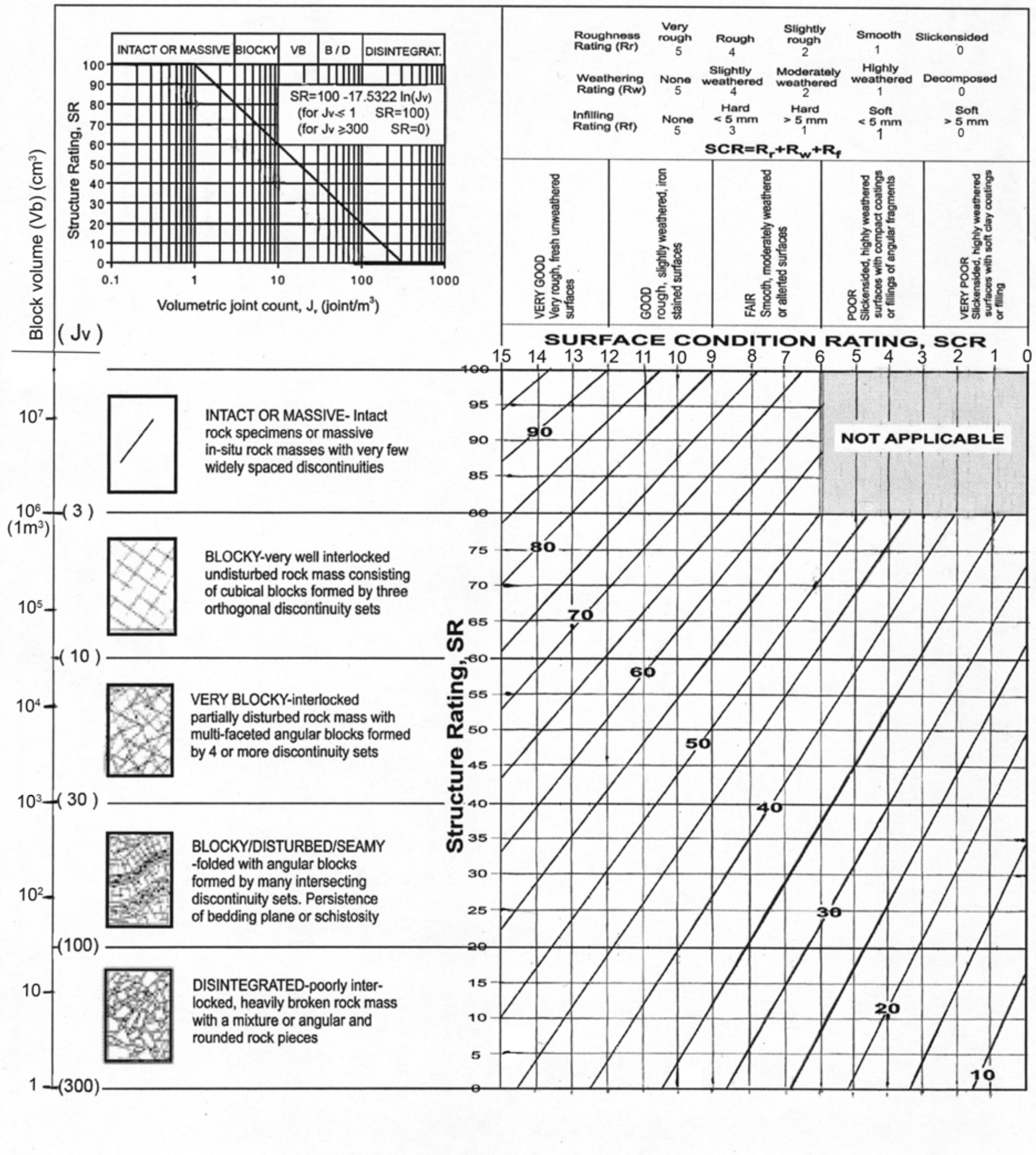
These changes based upon the fact that the limit between massive and blocky rock masses corresponds to Jv value of 3 joint/m^3 (equal to a block volume value of $1m^3$) (Palmstrom, 2000 & 2005), between blocky and very blocky rock masses corresponds to Jv value of 10 joint/m^3 (equal to a block volume value of $3*10^4cm^3$) and so on between other rock mass categories as in table (4-20). Also, other authors assume a block volume of $1m^3$ as a limit between massive and blocky rock masses (Cai et al., 2004), but Sonmez and Ulusay consider Jv value of 1 joint/m^3 for the mentioned limit, this value of Jv (Jv= 1 joint/m^3) corresponds to a block volume of $27m^3$ or more, which is far from reality.

Another change in the quantitative modified Sonmez – Ulusay GSI chart is in the value of SCR , this value ranges from 0 to 18(based on the sum of rating of roughness, weathering and infilling as the same used in RMR_{89}).

Because the GSI based on the RMR_{76} (Hoek et al., 1995), then the roughness, weathering and infilling ratings (SCR) must be based on the RMR_{76}, in which the sum of these three parameters ranges from 0 to 15.

On the basis of these changes, a new GSI-Chart is proposed, as it is shown in the table (4-20). This new GSI-Chart is used in this thesis, except it is unused for Flysch and Molasses deposits.

Table(4-20) The modified quantitative GSI system proposed in this study



4-4-4 Estimation of the GSI from RMR

In using Bieniawski's 1976 Rock Mass Rating to estimate the value of GSI , table (4-4) should be used to calculate the rating for the first four parameters. The rock mass should be assumed to be completely dry and a rating of 10 assigned to the ground water value. Very favorable joint orientations should be assumed and the adjustment for joint orientation value is set to zero. The final rating, called RMR_{76}, can then be used to estimate the value of GSI (Hoek et al., 1995) as follows:

$$GSI = RMR_{76} \text{ (for } RMR_{76} > 18) \ldots\ldots\ldots\ldots (4\text{-}23)$$

For $RMR_{76} < 18$ Bieniawski's 1976 classification can not be used to estimate GSI.

Bieinawski's 1989 classification, given in table (4-2), can be used to estimate the value of GSI in a similar manner to that described for the 1976 version. In this case, a value of 15 is assigned to the groundwater rating and the adjustment for joint orientation is again set to zero. The final rating, called RMR_{89} , can be used to estimate the value of GSI (Hoek et al ., 1995) as follows:

$$GSI= RMR_{89} - 5 \text{ (for } RMR>23) \ldots\ldots\ldots\ldots (4\text{-}24)$$

For RMR_{89} <23 Bieniawsk's 1989 classification can not be used to estimate GSI.

GSI system does not suggest a direct correlation between rock mass quality and GSI value. However, it is suggested that GSI can be related to RMR_{89} by GSI= RMR_{89} -5 , for reasonable good quality rock mass (where RMR_{89} has the Groundwater rating set to 15 and the Adjustment for joint orientation set to zero) (Hoek et al.,1995; Hoek and Brown,1997) . An approximate classification of rock mass quality and GSI is therefore suggested in table (4-21), based on the correlation between RMR and GSI.

Table (4-21) GSI Rock mass classes determined from RMR_{89}
(Modified from Bieniawski , 1989)

GSI Value	76-95	56-75	36-55	16-35	< 16
Rock mass quality	Very good	Good	Fair	Poor	Very Poor

Table (4-21) can be used in applying guidelines for excavation and supporting of 10m span tunnels, as in table (4-3) (the same guidelines for RMR_{89}).

4-4-5 Links between GSI and rock mass properties

Serafim- Pereira's equation for estimating the deformation modulus of the rock mass has been found to work well for better quality rocks. However, for many of the poor quality rock it appears to predict deformation modulus values which are too high. Because of this and based upon practical observations and back analysis of excavation behavior in poor quality rock masses, the following modifications to Serafim and Pereira's equation were proposed (Hoek and Brown, 1997):

$$Em\ (GPa) = \sqrt{(\sigma_{ci}/100)} \times 10^{((GSI-10)/40)} \quad (\text{for } \sigma_{ci} < 100MPa) \ldots\ldots (4\text{-}25)$$

$$Em\ (GPa) = 10^{((GSI-10)/40)} \quad (\text{for } \sigma_{ci} > 100MPa) \ldots\ldots\ldots\ldots (4\text{-}26)$$

Where: Em=Deformation modulus of the rock mass

σ_{ci}=Unconfined compressive strength of the intact rock.

Equations 4-25 and 4-26 have been modified by the inclusion of the disturbance factor (D) to allow for the effects of blast damage and stress relaxation (Hoek et al.,2002) as follows:

$$Em\ (GPa) = (1-D/2)\sqrt{(\sigma_{ci}/100)} \times 10^{((GSI-10)/40)} \quad (\text{for } \sigma_{ci}<100MPa) .. (4\text{-}27)$$

$$Em\ (GPa) = (1-D/2) \times 10^{((GSI-10)/40)} \quad (\text{for } \sigma_{ci}>100MPa) \ldots\ldots (4\text{-}28)$$

Table (4-22) gives guidelines for estimating disturbance factor (D) in tunnels, slopes and pit- quarries, but not for dams.

Excavations for dam's foundation are, as a rule, very careful, D should be very low, but it can not be zero. Tentative guidelines are as follows (Romana, 2003a):

- Good rock mass, normal blasting → D= 0.4

- Any rock mass, controlled blasting → D= 0.2

- Poor rock mass, mechanical excavation → D= 0.2

Another equation was proposed for estimating the deformation modulus of the rock mass as follows (Hoek and Diederichs 2006):

$$Em\ (MPa) = 100000((1-D/2) / (1+e^{((75+25D-GSI)/11)}) \ldots\ldots (4\text{-}29)$$

Also a detailed analysis of the Chinese and Taiwanese data by Hoek and Diederichs (2006) resulted in the following equation:

$$Em\ (MPa) = Ei\ (0.02 + (1-D/2) / (1+e^{((60+15D-GSI)/11)})) \ldots\ldots (4\text{-}30)$$

Where: Ei=Modulus of elasticity of intact rock and can be estimated from Ei=MRσ_{ci} as previously discussed.

Table(4-22)Guidelines for estimating disturbance factor(D)(Hoek et al.,2002)

Appearance of rock mass	Description of rock mass	Suggested value of D
	Excellent quality controlled blasting or excavation by Tunnel Boring Machine results in minimal disturbance to the confined rock mass surrounding a tunnel.	$D = 0$
	Mechanical or hand excavation in poor quality rock masses (no blasting) results in minimal disturbance to the surrounding rock mass. Where squeezing problems result in significant floor heave, disturbance can be severe unless a temporary invert, as shown in the photograph, is placed.	$D = 0$ $D = 0.5$ No invert
	Very poor quality blasting in a hard rock tunnel results in severe local damage, extending 2 or 3 m, in the surrounding rock mass.	$D = 0.8$
	Small scale blasting in civil engineering slopes results in modest rock mass damage, particularly if controlled blasting is used as shown on the left hand side of the photograph. However, stress relief results in some disturbance.	$D = 0.7$ Good blasting $D = 1.0$ Poor blasting
	Very large open pit mine slopes suffer significant disturbance due to heavy production blasting and also due to stress relief from overburden removal. In some softer rocks excavation can be carried out by ripping and dozing and the degree of damage to the slopes is less.	$D = 1.0$ Production blasting $D = 0.7$ Mechanical excavation

The simplified Hoek and Diederich's equation (4-29) can be used where only GSI (or RMR) data are available. The more detailed Hoek and Diederichs equation (4-30) can be used where reliable estimates of the intact rock strength are available.

4-5 The Hoek- Brown failure criterion

The Hoek – Brown failure criterion was originally developed for estimating the strengths of hard rock masses. It was introduced in 1980(Hoek and Brown, 1980b), and based upon experience in using the criterion on a number of projects, an updated version was published in 1988 (Hoek and Brown, 1988) and a modified criterion was published in 1992 (Hoek et al., 1992).

The original criterion has been found to work well for most rocks of good to reasonable quality in which the rock mass strength is controlled by tightly interlocking angular rock pieces. The failure of such rock masses can be defined by the following equation (Hoek and Brown, 1980b; Hoek and Brown, 1988):

$$\sigma'_1 = \sigma'_3 + \sigma_{ci} \left(m_b \left(\sigma'_3 / \sigma_{ci}\right) + s\right)^{0.5} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots(4\text{-}31)$$

Where: m_b is the value of the constant m for the rock mass

s is a constant which depends upon the characteristics of the rock mass:

σ_{ci} is the unconfined compressive strength of the intact rock

σ'_1 and σ'_3 are major and minor principal effective stresses respectively.

For poor quality rock masses in which the tight interlocking has been partially destroyed by shearing or weathering, the rock mass has no tensile strength or cohesion and specimens will fall apart without confinement. For such rock masses the modified criterion is more appropriate and this is obtained by putting s=0 (Hoek et al., 1992) which gives:

$$\sigma'_1 = \sigma'_3 + \sigma_{ci} \left(m_b \left(\sigma'_3 / \sigma_{ci}\right)\right)^{a} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots(4\text{-}32)$$

Where: a is a constant which depends upon the characteristics of the rock mass.

To estimate the rock mass constants (m_b and s) of the Hoek- Brown criterion, there are relationships between the RMR from Bieniawski's 1976 rock mass classification , assuming completely dry conditions and a very favourable joint orientation (Hoek and Brown , 1988) these relationships are as follows:

Undisturbed or interlocking rock masses:

m_b=mi exp ((RMR-100)/28)…………………..(4-33)

s = exp ((RMR-100)/9)………………………..(4-34)

Disturbed rock masses:

m_b= mi exp ((RMR-100)/14)…………………(4-35)

s= exp ((RMR-100)/6)………………………...(4-36)

Where mi is the value of m for the intact rock, and can be estimated from the last version as in table (4-23) (where no triaxial test is done).

The original criterion, with its bias towards hard rock, was based upon the assumption that rock mass failure is controlled by translation and rotation of individual rock pieces, separated by numerous joint surfaces. Failure of the intact rock was assumed to play no significant role in the overall failure processes and it was assumed that the joint pattern was 'chaotic' , so that there are no preferred failure directions and the rock mass can be treated as isotropic (Hoek and Brown,1980b).

The influence of joint orientation should be taken into account in deciding whether or not the Hoek – Brown failure criterion is applicable. This must be based on judgment of potential anisotropy of the rock mass, block size in relation to size of the excavation, and mode of failure(structural control versus rock mass failure) (Hoek and Brown,1988; Hoek et al.1995; Sjoberg,1997), as defined in Fig (4-7).

It soon became evident that the modified criterion was too conservative when used for better quality rock masses and a 'generalized' failure criterion was proposed (Hoek, 1994; Hoek et al., 1995) by the following equation:

$\acute{\sigma}_1 = \acute{\sigma}_3 + \sigma_{ci} (m_b (\acute{\sigma}_3 / \sigma_{ci}) + s)^a$…………….(4-37)

This generalized criterion incorporated both the original and the modified criteria with a 'switch' at an RMR value of approximately 25. Hence for very good- fair quality rock masses (undisturbed: RMR>25), the original Hoek – Brown criterion is used while, for poor – very poor rock masses (disturbed: RMR< 25) , the modified criterion with zero tensile strength is used.

Also with introducing the GSI to overcome the deficiencies in Bieniawski's RMR for very poor quality rock masses, the RMR is replaced by GSI in estimating the Hoek – Brown criterion's material constants (m_b , s and a) (parameters which describe the rock mass strength characteristics) (Hoek, 1994; Hoek et al., 1995; Hoek and Brown , 1997) as follows:

m_b=mi exp ((GSI-100)/28)……………(4-38)

For GSI > 25 (good to reasonable quality 'undisturbed' rock mass), the original Hoek-Brown criterion is applicable with:

s=exp ((GSI – 100) / 9) ………………(4-39)

a=0.5 …………………………………..(4-40)

For GSI < 25 (very poor quality 'disturbed' rock mass), the modified Hoek-Brown criterion is applicable with:

s=0 ……………………………………....(4-41)

a=0.65 – (GSI / 200) …………………....(4-42)

The choice of GSI equal to 25 for the switch between the original and modified criteria is purely arbitrary. It could be argued that a switch at GSI equals to 30 would not introduce a discontinuity in the value of a, but extensive trials have shown that the exact location of this switch has negligible practical significance (Hoek and Brown, 1997).

As a result of the evolution of the Hoek-Brown criterion, a disturbance factor (D) was introduced in estimating the rock mass constants (m_b and s), and a new equation was proposed in estimating the 'a' rock mass constant (Hoek et al., 2002) as follows:

$m_b = m_i$ exp((GSI-100) / (28-14D))……………..(4-43)

s= exp ((GSI-100) / (9-3D))……………………(4-44)

a = 1/2+1/6($e^{-GSI/15}$ – $e^{-20/3}$)……………………(4-45)

Where D is a disturbance factor, which was mentioned previously.

It is noted that the "switch" at GSI equals to 25 for the coefficients s and a (Hoek,1994; Hoek et al.,1995; Hoek and Brown,1997) has been eliminated in equations 4-44 and 4-45 which give smooth continuous transitions for the entire range of GSI values.

Table (4-23) Values of the constant mi for intact rock, by rock group**. Note that values in parenthesis are estimates. (In Marinos and Hoek, 2001)

Rock type	Class	Group	Texture: Coarse	Texture: Medium	Texture: Fine	Texture: Very fine
SEDIMENTARY	Clastic		Conglomerates (21 ± 3) Breccias (19 ± 5)	Sandstones 17 ± 4	Siltstones 7 ± 2 Greywackes (18 ± 3)	Claystones 4 ± 2 Shales (6 ± 2) Marls (7 ± 2)
SEDIMENTARY	Non-Clastic	Carbonates	Crystalline Limestone (12 ± 3)	Sparitic Limestones (10 ± 2)	Micritic Limestones (9 ± 2)	Dolomites (9 ± 3)
SEDIMENTARY	Non-Clastic	Evaporites		Gypsum 8 ± 2	Anhydrite 12 ± 2	
SEDIMENTARY	Non-Clastic	Organic				Chalk 7 ± 2
METAMORPHIC	Non Foliated		Marble 9 ± 3	Hornfels (19 ± 4) Metasandstone (19 ± 3)	Quartzites 20 ± 3	
METAMORPHIC	Slightly foliated		Migmatite (29 ± 3)	Amphibolites 26 ± 6	Gneiss 28 ± 5	
METAMORPHIC	Foliated*			Schists 12 ± 3	Phyllites (7 ± 3)	Slates 7 ± 4
IGNEOUS	Plutonic	Light	Granite 32 ± 3 Granodiorite (29 ± 3)	Diorite 25 ± 5		
IGNEOUS	Plutonic	Dark	Gabbro 27 ± 3 Norite 20 ± 5	Dolerite (16 ± 5)		
IGNEOUS	Hypabyssal		Porphyries (20 ± 5)		Diabase (15 ± 5)	Peridotite (25 ± 5)
IGNEOUS	Volcanic	Lava		Rhyolite (25 ± 5) Andesite 25 ± 5	Dacite (25 ± 3) Basalt (25 ± 5)	
IGNEOUS	Volcanic	Pyroclastic	Agglomerate (19 ± 3)	Volcanic breccia (19 ± 5)	Tuff (13 ± 5)	

* These values are for intact rock specimens tested normal to bedding or foliation. The value of m_i will be significantly different if failure occurs along a weakness plane.

** Note that this table contains several changes from previously published versions, These changes have been made to reflect data that has been accumulated from laboratory tests and the experience gained from discussions with geologists and engineering geologists.

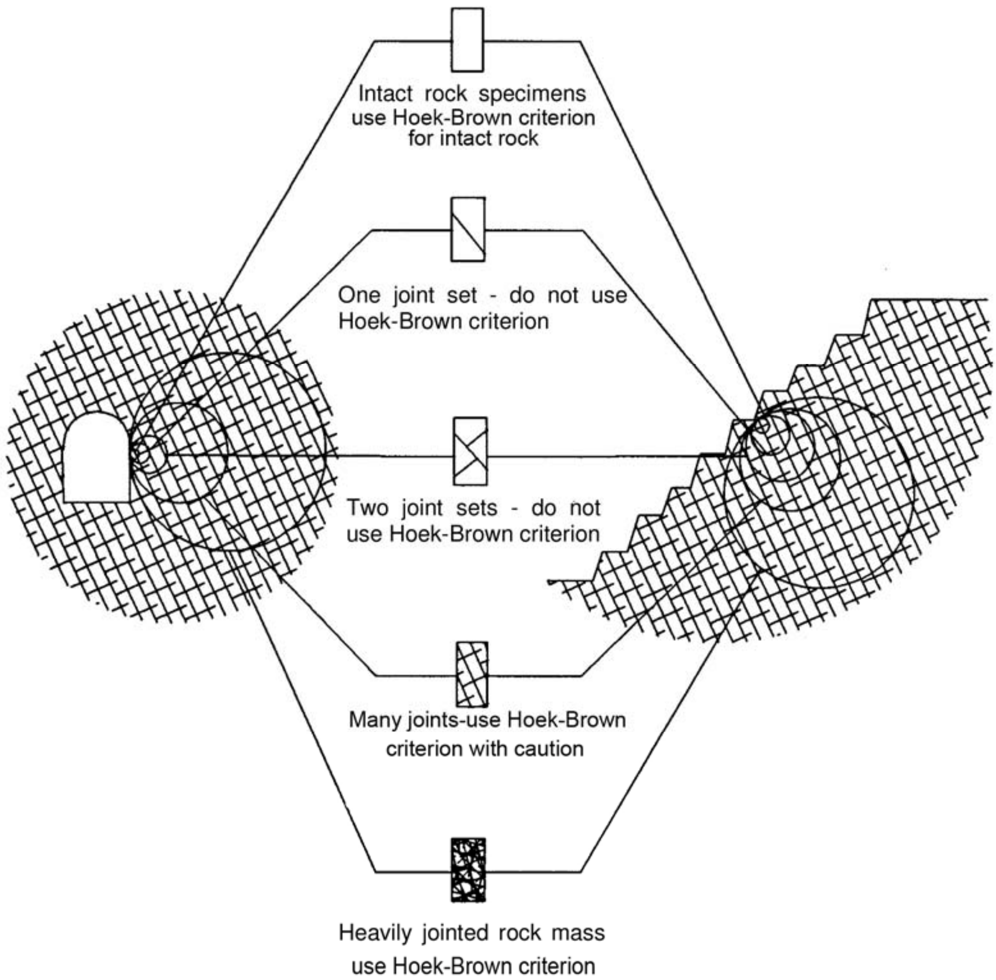


Fig (4-7) Rock mass conditions under which the Hoek-Brown failure criterion can be applied (After Hoek et al., 1995)

The unconfined compressive strength ($б_{cm}$) of the rock mass is obtained by setting $б'_3 = 0$ in equation (4-37) (generalized criterion equation), giving:

$б_{cm} = б_{ci} . s^a$ ……………………(4-46)

and the tensile strength ($б_t$) of the rock mass is:

$б_t = -(s\ б_{ci} / m_b)$ ………………..(4-47)

Equation 4-47 is obtained by setting $б'_1 = б'_3 = б_t$ in equation 4-37. This represents a condition of biaxial tension, for brittle materials, the uniaxial tensile strength is equal to the biaxial tensile strength (Hoek, 1983).

Since most geotechnical software is still written in terms of the Mohr-Coulomb failure criterion in which the rock mass strength is defined by the angle of friction and the cohesive strength, it is necessary to determine equivalent angles of friction (Φ́) and cohesive strengths (ć) for each rock mass and stress range. The processes of estimating the angle of friction and the cohesive strength are well established in 1997 paper of Hoek and Brown, and the newest and most precise one is that by Hoek et al.(2002).

4-6 Some explanations about Hoek-Brown failure criterion:

4-6-1 Selection of $б_{ci}$ and m_i for flysch:

In addition to the GSI values presented in table (4-17), it is necessary to consider the selection of other "intact" rock properties $б_{ci}$ and m_i for heterogeneous rock masses such as Flysch. Because the sandstone layers are usually separated from each other by weaker layers of siltstone or shales, rock-to-rock contact between blocks of sandstone may be limited. Consequently, it is not appropriate to use the properties of the sandstone to determine the overall strength of the rock mass. On the other hand, using the "intact" properties of the siltstone or shale only is too conservative since the sandstone skeleton certainly contributes to the rock mass strength. Therefore, it is proposed that a 'weighted average' of the intact strength properties of the strong and weak layers should be used (Marinos and Hoek, 2001). Suggested values for the components of this weighted average are given in table (4-24).

Table (4-24) Suggested proportions of parameters $б_{ci}$ and mi for estimating rock mass properties for Flysch.(Marinos and Hoek, 2001)

Flysch type see table (4-17)	Proportions of values for each rock type to be included in rock mass property determination
A and B	Use values for sandstone beds
C	Reduce sandstone values by 20% and use full values for siltstone
D	Reduce sandstone values by 40% and use full values for siltstone
E	Reduce sandstone values by 40% and use full values for siltstone
F	Reduce sandstone values by 60% and use full values for siltstone
G	use values for siltstone or shale
H	use values for siltstone or shale

4-6-2 Rock mass strength

The unconfined compressive strength (σ_{cm}) of the rock mass is given by equation 4-46. Failure initiates at the boundary of an equation when σ_{cm} is exceeded by the stress induced on that boundary. The failure propagates from this initiation point to a biaxial stress field and it eventually stabilizes when the local strength, defined by equation 4-37, is higher than the induced stresses σ'_1 and σ'_3. Most numerical models can follow this process of fracture propagation and this level of detailed analysis is very important when considering the stability of excavations in rock and when designing support systems (Hoek et al., 2002).

However, there are times when it is useful to consider the overall behavior of a rock mass rather than the detailed failure propagation process described above. For example, when considering the strength of a pillar, it is useful to have an estimate of the overall strength of the pillar rather than a detailed knowledge of the extent of fracture propagation in the pillar. This leads to the concept of a "global rock mass strength" (σ'_{cm}), and this could be estimated from the Mohr-Coulomb relationship as follows (Hoek and Brown, 1997):

$\sigma'_{cm} = (2c' \cos\Phi') / (1-\sin\Phi')$ ………………..(4-48)

Where: c' = cohesion of the rock mass, Φ' = friction angle of the rock mass

4-6-3 Windows programme "RocLab"

A number of uncertainties and practical problems in using the Hoek-Brown failure criterion have been addressed in 2002 edition. Wherever possible, an attempt has been made to provide a rigorous and unambiguous method for calculating or estimating the input parameters required for the analysis. These methods have been implemented in a windows program called "RocLab" that can be downloaded (free) from www.rocscience.com. (Hoek et al., 2002).

Chapter Five

Rock mass evaluation

5-1 Preface

To evaluate the rock masses, three surface sections, three boreholes and three topographic profiles (a-b, c-d & e-f) were selected in Darband Basara valley, Fig (5-1), to find the optimum profile site for dam construction after evaluating each of them.

Each surface section and borehole was divided into units depending on the guidelines followed by Bieniawski (1989) as formerly pointed out in chapter four. The result of this division (for all surface sections and boreholes) is that the rock masses are composed of 30 units.

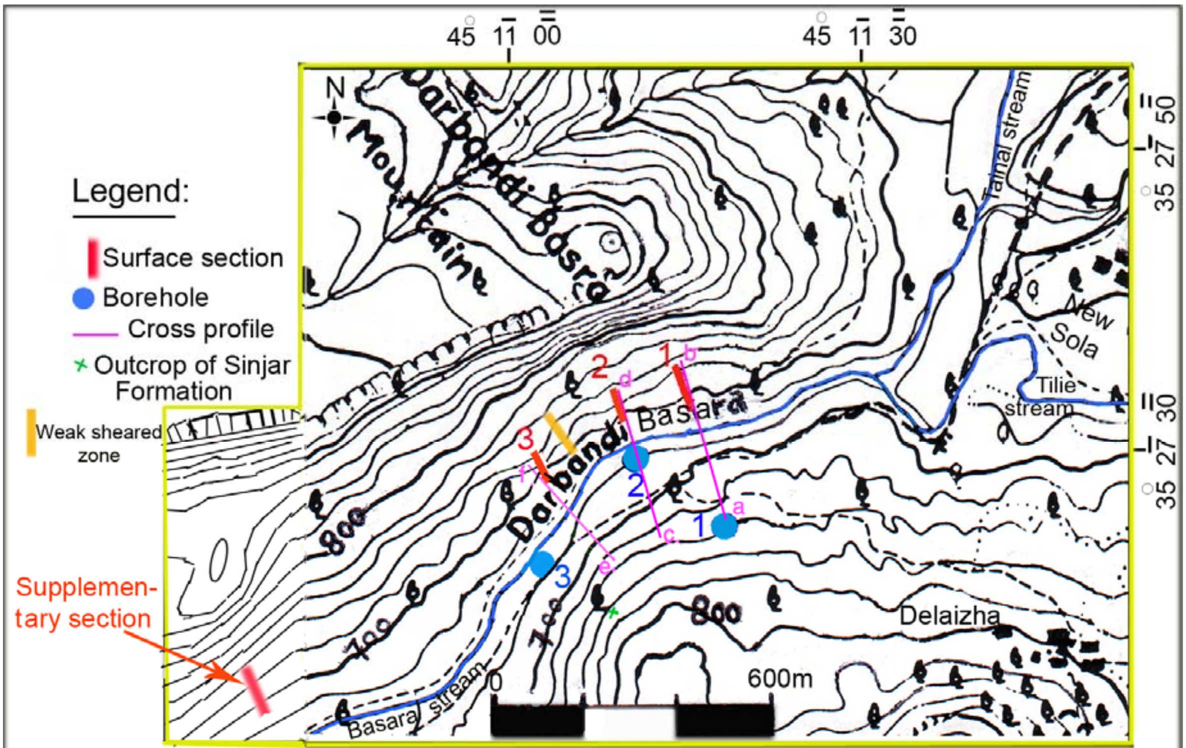


Fig (5-1) Location map of the proposed Basara dam site, showing surface sections, boreholes and cross profiles

5-2 Site conditions for rock masses in the surface sections and boreholes

The rock masses in the surface sections are not covered by soil. It means that they are exposed directly on the slope surface. But the rock masses in the boreholes are covered by soils and drifts.

Table (5-1) shows the vertical thickness of the soils and drifts in the bore holes and groundwater table elevation at each one.

Figs. (5-2, 5-3 & 5-4) illustrate the site condition for rock masses in the boreholes no. 1, 2 and 3, in which the depth, Rock Quality Designation (RQD) and lithology of each rock mass unit are shown in each borehole.

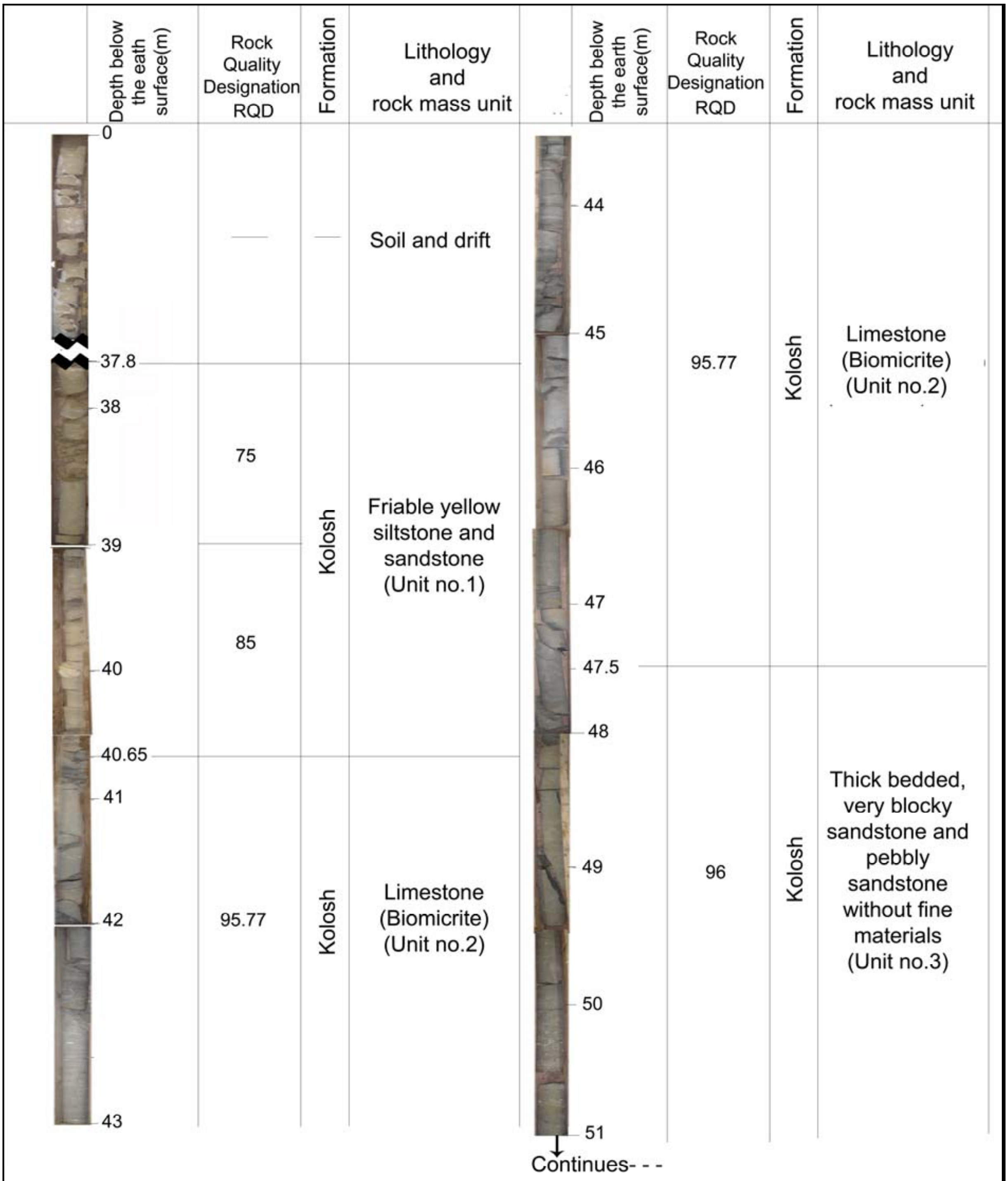
Table (5-1) Vertical thickness of the soil and drift, and groundwater table elevation at boreholes no. 1, 2 & 3.

Borehole No.	Elevation of bore hole above sea level (a.s.l) (m)	Thickness of the soil and drift (m)	Elevation of bedrock mass(a.s.l)(m)	Depth of ground-water below earth surface(m)	Elevation of groundwater (a.s.l) (m)
1	739	37.80	701.20	11	728
2	680	19.50	660.50	5.50	674.50
3	672	13	659	6.70	665.30

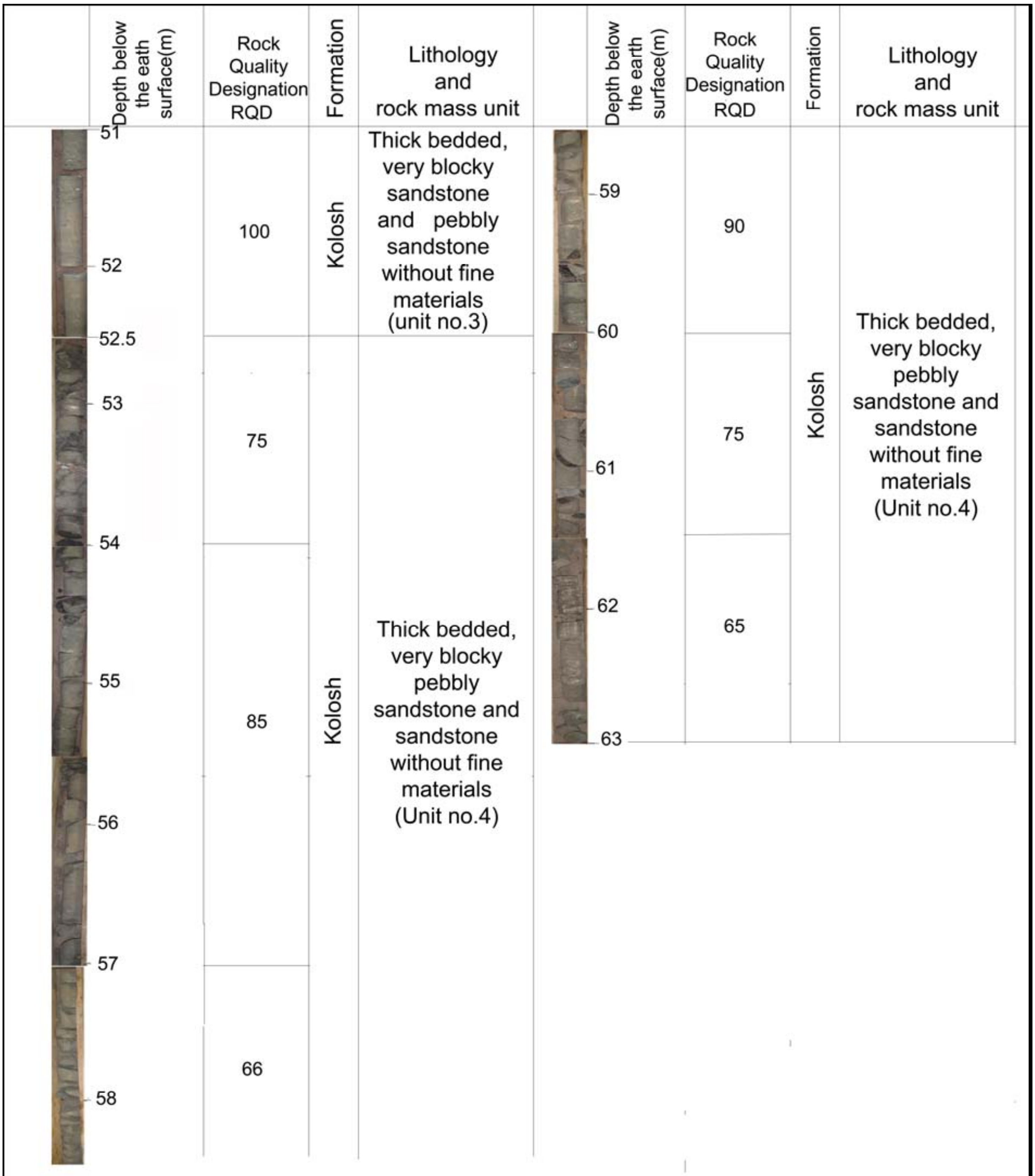
From table (5-1) it will be noted that the thickness of the soil and drift in borehole no.1 is much more than its thickness in boreholes 2 and 3. This is due to the presence of slid soil masses in this site, and this is concluded from field observations, which is leaving a scar in the upper part (see the upward shifting of the contour lines near borehole no.1,Fig "5-1"), and accumulation of drifts at the toe of the slope in the field, so it will be expected that the thickness of the soils and drifts will be decreased in the upper slopes on the left bank as we move downstream.

5-3 Rock mass evaluation of surface sections and boreholes

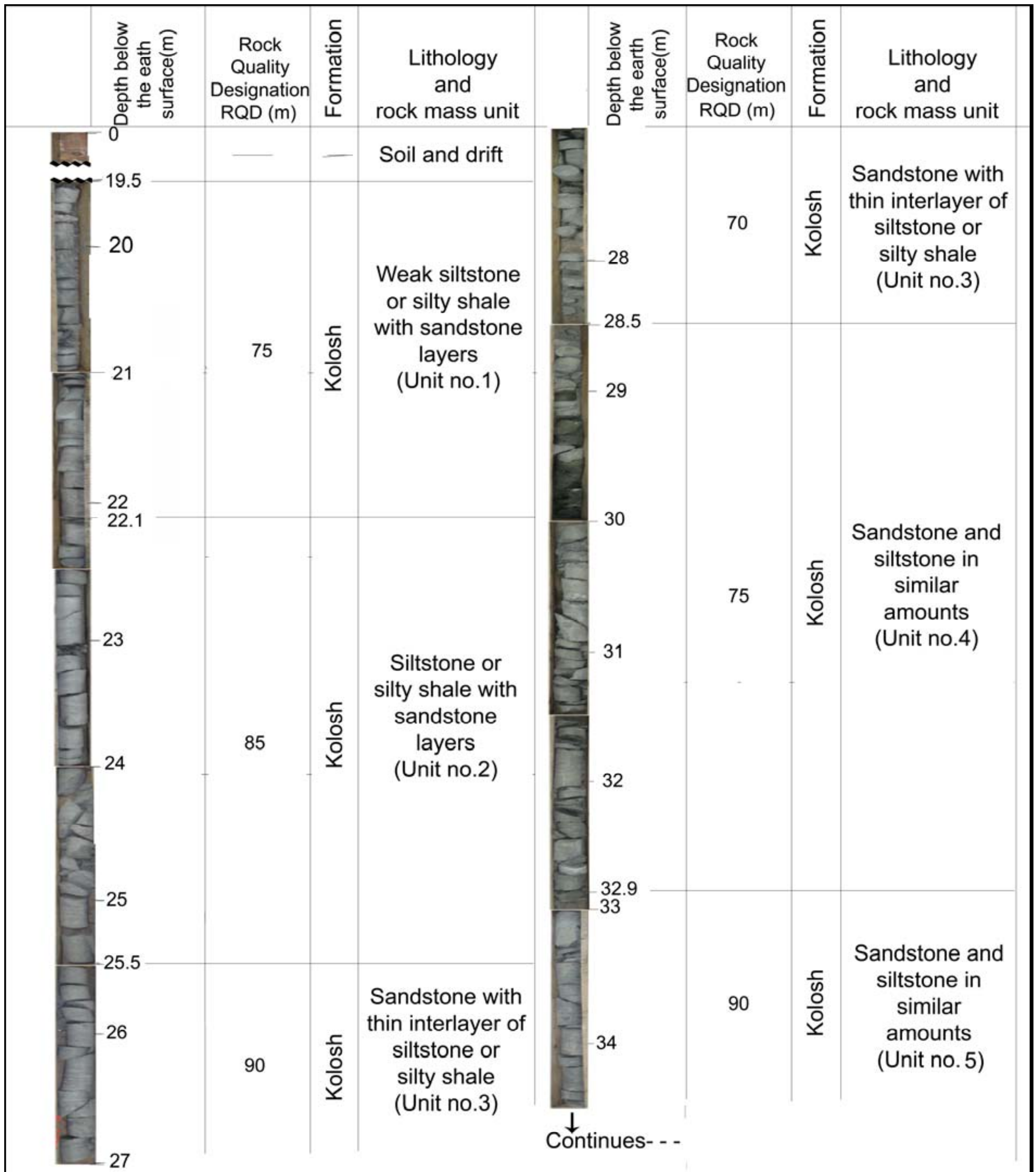
Rock mass characterization for each unit of surface sections and bore holes was assigned from various sources, some of them related to previous chapters, such as rock type and unconfined compressive strength ($б_{ci}$) of intact rock (chapter three), "condition of discontinuities, groundwater condition and strike & dip orientation of foundation rocks" from field or laboratory measurements and description, and comparison of these information with standard tables and figures presented in chapter four. Other characteristics, such as volumetric joint count (Jv), Rock Quality Designation (RQD) and block volume (Vb) for surface section units were measured or estimated (except unit no.1-surface section no.1 of Gercus Formation, due to difficulty in obtaining measurements about joint sets) by equations related to each one , which depends on the field measurements of discontinuities spacing.



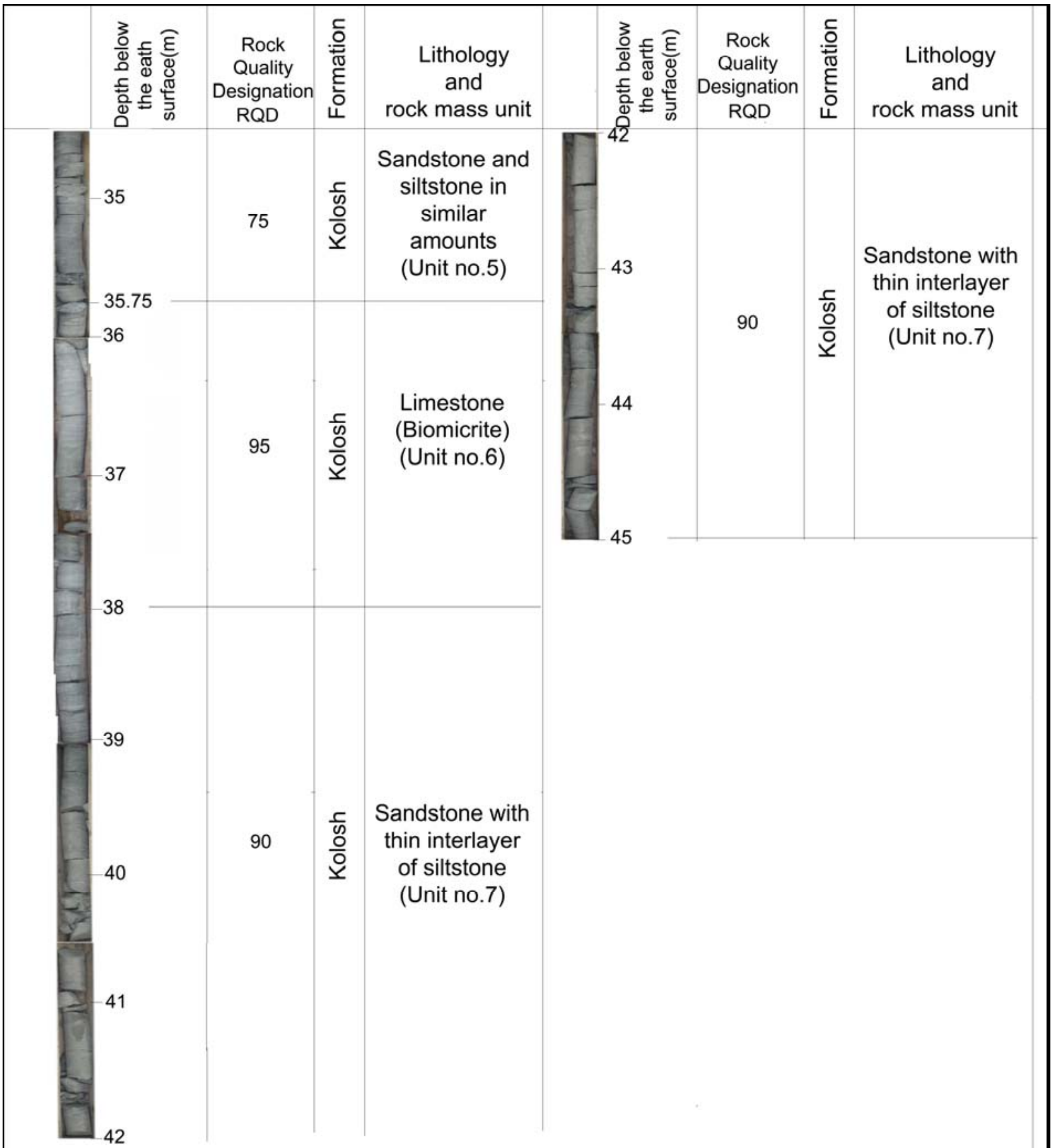
Fig(5-2) Vertical section at borehole no.1



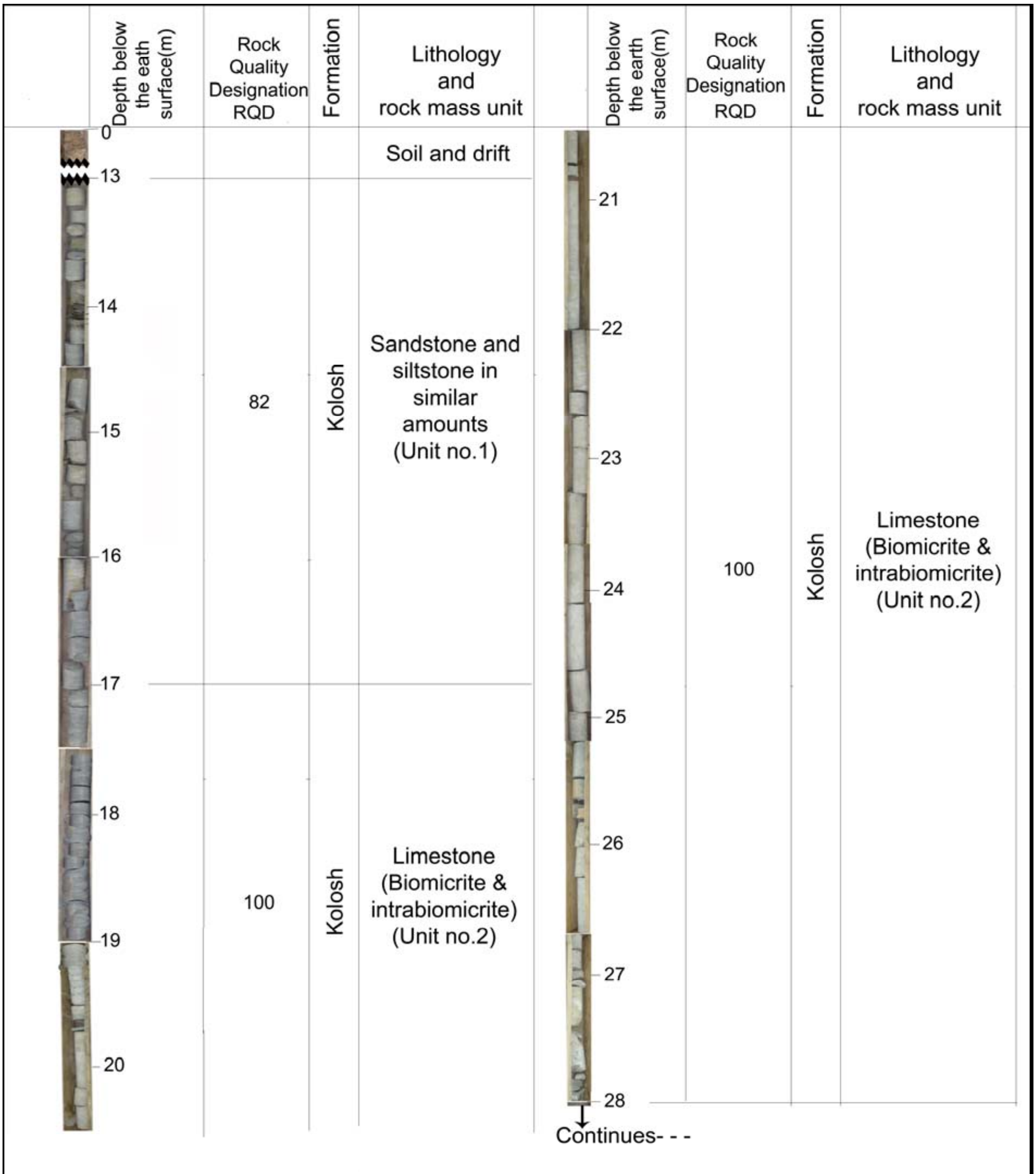
Complementary of Fig (5-2)



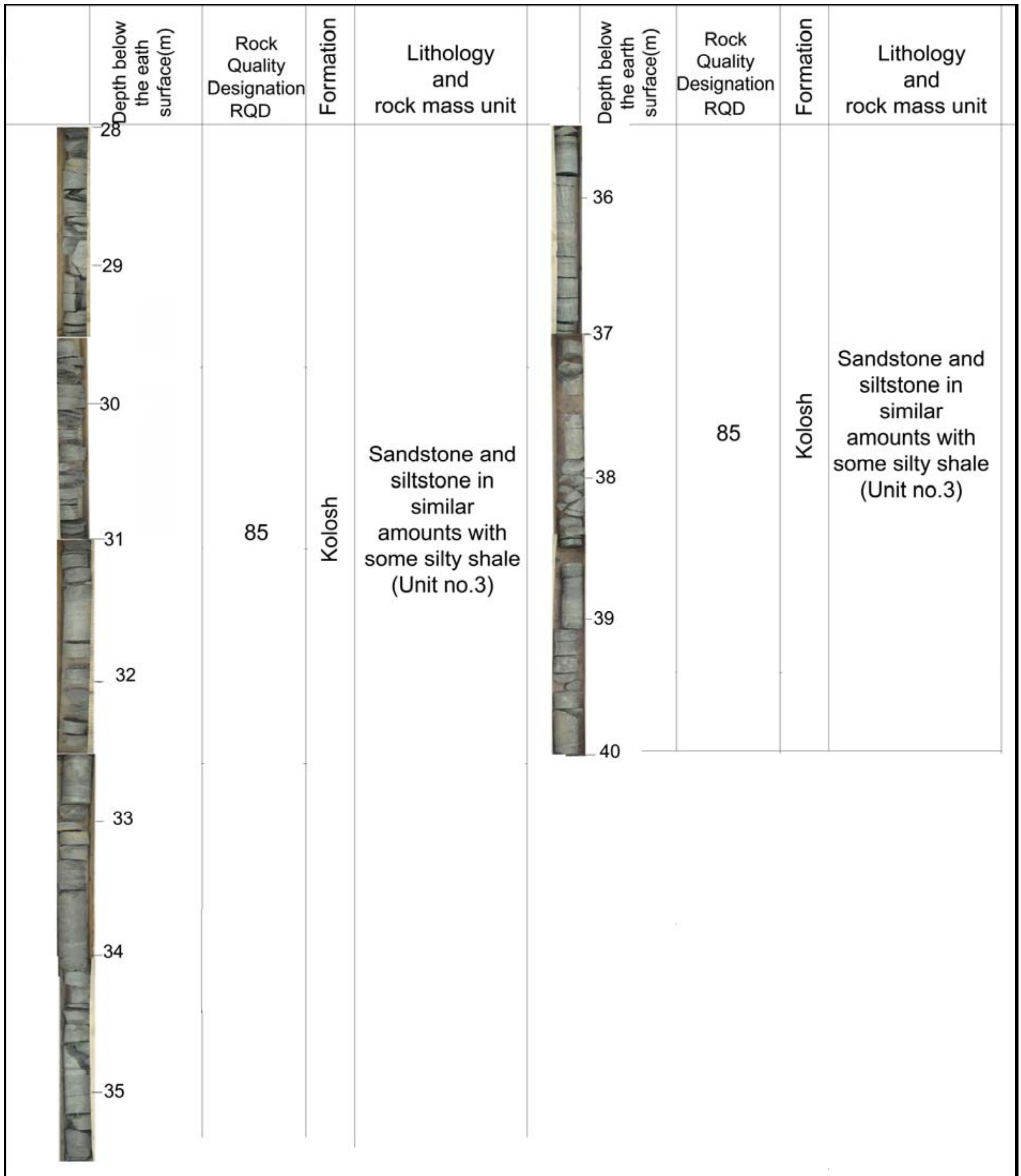
Fig(5-3) Vertical section at borehole no. 2



Complementary of Fig (5-3)



Fig(5-4) Vertical section at borehole no. 3



Complementary of Fig (5-4)

The results of Jv, Vb and RQD are shown in tables (5-2, 5-3 & 5-4) and the detailed information of table (5-4) is shown in Appendix-A (Tables “A-1, A-2, ……………, A-12 and A-13”).

Table (5-2) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.1- unit no.2 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)	Set spacing and frequency				Average spacing(m)	Average frequency*
	Spacing (m)		Max. frequency	Min. frequency		
	Min.	Max.				
Bedding plane (S_o)	0.2	1.2	5/m	0.83/m	0.70	1.42
Joint set 1 (S_1)	0.2	1	5/m	1/m	0.60	1.66
Joint set 2 (S_2)	0.25	1.25	4/m	0.80/m	0.75	1.33
Random joint **						
Calculations: Volumetric joint count Jv=∑ freguencies						4.41 Average Jv
Calculations: Block volume*** $Vb_0=S_0*S_1*S_2$					0.315 m^3 Average Vb_o	

- RQD = 110 -2.5 Jv………..(Palmstrom, 2005)

RQD = 110 - (2.5 * 4.41) = 98.97

-Equivalent block volume: Vb =βJv^{-3} …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β =20+7(S max./S min.)(3/nj)…...(Palmstrom,1995 and 1996b)

= 20+21 (S max./S min * nj)

Where: S max=Maximum average spacing. S min=Minimum average spacing

nj is an adjusting factor where less or more than three sets occur, which represents a rating for the actual number of joint sets.

-The ratings of nj are given as:

3 joint sets + random → nj=3.5 , 3 joint sets → nj=3

2 joint sets + random → nj=2.5 , 2 joint sets → nj=2

1 joint set + random → nj=1.5 , 1 joint sets only → nj=1

- β = 20 +21 (0.75 / 0.60 * 3) = 28.75

- Vb = 28.75 * (1 / 4.41^3) m^3 = 0.335m^3 = 3.35 * 10^5 cm^3

* Average frequency=1/Average spacing……..(Palmstrom, 2005)

**For random joints, a spacing of 5m for each random joint is used in the *Jv* calculation.

***Vb_o = Block volume for joint intersection at approximately right angles.

Table (5-3) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.1- unit no.3 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)	Set spacing and frequency				Average spacing(m)	Average frequency*
	Spacing (m)		Max. frequency	Min. frequency		
	Min.	Max.				
Bedding plane (S_o)	0.3	1.5	3.33/m	0.660/m	0.90	1.11
Joint set 1 (S_1)	0.3	1.6	3.33/m	0.625/m	0.95	1.05
Joint set 2 (S_2)	0.4	1.8	2.50/m	0.550/m	1.10	0.91
Random joint **						
Calculations: Volumetric joint count Jv=∑ freguencies						3.07 Average Jv
Calculations: Block volume*** $Vb_0=S_0*S_1*S_2$					0.94 m^3 Average Vb_o	

- RQD = 110 -2.5 Jv = 100 (because Jv < 4)

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (1.10 / 0.90 * 3) = 28.55

- Vb = 28.55 * (1 / 3.07^3) m^3 = 0.986 m^3 = 9.86 * 10^5 cm^3

* Average frequency=1/Average spacing.
**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (5 - 4) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in the rock mass surface sections

Surface section	Geologic unit	Rock mass units	Jv (joints/m^3)	RQD	Vb (m^3)
1	Sinjar Formation	4	75	0	64 *$10^{-6}$$m^3$ =64 cm^3
		5	4.03	99.92	0.445 m^3 =4.45 *$10^5$$cm^3$
2		1	3.86	100	0.534 m^3 =5.34 *10cm^3
		2	3.91	100	0.529 m^3 =5.29 *$10^5$$cm^3$
		3	4.55	98.62	0.307 m^3 =3.07 *$10^5$$cm^3$
		4	5.97	95.07	0.141 m^3 =1.41 *$10^5$$cm^3$
		5	3.50	100	0.639 m^3 =6.39 *$10^5$$cm^3$
3		1	5.48	96.30	0.18 m^3 =1.8 *$10^5$$cm^3$
		2	4.34	99.15	0.444 m^3 =4.44 *$10^5$$cm^3$
		3	13.99	75.02	0.012 m^3 =1.2 *$10^4$$cm^3$
		4	3.85	100	0.475 m^3 =4.75 *$10^5$$cm^3$
	Unconformity between Sinjar and Kolosh formation	5	9.28	86.80	0.038 m^3 =3.8 *$10^4$$cm^3$
		6	19.04	62.40	0.0059 m^3 =5.9 *$10^3$$cm^3$

Each borehole contains a unit of carbonate rocks of Kolosh Formation (units 2, 6 and 2 in the boreholes no.1, 2 and 3 respectively), which are treated separately from Flysch rocks of Kolosh Formation, though they also belong to

Kolosh Formation. These carbonate units are characterized by obvious discontinuities.

The Palmstrom method for calculating the weighted joint density (wJd) (which is also equal to the volumetric joint count, wJd ≈ Jv) (Palmstrom, 1996b,) was followed in these three carbonate units, as in the tables (5-5, 5-6 & 5-7).

Most of the limestone rock masses in the proposed dam site have slightly long (prismatic) and slightly flat shapes. According to Palmstrom (2005) the block shape factor (β) of 30 is suitable as an average value for these shapes; therefore the value of 30 was taken for (β) in these three carbonate units, then the RQD, block volume(Vb) and equivalent block diameter (Db) (which is equal to the average spacing (S_{avg}) of the discontinuity sets) can be calculated, as shown in table (5-8). (**Note:** Db can be calculated from the following equation (Palmstrom, 1996a)):

$$Db = \beta_o / \beta\ (Vb)^{1/3} = 27 / \beta\ (Vb)^{1/3}$$

Where: β_o = Block shape factor for cube blocks and equals to 27.

The average dip and strike orientation of significant discontinuities (bedding plane) were estimated by Schmidt equal area stereographic projection, Fig (5-5, 5-6 & 5-7),then these averages were compared with table (4-9) to determine “the condition of the strike and dip of foundation rocks relative to the dam project in the RMR classification system, as very favorable,…………, very unfavorable, as in tables (4-2 & 4-4)”, or “finding the rating of the adjusting factor for discontinuity orientation (dip) related to dam stability (R_{STA}) from table (4-9) in the DMR classification system”.

In the DMR classification system, the geometric correcting factor (CF) for relative orientation of discontinuities and dam axis was found because the dip orientation of the significant discontinuities is not parallel to the upstream-downstream direction of the dam axes (profiles), then this factor must be multiplied by R_{STA} to determine the final rating of strike and dip orientation in the DMR, which is equal to R_{STA}*CF, as in table (5-9).

Table (5-5) The calculation of the weighted joint density (wJd) from registration of jointing in borehole no.1- unit no.2(carbonate rocks) (at depth 40.65- 47.50m below the ground surface)

Angle of interval δ	Rating of *fi* factor	Number of joints(n) with each interval	Number of weighted joints in each interval n* *fi*	Total number of weighted joints Nw=Σ n* *fi*	Leng -th(L) (m)	wJd = (1/L) Nw
>60°	1	5	5	39	6.85	5.69
31-60	1.5	10	15			
16-30	3.5	2	7			
<16°	6	2	12			
Note: wJd ≈ Jv . Where: Jv = Volumetric joint count = 5.69						

Table (5-6) The calculation of the weighted joint density (wJd) from registration of jointing in borehole no.2 - unit no.6(carbonate rocks) (at depth 35.75- 38m below the ground surface)

Angle of int erval δ	Rating of *fi* factor	Number of joints(n) with each interval	Number of weighted joints in each interval n* *fi*	Total number of weighted joints Nw=Σ n* *fi*	Leng -th(L) (m)	wJd = (1/L) Nw
>60°	1	6	6	13.5	2.25	6
31-60	1.5	1	1.5			
16-30	3.5	0	0			
<16°	6	1	6			
Note: wJd ≈ Jv . Where: Jv = Volumetric joint count = 6						

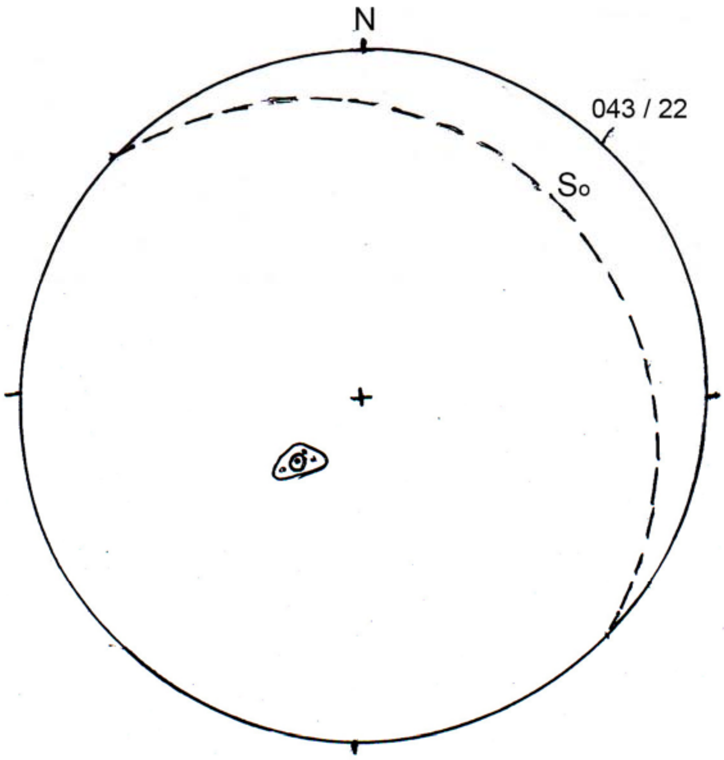
Table (5-7) The calculation of the weighted joint density (wJd) from registration of jointing in borehole no.3 - unit no.2(carbonate rocks) (at depth 17- 28m below the ground surface)

Angle of interval δ	Rating of *fi* factor	Number of joints(n) with each interval	Number of weighted joints in each interval n* *fi*	Total number of weighted joints Nw=Σ n* *fi*	Leng -th(L) (m)	wJd = (1/L) Nw
>60°	1	1	1	26	11	2.36
31-60	1.5	0	0			
16-30	3.5	2	7			
<16°	6	3	18			
Note: wJd ≈ Jv . Where: Jv = Volumetric joint count = 2.36						

Table (5- 8) Calculation of the RQD, Vb & Db (average spacing) in boreholes no. 1, 2 & 3 for the units 2, 6 & 2 respectively (Note: Values of Jv are from tables 5-5, 5-6 & 5-7)

B.h. no.	Unit	Depth below surface (m)	Unit thickness(m)	Jv (joint/m^3)	RQD	β	Vb(m^3) = β Jv^{-3}	Db(m) = $S_{avg.}$ =27/ β $(Vb)^{1/3}$
1	2	40.65-47.50	6.85	5.69	95.77	30	0.162	0.490
2	6	35.75-38	2.25	6	95	30	0.138	0.465
3	2	17-28	11	2.36	100	30	2.280	1.184

Where: B.h = Bore hole. Jv = Volumetric joint count.
Vb = Block volume. Db= Equivalent block diameter=Average spacing.



Fig(5-5) Stereographic projection showing the great circle (S_o) for average attitude (043 / 22) of bedding plane at surface section no.1

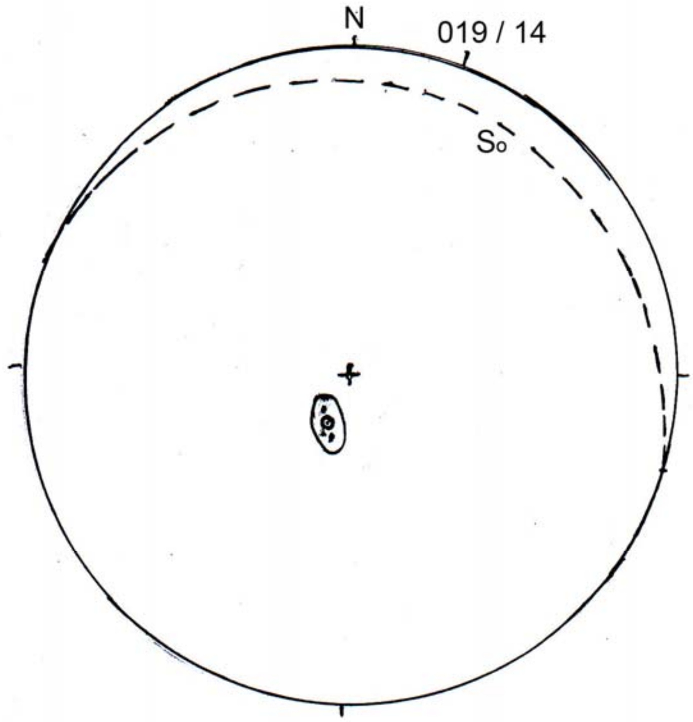


Fig (5-6) Stereographic projection showing the great circle (S_o) for average attitude(019/14) of bedding plane at surface section no.2

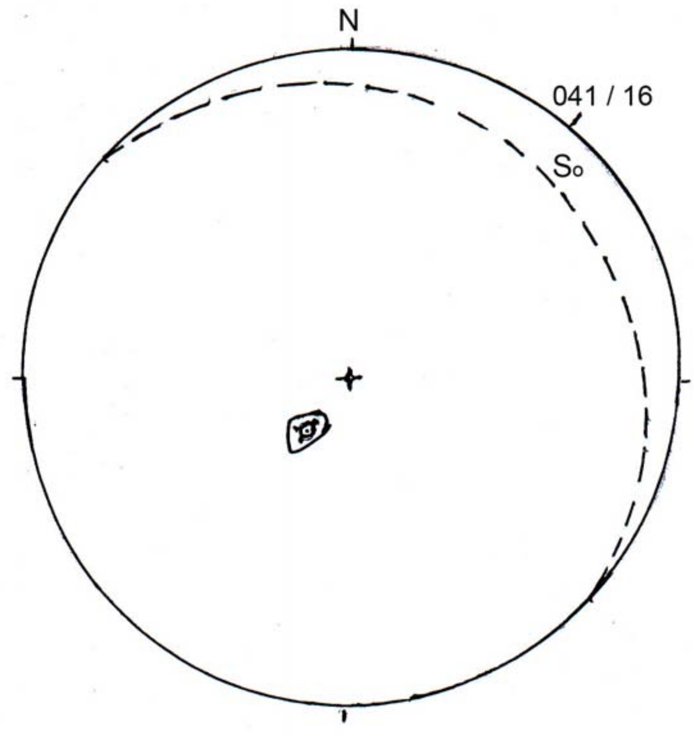


Fig (5- 7) Stereographic projection showing the great circle (S_o) for average attitude(041/16) of bedding plane at surface section no.3

Table (5- 9) Geometric correcting factor (CF), adjusting factor for joint orientation (R_{STA}) for each of gravity and fill dams. Apparent dip angle, dam axis orientation and horizontal distance of the proposed dam profiles

	Profile a-b (dam axis)		Profile c-d (dam axis)		Profile e-f (dam axis)		Remarks
	Gravity dam	Fill dam	Gravity dam	Fill dam	Gravity dam	Fill dam	
α_d	N 75 E		N 75 E		N 50 E		From Fig(5-1)
α_j	N 43 E		N 19 E		N 41 E		From Fig (5-5, 5-6 & 5-7)
CF	0.22		0.029		0.71		
Dip angle(bedding plane)	22° upstream		14° upstream		16° upstream		From Fig (5-5, 5-6 & 5-7)
Apparent dip angle (along dam axis)	12° upstream		12° upstream		3° upstream		From normal projection
R_{STA}	-7	0	-7	0	-7	0	Comparison of dip with table (4-9)
R_{STA} * CF	-1.54	0	-0.20	0	-4.97	0	
Dam axis orientation	N15W – S15E		N15W – S15E		N40W – S40E		From Fig(5-1)
Horizontal distance(m)	340		318		274		

Where: α_d = Upstream- Downstream direction of the dam axis.
α_j = Dip direction of significant governing discontinuities (bedding plane) for dam stability.
CF = Geometric correcting factor for relative orientation of significant discontinuities (joints) and dam axis.
$= (1 - \sin | \alpha_d - \alpha_j |)^2$
R_{STA} = Adjusting factor for discontinuity orientation and angle (dip) related to dam stability (Note: It is estimated from comparison of dip angle and direction with table (4 – 9).

The material constant (mi) of intact rock that is used in estimation of the rock mass strength is quoted from table (4-25). Fortunately, in terms of the estimation of rock mass strength, the value of the constant mi is the least sensitive of the three parameters required (the other two parameters are GSI and $б_{ci}$). The average values given in table (4-25) are sufficiently accurate for most practical applications (Marinos and Hoek, 2001).

The modulus ratio (MR) of intact rock is obtained from table (4-7), the value of MR from this table was used in the calculation of the intact modulus (Ei).

The summary of rock mass characterization for surface sections and bore holes are shown in tables (5-10, 5-11, 5-12, 5-13, 5-14, 5-15& 5-16):

Table (5-10) Rock mass characterization in surface section no. 1

Geologic unit		Gercus Formation	Sinjar Formation				Remarks
Rock mass unit		1	2	3	4	5	
Elevation(a.s.l) (m)		725 - 719	719 - 703	703 - 683	683 - 681	681 - 677	
Thickness of the unit (m)		6	16	20	2	4	
Rock type		a)Siltstone b)Silty Shale c)Sandstone; Mol-asse (M6)*	a) Micrite b) Biomicrite	a) Biomicrite b) Intrabiomicrite	Sandy Marlstone	Biomicrite	From tables (3-1) , (3-3) &(3-6)
Strength of intact rock material $UCS_{(50)}$ (MPa)		a) 18.105 b) 7 c)25.939; Avg. 20.58	a) 40.14 b) 75.28; Avg. 57.71	a) 56.59 b) 53.22; Avg. 54.905	10	75.54	
RQD		-	98.97	100	0	99.92	From tables (5-2, 5-3 & 5-4)
Average Spacing of discontinu-ites (m)	Set o (S_o)	0.15	0.70	0.90	0.04	0.85	From tables (5-2), (5-3),(A-1) & (A-2) (A = Appendix A)
	Set 1 (S_1)	No measurements of joint sets, because most of the unit is covered by soil	0.60	0.95	0.04	0.75	
	Set 2 (S_2)		0.75	1.10	0.04	0.65	
	Set 3 (Random)						
	Min. Spacing	-	0.60	0.90	0.04	0.65	
Condition of discontinuities		Smooth surfaces, moderately weathered and altered surfaces, no infilling, separation (1-2mm).	Rough surfaces, slightly weathered; no infling, no separation.	Rough-very rough surfaces, slightly weathered, hard filling < 5mm, no separation.	Smooth-slightly rough surfaces, slightly-mod. Weathered, hard filling < 5mm (some discontinuities)	Rough-very rough surfaces, slightly weathered, no infilling, separation < 1mm.	From field observations
Ground water condition	RMR	Dry	Dry	Dry	Dry	Dry	From field
	DMR (r_u)**	0.25	0.25	0.25	0.25	0.25	
***Strike and dip orientation of foundation rocks	Fill dam	Very favorable	Very favorable	Very favorable	Very favorable	Very favorable	Comparison of (Dip = 043 / 22) with table (4-9)
	Gravity dam	Fair	Fair	Fair	Fair	Fair	
Block volume (Vb) (Aeverage)		-	0.335 m^3 = 3.35 * 10^5 cm^3	0.986 m^3 = 9.86 * 10^5 cm^3	64 * 10^{-6} m^3 = 64 cm^3	0.445 m^3 = 4.45 * 10^5 cm^3	From table (5-2), (5-3) & (5-4)
Volumetric joint count (jointl/m^3)		-	4.41	3.07	75	4.03	
Material constant of intact rock (mi)		13 (From table"4-25" & dep-ending on the table"4-26")	9	9	7	9	From table (4-23)
Modulus ratio of intact rock (MR)		315(From table"4-7" & dep-ending on the table"4-26")	900	900	175	900	From table (4-7)

Where: (M6)* =Type of molasse deposits (it has the GSI = 33 "from rock type , condition of discontinuities, as in the table (5-24)".
(r_u)** = Water pressure ratio = 0.25 (mean value, when rock is saturated, as in the case of dam for upstream parts) (Romana, 2003a)
***Dip (average) = 043 / 22 (from Fig "5-5")

Table (5-11) Rock mass characterization in surface section no. 2

Geologic unit		Sinjar Formation					Remarks
Rock mass unit		1	2	3	4	5	
Elevation(a.s.l) (m)		735 - 722	722 - 712	712 - 704	704 - 697	697 - 692	
Thickness of the unit (m)		13	10	8	7	5	
Rock type		Intramicrite	a)Micrite b)Intramicrite	a)Intramicrite b)Intramicrite c)Intramicrite	Biomicrite	a)Biomicrite b)Intrabiomicrite	From tables (3-1) & (3-3)
Strength of intact rock material $UCS_{(50)}$ (MPa)		62.40	a)51.11 b)44.70 Avg.= 47.905	a)62.09 b)55.09 c)47.67 Avg. 54.95	41.27	a)86.52 b)62.16 Avg. 74.34	
RQD		100	100	98.62	95.07	100	From table (5-4)
Average Spacing of discontinu-ites (m)	Set o (S_o)	0.65	0.80	0.725	0,45	0.85	From tables (A-3), (A-4), (A-5), (A-6) & (A-7)
	Set 1 (S_1)	0.75	0.60	0.60	0.45	0.90	
	Set 2 (S_2)	1	1	0.90	0.65	0.825	
	Set 3 (Random)			5 / 2 = 2.5			
	Min. Spacing	0.65	0.60	0.60	0.45	0.825	
Condition of discontinuities		Rough surfaces, slightly weathered, no infilling, no separation.	Rough surfaces, slightly weathered, no infilling, no separation	Rough-slightly rough surfaces, slightly weathered,no infilling, separation < 1mm	Slightly rough surfaces, slightly weathered, no infilling ,separation > 5mm	Rough surfaces, non-slightly weathered, no infill-ing, no separation.	From field observations
Ground water condition	RMR	Dry	Dry	Dry	Dry	Dry	From field
	DMR (r_u)*	0.25	0.25	0.25	0.25	0.25	
**Strike and dip orientation of foundation rocks	Fill dam	Very favorable	Very favorable	Very favorable	Very favorable	Very favorable	Comparison of (Dip = 019 / 14) with table (4-9)
	Gravity dam	Fair	Fair	Fair	Fair	Fair	
Block volume (Vb) (Aeverage)		0.534 m^3 = 5.34 * 10^5 cm^3	0.529 m^3 = 5.29 * 10^5 cm^3	0.307 m^3 = 3.07 * 10^5 cm^3	0.141 m^3 = 1.41 * 10^5 cm^3	0.639 m^3 = 6.39 * 10^5 cm^3	From table (5-4)
Volumetric joint count (jointl/m^3)		3.86	3.91	4.55	5.97	3.50	
Material constant of intact rock (mi)		9	9	9	9	9	From table (4-23)
Modulus ratio of intact rock (MR)		900	900	900	900	900	From table (4-7)

Where: (r_u)* = Water pressure ratio = 0.25 (mean value, when rock is saturated, as in the case of dam for upstream parts) (Romana, 2003a)
**Dip (average) = 019 / 14 (from Fig "5-6")

Table (5-12) Rock mass characterization in surface section no. 3

Geologic unit		Sinjar formation				Unconformity		Remarks
Rock mass unit		1	2	3	4	5	6	
Elevation(a.s.l) (m)		745-731	731-711	711-704.5	704.5-700	700-694	694-686	
Thickness of the unit (m)		14	20	6.5	4.5	6	8	
Rock type		a)Micrite b)Biomicrite	a)Intramicrite b)Intramicrite	Biomicrite	a)Biomicrite b) Intrabiomicrite	Pelintrabiosparite	Pelintrabiosparite	From tables (3-1) & (3-3)
Strength of intact rock material $UCS_{(50)}$ (MPa)		a)53.34 b)73.08 Avg.=63.21	a)92.26 b)56.88 Avg.=74.57	77.35	a)86.18 b)53.55 Avg.=69.86	67.73	68.67	
RQD		96.3	99.15	75.02	100	86.8	62.4	From table(5-4)
Average Spacing of discontinuites (m)	Set o (S_o)	0.625	0.85	0.15	0.80	0.25	0.10	From tables (A-8), (A-9), (A-10), (A-11), (A-12)& (A-13)
	Set 1 (S_1)	0.45	1.05	0.30	0.87	0.375	0.175	
	Set 2 (S_2)	0.60	0.45	0.25	0.95	0.45	0.30	
	Set 3 (Random)				5/2=2.5	5/2=2.5		
	Min. Spacing	0.45	0.45	0.15	0.80	0.25	0.10	
Condition of discontinuities		Very rough-rough surfaces, non-slightly weathered, no infling, no separation.	Rough-slightly rough surfaces, slightly weath-ered, no infilling, separation (1-5mm)	Rough surfaces, slightly- mod., weathered, no infilling, separation <1mm	Slightly rough-rough surfaces, slightly weathered, no infilling, separation <1mm	Slightly rough surfaces, slightly weathered, no infilling, separation < 1mm	Slightly rough surfaces , slightly weathered, no in-filling, separation (1-2mm), persis-tence (20cm-3m)	From field observations
Ground water condition	RMR	Dry	Dry	Dry	Dry	Dry	Dry	From field
	DMR (r_u)*	0.25	0.25	0.25	0.25	0.25	0.25	
**Strike and dip orientation of foundation rocks	Fill dam	Very favorable	Very favorable	Very favorable	Very favorable	Very favorable	Very favorable	Comparison of (Dip=041 / 16) with table (4-9)
	Gravity dam	Fair	Fair	Fair	Fair	Fair	Fair	
Block volume (Vb) (Aeverage)		$0.18m^3$ $=1.8*10^5cm^3$	$0.4444m^3$ $=4.44*10^5cm^3$	$0.012m^3$ $=1.2*10^4cm^3$	$0.475m^3$ $=4.75*10^4cm^3$	$0.038m^3$ $=3.8*10^4cm^3$	$0.0059m^3$ $=5.9*10^3cm^3$	From table (5-4)
Volumetric joint count ($jointl/m^3$)		5.48	4.34	13.99	3.85	9.28	19.04	
Material constant of intact rock (mi)		9	9	9	9	10	10	From table (4-23)
Modulus ratio of intact rock (MR)		900	900	900	900	700	700	From table (4-7)

Where: (r_u)* = Water pressure ratio = 0.25 (mean value, when rock is saturated, as in the case of dam for upstream parts) (Romana, 2003a).
**Dip (average) = 041 / 16 (from Fig "5-7").

Table (5-13) Rock mass characterization of blocky carbonate rocks in boreholes no. 1, 2 & 3

<table>
<tr><td colspan="2">Geologic unit</td><td colspan="5">Kolosh Formation</td><td>Remarks</td></tr>
<tr><td colspan="2">Bore hole no.</td><td>1</td><td colspan="2">2</td><td colspan="2">3</td><td></td></tr>
<tr><td colspan="2">Rock mass unit</td><td>2</td><td colspan="2">6</td><td colspan="2">2</td><td></td></tr>
<tr><td colspan="2">Depth below surface(m)</td><td>40.65 – 47.50</td><td colspan="2">35.75 – 38</td><td colspan="2">17 - 28</td><td></td></tr>
<tr><td colspan="2">Elevation(a.s.l)(m)</td><td>698.35 – 691.50</td><td colspan="2">644.25 – 642</td><td colspan="2">655 - 644</td><td></td></tr>
<tr><td colspan="2">Thickness of the unit (m)</td><td>6.85</td><td colspan="2">2.25</td><td colspan="2">11</td><td></td></tr>
<tr><td colspan="2">Rock type</td><td>Biomicrite</td><td colspan="2">a) Biomicrite
b) Biomicrite</td><td colspan="2">a) Biomicrite
b) Biomicrite
c) Intrabiomicrite</td><td rowspan="2">From tables (3-2) & (3-4)</td></tr>
<tr><td colspan="2">Strength of intact rock material $UCS_{(50)}$(MPa)</td><td>42.512</td><td>a) 58.514
b) 44</td><td>Average
51.257</td><td>a) 51.744
b) 55.185
c) 53.271</td><td>Average
53.40</td></tr>
<tr><td colspan="2">RQD (%)</td><td>95.77</td><td colspan="2">95</td><td colspan="2">100</td><td rowspan="3">From table (5-8)</td></tr>
<tr><td rowspan="2">Spacing (m)</td><td>Average</td><td>0.490</td><td colspan="2">0.465</td><td colspan="2">1.184</td></tr>
<tr><td>Min.</td><td>0.490</td><td colspan="2">0.465</td><td colspan="2">1.184</td></tr>
<tr><td colspan="2">Surface condition of discontinuities (predominantly bedding planes)</td><td>Slightly rough-rough surfaces,non-slightly weathered, no infill-ing,separation<1mm</td><td colspan="2">Slightly rough-rough surfaces,non-slightly weathered, no infill-ing, separation(1-2mm).</td><td colspan="2">Very rough surfaces , non-slightly weath-ered, hard filling (Pyrite) < 5mm, no separation.</td><td>From descri-ption of bore holes in laboratory</td></tr>
<tr><td rowspan="2">Ground water condition</td><td>RMR</td><td>Saturated</td><td colspan="2">Saturated</td><td colspan="2">Saturated</td><td>From field</td></tr>
<tr><td>DMR(r_u)*</td><td>0.25</td><td colspan="2">0.25</td><td colspan="2">0.25</td><td></td></tr>
<tr><td rowspan="2">**Strike and dip orientation of foundation rocks</td><td>Fill dam</td><td>Very favorable</td><td colspan="2">Very favorable</td><td colspan="2">Very favorable</td><td rowspan="2">Comparison of dip with table (4-9)</td></tr>
<tr><td>Gravity dam</td><td>Fair</td><td colspan="2">Fair</td><td colspan="2">Fair</td></tr>
<tr><td colspan="2">Material constant of intact rock (mi)</td><td>9</td><td colspan="2">9</td><td colspan="2">9</td><td>From table (4-23)</td></tr>
<tr><td colspan="2">Modulus ratio of intact rock (MR)</td><td>900</td><td colspan="2">900</td><td colspan="2">900</td><td>From table (4-7)</td></tr>
<tr><td colspan="2">Block volume (Vb) (Average)</td><td>0.162 m^3
= 1.62 * $10^5 cm^3$</td><td colspan="2">0.138 m^3
= 1.38 * $10^5 cm^3$</td><td colspan="2">2.28 m^3
= 2.28 * $10^6 cm^3$</td><td>From table (5-8)</td></tr>
<tr><td colspan="2">Volumetric joint count (Jv) (joint / m^3)</td><td>5.69</td><td colspan="2">6</td><td colspan="2">2.36</td><td>From tables (5-5,5-6 & 5-7)</td></tr>
</table>

(r_u)* = Water pressure ratio = 0.25 (mean value, when rock is saturated, as in the case of dam for Upstream parts) (Romana, 2003a).

(**):- Dip angle of significant discontinuity (bedding plane here) > 10° (from cores observation).

- Dip direction (according to their nearness to the surface sections) :
  - Bore hole no. 1 ≈ 043 (N 43 E) (Supported by section no. 1)
  - Bore hole no. 2 ≈ 019 (N 19 E) (Supported by section no. 2)
  - Bore hole no. 3 ≈ 041 (N 41 E) (Supported by section no. 3)

Note: (1) Elevation of the bore holes no.1, 2 & 3 are 739, 680 & 672m above sea level respectively

(2) Elevation of the ground water table (G.W.T) :
- At bore hole no.1 = 11m below the earth surface (728m above sea level)
- At bore hole no.2 = 5.5m below the earth surface (674.5m above sea level)
- At bore hole no.3 = 6.7m below the earth surface (665.3m above sea level)

Table (5-14) Rock mass characterization and Geological Strength Index (GSI) of Flysch rocks in borehole no. 1

Geologic unit	Kolosh Formation			Remarks
Borehole no.	1			
Rock mass unit	1	3	4	
Depth below surface(m)	37.80 – 40.65	47.50 – 52.50	52.50 - 63	
Elevation(a.s.l)(m)	701.20 – 698.35	691.50 – 686.50	686.50 - 676	
Thickness of the unit (m)	2.85	5	10.5	
Rock type	a)Sandstone. b)Siltstone. (Friable yellow Siltstone and sandstone).	a)Sandstone b)Pebbly sandstone (Thick bedded, very blocky sandstone and pebbly sandstone without fine materials).	a) Pebbly sandstone b) Sandstone (Thick bedded, very blocky pebbly sand-stone and sandstone without fine materials).	From tables (3-2), (3-4) & 3-6) From field observation
Strength of intact rock material $UCS_{(50)}$ (MPa)	a) 5 b) 5 Weighted average: 5	a) 31.167 b)21.054 Weighted average: 26.11	a) 17.50 b) 26.31 Weighted average: 21.90	From tables (3-2) (3-4) & 3-6)& weighted average depending on the table (4-24)
Surface condition of discontinuities (predominantly bedding planes)	Very smooth occa-sionally slicken-sided surfaces, moderately weath-ered, no infilling, separation <1mm	Slightly- rough surfaces, slightly weathered surfaces, no infilling, no separation.	Slightly- rough surfaces, slightly weathered surfaces, no infilling, separation <1mm	From field observation
Ground water condition: RMR	Saturated	Saturated	Saturated	From field
Ground water condition: DMR(r_u)*	0.25	0.25	0.25	
**Strike and dip orientat-ion of found-ation rocks: Fill dam	Very favorable	Very favorable	Very favorable	Comparison of dip with table (4-9)
**Strike and dip orientat-ion of found-ation rocks: Gravity dam	Fair	Fair	Fair	
Material constant of intact rock (mi)	Weighted average 13	Weighted average 17	Weighted average 17	From table(4-23) and depending on the table (4-24)
Modulus ratio of intact rock (MR)	Weighted average 315	Weighted average 275	Weighted average 275	From table(4-7) & depending on the table(4-24)
Type of flysch	E	A	A	From rock type, surface condition of discontinuities & depending on the table(5-25)
Geological Strength Index (GSI)	30	57	55	

Where:(r_u)* = Water pressure ratio = 0.25 (mean value, when rock is saturated, as in the case of dam for Upstream parts) (Romana, 2003a).
(**):- Dip of significant discontinuity (bedding plane here)(20<dip angle<30° (from cores observation).
- Dip direction ≈ 043(N43E) (supported by surface section no. 1, because bore hole no. 1 is near to it more than other sections).

Note: (1) Elevation of the bore hole no. 1 is 739m above sea level.
(2) Ground water table (G.W.T) is at depth 11m below the earth surface(=728m above sea level)

Table (5-15) Rock mass characterization and Geological Strength Index (GSI) of Flysch rocks in borehole no. 2

Geologic unit	Kolosh Formation						Remarks
Borehole no.	**2**						
Rock mass unit	1	2	3	4	5	7	
Depth below surface(m)	19.50 – 22.10	22.10 – 25.50	25.50 – 28.50	28.50 – 32.90	32.90 – 35.75	38 - 45	
Elevation(a.s.l)(m)	660.50 – 657.90	657.90 – 654.50	654.50 – 651.50	651.50 – 647.10	647.10 – 644.25	642 - 635	
Thickness of the unit (m)	2.60	3.40	3	4.40	2.85	7	
Rock type	a) Siltstone b) Sandstone (Weak siltstone or silty shale with sandstone layers).	a) Siltstone b) Sandstone (Siltstone or silty shale with sandstone layers).	a)Sandstone b)Siltstone (Sandstone with thin interlayer of silt-stone or silty shale).	a)Sandstone b)Siltstone (Sandstone and siltstone in similar amounts).	a)Sandstone b)Siltstone (Sandstone and siltstone in similar amounts).	a)Sandstone b)Siltstone (Sandstone with thin interlayer of siltstone).	From tables (3-2) & (3-4); From field observation
Strength of intact rock material $UCS_{(50)}$ (MPa)	a)17.48 b)26.13; Weighted average 22.67	a)23.71 b)28.52; Weighted average 26.59	a)34.65 b)27.93; Weighted average 34.65	a)31.92 b)26.04; Weighted average 30.74	a)34.12 b)29.39; Weighted average 33.17	a)38.05 b)34.63; Weighted average 38.05	From tables (3-2) & (3-4) and weighted average depending on the table (4-24)
Surface condition of discontinuities (predominantly bedding planes)	Smooth-very smooth surfaces, occasion-ally slickensided, slightly weathered surfaces, no infilling.	Smooth-very smooth surfaces, occasionally slicken-sided, slightly wea-thered, no infilling.	Rough-smooth surfaces, slightly weathered surfaces, no infilling.	Smooth- rough surfaces, moderately weathered surfaces, no infilling.	Smooth- rough surfaces, slightly weathered surfaces, no infilling.	Rough-smooth surfaces, slightly weathered surfaces, no infilling.	From field observation
Ground water condition – RMR	Saturated	Saturated	Saturated	Saturated	Saturated	Saturated	From field
Ground water condition – DMR(r_u)*	0.25	0.25	0.25	0.25	0.25	0.25	
**Strike and dip orientation of foundation rocks – Fill dam	Very favorable	Very favorable	Very favorable	Very favorable	Very favorable	Very favorable	Comparison of dip with table (4-10)
**Strike and dip orientation of foundation rocks – Gravity dam	Fair	Fair	Fair	Fair	Fair	Fair	
Material constant of intact rock (mi)	Weighted average 13	Weighted average 13	Weighted average 17	Weighted average 15	Weighted average 15	Weighted average 17	From table(4-23) and depending on the table (4-24)
Modulus ratio of intact rock (MR)	Weighted average 315	Weighted average 315	Weighted average 275	Weighted average 295	Weighted average 295	Weighted average 275	From table (4-7) and depending on the table (4-24)
Type of flysch	E	D	B	C	C	B	From rock type, surface condition of discontinuities & depending on the table (5-25)
Geological Strength Index (GSI)	32	35	44	39	41	44	

Where: (r_u)* = Water pressure ratio = 0.25 (mean value, when rock is saturated, as in the case of dam for upstream parts) (Romana, 2003a).
**Dip angle bedding plane (10<dip angle<20°) (from cores observation), Dip direction ≈ 019 (supported by surface section no. 2 because it is near to it more than other section). Note: (1) Elevation of the bore hole no. 2 is 680m above sea level. (2)Ground water table is at depth 5.5m below the earth surface (= 674.5 m above sea level).

Table (5-16) Rock mass characterization and Geological Strength Index (GSI) of Flysch rocks in borehole no. 3

Geologic unit		Kolosh Formation				Remarks
Borehole no.		3				
Rock mass unit		1		3		
Depth below surface(m)		13-17 m		28-40 m		
Elevation(a.s.l)(m)		659-655		644-632		
Thickness of the unit (m)		4		12		
Rock type		a)Sandstone. b)Siltstone. (Sandstone and siltstone in similar amounts)		a)Sandstone b)Siltstone c)Silty shale (Sandstone and siltstone with silty shale in similar amounts).		From tables (3-2) & (3-4) From field observation
Strength of intact rock material $UCS_{(50)}$ (MPa)		a) 27.72 b) 23.31	Weighted average 26.83	a) 36.213 b) 29.028 c) 22.512	Weighted average 34.12	From tables (3-2) & (3-4) and weighted average depending on the table (4-24)
Surface condition of discontinuities (predominantly bedding planes)		Smooth surfaces, slightly-moderately weathered surfaces, no infilling.		Smooth-rough surfaces, slightly weathered surfaces, no infilling.		From field observation
Ground water condition	RMR	Saturated		Saturated		From field
	DMR(r_u)*	0.25		0.25		
**Strike and dip orientation of foundation rocks	Fill dam	Very favorable		Very favorable		Comparison of dip with table (4-9)
	Gravity dam	Fair		Fair		
Material constant of intact rock (mi)		Weighted average 15		Weighted average 15		From table(4-23) and depending on the table (4-24)
Modulus ratio of intact rock (MR)		Weighted average 295		Weighted average 295		From table(4-7) & depending on the table (4-24)
Type of Flysch		C		C		From rock type, surface condition of discontinuities & depending on the table (5-25)
Geological Strength Index (GSI)		40		42		

Where: (r_u)* = Water pressure ratio = 0.25 (mean value, when rock is saturated, as in the case of dam for Upstream parts) (Romana, 2003a).
(**):- Dip of significant discontinuity (bedding plane here) (10° < dip angle < 20°) (from cores observation).
- Dip direction ≈ 041(N41E) (supported by surface section no. 3, because bore hole no. 3 is Near to it more than other sections).

Note: (1) Elevation of the bore hole no. 3 is 672m above sea level.
(2) Ground water table (G.W.T) is at depth 6.7m below the earth surface(=665.3m above sea level)

The required characteristics for determination of GSI for rock mass units were assigned, which are Structure Rating (SR) (which is depending on the volumetric joint count (Jv)) and Surface Condition Rating (SCR) (which is the sum of Roughness Rating (Rr), Weathering Rating (Rw) and Infilling Rating (Rf), table (5-17)). The results of GSI determination for Sinjar and Kolosh carbonate rocks are plotted on table (5-18) and transferred also to table (5-17). These tables do not include the results of GSI determination for Flysch and Molasse because they were determined by their own GSI-charts.

For correlation between the proposed GSI-chart in this study with those GSI charts of Hoek and of Sonmez & Ulusay, the results of this study are plotted on the three charts (using tables 4-20, 4-16 & 4-19) and the correlation results are shown in table (5-19) and in Fig (5-8).

The proposed GSI-chart also proved to be highly precise and gave GSI values very close to $RMR_{BD(1976)}$ as shown in tables (5-20, 5-21, 5-22 & 5-23) and more precise than Hoek's chart which is based on qualitative description of the rock mass.

After the specification of the rock mass characterization for each rock mass unit, the required characteristics (parameters) of the rock mass classification system (RMR and DMR) were rated, then the rock mass classifications were determined, as in tables (5-20, 5-21, 5-22 & 5-23).

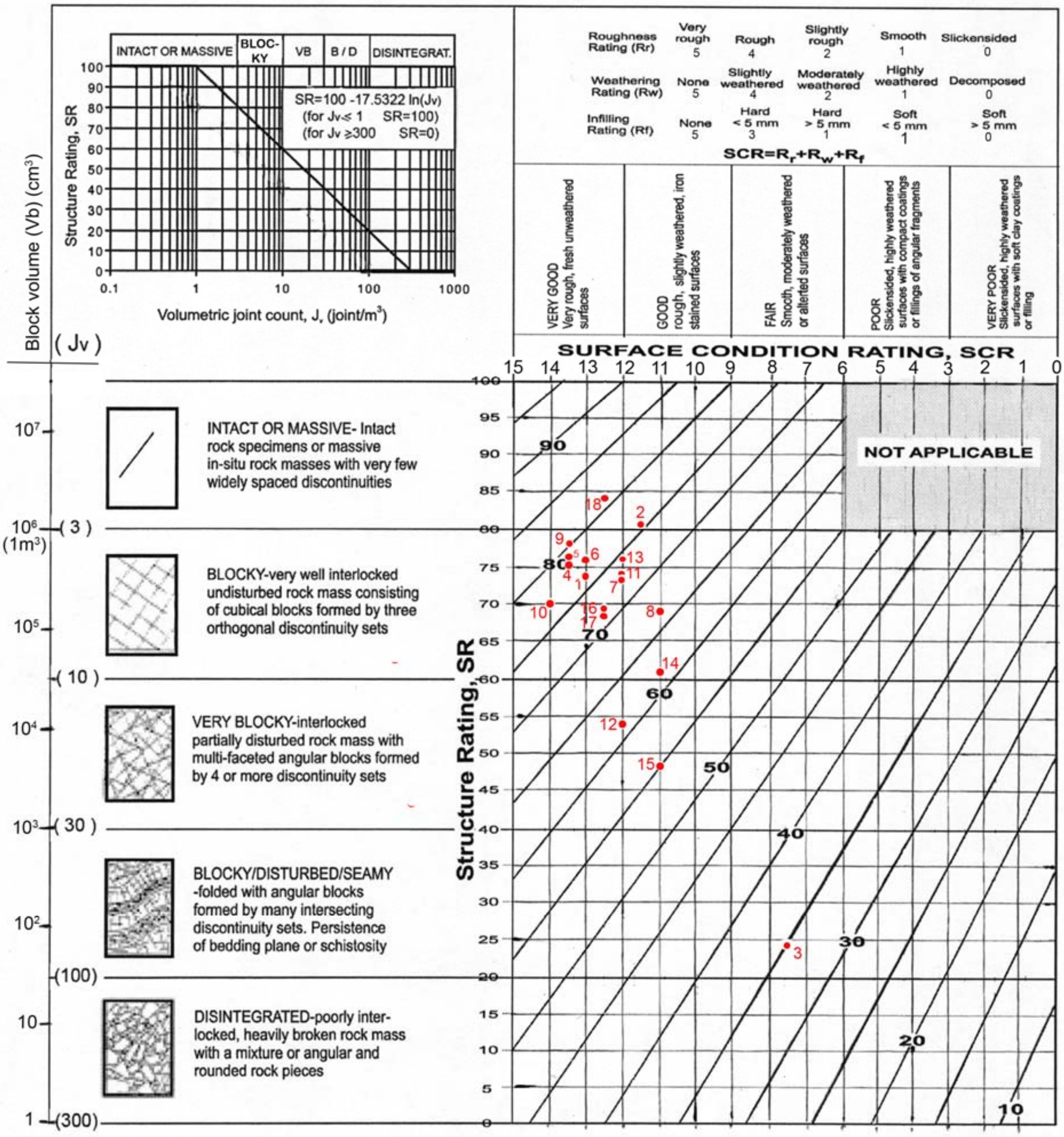
The GSI determination for Molasse rocks of Gercus Formation (Unit no.1 in the surface section no.1) and for Flysch rocks of Kolosh Formation (bore holes no.1, 2 and 3) depends on the determination of type of Molasse, Flysch and also on the determination of surface condition of discontinuities (predominantly bedding plane). These characterizations were assigned for each of Molasse and Flysch types, as in tables (5-10, 5-14, 5-15 & 5-16). From these characterization and depending on tables (4-18 & 4-17), the GSI of Molasse and Flysch units were determined, as in tables (5-24 & 5-25), which were also transferred to tables (5-10, 5-14, 5-15 & 5-16).

Table (5-17) Structure Rating and Surface Condition Rating of discontinuities for determination of GSI of blocky carbonate rock masses in the study area

Geologic unit	Surface section no.	Bore hole no.	Unit no.	Depth below surface (m)	Elevation above sea level (m)	Thickness of the unit (m)	Jv joint / m^3	SR	Condition of discontinuities	R_r	R_w	R_f	SCR = R_r+ R_w+ R_f	*GSI
Sinjar Formation	1	-	2	-	719 – 703	16	4.41	73.98	Rough surfaces, slightly weathered, no infilling	4	4	5	13	76.5
		-	3	-	703 – 683	20	3.07	80.33	Rough-very rough surfaces, slightly weathered, Hard filling < 5mm	4.5	4	3	11.5	75.5
		-	4	-	683 – 681	2	75	24.30	Smooth-slightly rough, slightly-mod. Weathered, hard filling < 5mm	1.5	3	3	7.5	35
		-	5	-	681 – 677	4	4.03	75.56	Rough-very rough surfaces, slightly weathered, no infilling	4.5	4	5	13.5	78
	2	-	1	-	735 – 722	13	3.86	76.32	Rough surfaces, none-slightly weathered , no infilling	4	4.5	5	13.5	79
		-	2	-	722 – 712	10	3.91	76.09	Rough surfaces, slightly weathered, no infilling	4	4	5	13	77.5
		-	3	-	712 – 704	8	4.55	73.43	Rough-slightly rough surfaces, slightly weathered, no infilling	3	4	5	12	73
		-	4	-	704 – 697	7	5.97	68.67	Slightly rough surfaces, slightly weathered, no infilling	2	4	5	11	67
		-	5	-	697 – 692	5	3.50	78.03	Rough surfaces, none-slightly weathered , no infilling	4	4.5	5	13.5	80
	3	-	1	-	745 – 731	14	5.48	70.17	Very rough-rough surfaces, none-slightly weathered, no infilling	4.5	4.5	5	14	77
		-	2	-	731 – 711	20	4.34	74.26	Rough-slightly rough surfaces, slightly weathered, no infilling	3	4	5	12	73
		-	3	-	711 – 704.5	6.50	13.99	53.74	Rough surfaces, slightly-mod. weathered, no infilling	4	3	5	12	61.5
		-	4	-	704.5 –700	4.50	3.85	76.36	Slightly rough-rough surfaces, slightly weathered, no infilling	3	4	5	12	74
Unconformity		-	5	-	700 – 694	6	9.28	60.94	Slightly rough surfaces, slightly weathered, no infilling	2	4	5	11	62
		-	6		694 – 686	8	19.04	48.34	Slightly rough surfaces, slightly weathered, no infilling	2	4	5	11	55
Kolosh Formation	-	1	2	40.6 – 47.5	698.35 – 691.5	6.85	5.69	69.51	Slightly rough-rough surfaces, none-slightly weathered, no infilling	3	4.5	5	12.5	72
	-	2	6	35.75 -38	644.25 – 642	2.25	6	68.58	Slightly rough-rough surfaces, none-slightly weathered, no infilling	3	4.5	5	12.5	71
	-	3	2	17 – 28	655 – 644	11	2.36	84.94	Very rough surfaces, none-slightly weathered, hard filling(Pyrite) < 5mm	5	4.5	3	12.5	81

Where: Jv = Volumetric joint count. SR (Structure Rating) = 100 – 17.5322 *ln* Jv. R_r = Roughness Rating. R_w = Weathering Rating. R_f = Infilling Rating. SCR = Surface Condition Rating. GSI = Geological Strength Index . Jv values and condition of discontinuities are taken from tables (5-10, 5-11, 5-12 & 5-13).
R_r , R_w & R_f values were estimated from comparison condition of Discontinuities with table (4-20) (chart of this study).
*GSI values were taken from table (5 – 18) (proposed chart in this study).

Table (5 -18) GSI determination for blocky carbonate rock masses in the surface sections (Sinjar Formation) and boreholes (Kolosh Formation) from the modified chart of this study



Note: 1 → S1-U2 (S=Surface section - U=Uint) 2 → S1-U3 3 → S1-U4 4 → S1-U5
5 → S2-U1 6 → S2-U2 7 → S2-U3 8 → S2-U4 9 → S2-U5
10 → S3-U1 11 → S3-U2 12 → S3-U3 13 → S3-U4 14 → S3-U5
15 → S3-U6 16 → B1-U2 (B=Bore hole) 17 → B2-U6 18 → B3-U2
(1 - 15) → Surface sections. (16 - 18) → Boreholes.

Table (5-19) Correlation among calculated GSI values plotted on this study chart, on Hoek-chart and on Sonmez & Ulusay-chart for blocky carbonate rock masses in the surface sections(Sinjar Formation) and boreholes (Kolosh Formation)

Geologic unit	Surface section no.	Bore hole no.	Unit no.	Jv joints / m^3	SR this study	Condition of discontinuities	This study					GSI by Hoek's chart	Sonmez & Ulusay					
							R_r	R_w	R_f	SCR=R_r + R_w+R_f	GSI		SR	R_r	R_w	R_f	SCR	GSI
Sinjar Formation	1	-	2	4.41	73.98	Rough surfaces, slightly weathered, no infilling	4	4	5	13	76.5	71	53.83	5	5	6	16	59
		-	3	3.07	80.33	Rough-very rough surfaces, slightly weathered, Hard filling < 5mm	4.5	4	3	11.5	75.5	71	60.17	5.5	5	4	14.5	58
		-	4	75	24.30	Smooth-slightly rough, slightly-mod. Weathered, hard filling < 5mm	1.5	3	3	7.5	35	33	4.24	2	4	4	10	33
		-	5	4.03	75.56	Rough-very rough surfaces, slightly weathered, no infilling	4.5	4	5	13.5	78	73	55.40	5.5	5	6	16.5	60.5
	2	-	1	3.86	76.32	Rough surfaces, none-slightly weathered , no infilling	4	4.5	5	13.5	79	74	56.16	5	5.5	6	16.5	61
		-	2	3.91	76.09	Rough surfaces, slightly weathered, no infilling	4	4	5	13	77.5	74	55.93	5	5	6	16	60
		-	3	4.55	73.43	Rough-slightly rough surfaces, slightly weathered, no infilling	3	4	5	12	73	68	53.28	4	5	6	15	56.5
		-	4	5.97	68.67	Slightly rough surfaces, slightly weathered, no infilling	2	4	5	11	67	64	48.53	3	5	6	14	52
		-	5	3.50	78.03	Rough surfaces, none-slightly weathered , no infilling	4	4.5	5	13.5	80	74	57.87	5	5.5	6	16.5	62.5
	3	-	1	5.48	70.17	Very rough-rough surfaces, none-slightly weathered, no infilling	4.5	4.5	5	14	77	72	50.03	5.5	5.5	6	17	60
		-	2	4.34	74.26	Rough-slightly rough surfaces, slightly weathered, no infilling	3	4	5	12	73	68	54.11	4	5	6	15	57
		-	3	13.99	53.74	Rough surfaces, slightly-mod. weathered, no infilling	4	3	5	12	61.5	57	33.62	5	4	6	15	47
		-	4	3.85	76.36	Slightly rough-rough surfaces, slightly weathered, no infilling	3	4	5	12	74	70	56.20	4	5	6	15	57.5
Unconformity		-	5	9.28	60.94	Slightly rough surfaces, slightly weathered, no infilling	2	4	5	11	62	58	40.81	3	5	6	14	48
		-	6	19.04	48.34	Slightly rough surfaces, slightly weathered, no infilling	2	4	5	11	55	55	28.23	3	5	6	14	45
Kolosh Formation	-	1	2	5.69	69.51	Slightly rough-rough surfaces, none-slightly weathered, no infilling	3	4.5	5	12.5	72	70	48.56	4	5.5	6	15.5	56
	-	2	6	6	68.58	Slightly rough-rough surfaces, none-slightly weathered, no infilling	3	4.5	5	12.5	71	70	48.44	4	5.5	6	15.5	55
	-	3	2	2.36	84.94	Very rough surfaces, none-slightly weathered, hard filling(Pyrite) < 5mm	5	4.5	3	12.5	81	76	64.77	6	5.5	4	15.5	62

Where: Jv = Volumetric joint count. SR (Structure Rating) = 100 – 17.5322 *ln* Jv (this study). SR = 79.8 – 17.5 *ln* Jv (Sonmez & Ulusay, 2002) R_r = Roughness Rating. R_w = Weathering Rating. R_f = Infilling Rating. SCR = Surface Condition Rating. Jv values and condition of discontinuities are taken from tables (5-10, 5-11, 5-12 & 5-13). R_r , R_w & R_f values were estimated from comparison condition of Discontinuities with table (4-20) (this study), and with table (4-19) (Sonmez & Ulusay, 1999 & 2002). GSI by Hoek's chart were estimated depending on table (4-16).

Table (5-20) Rating of rock mass parameters and values of the rock mass classification systems in surface section no. 1

			1	2	3	4	5	Remarks
Geologic unit			Gercus Fn.	Sinjar Formation				Remarks
Rock mass unit			1	2	3	4	5	
Elevation above sea level (= a.s.l) (m)			725 - 719	719 - 703	703 - 683	683 - 681	681 - 677	
Thickness of the unit (m)			6	16	20	2	4	
Rating of parameters	Strength of intact rock (UCS)		3	6.1	6	2	7.6	Comparison of their values with Fig(4-1 & 4-2)
	RQD		-	19.8	20	3	20	
	Spacing of discontinuities (Min.)*	RMR $_{(1976)}$		20	20	5	20	Comparison of their values in the table (5-10) with the tables (4-2) , & (4-4)
		RMR $_{(1989)}$		10	15	5	15	
	Condition of discontinuities	RMR $_{(1976)}$		21	19.5	15.5	19.5	
		RMR $_{(1989)}$		26	24.5	20.5	24.5	
	Ground water condition	RMR $_{(1976)}$	10	10	10	10	10	
		RMR $_{(1989)}$	15	15	15	15	15	
		DMR	**5	**5	**5	**5	**5	
	Strike and dip orientation of foundation rocks	Fill dam	0	0	0	0	0	Comparison its condition in table (5-10) with table (4 -9)
		Gravity dam	-7	-7	-7	-7	-7	
	R_{STA}	Fill dam	0	0	0	0	0	From table (5-9)
		Gravity dam	-7	-7	-7	-7	-7	
	CF		0.22	0.22	0.22	0.22	0.22	
	R_{STA} * CF	Fill dam	0	0	0	0	0	
		Gravity dam	-1.54	-1.54	-1.54	-1.54	-1.54	
Rock mass classification system	RMR $_{(1976)}$	Fill dam	33	76.9	75.5	35.5	77.1	
		Gravity dam	26	69.9	68.5	28.5	70.1	
	$RMR_{B\ (1976)}$		33	76.9	75.5	35.5	77.1	
	$RMR_{BD\ (1976)}$		33	76.9	75.5	35.5	77.1	
	RMR $_{(1989)}$	Fill dam	38	76.9	80.5	40.5	82.1	
		Gravity dam	31	69.9	73.5	33.5	75.1	
	$RMR_{B\ (1989)}$		38	76.9	80.5	40.5	82.1	
	$RMR_{BD\ (1989)}$		38	76.9	80.5	40.5	82.1	
	DMR_{STA} ($RMR_{BD\ (1989)}$ + R_{STA} * CF)	Fill dam	38	76.9	80.5	40.5	82.1	
		Gravity dam	36.46	75.36	78.96	38.96	80.56	
	DMR_{DEF} ($RMR_{BD\ (1976)}$ – 5)		28	71.9	70.5	30.5	72.1	
	GSI (Geological Strength Index)		33	76.5	75.5	35	78	From tables (5-24), (5-17 & 5-18)

* Rating of the minimum spacing must be used (Edelbro, 2003)

**In DMR → Water rating (WR)= 5 when water pressure ratio (r_u)=0.25 (Romana,2003a,2003b&2004)

Where: RMR = Rock Mass Rating (the sum of the rating of the six parameter).

RMR $_{(1976\ or\ 1989)}$ = Rock Mass Rating related to the year of that version.

DMR = Dam Mass Rating.

RMR_B = Basic RMR, with no adjusting factor for joint orientation.

$RMR_{BD(1976)}$ = Basic dry RMR (the addition of the first four parameters of $RMR_{B\ (1976)}$ plus 10.

$RMR_{BD(1989)}$ = Basic dry RMR (the addition of the first four parameters of $RMR_{B\ (1989)}$ plus 15.

DMR_{STA} = DMR related to dam stability.

DMR_{DEF} = RMR related to relative deformability, with WR(water rating) = 5, and no adjusting for discontinuities orientation.

Table (5-21) Rating of rock mass parameters and values of the rock mass classification systems in surface section no. 2

Surface section no.			2					Remarks
Rock mass unit			1	2	3	4	5	
Elevation above sea level (m)			735-722	722-712	712-704	704-697	697-692	
Thickness of the unit (m)			13	10	8	7	5	
Rating of parameters	Strength of intact rock (UCS)		6.5	5.4	6	4.75	7.6	Comparison of their values with Fig(4-1 & 4-2)
	RQD		20	20	19.7	19	20	
	Spacing of discontinuities (Min.)*	RMR $_{(1976)}$	20	20	20	20	20	Comparison of their values in the table (5-11) with the tables (4-2) , & (4-4)
		RMR $_{(1989)}$	15	15	15	10	15	
	Condition of discontinuities	RMR $_{(1976)}$	21.5	21	18	14	21.5	
		RMR $_{(1989)}$	26.5	26	23	19	26.5	
	Ground water condition	RMR $_{(1976)}$	10	10	10	10	10	
		RMR $_{(1989)}$	15	15	15	15	15	
		DMR	**5	**5	**5	**5	**5	
	Strike and dip orientation of foundation rocks	Fill dam	0	0	0	0	0	Comparison its condition in table (5-11) with table (4 - 9)
		Gravity dam	-7	-7	-7	-7	-7	
	R_{STA}	Fill dam	0	0	0	0	0	From table (5-9)
		Gravity dam	-7	-7	-7	-7	-7	
	CF		0.029	0.029	0.029	0.029	0.029	
	R_{STA} * CF	Fill dam	0	0	0	0	0	
		Gravity dam	-0.20	-0.20	-0.20	-0.20	-0.20	
Rock mass classification system	RMR $_{(1976)}$	Fill dam	78	76.4	73.7	67.75	79.1	
		Gravity dam	71	69.4	66.7	60.75	72.1	
	$RMR_{B\ (1976)}$		78	76.4	73.7	67.75	79.1	
	$RMR_{BD\ (1976)}$		78	76.4	73.7	67.75	79.1	
	RMR $_{(1989)}$	Fill dam	83	81.4	78.7	67.75	84.1	
		Gravity dam	76	74.4	71.7	60.75	77.1	
	$RMR_{B\ (1989)}$		83	81.4	78.7	67.75	84.1	
	$RMR_{BD\ (1989)}$		83	81.4	78.7	67.75	84.1	
	DMR_{STA} ($RMR_{BD\ (1989)}$ + R_{STA} * CF)	Fill dam	83	81.4	78.7	67.75	84.1	
		Gravity dam	82.80	81.20	78.50	67.55	83.90	
	DMR_{DEF} ($RMR_{BD\ (1976)}$ – 5)		73	71.4	68.7	62.75	74.1	
	GSI (Geological Strength Index)		79	77.5	73	67	80	From tables (5-17 & 5-18)

* Rating of the minimum spacing must be used (Edelbro, 2003)

**In DMR → Water rating (WR)= 5 when water pressure ratio (r_u)=0.25 (Romana,2003a,2003b&2004)

Where: RMR = Rock Mass Rating (the sum of the rating of the six parameter).

RMR $_{(1976\ or\ 1989)}$ = Rock Mass Rating related to the year of that version.

DMR = Dam Mass Rating.

RMR_B = Basic RMR, with no adjusting factor for joint orientation.

$RMR_{BD(1976)}$ = Basic dry RMR (the addition of the first four parameters of $RMR_{B\ (1976)}$ plus 10.

$RMR_{BD(1989)}$ = Basic dry RMR (the addition of the first four parameters of $RMR_{B\ (1989)}$ plus 15.

DMR_{STA} = DMR related to dam stability.

DMR_{DEF} = RMR related to relative deformability, with WR(water rating) = 5, and no adjusting for discontinuities orientation.

Table (5-22) Rating of rock mass parameters and values of the rock mass classification systems in surface section no. 3

Geologic unit			Sinjar Formation				Unconformity		Remarks
Rock mass unit			1	2	3	4	5	6	
Elevation (a.s.l) (m)			745 - 731	731 - 711	711 – 704.5	704.5 - 700	700 - 694	694 - 686	
Thickness of the unit (m)			14	20	6.5	4.5	6	8	
Rating of parameters	Strength of intact rock (UCS)		6.4	7.7	7.8	7.2	7	7.2	Comparison of their values with Fig(4-1 & 4 -2)
	RQD		19.1	19.8	15	20	17.3	12.6	
	Spacing of discontinuities (Min.)*	$RMR_{(1976)}$	20	20	10	20	10	10	Comparison of their values in the table (5-12) with the tables (4-2) , & (4-4)
		$RMR_{(1989)}$	10	10	8	15	10	8	
	Condition of discontinuities	$RMR_{(1976)}$	22	16	18	18	17	16	
		$RMR_{(1989)}$	27	21	23	23	22	21	
	Ground water condition	$RMR_{(1976)}$	10	10	10	10	10	10	
		$RMR_{(1989)}$	15	15	15	15	15	15	
		DMR	**5	**5	**5	**5	**5	**5	
	Strike and dip orientation of foundation rocks	Fill dam	0	0	0	0	0	0	Comparison its condition in table (5-12) with table (4 -9)
		Gravity dam	-7	-7	-7	-7	-7	-7	
	R_{STA}	Fill dam	0	0	0	0	0	0	From table (5-9)
		Gravity dam	-7	-7	-7	-7	-7	-7	
	CF		0.71	0.71	0.71	0.71	0.71	0.71	
	R_{STA} * CF	Fill dam	0	0	0	0	0	0	
		Gravity dam	-4.97	-4.97	-4.97	-4.97	-4.97	-4.97	
Rock mass classification system	$RMR_{(1976)}$	Fill dam	77.5	73.5	60.8	75.2	61.3	55.8	
		Gravity dam	70.5	66.5	53.8	68.2	54.3	48.8	
	$RMR_{B\ (1976)}$		77.5	73.5	60.8	75.2	61.3	55.8	
	$RMR_{BD\ (1976)}$		77.5	73.5	60.8	75.2	61.3	55.8	
	$RMR_{(1989)}$	Fill dam	77.5	73.5	68.8	80.2	71.3	63.8	
		Gravity dam	70.5	66.5	61.8	73.2	64.3	56.8	
	$RMR_{B\ (1989)}$		77.5	73.5	68.8	80.2	71.3	63.8	
	$RMR_{BD\ (1989)}$		77.5	73.5	68.8	80.2	71.3	63.8	
	DMR_{STA} ($RMR_{BD\ (1989)}$ + R_{STA} * CF)	Fill dam	77.5	73.5	68.8	80.2	71.3	63.8	
		Gravity dam	72.53	68.53	63.83	75.23	66.33	58.83	
	DMR_{DEF} ($RMR_{BD\ (1976)}$ – 5)		72.5	68.5	55.8	70.2	56.3	50.8	
	GSI (Geological Strength Index)		77	73	61.5	74	62	55	From tables (5-17&5-18)

* Rating of the minimum spacing must be used (Edelbro, 2003)

**In DMR → Water rating (WR)= 5 when water pressure ratio (r_u)=0.25 (Romana,2003a,2003b&2004)

Where: RMR = Rock Mass Rating (the sum of the rating of the six parameter).

$RMR_{(1976\ or\ 1989)}$ = Rock Mass Rating related to the year of that version.

DMR = Dam Mass Rating.

RMR_B = Basic RMR, with no adjusting factor for joint orientation.

$RMR_{BD(1976)}$ = Basic dry RMR (the addition of the first four parameters of $RMR_{B\ (1976)}$ plus 10.

$RMR_{BD(1989)}$ = Basic dry RMR (the addition of the first four parameters of $RMR_{B\ (1989)}$ plus 15.

DMR_{STA} = DMR related to dam stability.

DMR_{DEF} = RMR related to relative deformability, with WR(water rating) = 5, and no adjusting for discontinuities orientation.

Table (5-23) Rating of rock mass parameters and values of the rock mass classification systems of the blocky carbonate rocks in boreholes no. 1,2&3

Geologic unit			Kolosh Formation			Remarks
Bore hole no.			1	2	3	
Rock mass unit			2	6	2	
Depth below surface (m)			40.65 – 47.50	35.75 - 38	17 - 28	
Elevation (a.s.l) (m)			698.35– 691.50	644.25 - 642	655 - 644	
Thickness of the unit (m)			6.85	2.25	11	
Rating of parameters	Strength of intact rock (UCS)		4.8	5.6	5.7	Comparison of their values with Fig(4-1 & 4-2)
	RQD		19.1	19	20	
	Spacing of discontinuities (Min.)*	RMR $_{(1976)}$	20	20	25	Comparison of their values or conditions in the table (5-13) with the tables (4-2) , & (4-4)
		RMR $_{(1989)}$	10	10	15	
	Condition of discontinuities	RMR $_{(1976)}$	18.5	16.5	20.5	
		RMR $_{(1989)}$	23.5	21.5	25.5	
	Ground water condition	RMR $_{(1976)}$	4	4	4	
		RMR $_{(1989)}$	4	4	4	
		DMR	**5	**5	**5	
	Strike and dip orientation of foundation rocks	Fill dam	0	0	0	Comparison its condition in table (5-13) with table (4 - 9)
		Gravity dam	-7	-7	-7	
	R_{STA}	Fill dam	0	0	0	***From table (5-9)
		Gravity dam	-7	-7	-7	
	CF		0.22	0.029	0.71	
	R_{STA} * CF	Fill dam	0	0	0	
		Gravity dam	-1.54	-0.20	-4.97	
Rock mass classification system	RMR $_{(1976)}$	Fill dam	66.4	65.1	75.2	
		Gravity dam	59.4	58.1	68.2	
	$RMR_{B\ (1976)}$		66.4	65.1	75.2	
	$RMR_{BD\ (1976)}$		72.4	71.1	81.2	
	RMR $_{(1989)}$	Fill dam	66.4	65.1	75.2	
		Gravity dam	59.4	58.1	68.2	
	$RMR_{B\ (1989)}$		66.4	65.1	75.2	
	$RMR_{BD\ (1989)}$		72.4	71.1	81.2	
	DMR_{STA} ($RMR_{BD\ (1989)}$ + R_{STA} * CF)	Fill dam	72.4	71.1	81.2	
		Gravity dam	70.86	70.90	76.23	
	DMR_{DEF} ($RMR_{BD\ (1976)}$ – 5)		67.4	66.1	76.2	
	GSI (Geological Strength Index)		72	71	81	From tables (5-17&5-18)

* Rating of the minimum spacing must be used (Edelbro, 2003).
**In DMR → Water rating (WR)= 5 when water pressure ratio (r_u)=0.25 (Romana,2003a,2003b&2004).
Where: RMR = Rock Mass Rating (the sum of the rating of the six parameter).
RMR $_{(1976\ or\ 1989)}$ = Rock Mass Rating related to the year of that version (=the addition of the six parameters).
DMR = Dam Mass Rating.
RMR_B = Basic RMR, with no adjusting factor for joint orientation.
$RMR_{BD(1976)}$ = Basic dry RMR (the addition of the first four parameters of $RMR_{B\ (1976)}$ plus 10.
$RMR_{BD(1989)}$ = Basic dry RMR (the addition of the first four parameters of $RMR_{B\ (1989)}$ plus 15.
DMR_{STA} = DMR related to dam stability.
DMR_{DEF} = RMR related to relative deformability, with WR(water rating) = 5, and no adjusting for discontinuities orientation.
***The same values of surface sections no. 1, 2 & 3 were used for bore holes no. 1, 2 & 3 Respectively, due to nearness of each one from surface sections.

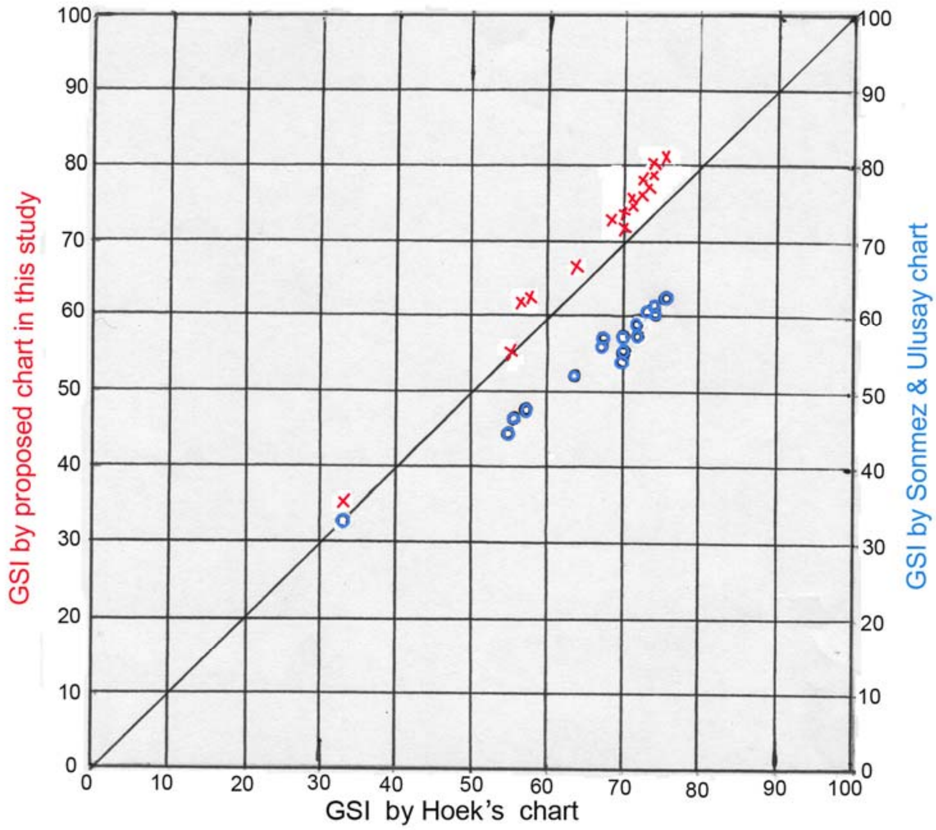


Fig (5-8) Correlation between the GSI values of Hoek-this study(red cross) and also between Hoek-Sonmez&Ulusay (blue circle)

Table(5-24) GSI determination for surface section no.1-unit no.1(S1-U1) of Molasse Gercus Formation

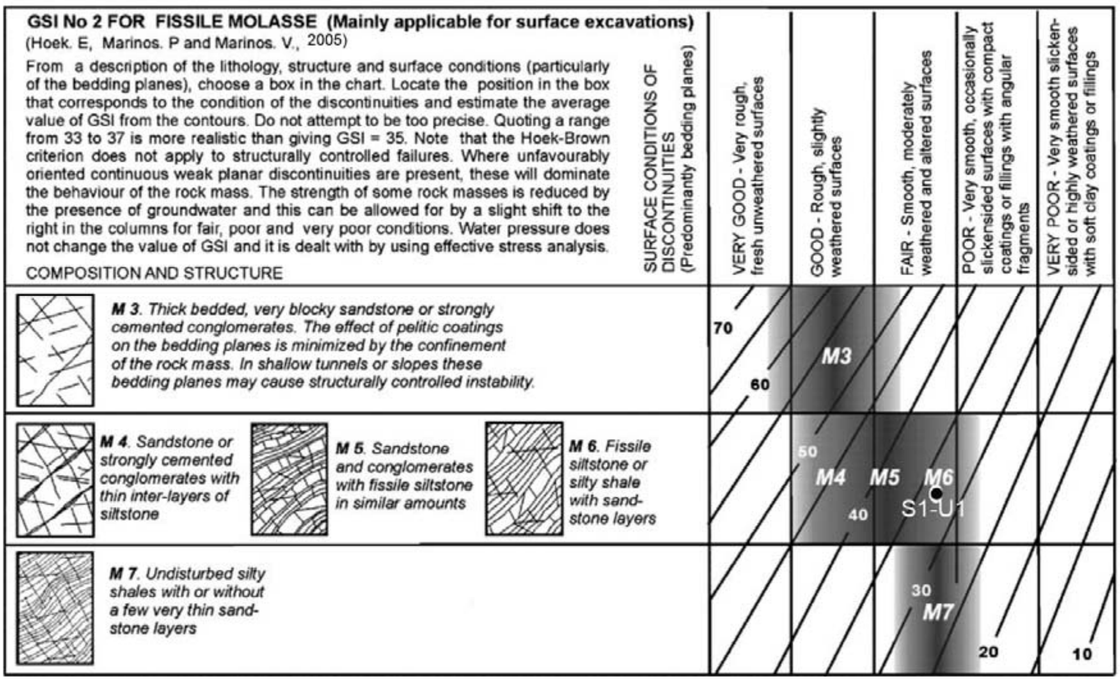
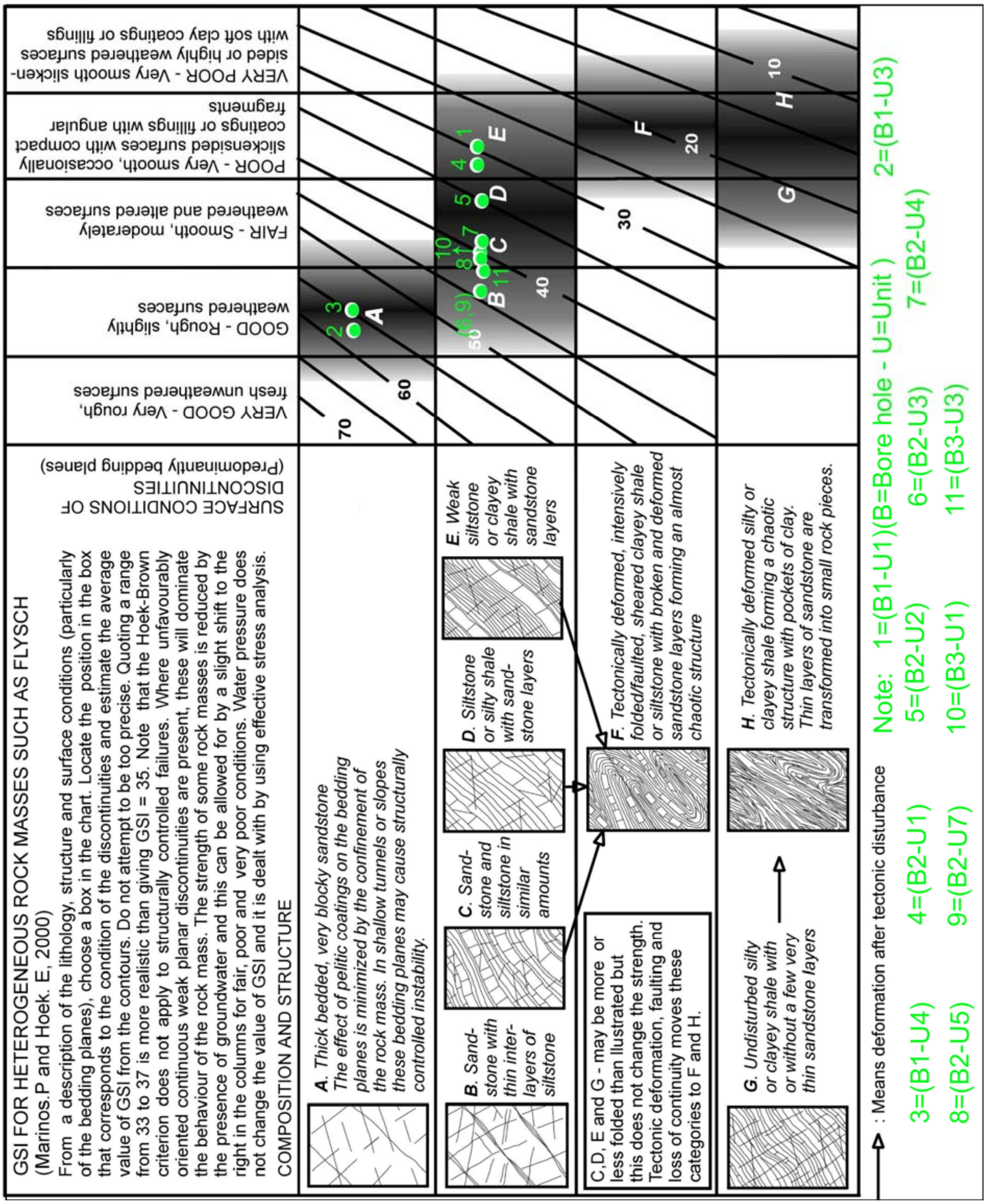


Table (5 - 25) GSI determination for bore hole units of Flysch Kolosh Formation



The RMR_{76}, RMR_{89} and DMR values for Molasse and Flysch rocks were calculated from GSI value, through the relationship between them, which were mentioned in chapter four. The results of these calculations are shown in table (5-20) for Molasses rocks of Gercus Formation (section no.1-unit no.1) and tables (5-26, 5-27 & 5-28) for Flysch rocks of Kolosh Formation.

Table (5-26) Rating of rock mass parameters (Ground water, Strike and dip orientation and R_{STA}) and values of the rock mass classification systems of Fysch rocks in borehole no. 1

Geologic unit			Kolosh Formation			Remarks
Bore hole no.			1			
Rock mass unit			1	3	4	
Depth below surface (m)			37.80-40.65	47.50-52.50	52.50-63	
Elevation above sea level (m)			701.2-698.35	691.5-686.5	686.5-676	
Thickness of the unit (m)			2.85	5	10.5	
Rock mass characterization	Ground water condition	RMR (1976)	4	4	4	Comparison its condition in table (5-14) with tables (4 -2) & (4 -4)
		RMR (1989)	4	4	4	
		DMR	*5	*5	*5	
	Strike and dip orientation of foundation rocks	Fill dam	0	0	0	Comparison its condition in table (5-14) with table (4 - 9)
		Gravity dam	-7	-7	-7	
	R_{STA}	Fill dam	0	0	0	**From table (5-9)
		Gravity dam	-7	-7	-7	
CF			0.22	0.22	0.22	
R_{STA} * CF		Fill dam	0	0	0	
		Gravity dam	-1.54	-1.54	-1.54	
Rock mass classification system	Geological Strength Index (GSI)		30	57	55	Tables (5-14 & 5-25)
	$RMR_{BD\ (1976)}$		30	57	55	
	$RMR_{B\ (1976)}$		24	51	49	
	RMR (1976)	Fill dam	24	51	49	
		Gravity dam	17	44	42	
	$RMR_{BD\ (1989)}$		35	62	60	
	$RMR_{B\ (1989)}$		24	51	49	
	RMR (1989)	Fill dam	24	51	49	
		Gravity dam	17	44	42	
	DMR_{STA} ($RMR_{BD\ (1989)}$ + R_{STA} * CF)	Fill dam	35	62	60	
		Gravity dam	33.46	60.46	58.46	
	DMR_{DEF} ($RMR_{BD\ (1976)}$ – 5)		25	52	50	

Where: $RMR_{BD(1976)}$ = Basic dry RMR = Geological Strength Index (GSI).(Hoek and Brown, 1997).
= (the addition of the first four parameters of $RMR_{B\ (1976)}$ plus 10).
$RMR_{BD(1989)}$ = Basic dry RMR = (GSI + 5) (Hoek and Brown, 1997).
= (the addition of the first four parameters of $RMR_{B\ (1989)}$ plus 15).
$RMR_{B\ (1976)}$ = Basic RMR (= $RMR_{BD(1976)}$ – 10 + Rating of ground water, with no adjusting factor for joint orientation).
$RMR_{B\ (1989)}$ = Basic RMR (= $RMR_{BD(1989)}$ – 15 + Rating of ground water, with no adjusting factor for joint orientation).
RMR $_{(1976\ or\ 1989)}$ = Rock Mass Rating related to the year of that version (= RMR_B + Rating of strike and dip orientation of foundation rocks “significant discontinuities,here bedding planes”).
DMR = Dam Mass Rating.
DMR_{STA} = DMR related to dam stability.
DMR_{DEF} = RMR related to relative deformability, with WR(water rating) = 5, and no adjusting for discontinuities orientation.
*In DMR → Water rating (WR)= 5 when water pressure ratio (r_u)=0.25 (Romana,2003a, 2003b&2004).
**The same values of surface section no.1 were used, because bore hole no.1 is near to it more than other sections, or it is along a-b profile (see Fig “5-1”).

Table (5-27) Rating of rock mass parameters (Ground water, Strike and dip orientation and R_{STA}) and values of the rock mass classification systems of Flysch rocks in borehole no. 2

Geologic unit			Kibosh Formation						Remarks
Bore hole no.			**2**						
Rock mass unit			1	2	3	4	5	7	
Depth below surface(m)			19.50 – 22.10	22.10 – 25.50	25.50 – 28.50	28.50 – 32.90	32.90 – 35.75	38 - 45	
Elevation(a.s.l)(m)			660.50 – 657.90	657.90 – 654.50	654.50 – 651.50	651.50 – 647.10	647.10 – 644.25	642 - 635	
Thickness of the unit (m)			2.60	3.40	3	4.40	2.85	7	
Rock mass characterization	Ground water condition	$RMR_{(1976)}$	4	4	4	4	4	4	Comparison its condition in table (5-15) with tables (4-2)&(4-4)
		$RMR_{(1989)}$	4	4	4	4	4	4	
		DMR	*5	*5	*5	*5	*5	*5	
	Strike and dip orientation of foundation rocks	Fill dam	0	0	0	0	0	0	Comparison its condition in table (5-15) with table (4- 9)
		Gravity dam	-7	-7	-7	-7	-7	-7	
	R_{STA}	Fill dam	0	0	0	0	0	0	**From table (5-9)
		Gravity dam	-7	-7	-7	-7	-7	-7	
CF (Geometric correction factor)			0.029	0.029	0.029	0.029	0.029	0.029	
R_{STA} * CF		Fill dam	0	0	0	0	0	0	
		Gravity dam	-0.20	-0.20	-0.20	-0.20	-0.20	-0.20	
Rock mass classification system	Geological Strength Index (GSI)		32	35	44	39	41	44	Tables(5-15 &5-25)
	$RMR_{BD\ (1976)}$		32	35	44	39	41	44	
	$RMR_{B\ (1976)}$		26	29	38	33	35	38	
	$RMR_{(1976)}$	Fill dam	26	29	38	33	35	38	
		Gravity dam	19	22	31	26	28	31	
	$RMR_{BD\ (1989)}$		37	40	49	44	46	49	
	$RMR_{B\ (1989)}$		26	29	38	33	35	38	
	$RMR_{(1989)}$	Fill dam	26	29	38	33	35	38	
		Gravity dam	19	22	31	26	28	31	
	DMR_{STA} ($RMR_{BD\ (1989)}$ + R_{STA} * CF)	Fill dam	37	40	49	44	46	49	
		Gravity dam	36.80	39.80	48.80	43.80	45.80	48.80	
	DMR_{DEF} ($RMR_{BD\ (1976)}$ -5)(Romana,2003)		27	30	39	34	36	39	

$RMR_{BD(1976)}$ = Basic dry RMR = Geological Strength Index (GSI).(Hoek and Brown, 1997) = (the addition of the first four parameters of $RMR_{B\ (1976)}$ plus10).
$RMR_{BD(1989)}$=Basic dry RMR = (GSI + 5) (Hoek and Brown, 1997) = (the addition of the first four parameters of $RMR_{B\ (1989)}$ plus 15).
$RMR_{B\ (1976)}$ = Basic RMR (= $RMR_{BD(1976)}$ – 10 + Rating of ground water, with no adjusting factor for joint orientation).
$RMR_{B\ (1989)}$ = Basic RMR (= $RMR_{BD(1989)}$ – 15 + Rating of ground water, with no adjusting factor for joint orientation).
$RMR_{(1976\ or\ 1989)}$ = Rock Mass Rating related to the year of that version (= RMR_B + Rating of strike and dip orientation of foundation rocks "significant discontinuities,here bedding planes").
DMR = Dam Mass Rating.
DMR_{STA} = DMR related to dam stability.
DMR_{DEF} = RMR related to relative deformability, with WR(water rating) = 5, and no adjusting for discontinuities orientation.
*In DMR → Water rating (WR)= 5 when water pressure ratio (r_u)=0.25 (Romana, 2003a, 2003b & 2004).
**The same values of surface section no.2 were used, because bore hole no.2 is near to it more than other sections, or it is along c-d profile (see Fig "5-1").

Table (5-28) Rating of rock mass parameters (Ground water, Strike and dip orientation and R_{STA}) and values of the rock mass classification systems of Flysch rocks in borehole no. 3

Geologic unit			Kolosh Formation		Remarks
Bore hole no.			3		
Rock mass unit			1	3	
Depth below surface (m)			13 - 17	28 - 40	
Elevation above sea level (m)			659 - 655	644 - 632	
Thickness of the unit (m)			4	12	
Rock mass characterization	Ground water condition	RMR (1976)	4	4	Comparison its condition in table (5-16) with tables (4-2) & (4-4)
		RMR (1989)	4	4	
		DMR	*5	*5	
	Strike and dip orientation of foundation rocks	Fill dam	0	0	Comparison its condition in table (5-16) with table (4-9)
		Gravity dam	-7	-7	
	R_{STA}	Fill dam	0	0	**From table (5-9)
		Gravity dam	-7	-7	
CF			0.71	0.71	
R_{STA} * CF		Fill dam	0	0	
		Gravity dam	- 4.97	- 4.97	
Rock mass classification system	Geological Strength Index (GSI)		40	42	Tables (5-16& 5-25)
	$RMR_{BD\ (1976)}$		40	42	
	$RMR_{B\ (1976)}$		34	36	
	RMR (1976)	Fill dam	34	36	
		Gravity dam	27	29	
	$RMR_{BD\ (1989)}$		45	47	
	$RMR_{B\ (1989)}$		34	36	
	RMR (1989)	Fill dam	34	36	
		Gravity dam	27	29	
	DMR_{STA} ($RMR_{BD\ (1989)}$ + R_{STA} * CF)	Fill dam	45	47	
		Gravity dam	40.03	42.03	
	DMR_{DEF} ($RMR_{BD\ (1976)}$ – 5)		35	37	

Where: $RMR_{BD(1976)}$ = Basic dry RMR = Geological Strength Index (GSI).(Hoek and Brown, 1997).
= (the addition of the first four parameters of $RMR_{B\ (1976)}$ plus 10).
$RMR_{BD(1989)}$ = Basic dry RMR = (GSI + 5) (Hoek and Brown, 1997).
= (the addition of the first four parameters of $RMR_{B\ (1989)}$ plus 15).
$RMR_{B\ (1976)}$ = Basic RMR (= $RMR_{BD(1976)}$ – 10 + Rating of ground water, with no adjusting factor for joint orientation).
$RMR_{B\ (1989)}$ = Basic RMR (= $RMR_{BD(1989)}$ – 15 + Rating of ground water, with no adjusting factor for joint orientation).
RMR (1976 or 1989) = Rock Mass Rating related to the year of that version (= RMR_B + Rating of strike and dip orientation of foundation rocks “significant discontinuities,here bedding planes”).
DMR = Dam Mass Rating.
DMR_{STA} = DMR related to dam stability.
DMR_{DEF} = RMR related to relative deformability, with WR(water rating) = 5, and no adjusting for discontinuities orientation.
*In DMR → Water rating (WR)= 5 when water pressure ratio (r_u)=0.25(Romana, 2003a, 2003b & 2004).
**The same values of surface section no.3 were used, because bore hole no.3 is near to it more than other sections (see Fig “5-1”).

The Geological Strength Index (GSI), unconfined compressive strength of intact rock ($б_{ci}$), material constant of intact rock (mi) and disturbance factor data of each unit were introduced into RocLab programme to find the mechanical properties (cohesion, friction angle, tensile strength, compressive strength, global strength and deformation modulus) of the rock mass, as in Fig (5-9 & 5-10) and Appendix-B (Fig "B-1, B-2,………, B-27 & B-28), and the summary of these six parameters is shown in table (5-29).(**Note:** the disturbance factor value of 0.2 was used in the RocLab programme because excavation for dams are, as a rule, very careful ("Romana, 2003a, 2003b & 2004").

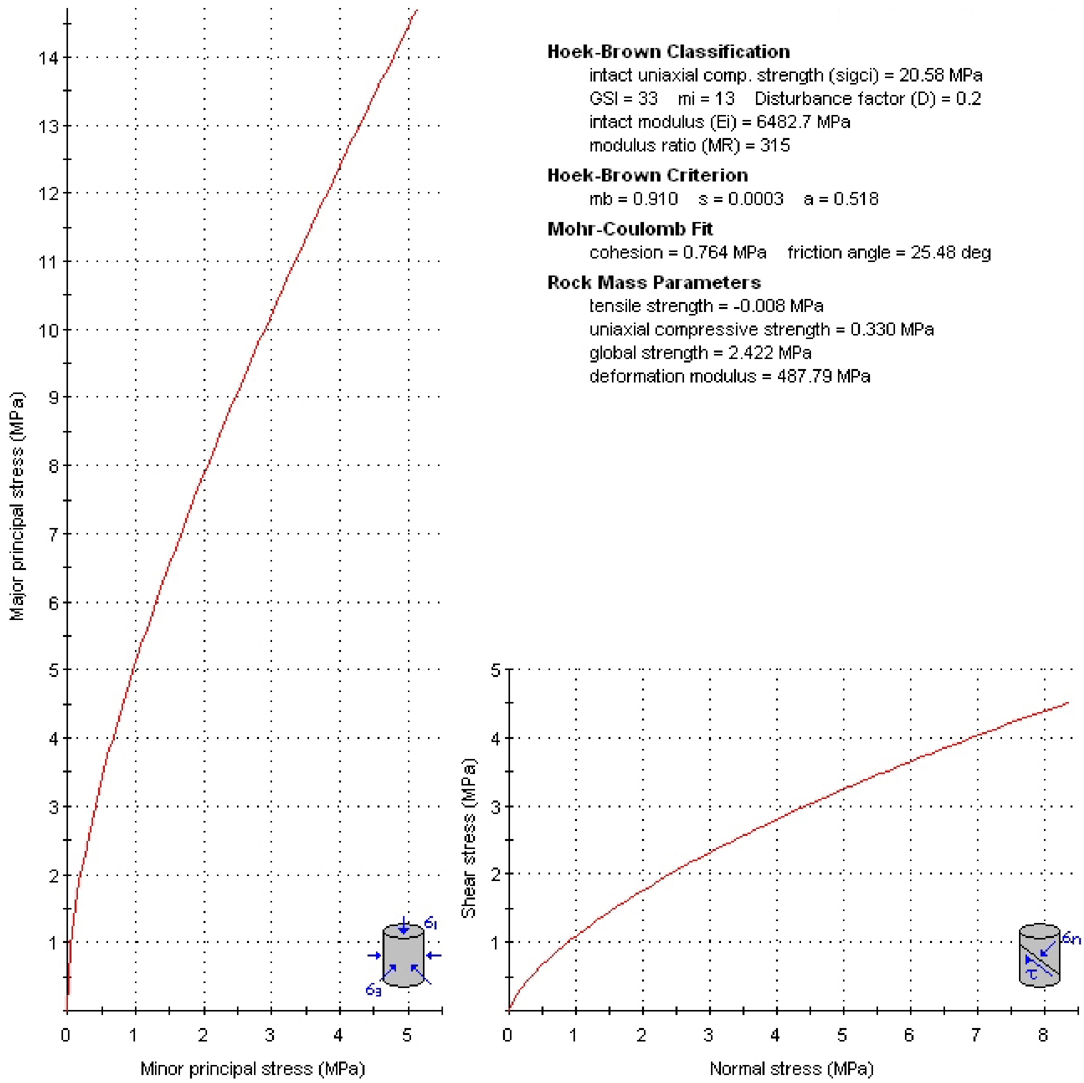
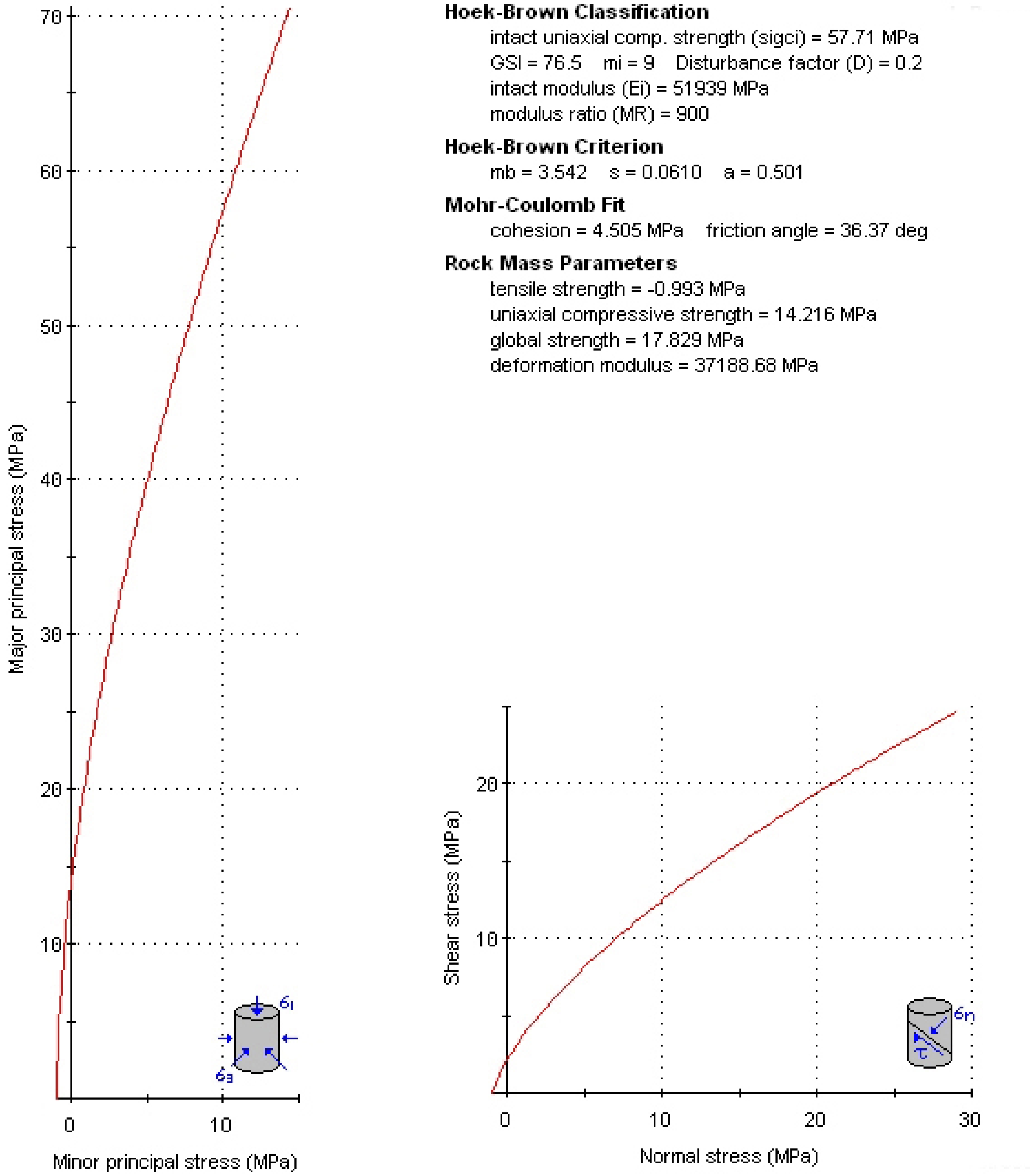


Fig (5 - 9) Analysis of rock strength for surface section no.1-unit no.1(Gercus Formation), using RocLab programme



Fig(5-10)Analysis of rock strength for surface section no.1 - unit no.2 (Sinjar Formation), using RocLab programme

Table (5-29) Analysis of rock strength for surface sections and bore holes, using RocLab programme

Geologic unit	Surface section no.	Bore hole no.	Unit no.	Depth below surface (m)	Elevation above sea level (m)	Thickness of the unit (m)	Cohesion (MPa)	Friction Angle (degree)	Tensile Strength (MPa)	Uniaxial compressive strength(MPa)	Global strength (MPa)	Deformation modulus (MPa)
Gercus Fn.	1	-	1	-	725 – 719	6	0.764	25.48	-0.008	0.330	2.422	518.76
Sinjar Formation		-	2	-	719 – 703	16	4.505	36.37	-0.993	14.216	17.829	37188.68
		-	3	-	703 – 683	20	4.169	36.08	-0.873	12.740	16.391	34654.89
		-	4	-	683 – 681	2	0.309	21.33	-0.008	0.185	0.906	149.55
		-	5	-	681 – 677	4	6.159	36.82	-1.464	20.354	24.607	50086.31
	2	-	1	-	735 – 722	13	5.243	37.10	-1.310	17.849	21.081	42099.07
		-	2	-	722 – 712	10	3.849	36.67	-0.893	12.528	15.328	31473.20
		-	3	-	712 – 704	8	3.907	35.32	-0.716	10.977	15.112	32716.10
		-	4	-	704 – 697	7	2.550	33.45	-0.334	5.750	9.483	20463.13
		-	5	-	697 – 692	5	6.444	37.39	-1.689	22.573	26.071	50971.04
	3	-	1	-	745 – 731	14	5.006	36.52	-1.132	16.044	19.871	41135.77
		-	2	-	731 – 711	20	5.302	35.32	-0.972	14.897	20.508	44397.44
		-	3	-	711 – 704.5	6.50	4.273	31.70	-0.405	7.729	15.321	30586.44
		-	4	-	704.5 –700	4.50	5.096	35.62	-0.986	14.818	19.842	42625.60
Unconformity		-	5	-	700 – 694	6	3.882	32.77	-0.332	6.976	14.229	21314.07
		-	6	-	694 – 686	8	3.467	30.50	-0.193	4.614	12.132	15055.89
Kolosh Formation	-	1	1	37.80 – 40.65	701.20 – 698.35	2.85	0.174	24.48	-0.001	0.064	0.542	98.73
	-		2	40.65 – 47.50	698.35 – 691.50	6.85	2.948	35.01	-0.512	7.999	11.328	24657.76
	-		3	47.50 – 52.50	691.50 – 686.50	5	1.591	35.73	-0.051	1.983	6.210	2514.72
	-		4	52.50 – 63	686.50 – 676	10.50	1.294	35.06	-0.036	1.471	4.976	1886.33
	-	2	1	19.50 – 22.10	660.5 – 657.9	2.60	0.825	25.15	-0.008	0.338	2.597	504.98
	-		2	22.10 – 25.5	657.90 – 654.50	3.40	1.028	26.14	-0.012	0.491	3.298	715.79
	-		3	25.50 – 28.50	654.50 – 651.50	3	1.720	31.37	-0.024	1.167	6.125	1484.99
	-		4	28.50 – 32.90	651.50 – 647.10	4.40	1.344	28.64	-0.016	0.745	4.532	1008.90
	-		5	32.90 – 35.75	647.10 – 644.25	2.85	1.502	29.31	-0.020	0.919	5.132	1245.48
	-		6	35.75 -38	644.25 – 642	2.25	3.469	34.70	-0.570	9.082	13.244	28914.45
	-		7	38 – 45	642 – 635	7	1.889	31.37	-0.026	1.281	6.726	1630.70
	-	3	1	13 – 17	659 – 655	4	1.194	28.97	-0.015	0.696	4.052	941.75
	-		2	17 – 28	655 – 644	11	4.780	37.67	-1.313	17.212	19.460	37166.51
	-		3	28 – 40	644 – 632	12	1.572	29.64	-0.023	1.009	5.406	1370.65

Carbonate Unit

After finishing the rock mass classification systems (RMR, DMR & GSI) and determining the mechanical properties of the rock mass by Hoek-Brown criterion using RocLab programme, the rock mass units of the surface sections and boreholes were evaluated as follows:

1- Evaluating all rock mass units for the degree of safety of the dam (Fill or Gravity dams) against horizontal sliding after filling the reservoir and this depends on the DMR_{STA} of each unit, as shown in table (5-30).

2- Evaluating all rock mass units for foundation excavation desirability and required consolidation grouting in the case of construction of different dams (Gravity, Rockfill and Earthfill), which depends on the $RMR_{BD(1989)}$ value, then comparing this value with table (4-10) and the results are shown in table (5-31).

3- Evaluating all rock mass units for the effect of E_c / E_m (deformation modulus of the dam / deformation modulus of the rock mass) on the proposed Basara Gravity (CVC, RCC) or Hardfill dams behaviors, E_c / E_m value shows its influence on the dam and the level of problems. These were done from comparing E_c / E_m value with table (4-12) and the results of this evaluation are shown in table (5-32).

4- Evaluating all rock mass units for the deformability problems, which depends on the DMR_{DEF} value of each unit and comparing this value with table (4-13) to determine the type of deformability problems as shown in table (5-33).

5- Evaluating all rock mass units by Hoek-Brown criterion, using RocLab programme in determining the mechanical properties (cohesion, friction angle, tensile strength, compressive strength, global strength and deformation modulus) of the rock mass, as shown in table (5-29).

6- Classification of the intact rock (which makes an important part of the rock mass) strength according to table (3-5).The results of this classification are shown in table (5-34), which indicates strength ranging from very weak – strong.

7- Comparison between unconfined compressive strength of the intact rock and global strength of the rock mass was done in table (5-34). This comparison reveals huge reduction in strength of the rock mass due to intersection of the rock mass by discontinuities in which the degree of this strength reduction depends on the spacing or frequency and surface conditions of discontinuities.

8- Comparison between estimated GSI (eighteen rock mass units “green colour”) by proposed GSI-chart in this study and $RMR_{BD(1976)}$ in table (5-34) reveals the precision of the GSI estimation because the values from the two classification systems are quite close to each other.

5-4 Rock mass evaluation of a-b, c-d & e-f profiles and choosing the optimum one among these for dam site:

For evaluation of the dam site, three profiles were selected and for drawing these three profiles a-b, c-d and e-f, a supplementary surface section was taken, as shown in Fig (5-1) and the detailed information about this section was recorded in table (5-35). This supplementary section provides information about units having no information in the profiles.

The above mentioned profiles were drawn and rock mass units were projected below each profile using the apparent dip in each one, forming the rock mass – valley section model, as shown in Fig (5-11, 5-12 & 5-13).

By comparing the rock mass units on both sides of each profile with the aid of tables (5-29, 5-30, 5-31, 5-32 & 5-33), it is concluded that the two sides are inhomogeneous, so that each profile is divided into two parts (sub profiles), which are represented relatively by strong rocks of Sinjar Formation in the right side and by weak rocks of Kolosh Formation in the left side, as shown in Figures (5-11, 5-12 & 5-13) and table (5-36), so this inhomogenity affects the mechanical properties of the rock mass units on both sides (banks) and hence the type of the proposed dam.

The profiles were evaluated on the basis of horizontal distance, soil and drift thickness, area of profiles before and after stripping of the soil and drift, effect of weak sheared zone and also on the relationship between rock mass units in the profiles and tables (5-29, 5-30, 5-31, 5-32 & 5-33), furthermore other parameters that are illustrated in table (5-36).

Table (5 - 30) The degree of safety of the dam against sliding from evaluation of DMR_{STA}

Geologic unit	Surface section no.	Bore hole no.	Rock mass unit	Depth below surface (m)	Elevation above sea level (m)	Thickness of the unit (m)	*DMR_{STA}		**Degree of safety of the dam against sliding	
							Fill dam	Gravity dam	Fill dam	Gravity dam
Gercus Fn.	1	-	1	-	725 – 719	6	38	36.46	Primary concern	Primary concern
Sinjar Formation		-	2	-	719 – 703	16	76.9	75.36	No primary concern	No primary concern
		-	3	-	703 – 683	20	80.4	78.86	No primary concern	No primary concern
		-	4	-	683 – 681	2	40.5	38.96	Primary concern	Primary concern
		-	5	-	681 – 677	4	82.1	80.56	No primary concern	No primary concern
	2	-	1	-	735 – 722	13	83	82.80	No primary concern	No primary concern
		-	2	-	722 – 712	10	81.4	81.20	No primary concern	No primary concern
		-	3	-	712 – 704	8	78.7	78.50	No primary concern	No primary concern
		-	4	-	704 – 697	7	67.75	67.55	No primary concern	No primary concern
		-	5	-	697 – 692	5	84.1	83.90	No primary concern	No primary concern
	3	-	1	-	745 – 731	14	77.5	72.53	No primary concern	No primary concern
		-	2	-	731 – 711	20	73.5	68.53	No primary concern	No primary concern
		-	3	-	711 – 704.5	6.50	68.8	63.83	No primary concern	No primary concern
		-	4	-	704.5 –700	4.50	80.2	75.23	No primary concern	No primary concern
Unconformity		-	5	-	700 – 694	6	71.3	66.33	No primary concern	No primary concern
		-	6	-	694 – 686	8	63.8	58.83	No primary concern	Primary concern
Kolosh Formation	-	1	1	37.80 – 40.65	701.20 – 698.35	2.85	35	33.46	Primary concern	Primary concern
	-		2	40.65 – 47.50	698.35 – 691.50	6.85	72.4	70.86	No primary concern	No primary concern
	-		3	47.50 – 52.50	691.50 – 686.50	5	62	60.46	No primary concern	No primary concern
	-		4	52.50 – 63	686.50 – 676	10.50	60	58.46	No primary concern	Primary concern
	-	2	1	19.50 – 22.10	660.50 – 657.90	2.60	37	36.80	Primary concern	Primary concern
	-		2	22.10 – 25.50	657.90 – 654.50	3.40	40	39.80	Primary concern	Primary concern
	-		3	25.50 – 28.50	654.50 – 651.50	3	49	48.80	Primary concern	Primary concern
	-		4	28.50 – 32.90	651.50 – 647.10	4.40	44	43.80	Primary concern	Primary concern
	-		5	32.90 – 35.75	647.10 – 644.25	2.85	46	45.80	Primary concern	Primary concern
	-		6	35.75 -38	644.25 – 642	2.25	71.1	70.90	No primary concern	No primary concern
	-		7	38 – 45	642 – 635	7	49	48.80	Primary concern	Primary concern
	-	3	1	13 – 17	659 – 655	4	45	40.03	Primary concern	Primary concern
	-		2	17 – 28	655 – 644	11	81.2	76.23	No primary concern	No primary concern
	-		3	28 – 40	644 – 632	12	47	42.03	Primary concern	Primary concern

*DMR_{STA} = ($RMR_{BD\ (1989)}$ + R_{STA} * CF) (From tables (5-20), (5-21), (5-22), (5-23), (5-24), (5-25) and (5-26).

**Degree of safety of the dam against sliding (Romana, 2003a, 2003b & 2004) :
(DMR_{STA} > 60 → No primary concern). (60 > DMR_{STA} > 30 → Primary concern). (DMR_{STA} < 30 → Serious concern).

Table (5 - 31) Tentative guidelines for the proposed Basara dam foundation excavation and consolidation grouting according to the table (4 – 10)

Geologic unit	Surface section no.	Bore hole no.	Rock mass unit	Depth below surface (m)	Elevation above sea level (m)	Thickness of the unit (m)	* RMR_{BD} (1989)	■Foundation excavation			■Consolidation grouting according to $RMR_{BD(1989)}$		
								**Gravity dam	***Rockfill dam	Earthfill dam	Gravity	Rockfill	Earthfill
Gercus Fn.	1	-	1	-	725 – 719	6	38	Not desira.	desirable	desirable	¿	Spot	None
Sinjar Formation		-	2	-	719 – 703	16	76.9	desirable	desirable	desirable	None	None	None
		-	3	-	703 – 683	20	80.5	desirable	desirable	desirable	None	None	None
		-	4	-	683 – 681	2	40.5	desirable.	desirable	desirable	Systematic	Spot	None
		-	5	-	681 – 677	4	82.1	desirable	desirable	desirable	None	None	None
	2	-	1	-	735 – 722	13	83	desirable	desirable	desirable	None	None	None
		-	2	-	722 – 712	10	81.4	desirable	desirable	desirable	None	None	None
		-	3	-	712 – 704	8	78.7	desirable	desirable	desirable	None	None	None
		-	4	-	704 – 697	7	67.75	desirable	desirable	desirable	None	None	None
		-	5	-	697 – 692	5	84.1	desirable	desirable	desirable	None	None	None
	3	-	1	-	745 – 731	14	77.5	desirable	desirable	desirable	None	None	None
		-	2	-	731 – 711	20	73.5	desirable	desirable	desirable	None	None	None
		-	3	-	711 – 704.5	6.50	68.8	desirable	desirable	desirable	None	None	None
		-	4	-	704.5 –700	4.50	80.2	desirable	desirable	desirable	None	None	None
Unconformity		-	5	-	700 – 694	6	71.3	desirable	desirable	desirable	None	None	None
		-	6		694 – 686	8	63.8	desirable	desirable	desirable	None	None	None
Kolosh Formation	-	1	1	37.80 – 40.65	701.20 – 698.35	2.85	35	Not desira.	desirable	desirable	¿	Spot	None
	-		2	40.65 – 47.50	698.35 – 691.50	6.85	72.4	desirable	desirable	desirable	None	None	None
	-		3	47.50 – 52.50	691.50 – 686.50	5	62	desirable	desirable	desirable	None	None	None
	-		4	52.50 – 63	686.50 – 676	10.50	60	desirable	desirable	desirable	Spot	None	None
	-	2	1	19.50 – 22.10	660.50 – 657.90	2.60	37	Not desira.	desirable	desirable	¿	Spot	None
	-		2	22.10 – 25.50	657.90 – 654.50	3.40	40	desirable	desirable	desirable	Systematic	Spot	None
	-		3	25.50 – 28.50	654.50 – 651.50	3	49	desirable	desirable	desirable	Systematic	Spot	None
	-		4	28.50 – 32.90	651.50 – 647.10	4.40	44	desirable	desirable	desirable	Systematic	Spot	None
	-		5	32.90 – 35.75	647.10 – 644.25	2.85	46	desirable	desirable	desirable	Systematic	Spot	None
	-		6	35.75 -38	644.25 – 642	2.25	71.1	desirable	desirable	desirable	None	None	None
	-		7	38 – 45	642 – 635	7	49	desirable	desirable	desirable	Systematic	Spot	None
	-	3	1	13 – 17	659 – 655	4	45	desirable	desirable	desirable	Systematic	Spot	None
	-		2	17 – 28	655 – 644	11	81.2	desirable	desirable	desirable	None	None	None
	-		3	28 – 40	644 – 632	12	47	desirable	desirable	desirable	Systematic	Spot	None

*$RMR_{BD(1989)}$ (Basic dry RMR = The addition of the first four parameters + 15), the values of $DMR_{BD(1989)}$ are from tables (5-20), (5-21), (5-22), (5-23), (5-24), (5-25) and (5-26).
**Gravity dams include "CVC, RCC and hardfill concrete. ***Rockfill dams included are the ones sensible to settlement (with concrete – CFRD – or asphaltic – AFRD – face upstream. ■The guidelines for foundation excavation and consolidation grouting are depending on the table (4 -10). –Note: Not desira.= Not desirable

Table(5-32) Effect of Ec / E_m on the proposed Basara Gravity (CVC, RCC) or Hardfill dams behaviors according to the table(4-12)

Geologic unit	Surface section no.	Bore hole no.	Rock mass unit	Depth below surface (m)	Elevation above sea level (m)	Thickness of the unit (m)	*DMR_{DEF}	**E_m (GPa)	***E_c / E_m			Influence on dam			Problems		
									**** CVC	***** RCC	****** Hardfill	CVC	RCC	Hard-fill ■	CVC	RCC	Hard-fill ■
Gercus Fn.	S1	-	1	-	725 – 719	6	28	2.81	10.6	7.11	3.55	Imp.	L.im	Neg.	Som	Non.	Non.
Sinjar Formation	S1	-	2	-	719 – 703	16	71.9	43.80	0.68	0.45	0.22	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	3	-	703 – 683	20	70.5	41.00	0.73	0.49	0.24	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	4	-	683 – 681	2	30.5	3.25	9.23	6.15	3.27	Imp.	L.im	Neg.	Som	Non.	Non.
		-	5	-	681 – 677	4	72.1	44.20	0.67	0.45	0.22	Neg.	Neg.	Neg.	Non.	Non.	Non.
	S2	-	1	-	735 – 722	13	73	46	0.65	0.43	0.21	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	2	-	722 – 712	10	71.4	42.80	0.70	0.46	0.23	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	3	-	712 – 704	8	68.7	37.40	0.80	0.53	0.26	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	4	-	704 – 697	7	62.75	25.50	1.17	0.78	0.39	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	5	-	697 – 692	5	74.1	48.20	0.62	0.41	0.20	Neg.	Neg.	Neg.	Non.	Non.	Non.
	S3	-	1	-	745 – 731	14	72.5	45	0.66	0.44	0.22	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	2	-	731 – 711	20	68.5	37	0.81	0.54	0.27	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	3	-	711 – 704.5	6.50	55.8	11.60	2.58	1.72	0.86	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	4	-	704.5 –700	4.50	70.2	40.40	0.74	0.49	0.24	Neg.	Neg.	Neg.	Non.	Non.	Non.
Unconformity		-	5	-	700 – 694	6	56.3	12.60	2.38	1.58	0.79	Neg.	Neg.	Neg.	Non.	Non.	Non.
		-	6		694 – 686	8	50.8	10.47	2.86	1.91	0.95	Neg.	Neg.	Neg.	Non.	Non.	Non.
Kolosh Formation	-	B1	1	37.80 – 40.65	701.20 – 698.35	2.85	25	2.37	12.6	8.43	4.21	Imp.	Imp.	L.im	Som	Som	Non.
	-		2	40.65 – 47.50	698.35 – 691.50	6.85	67.4	34.80	0.86	0.57	0.28	Neg.	Neg.	Neg.	Non.	Non.	Non.
	-		3	47.50 – 52.50	691.50 – 686.50	5	52	11.22	2.67	1.78	0.89	Neg.	Neg.	Neg.	Non.	Non.	Non.
	-		4	52.50 – 63	686.50 – 676	10.50	50	10	3	2	1	Neg.	Neg.	Neg.	Non.	Non.	Non.
	-	B2	1	19.50 – 22.10	660.50 – 657.90	2.60	27	2.66	11.2	7.51	3.75	Imp.	Neg.	Neg.	Som	Non.	Non.
	-		2	22.10 – 25.50	657.90 – 654.50	3.40	30	3.16	9.49	6.32	3.16	Imp.	Neg.	Neg.	Som	Non.	Non.
	-		3	25.50 – 28.50	654.50 – 651.50	3	39	5.30	5.66	3.77	1.88	L.im	Neg.	Neg.	Non.	Non.	Non.
	-		4	28.50 – 32.90	651.50 – 647.10	4.40	34	3.98	7.53	5.02	2.51	L.im	Neg.	Neg.	Non.	Non.	Non.
	-		5	32.90 – 35.75	647.10 – 644.25	2.85	36	4.46	6.72	4.48	2.24	L.im	Neg.	Neg.	Non.	Non.	Non.
	-		6	35.75 -38	644.25 – 642	2.25	66.1	32.2	0.93	0.62	0.31	Neg.	Neg.	Neg.	Non.	Non.	Non.
	-		7	38 – 45	642 – 635	7	39	5.30	5.66	3.77	1.88	L.im	Neg.	Neg.	Non.	Non.	Non.
	-	B3	1	13 – 17	659 – 655	4	35	4.21	7.12	4.75	2.37	L.im	Neg.	Neg.	Non.	Non.	Non.
	-		2	17 – 28	655 – 644	11	76.2	52.4	0.57	0.38	0.19	Neg.	Neg.	Neg.	Non.	Non.	Non.
	-		3	28 – 40	644 – 632	12	37	4.73	6.34	4.22	2.11	L.im	Neg.	Neg.	Non.	Non.	Non.

*DMR_{DEF} (RMR related to deformability = $RMR_{BD\ (1976)}$ – 5), the values of DMR_{DEF} are from tables (5-20), (5-21), (5-22), (5-23), (5-24), (5-25) and (5-26).
** E_m (Deformation modulus of the rock mass), according to "Romana, 2003a", if $RMR_{BD} > 60$ or $DMR_{DEF} > 55$ the $E_m = 2RMR – 100$ was used, and if $RMR_{BD} < 60$ or $DMR_{DEF} < 55$ the $E_m = 10^{(RMR-10)/40}$ was used (Note: Here RMR = DMR_{DEF}). ***E_c / E_m and its guidelines are based on the table (4-12). ****CVC = Conventional vibrated concrete dam (Gravity dam, having E_c = 30 GPa). *****RCC = Roller compacted concrete dam (Gravity dam, having E_c = 20 GPa). ******Hardfill dam (having E_c = 10 GPa). ■The same guidelines for Gravity dam was also used for hardfill due to the lack of guidelines for Hardfill dam.
Note about abbreviations: Neg.= Negligible, L.im.= Low importance, Imp.= Important, Som.= Some, Non.= None.

Table (5-33) Deformability problems in the proposed Basara dam depending on the value of DMR_{DEF} and dam type & height according to the table (4-13)

Geologic unit	Surface section no.	Bore hole no.	Rock mass unit	Depth below surface (m)	Elevation above sea level (m)	Thickness of the unit (m)	*DMR_{DEF}	Deformability problems			*****Dam height (m)
								**CVC dam	***RCC dam	****Hardfill dam	
Gercus Fn.	S1	-	1	-	725 – 719	6	28	Serious prob.	Serious prob.	Serious prob.	60
Sinjar Formation	S1	-	2	-	719 – 703	16	71.90	Normal	Normal	Normal	
		-	3	-	703 – 683	20	70.50	Normal	Normal	Normal	
		-	4	-	683 – 681	2	30.50	Serious prob.	Serious prob.	problems	
		-	5	-	681 – 677	4	72.10	Normal	Normal	Normal	
	S2	-	1	-	735 – 722	13	73	Normal	Normal	Normal	
		-	2	-	722 – 712	10	71.40	Normal	Normal	Normal	
		-	3	-	712 – 704	8	68.70	Normal	Normal	Normal	
		-	4	-	704 – 697	7	62.75	Normal	Normal	Normal	
		-	5	-	697 – 692	5	74.10	Normal	Normal	Normal	
	S3	-	1	-	745 – 731	14	72.50	Normal	Normal	Normal	
		-	2	-	731 – 711	20	68.50	Normal	Normal	Normal	
		-	3	-	711 – 704.5	6.50	55.80	Normal	Normal	Normal	
		-	4	-	704.5 –700	4.50	70.20	Normal	Normal	Normal	
Unconformity		-	5	-	700 – 694	6	56.30	Normal	Normal	Normal	
		-	6		694 – 686	8	50.80	Normal	Normal	Normal	
Kolosh Formation	-	B1	1	37.80 – 40.65	701.20 – 698.35	2.85	25	Serious prob.	Serious prob.	Serious prob.	
	-		2	40.65 – 47.50	698.35 – 691.50	6.85	67.40	Normal	Normal	Normal	
	-		3	47.50 – 52.50	691.50 – 686.50	5	52	Normal	Normal	Normal	
	-		4	52.50 – 63	686.50 – 676	10.50	50	Norm. - prob.	Normal	Normal	
	-	B2	1	19.50 – 22.10	660.50 – 657.90	2.60	27	Serious prob.	Serious prob.	Serious prob.	
	-		2	22.10 – 25.50	657.90 – 654.50	3.40	30	Serious prob.	Serious prob.	Se.prob.-prob	
	-		3	25.50 – 28.50	654.50 – 651.50	3	39	Serious prob.	problems	problems	
	-		4	28.50 – 32.90	651.50 – 647.10	4.40	34	Serious prob.	Serious prob.	problems	
	-		5	32.90 – 35.75	647.10 – 644.25	2.85	36	Serious prob.	problems	problems	
	-		6	35.75 -38	644.25 – 642	2.25	66.10	Normal	Normal	Normal	
	-		7	38 – 45	642 – 635	7	39	Serious prob.	problems	problems	
	-	B3	1	13 – 17	659 – 655	4	35	Serious prob.	Se.prob.-prob	problems	
	-		2	17 – 28	655 – 644	11	76.20	Normal	Normal	Normal	
	-		3	28 – 40	644 – 632	12	37	Serious prob.	problems	problems	

*DMR_{DEF} (RMR related to deformability = $RMR_{BD\ (1976)}$ – 5), the values of DMR_{DEF} are from tables (5-20), (5-21), (5-22), (5-23), (5-24), (5-25) and (5-26).
**CVC = Conventional vibrated concrete dam (Gravity dam, having E_c = 30 GPa). ***RCC = Roller compacted concrete dam (Gravity dam, having E_c = 20 GPa).
****Hardfill dam (having E_c = 10 GPa). *****The proposed height of the proposed **Basara** dam is **60m** from the valley floor. Se.=Serious. Prob.=Problems.

Table (5-34) Classification of intact rock strength, comparison between unconfined compressive strength of intact rock and global strength of rockmass and also between GSI and $RMR_{BD(1976)}$

Geologic unit	Surface section no.	Bore hole no.	Rock mass unit	Depth below surface (m)	Elevation above sea level (m)	Thickness of the unit (m)	Unconfined compressive strength of intact rock (MPa)	Field estimate of strength of intact rock according to the table (3 – 5)	Global strength of the rock mass (MPa)	Geological Strength Index (GSI)	$RMR_{BD(1976)}$
Gercus Fn.	S1	-	1	-	725 – 719	6	20.58	Weak	2.422	33	33
Sinjar Formation	S1	-	2	-	719 – 703	16	57.71	Strong	17.829	76.5	76.9
		-	3	-	703 – 683	20	54.905	Strong	16.391	75.5	75.4
		-	4	-	683 – 681	2	10	Weak	0.906	35	35.5
		-	5	-	681 – 677	4	75.54	Strong	24.607	78	77.1
	S2	-	1	-	735 – 722	13	62.40	Strong	21.081	79	78
		-	2	-	722 – 712	10	47.905	Moderately strong	15.328	77.5	76.4
		-	3	-	712 – 704	8	54.95	Strong	15.112	73	73.7
		-	4	-	704 – 697	7	41.27	Moderately strong	9.483	67	67.75
		-	5	-	697 – 692	5	74.34	Strong	26.071	80	79.1
	S3	-	1	-	745 – 731	14	63.21	Strong	19.871	77	77.5
		-	2	-	731 – 711	20	74.57	Strong	20.508	73	73.5
		-	3	-	711 – 704.5	6.50	77.35	Strong	15.321	61.5	60.8
		-	4	-	704.5 –700	4.50	69.86	Strong	19.842	74	75.2
Unconformity		-	5	-	700 – 694	6	67.73	Strong	14.229	62	61.3
		-	6	-	694 – 686	8	68.67	Strong	12.132	55	55.8
Kolosh Formation	-	B1	1	37.80 – 40.65	701.20 – 698.35	2.85	5	Very weak - weak	0.542	30	30
	-		2	40.65 – 47.50	698.35 – 691.50	6.85	42.512	Moderately strong	11.328	72	72.4
	-		3	47.50 – 52.50	691.50 – 686.50	5	26.11	Moderately strong	6.210	57	57
	-		4	52.50 – 63	686.50 – 676	10.50	21.90	Weak	4.976	55	55
	-	B2	1	19.50 – 22.10	660.5 – 657.9	2.60	22.67	Weak	2.597	32	32
	-		2	22.10 – 25.5	657.90 – 654.50	3.40	26.59	Moderately strong	3.298	35	35
	-		3	25.50 – 28.50	654.50 – 651.50	3	34.65	Moderately strong	6.125	44	44
	-		4	28.50 – 32.90	651.50 – 647.10	4.40	30.74	Moderately strong	4.532	39	39
	-		5	32.90 – 35.75	647.10 – 644.25	2.85	33.17	Moderately strong	5.132	41	41
	-		6	35.75 -38	644.25 – 642	2.25	51.257	Strong	13.244	71	71.1
	-		7	38 – 45	642 – 635	7	38.05	Moderately strong	6.726	44	44
	-	B3	1	13 – 17	659 – 655	4	26.83	Moderately strong	4.052	40	40
	-		2	17 – 28	655 – 644	11	53.40	Strong	19.460	81	81.2
	-		3	28 – 40	644 – 632	12	34.12	Moderately strong	5.406	42	42

Note: Green colour (carbonate rock masses) give GSI values from this study chart (Table 5-18).

Table (5 – 35) Supplementary surface section of horizontal – subhorizontal attitude, showing the Unconformity between Sinjar and Kolosh Formations and the upper part of Kolosh Formation

Elevation above sea level (m)	Thickness (m)	Geologic unit	Lithology	Detailed Lithology	Remarks on detailed lithology
-	-	Sinjar Fn.	Limestone	-	-
738 - 730	8	Unconformity	Limestone	Pelintrabiosparite (like S3 – U5)	From comparison of thin section of these three units with equivalent unit to each one
730 - 718	12			Pelintrabiosparite (like S3 – U6)	
718 - 792	26	Kolosh Fn.	Blocky sandstone and pebbly sandstone	Flysch of type A (like B1-U3 & B1-U4)	

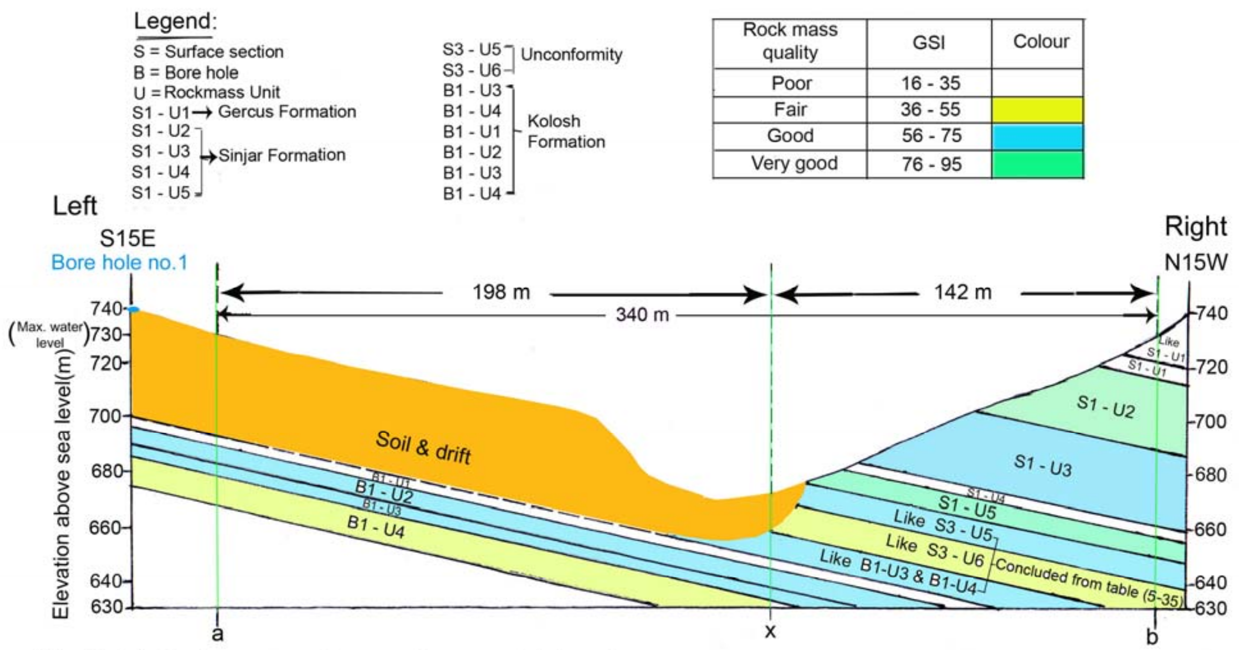


Fig (5-11) Rock mass valley section model showing the rock mass units along a-b topographic profile at the proposed Basara dam site

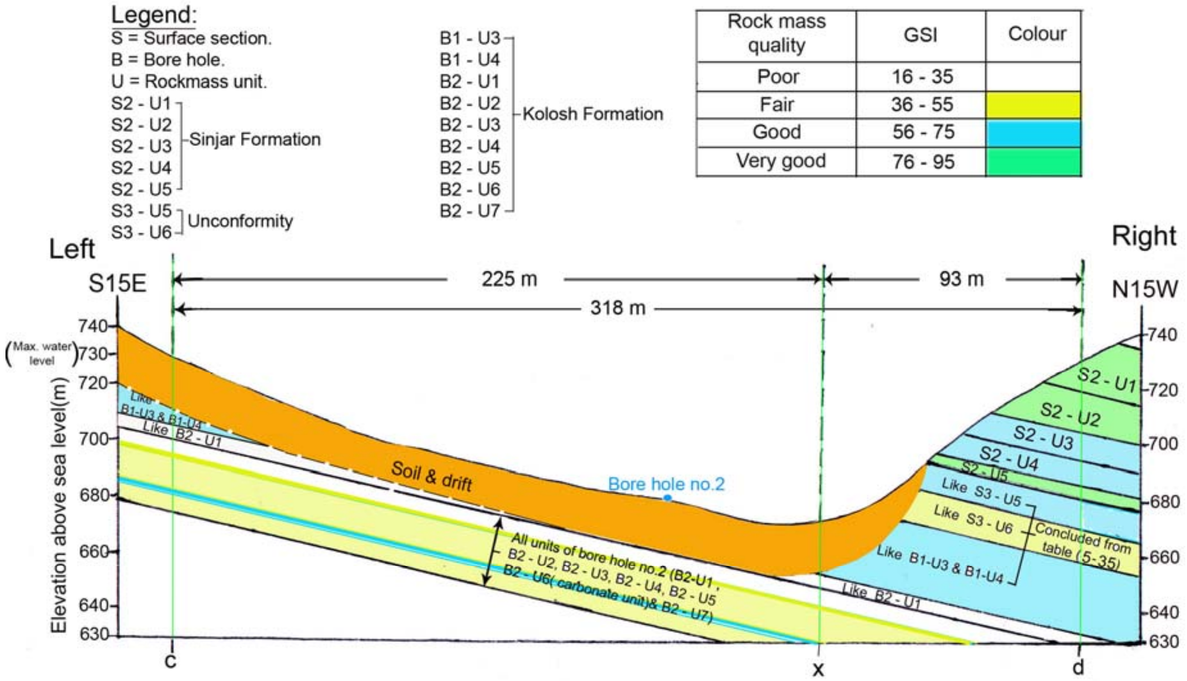


Fig (5 -12)Rock mass-valley section model showing the rock mass units along c-d topographic profile at the proposed Basara dam site

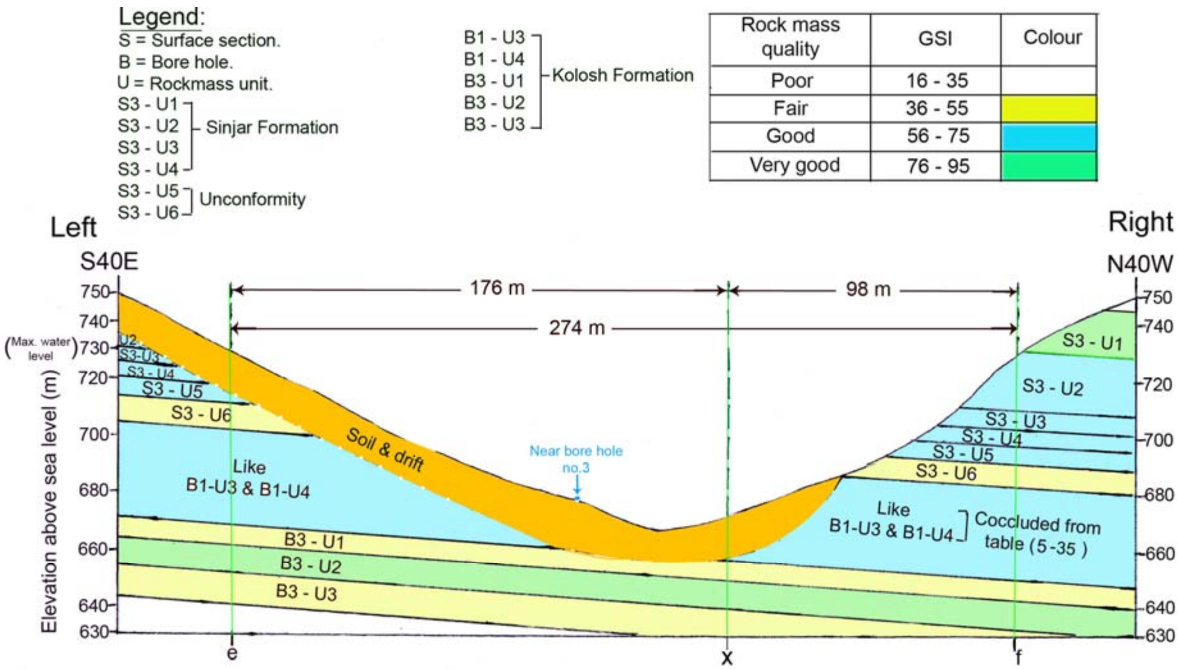


Fig (5-13) Rock mass-valley section model showing the rock mass units along e-f topographic profile at the proposed Basara dam site

Table (5 –36) Comparison between proposed dam profiles of Figures (5 – 11, 5 – 12 & 5 – 13)

Dam profile		a - b		c - d		e - f		Remarks
Horizontal distance (m)		340		318		274		From Figs (5-1), (5-11), (5-12) & (5-13)
Soil and drift thickness (m)		0 – 37.5		0 – 19.5		0 - 17		
Cross sectional area (m^2)	Before stripping soil & drift	9400		10640		9900		
	After stripping soil & drift	15920		14900		12780		
Sub – profile		a – x (left)	x – b (right)	c – x (left)	x – d (right)	e – x (left)	x – f (right)	
Horizontal distance (m)		198	142	225	93	176	98	
Soil and drift thickness (m)		12 – 37.5	0 – 12 (small part)	17 – 19.5	0 – 17 (small part)	14 - 17	0 – 14 (small part)	
Effect of weak sheared zone		Has no effect		Has no effect		Has an effect		
The degree of safety of the dam against sliding		No primary concern except B1-U1	Primary concern for S1-U1 & S1-U4	Has primary concern except B2-U6	No primary concern	Has primary concern except for B3-U2	No primary concern	Comparison of profiles in Fig(5-11, 5-12 & 5-13) with table (5-30)
Foundation excavation	Gravity dam	Not desirable for B1-U1 only	Not desirable for S1-U1	Not desirable for B2-U1 only	desirable	desirable	desirable	Comparison of profiles in Fig(5-11, 5-12 & 5-13) with table (5-31)
	Rockfil I& Earthfill dam	desirable		desirable		desirable		
Effect of E_c / E_m on the dam	CVC	Negligible, except for B1-U1 (imporant –low important) near the middle part	Important-low impor. for S1-U1 & S1-U4	Important-low impor. for all units of Bh.2	Negligible	Low important for B3-U1 in the middle part	Negligible	Comparison of profiles in Fig(5-11, 5-12 & 5-13) with table (5-32)
	RCC			Negligible	Negligible	Negligible	Negligible	
	Hardfill		Negligible	Negligible	Negligible	Negligible	Negligible	
Problems of E_c / E_m on the dam	CVC	Some problems only for B1-U1	Some problems for S1-U1 & S1-U4	Some problems for B2-U1 & B2-U2	None	None	None	
	RCC		None	None	None	None	None	
	Hardfill	None	None	None	None	None	None	
Deformability problems	CVC	Serious problems for B1-U1(Bore hole no.1-Unit no.1) only	Serious problems-problems for S1-U1 (Surface section no.1-Unit no.1) & S1-U4	Serious problems-problems for all units of Bh.2, except B2-U6	Normal	Serious problems-problems for B3-U1 only in the middle part	Normal	Comparison of profiles in Fig(5-11, 5-12 & 5-13) with table (5-33)
	RCC				Normal		Normal	
	Hardfill				Normal		Normal	
Weighted average of global strength of the rockmass (MPa)		6.444	13.42	5.663	14.472	9.417	14.084	Depending on the global strength, deformation modulus in the table (5-29) and thickness of each unit in the profiles
Weighted average of deformat-ion modulus of the rockmass		7997.712	25544.867	3851.577	24862.872	11526.066	25149.688	
Effect of E_c / $E_{m\ (average)}$ on the dam	CVC	Low important	Negligible	Low important	Negligible	Negligible	Negligible	Comparison of E_c / $E_{m\ (average)}$ with table (4-12)
	RCC	Negligible	Negligible	Low important	Negligible	Negligible	Negligible	
	Hardfill	Negligible	Negligible	Negligible	Negligible	Negligible	Negligible	
Problems of E_c/E_{m} $_{(average)}$ on the dam	CVC	None	None	None	None	None	None	
	RCC	None	None	None	None	None	None	
	Hardfill	None	None	None	None	None	None	
$DMR_{DEF\ (average)}$		52.29	59.41	39.80	61.68	51.75	58.82	Depending on the thickness of each units
Deformability problem(average)	CVC	Normal	Normal	Serious problems	Normal	Normal	Normal	Comparison of $DMR_{DEF\ (average)}$ with table (4 -13)
	RCC	Normal	Normal	problems	Normal	Normal	Normal	
	Hardfill	Normal	Normal	Normal	Normal	Normal	Normal	

Choosing the optimum profile among these three profiles was done by comparing the evaluated parameters of each profile with each other in table (5-36) and also with the aid of Figures (5-11, 5-12 & 5-13) as follows:

1- From comparing among the three profiles on the basis of their length (horizontal distance), the e-f profile is seen primarily to be more suitable because it is the shortest one, secondly the c-d profile and then a-b profile as a finalist.

2- After finding the area of each profile, before and after stripping (removing) the soil and drift in each one, as in table (5-36), the e-f profile is primarily considered the most suitable one after stripping the soil and drift. This is due to less required materials for construction of the dam, secondly the c-d profile and finally the a-b profile.

3- The area of the soil and drift in each profile (=Area of the profile after stripping the soil and drift – Area of the profile before stripping the soil and drift) is 6520m^2, 4260m^2 and 2880m^2 in a-b, c-d & e-f profiles respectively. From comparing these, the e-f profile appears to be the best one because it requires the least effort and cost during stripping the soil and drift.

From the above points and also due to the presence of somewhat weak rocks of Gercus Formation above 719m from sea level, presence of serious problems for deformability resulted from low value of DMR_{BD} and some problems resulted from E_c / E_m for S1-U1 (Surface section no.1 – Unit no.1) and S1-U4 in the right side of the a-b profile, it will be concluded that the a-b profile is the least suitable one and can be excluded from comparison, so the choice of the optimum profile is restricted between b-c and e-f profiles.

Though the e-f profile is better than the c-d profile on the basis of the above mentioned points, but the final decision in choosing between these two profiles depends also on the comparison between them on the basis of the geotechnical properties that are recorded in table (5-36), this comparison is as follows:

The comparison between c-d and e-f profiles on the basis of the detailed and overall (based on the average values of the characteristics)

characteristics (parameters) that are present in table (5-36) reveals that the e-f profile has more suitable parameters than the c-d profile, but if they are compared on the basis of the effect of weak sheared zone(which is obvious in Fig (5-1)) on each one, the c-d profile is more suitable than e-f profile, because this weak sheared zone does not fall within the dam body or the reservoir, but in the case of constructing the dam along e-f profile, the effect of this weak sheared zone after filling the reservoir will be too much and causes instability and with the passage of time this instability is increased until it reaches the stage that the right side (bank) may fail and eventually the overall failure of the dam may occur, because in this case the weak sheared zone is located within the dam or reservoir. Finally, the choice of the optimum profile will be for c-d profile.

Questions which will be raised at choosing c-d profile are:

1- What type of dam is to be constructed?

2- How much unconsolidated materials (soil and drift) are required to be excavated?

3- What are the present problems according to the type of the dam?

The above questions may be answered as follows:

1- The right side (x-d) of the profile is adequate for all dam types and the left side (c-x) is adequate for Hardfill, Rockfill and Earthfill dams primarily or Gravity dam of RCC (Roller Compacted Concrete) type with some problems.

2- The area of the soil and drift that must be stripped until reaching the fresh rocks is equal to 4260m^2 (3840m^2 on the left side and 420m^2 on the right side) with cleaning the exposed surface of strong rocks on the right side until it reaches fresh rocks.

3- On the right side (x-d), there are no problems, whatever the type of the dam is, but on the left side (c-x) there are serious problems in the case of constructing CVC (Conventional Vibrated Concrete) - gravity dam and some problem of low importance in the case of RCC-gravity dam.

From the above questions and their answers, we can take into account the following proposed dam types at c-d profile:

1- Constructing Hardfill, Rockfill or Earthfill dams without any problem.

2-Constructing a composite dam due to the presence of inhomogeneous foundation rocks on the two sides, in which the left part (“c-x”=225m) (the relatively weak rock foundation) will be constructed of Hardfilll, Rockfill or Earthfill dam and the right part (“x-d”=93m) (the relatively strong rock foundation) will be constructed of RCC-Gravity dam without any problem.

Chapter Six

Conclusions and Recommendations

6-1 Conclusions

This study has led to the following conclusions:

1-The Dam Mass Rating (DMR) system is seen to be much suitable in the evaluation of the dam site foundation as follows:

1-1 The DMR_{STA} (RMR related to dam stability) values for the right side (bank) of c-d & e-f profiles range between 58.83 - 84.10 and of a-b profile between 36.46 – 82.10. These values indicate good stability of foundation rocks of the right part of c-d & e-f profiles and presence of some instability of primary effect (not serious effect) in the surface section no.1 – unit no.1(Gercus Formation) and surface section no.1 – unit no.4(Sinjar Formation) on the right side of a-b profile, because these two units are weak rocks.

The DMR_{STA} values of the left side of the a-b, c-d & e-f profiles range between 33.46 – 72.40, 36.80 – 71.10 & 40.03 – 81.20 respectively. These values indicate no instability for the foundation rocks on the left side of a-b profile and some instability of primary effect for the first upper unit (B1-U1), presence of instability of primary effect in all rock mass units of c-d & e-f profiles, except the two carbonate rock mass units.

1-2 The desirability for dam foundation excavation was evaluated from the $RMR_{BD(1989)}$ values which range between 35 – 84.1. These values reveal that the foundation rocks are desirable and can be excavated for rockfill and Earthfill dams, but for Gravity dam, the first (upper) rock mass units in each of the surface section no.1, borehole no.1 and borehole no.2 are not desirable, as shown in table (5-31). This means that they must be removed in the case of gravity dam.

1-3 The $RMR_{BD(1989)}$ values which range between 35 – 84.1 in table (5-31) indicate that the foundation rocks do not require grouting when the type of dam is Earthfill. Some rock mass units require grouting (spot grouting) especially for Flysch Kolosh Formation when the type of dam is Rockfill and require systematic grouting especially for Flysch Kolosh when the type of dam is Gravity.

1-4 The E_c / Em (deformation modulus of the dam / deformation modulus of foundation rocks) values for each of CVC-Gravity dam, RCC-Gravity dam and Hardfill dam range between 0.57–12.6, 0.38–8.43 and 0.19-4.21 for each one respectively. These values indicate no problems in the case of Hardfill dam construction, some problems of low important influence in the case of the RCC -Gravity dam and problems of low important – important influence for most of the rock mass units on the left side in the case of CVC –Gravity dam construction.

1-5 The DMR_{DEF} (RMR related to relative deformability) values range between 25-76.20, which indicate some - serious deformability problems on the left sides of the selected profiles for the dam. The degree of problems decreases whenever the dam type changed from CVC to RCC and from RCC to Hardfill dam and this conclusion is observed in table (5-33).

2-This study proposed a modified GSI chart for GSI determination of blocky rocks based on quantitative study. This chart proved to be highly precise and gave GSI values very close to $RMR_{BD(1976)}$ (as shown in table 5-34)and more precise than Hoek's chart (which is based on qualitative description of the rock mass).

3- There is direct relationship between the mechanical properties and GSI or RMR values (tables ""5-29 & 5-34"). Whenever the GSI or RMR has higher values, the mechanical properties will also have higher values. This reflects the fact that the profile which is characterized by the rock masses having high GSI or RMR values, is considered to be the best for the dam site, so the e-f profile is the best one, but the presence of weak sheared zone in it makes c-d profile the best.

4- The rock mass – valley section model proposed in this study proved to be very effective in differentiating between different proposed dam sites, and helps to choose the optimum one for dam construction.

This model combines the mechanical properties of the rocks beneath the profile with the topographic and cross-sectional area variations above the profile that give the best choice for dam site.

Accordingly, it is concluded from comparison among the three studied models of profiles a-b, c-d and e-f that the c-d profile model is the best site for dam construction for many reasons: (1)Its rocks have good mechanical properties like e-f profile (better than a-b profile).,(2)It is devoid of weak sheared zone (in comparison with the right bank side of the e-f profile), and devoid of weak rock in comparison with the right side bank of a-b profile., (3)It has about half thickness of soil and drift on the left bank than a-b profile, and (4)It has less horizontal distance and cross-sectional area (after stripping) than a-b profile.

5-The a-b profile is characterized by more horizontal distance and soil plus drift thickness on the left side than the c-d and e-f profiles, also the upper first rock mass unit on the right side of a-b is a very weak rock mass which is characterized by serious deformability problems; therefore, it is concluded that the a-b profile would be excluded from choosing for the dam site.

6- Though the e-f profile is characterized by less horizontal distance and area, less deformability problems in the foundation rocks and has more desirable foundation rocks than that of c-d profile, as shown in table (5-36), but due to the presence of a weak sheared zone near the right side of the e-f profile, it is concluded that the c-d profile is the best one for the dam site, because in the case of choosing the e-f profile, the effect of the mentioned weak sheared zone will be too much on the stability of the dam after constructing and filling the reservoir with water and causes instability which with the passage of time this instability will increase until it reaches a stage that the right side may fail and then the overall failure of the dam may occur.

7- The geological map shows that most parts of the proposed dam reservoir are located within a synclinal structure (New Sola – Qazanqaya syncline), which gives suitable structural configuration in collecting water, especially collecting larger amount of groundwater into the basin.

8-The convenient height of the proposed dam is estimated to be 60m above the valley floor. At the proposed dam site, the present water level of Basara stream is 670m above mean sea level and when the dam is constructed with 60m high, the water level in the reservoir will reach an elevation of 730m above mean sea level. This proposed height is based on the meteorological data that the total amount of inflow from catchment area is equal to $131x10^6 m^3$/year, which corresponds to an elevation of 730m above mean sea level (depending on the comparison between the total discharge and reservoir volume, as shown in Fig “2-32“).

6-2 Recommendations

This study also proposes the following recommendations:

1-The proposed dam along c-d profile gives the following options to the type of the dam:

A-It is recommended to construct Hardfill, Rockfill or Earthfill dams without any problem.

B-The proposed dam site lies on the strong Sinjar Formation in the right side and the weak Kolosh Formation in the left side; therefore, it is possible to recommend to construct a composite dam in which the left side (the relatively weak rock foundation) (“c-x“=225m) composes the Hardfill, Rockfill or Earthfill dam types and the right side (the relatively strong rock foundation) (“x-d“=93m) composes the RCC – Gravity dam type without any problem.

2- Though the carbonate rock mass units don’t require grouting according to the DMR system, but due to the presence of joint sets and systems in them, it is possible to recommend a systematic grouting for carbonate rock mass units during construction of the dam because these joints act as avenues for moving solutions through them and dissolving carbonate rocks which create cavities in them.

3- The e-f profile remains as the most suitable site than c-d profile for dam site if the future geophysical study (2D – Electric Resistivity method) reveals that the weak sheared zone on the right side of e-f profile does not extend to a great depth, this means that if it affects only the shallow surface rock mass units.

4- Though there is no any field evidence along Basara gorge for recognizing the fault, but it is recommended to carry out a geophysical study (Seismic method) to check whether the Basara gorge is a fault or not and to ensure that it is active or not in the case of fault presence.

5- Slope stability assessment study is necessary near the dam site, for the reservoir and the slopes surrounding the reservoir, which may create instability and problems in the reservoir and near the dam site; therefore, it is recommended to carry out such study in the future for the reservoir slopes in the study area.

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Appendix – A: Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface sections.

Table (A-1) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.1, unit no.4 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency**
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)						0.04*	25
Joint set 1 (S_1)						0.04	25
Joint set 2 (S_2)						0.04	25
Random joint ***							
Calculations	Volumetric joint count Jv=∑ freguencies						75 Average Jv
	Block volume**** $Vb_0=S_0*S_1*S_2$					$64*10^{-6}$ m^3 Average Vb_o	

- RQD = 110 -2.5 Jv = 0 (because Jv > 44)

-Equivalent block volume: Vb =βJv $^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.04 / 0.04 * 3) = 27

- Vb = 27 * (1 / 75^3) m^3 = $64*10^{-6}$ m^3 = 64 cm^3

*From field measurements. ** Average frequency=1/Average spacing.

***For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

****Block volume for joint intersection at approximately right angles.

Table (A - 2) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.1, unit no.5 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.3	1.4	3.33/m	0.71/m	0.85	1.17
Joint set 1 (S_1)		0.3	1.2	3.33/m	0.83/m	0.75	1.33
Joint set 2 (S_2)		0.2	1.1	5/m	0.91/m	0.65	1.53
Random joint **							
Calculations	Volumetric joint count Jv=∑ freguencies						4.03 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					$0.414\ m^3$ Average Vb_o	

- RQD = 110 -2.5 Jv = 110 – (2.5 * 4.03) = 99.92

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.85 / 0.65 * 3) = 29.15

- Vb = 29.15 * (1 / 4.03^3) m^3 = 0.445 m^3 = 4.45 * 10^5 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 3) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.2, unit no.1 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.3	1	3.33/m	1/m	0.65	1.53
Joint set 1 (S_1)		0.3	1.2	3.33/m	0.830/m	0.75	1.33
Joint set 2 (S_2)		0.4	1.6	2.5/m	0.625/m	1	1
Random joint **							
Calculations	Volumetric joint count Jv=∑ freguencies						3.86 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					$0.487\ m^3$ Average Vb_0	

- RQD = 110 -2.5 Jv = 100 (because Jv < 4)

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and 1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (1 / 0.65 * 3) = 30.76

- Vb = 30.76 * (1 / 3.86^3) m^3 = 0.534 m^3 = 5.34 * 10^5 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 4) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.2, unit no.2(Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.2	1.4	5/m	0.71/m	0.80	1.25
Joint set 1 (S_1)		0.2	1	5/m	1/m	0.60	1.66
Joint set 2 (S_2)		0.5	1.5	2/m	0.66/m	1	1
Random joint **							
Calculations	Volumetric joint count Jv=∑ freguencies						3.91 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					0.48 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 100 (because Jv < 4)

-Equivalent block volume: Vb =βJv^{-3} …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (1 / 0.6 * 3) = 31.66

- Vb = 31.66 * (1 / 3.91^3) m^3 = 0.529 m^3 = 5.29 * 10^5 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 5) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.2, unit no.3 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.25	1.2	4/m	0.83/m	0.725	1.38
Joint set 1 (S_1)		0.20	1	5/m	1/m	0.60	1.66
Joint set 2 (S_2)		0.40	1.4	2.5/m	0.71/m	0.90	1.11
Random joint **		5		2/5=0.40/m	2/5=0.40/m	5	2 / 5 = 0.40
Calculations	Volumetric joint count Jv=∑ freguencies						4.55 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					0.391 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 110 – (2.5 * 4.55) = 98.62

-Equivalent block volume: Vb =βJv^{-3} …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.90 / 0.60 * 3.5) = 29

- Vb = 29 * (1 / 4.55^3) = 0.307 m^3 = 3.07 * 10^5 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 6) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.2, unit no.4 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.2	0.7	5/m	1.42/m	0.45	2.22
Joint set 1 (S_1)		0.2	0.7	5/m	1.42/m	0.45	2.22
Joint set 2 (S_2)		0.3	1.0	3.33/m	1/m	0.65	1.53
Random joint **							
Calculations	Volumetric joint count Jv=∑ freguencies						5.97 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					0.131 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 110 – (2.5 * 5.97) = 95.07

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.65 / 0.45 * 3) =30.11

- Vb = 30.11 * (1 / 5.97^3) m^3 = 0.141 m^3 = 1.41 * 10^5 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 7) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.2, unit no.5 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.3	1.4	3.33/m	0.71/m	0.85	1.18
Joint set 1 (S_1)		0.3	1.5	3.33/m	0.66/m	0.90	1.11
Joint set 2 (S_2)		0.25	1.4	4/m	0.71/m	0.825	1.21
Random joint **							
Calculations	Volumetric joint count Jv=∑ freguencies						3.50 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					0.631 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 100 (because Jv < 4)

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.90 / 0.85 * 3) = 27.41

- Vb = 27.41 * (1 / 3.5^3) m^3 = 0.639 m^3 = 6.39 * 10^5 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 8) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.3, unit no.1 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.25	1	4/m	1/m	0.625	1.60
Joint set 1 (S_1)		0.15	0.75	6.66/m	1.33/m	0.45	2.22
Joint set 2 (S_2)		0.20	1	5/m	1/m	0.60	1.66
Random joint **							
Calculations	Volumetric joint count Jv=∑ freguencies						5.48 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					0.168 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 110 – (2.5 * 5.48) = 96.30

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.625 / 0.45 * 3) = 29.72

- Vb = 29.72 * (1 / 5.48^3) m^3 = 0.18 m^3 = 1.8 * 10^5 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 9) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.3, unit no.2 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.2	1.5	5/m	0.66/m	0.85	1.17
Joint set 1 (S_1)		0.3	1.8	3.33/m	0.55/m	1.05	0.95
Joint set 2 (S_2)		0.2	0.7	5/m	1.42/m	0.45	2.22
Random joint **							
Calculations	Volumetric joint count Jv=∑ freguencies						4.34 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					0.401 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 110 – (2.5 * 4.34) = 99.15

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (1.05 / 0.45 * 3) = 36.33

- Vb = 36.33 * (1 / 4.34^3) m^3 = 0.444 m^3 = 4.44 * $1o^5$ cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 10) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.3, unit no.3 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)	Set spacing and frequency				Average spacing(m)	Average frequency*
	Spacing (m)		Max. frequency	Min. frequency		
	Min.	Max.				
Bedding plane (S_o)	0.10	0.20	10/m	5/m	0.15	6.66
Joint set 1 (S_1)	0.15	0.45	6.66/m	2.22/m	0.30	3.33
Joint set 2 (S_2)	0.10	0.40	10/m	2.5/m	0.25	4
Random joint **						
Calculations: Volumetric joint count Jv=∑ freguencies						13.99 Average Jv
Calculations: Block volume*** $Vb_0=S_0*S_1*S_2$					0.011 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 110 – (2.5 * 13.99) = 75.02

-Equivalent block volume: Vb =βJv $^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.30 / 0.15 * 3) = 34

- Vb = 34 * (1 /13.99^3)m^3 = 0.012 m^3 = 1.2 * 10^4 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 11) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.3, unit no.4 (Sinjar Formation)

Discontinuities (Bedding plane and Joints)	Set spacing and frequency				Average spacing(m)	Average frequency*
	Spacing (m)		Max. frequency	Min. frequency		
	Min.	Max.				
Bedding plane (S_o)	0.30	1.30	3.33/m	0.77/m	0.80	1.25
Joint set 1 (S_1)	0.25	1.50	4/m	0.66/m	0.87	1.15
Joint set 2 (S_2)	0.30	1.60	3.33/m	0.62/m	0.95	1.05
Random joint **	5		2/5=0.40/m	2/5=0.40/m	5	2 / 5 = 0.40
Calculations: Volumetric joint count Jv=∑ freguencies						3.85 Average Jv
Calculations: Block volume*** $Vb_0=S_0*S_1*S_2$					0.661 m^3 Average Vb_0	

-RQD = 110 -2.5 Jv = 100 (because Jv < 4)

-Equivalent block volume: Vb =βJv $^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20+21 (0.95 / 0.80 * 3.5) = 27.125

- Vb = 27.125 * (1 / 3.85^3) m^3 = 0.475 m^3 = 4.75 * 10^5 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 12) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.3, unit no.5 (Unconformity between Sinjar & Kolosh Formations)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.10	0.40	10/m	2.50/m	0.25	4
Joint set 1 (S_1)		0.15	0.60	6.66/m	1.66/m	0.375	2.66
Joint set 2 (S_2)		0.20	0.70	5/m	1.42/m	0.45	2.22
Random joint **		5		2/5=0.40/m	2/5=0.40/m	5	2 / 5 = 0.40
Calculations	Volumetric joint count Jv=∑ freguencies						9.28 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					0.042 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 110 – (2.5 * 9.28) = 86.8

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.45 / 0.25 * 3.5) = 30.8

- Vb = 30.8 * (1 / 9.28^3) m^3 = 0.038 m^3 = 3.8 * 10^4 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Table (A - 13) Volumetric joint count (Jv), Rock Quality Designation (RQD) and Block volume (Vb) measurements from joint sets observed in a rock surface section no.3, unit no.6 (Unconformity between Sinjar & Kolosh Formations)

Discontinuities (Bedding plane and Joints)		Set spacing and frequency				Average spacing(m)	Average frequency*
		Spacing (m)		Max. frequency	Min. frequency		
		Min.	Max.				
Bedding plane (S_o)		0.05	0.15	20/m	6.66/m	0.10	10
Joint set 1 (S_1)		0.05	0.30	20/m	3.33/m	0.175	5.71
Joint set 2 (S_2)		0.10	0.50	10/m	2/m	0.30	3.33
Random joint **							
Calculations	Volumetric joint count Jv=∑ freguencies						19.04 Average Jv
	Block volume*** $Vb_0=S_0*S_1*S_2$					0.0052 m^3 Average Vb_0	

- RQD = 110 -2.5 Jv = 110 – (2.5 * 19.04) = 62.40

-Equivalent block volume: $Vb = \beta Jv^{-3}$ …………..(Palmstrom,1995 and1996b)

Where: β is the block shape factor.

β = 20+21 (S max./S min * nj) = 20 + 21 (0.30 / 0.10 * 3) = 41

- Vb = 41 * (1 / 19.04^3) m^3 = 0.0059 m^3 = 5.9 * 10^3 cm^3

* Average frequency=1/Average spacing.

**For random joints, a spacing of 5m for each random joint is used in the Jv calculation.

***Block volume for joint intersection at approximately right angles.

Appendix – B: Analysis of rock strength for surface sections and boreholes, using RocLab programme

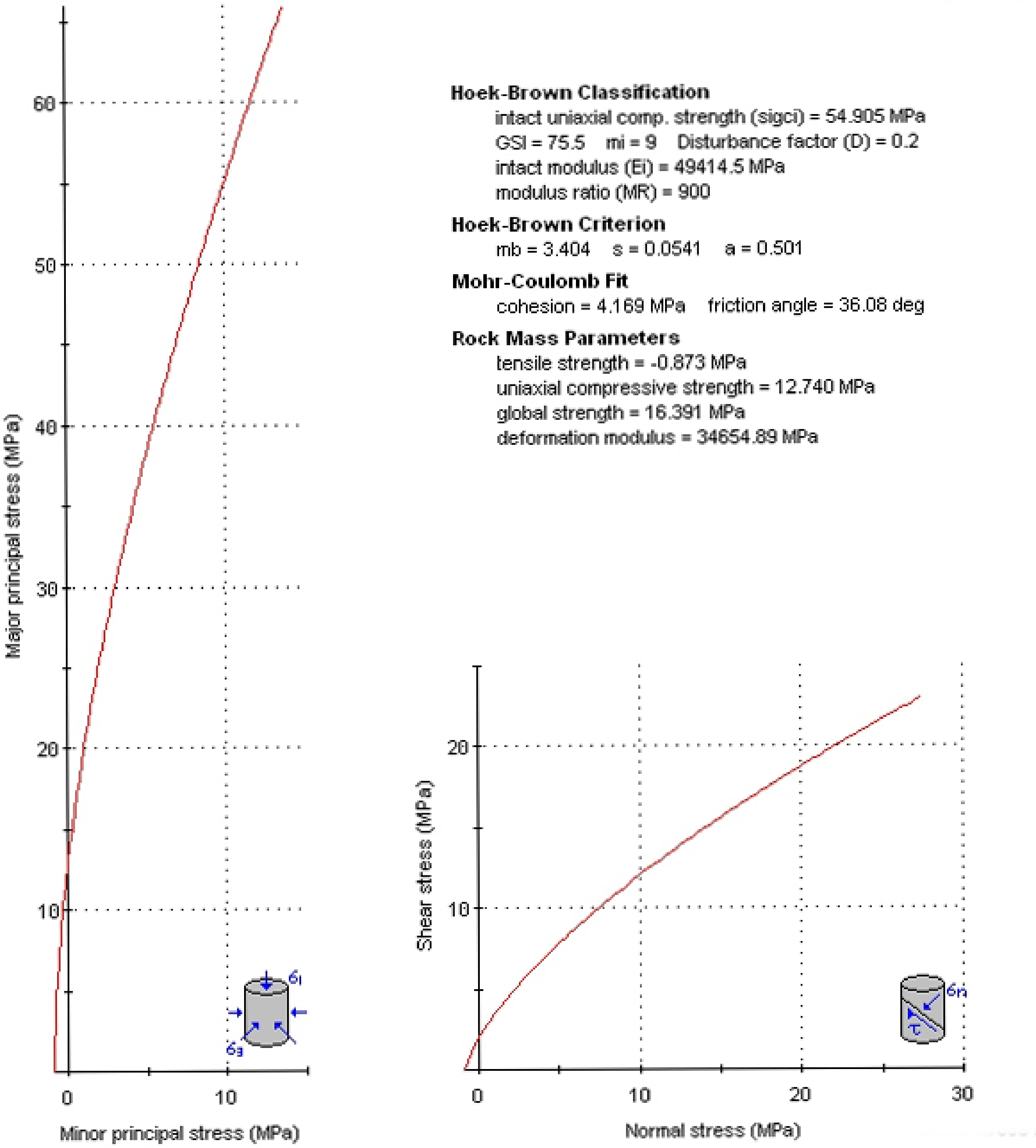


Fig (B -1) Analysis of rock strength for surface section no.1 - unit no.3 (Sinjar Formation), using RocLab programme

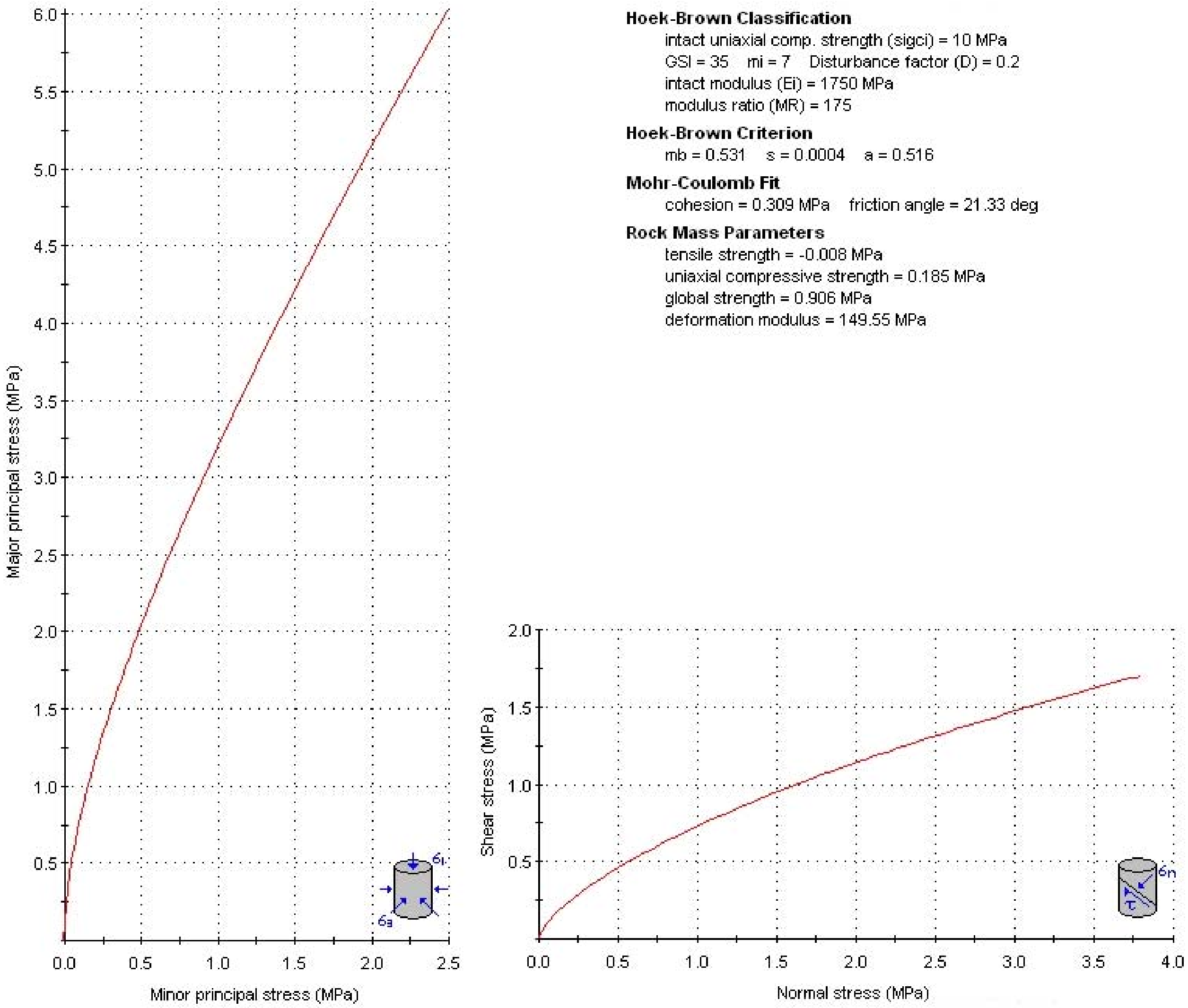


Fig (B - 2) Analysis of rock strength for surface section no.1 - unit no.4 (Sinjar Formation), using RocLab programme

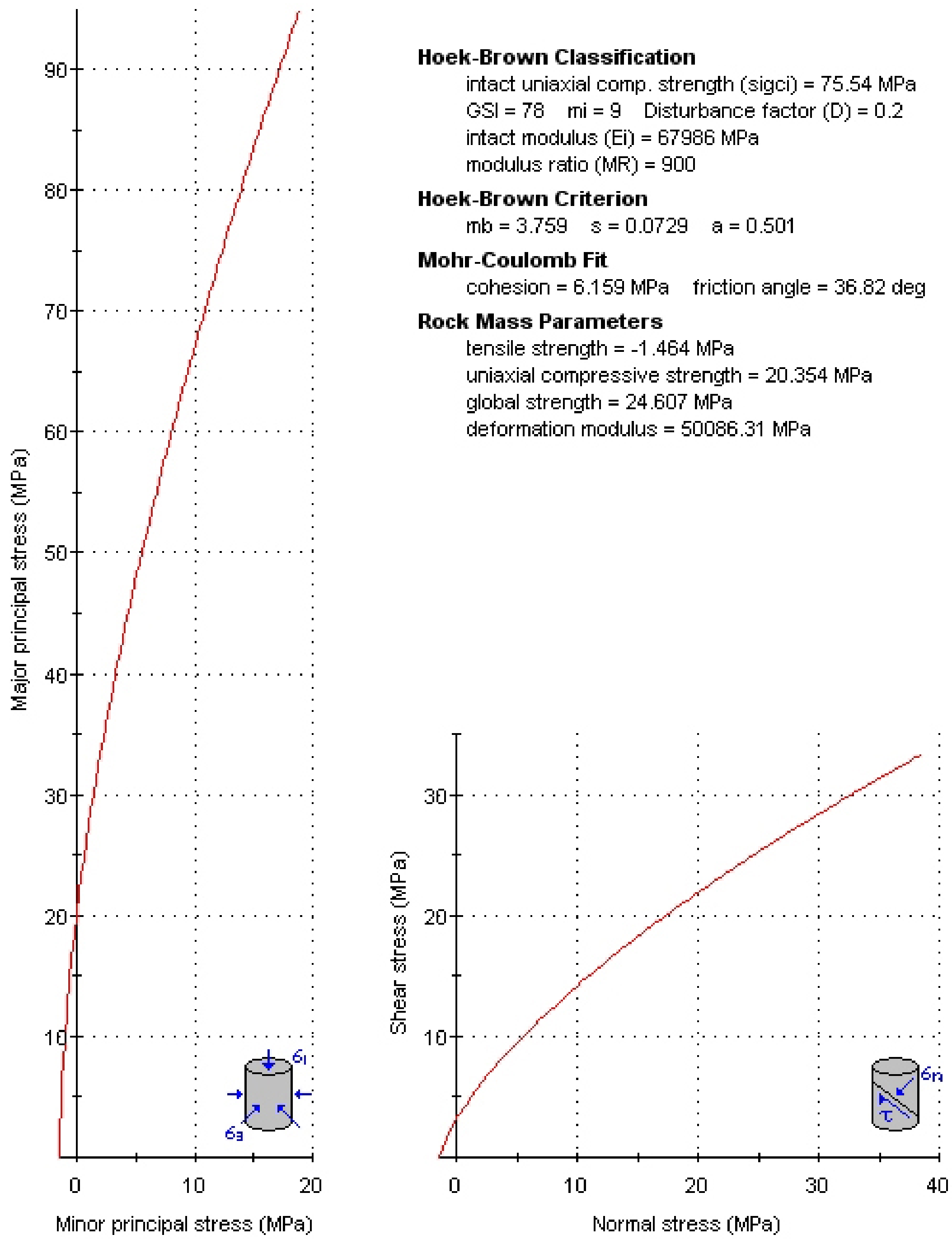


Fig (B - 3) Analysis of rock strength for surface section no.1 - unit no.5 (Sinjar Formation), using RocLab programme

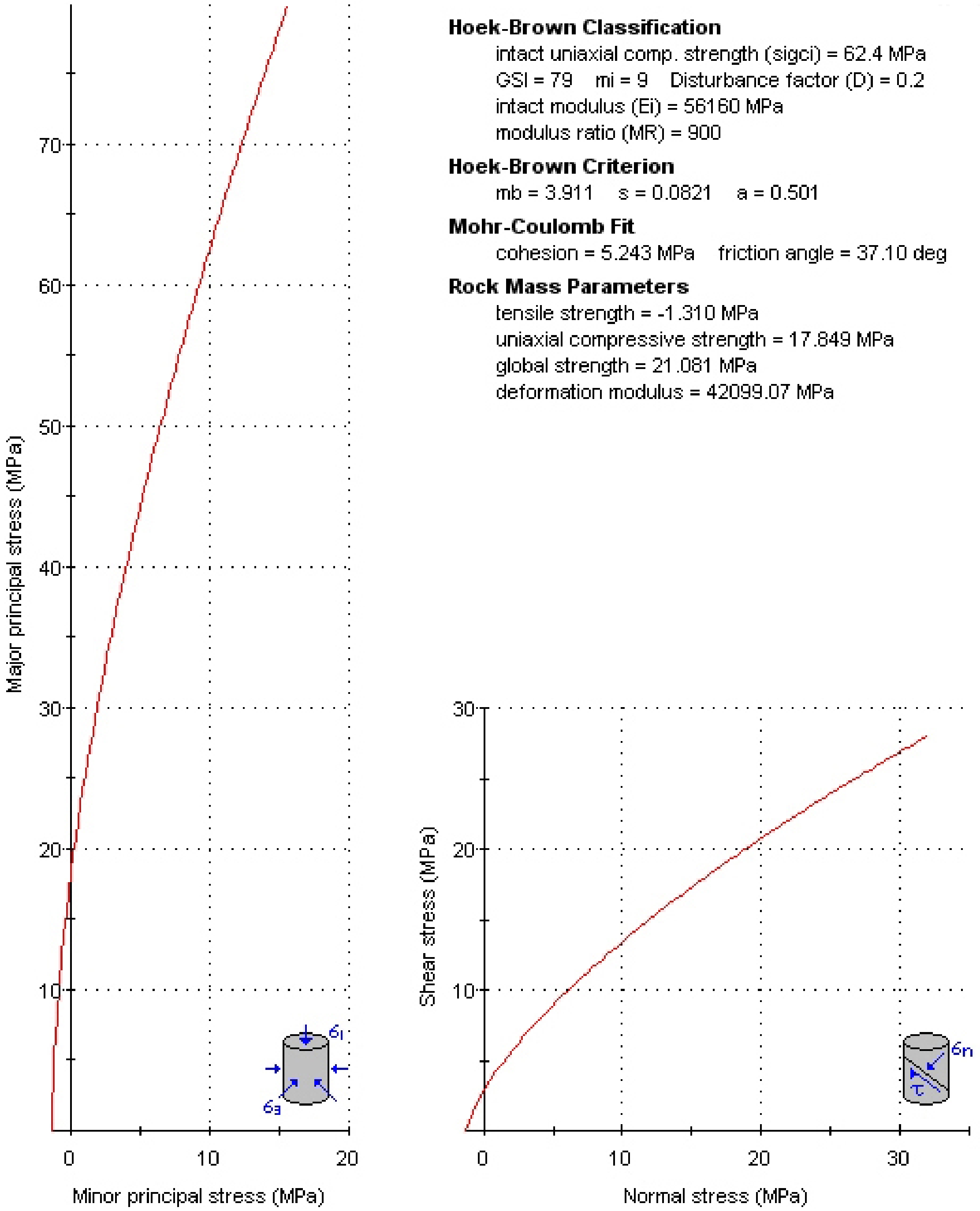


Fig (B -4) Analysis of rock strength for surface section no.2 - unit no.1 (Sinjar Formation), using RocLab programme

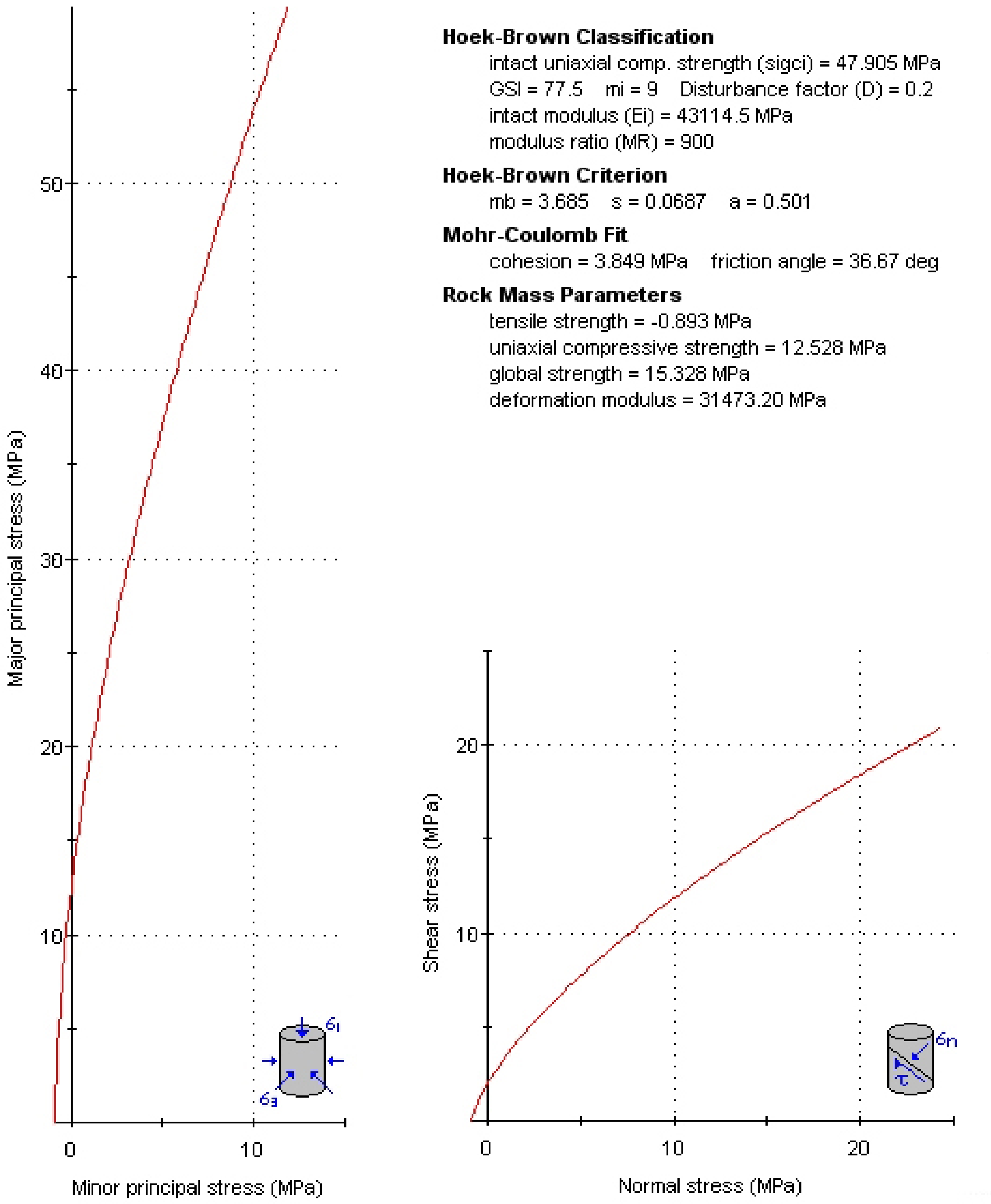


Fig (B - 5) Analysis of rock strength for surface section no.2 - unit no.2 (Sinjar Formation), using RocLab programme

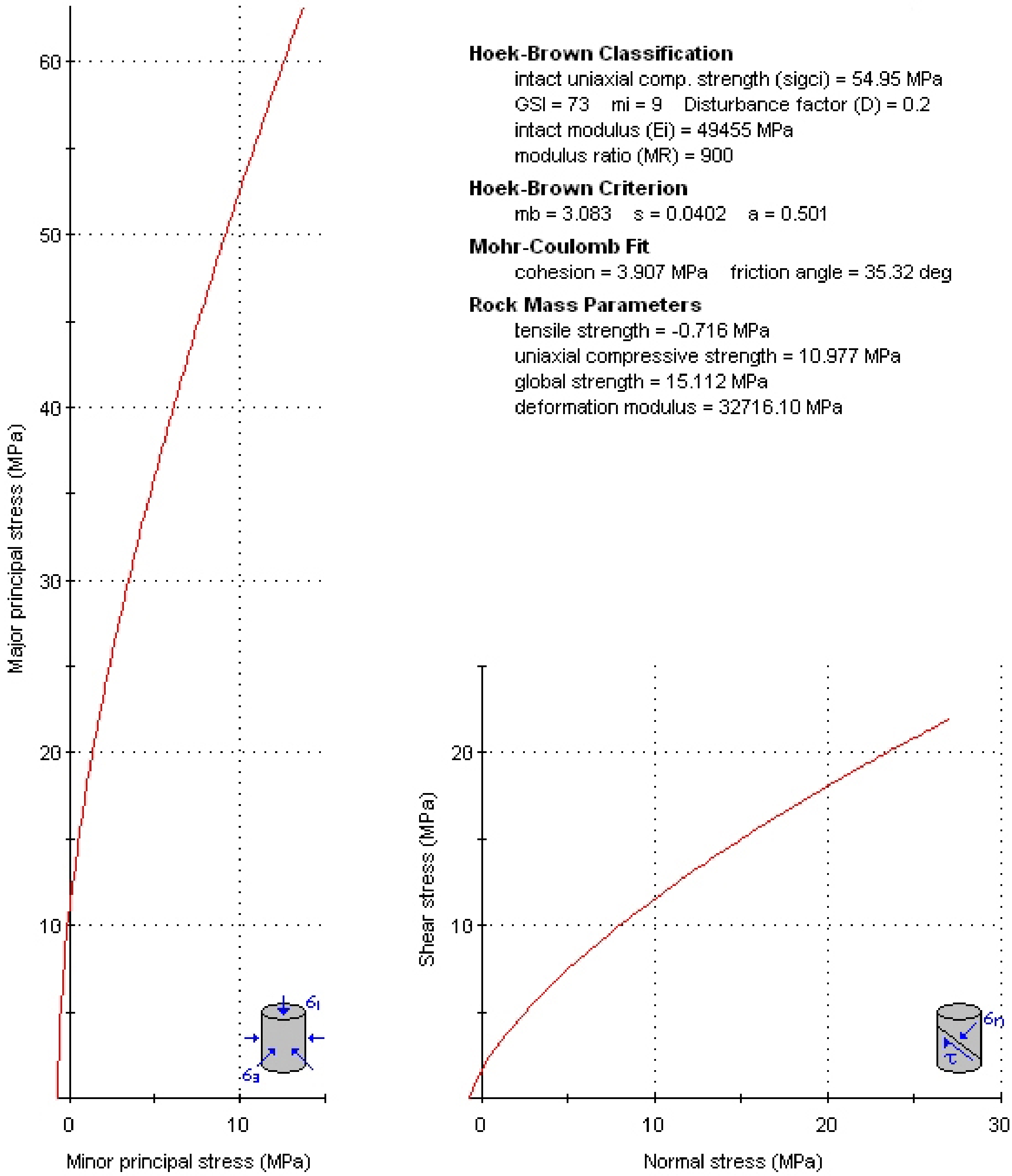


Fig (B -6) Analysis of rock strength for surface section no.2 - unit no.3 (Sinjar Formation), using RocLab programme

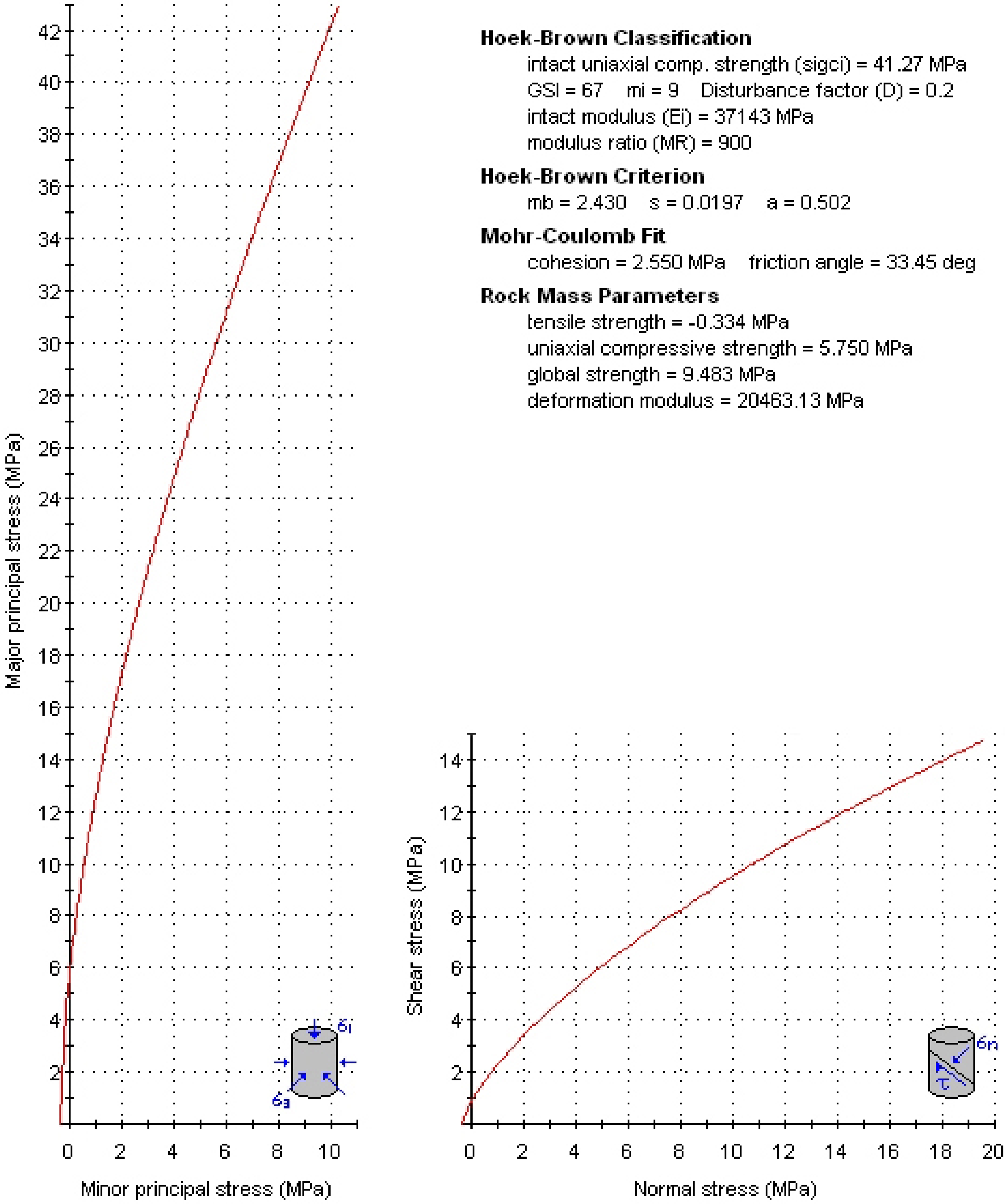


Fig (B - 7) Analysis of rock strength for surface section no.2 - unit no.4 (Sinjar Formation), using RocLab probramme

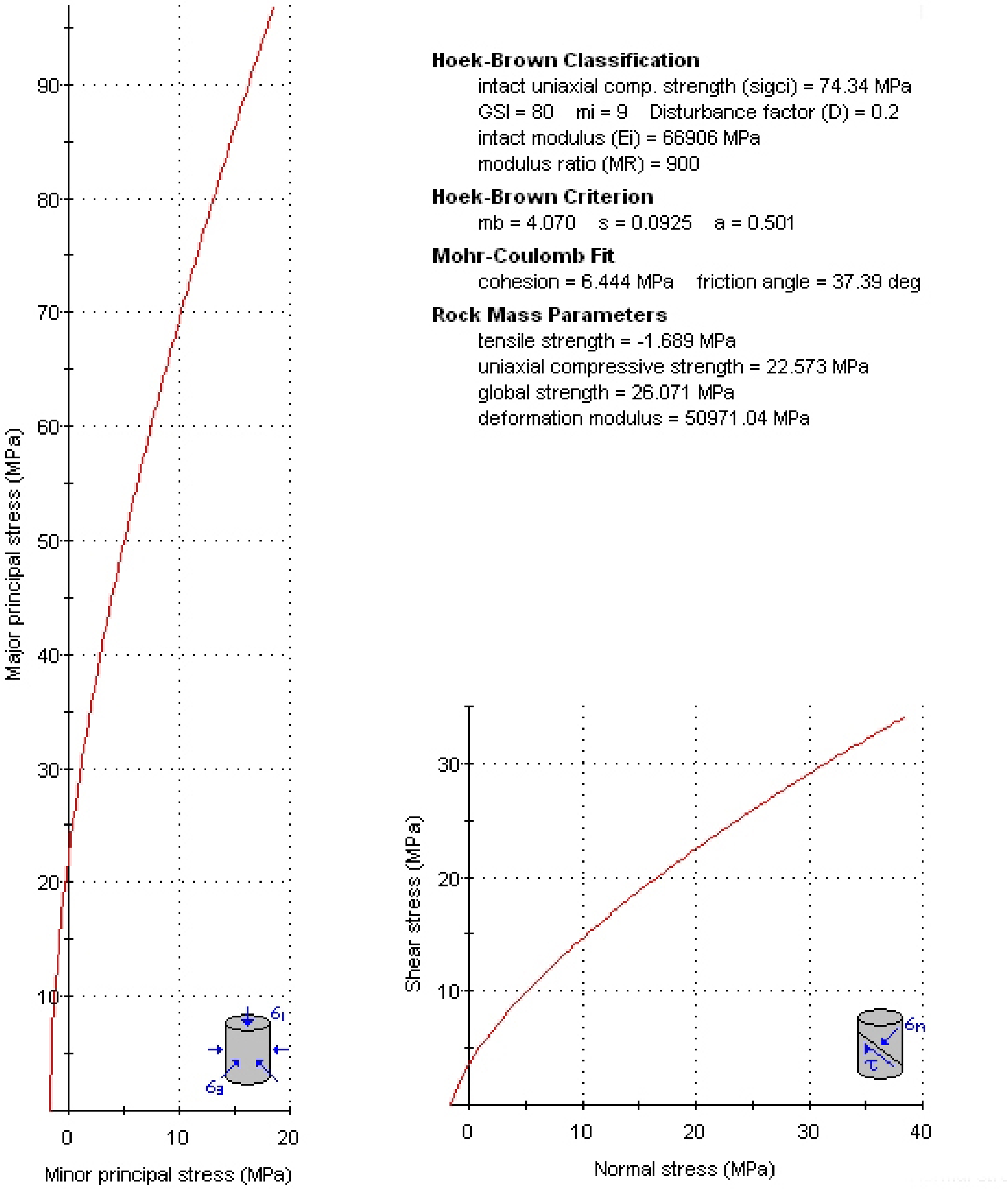


Fig (B - 8) Analysis of rock strength for surface section no.2 - unit no.5 (Sinjar Formation), using RocLab programme

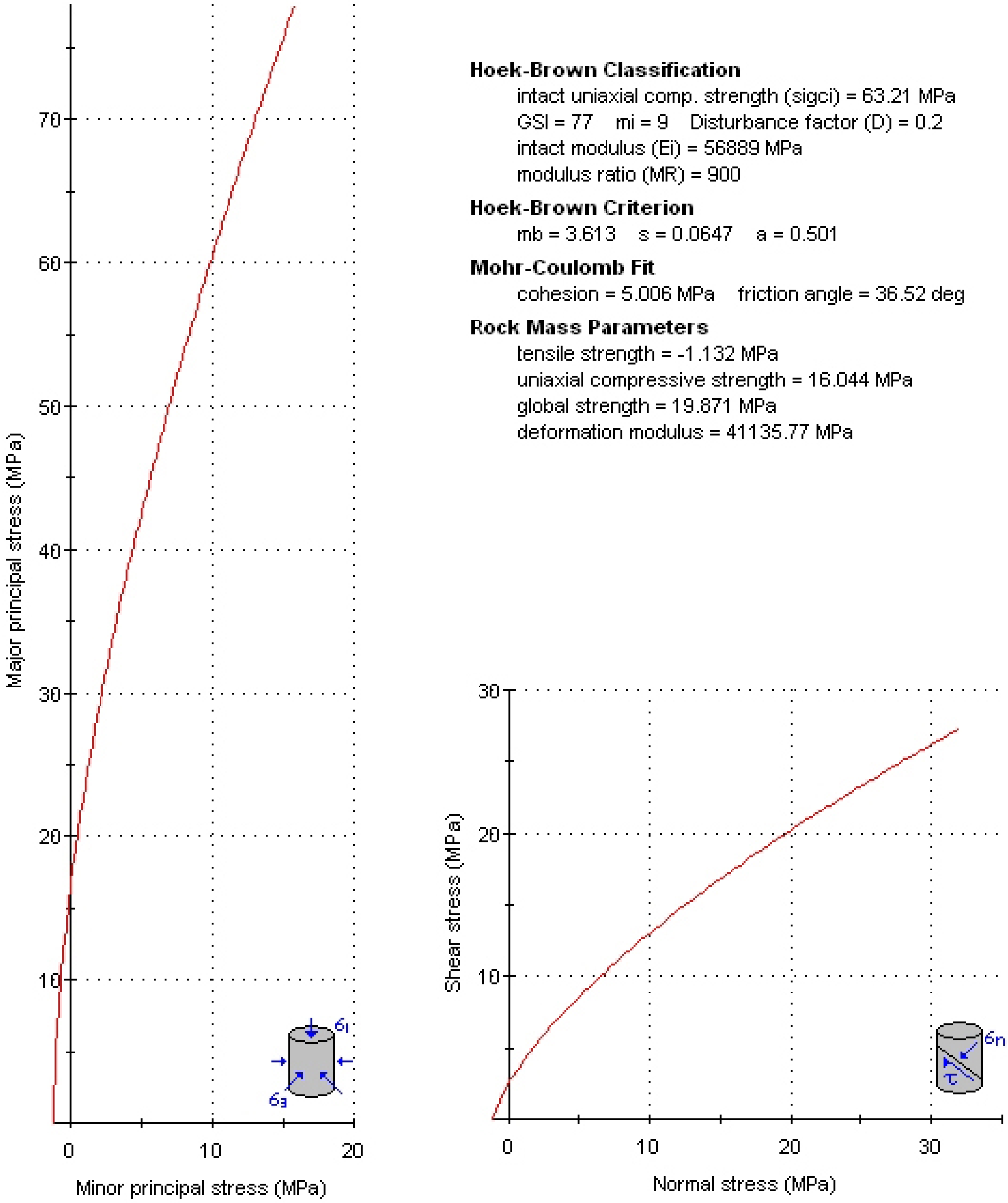


Fig (B - 9) Analysis of rock strength for surface section no.3 - unit no.1 (Sinjar Formation), using RocLab programme

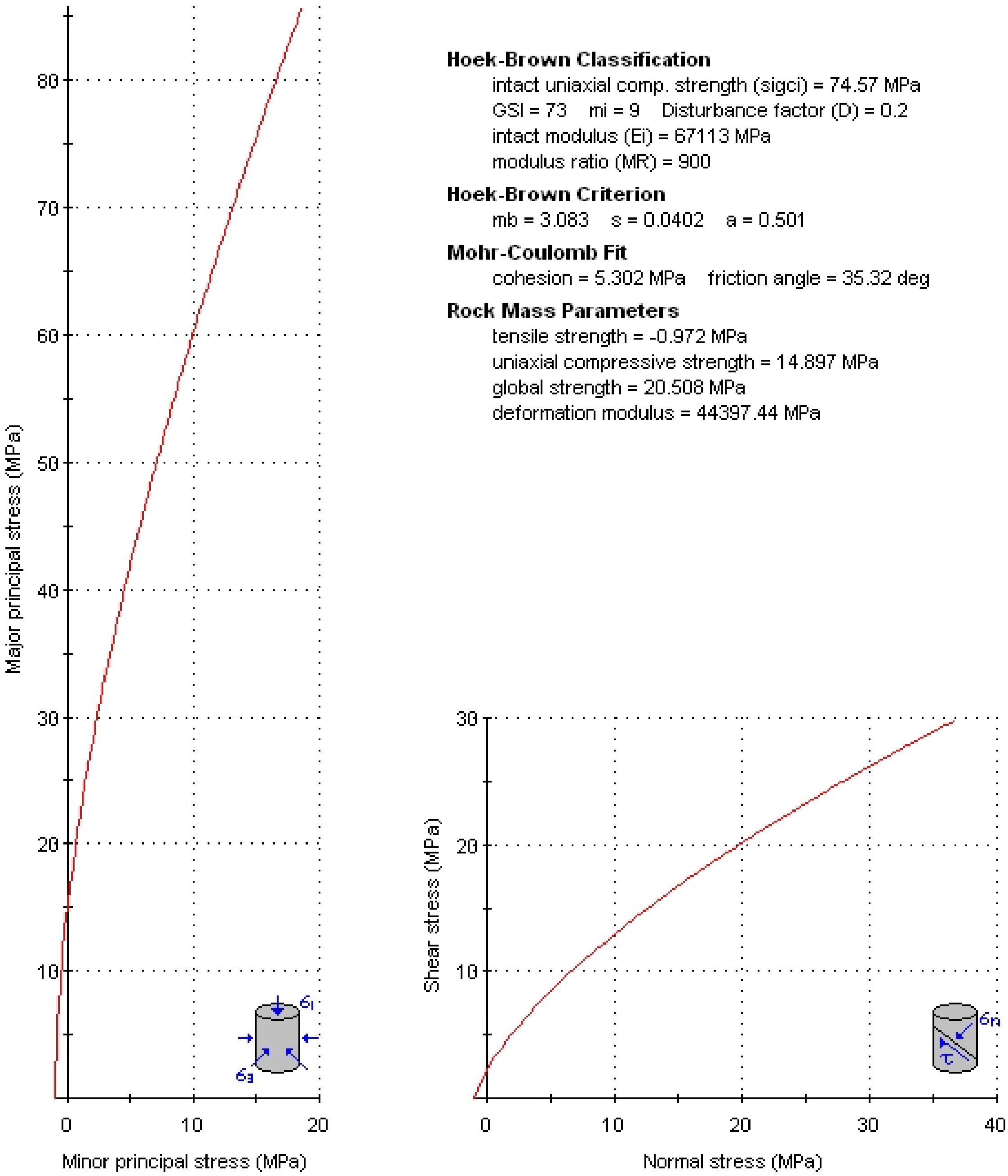


Fig (B - 10) Analysis of rock strength for surface section no.3 - unit no.2 (Sinjar Formation), using RocLab programme

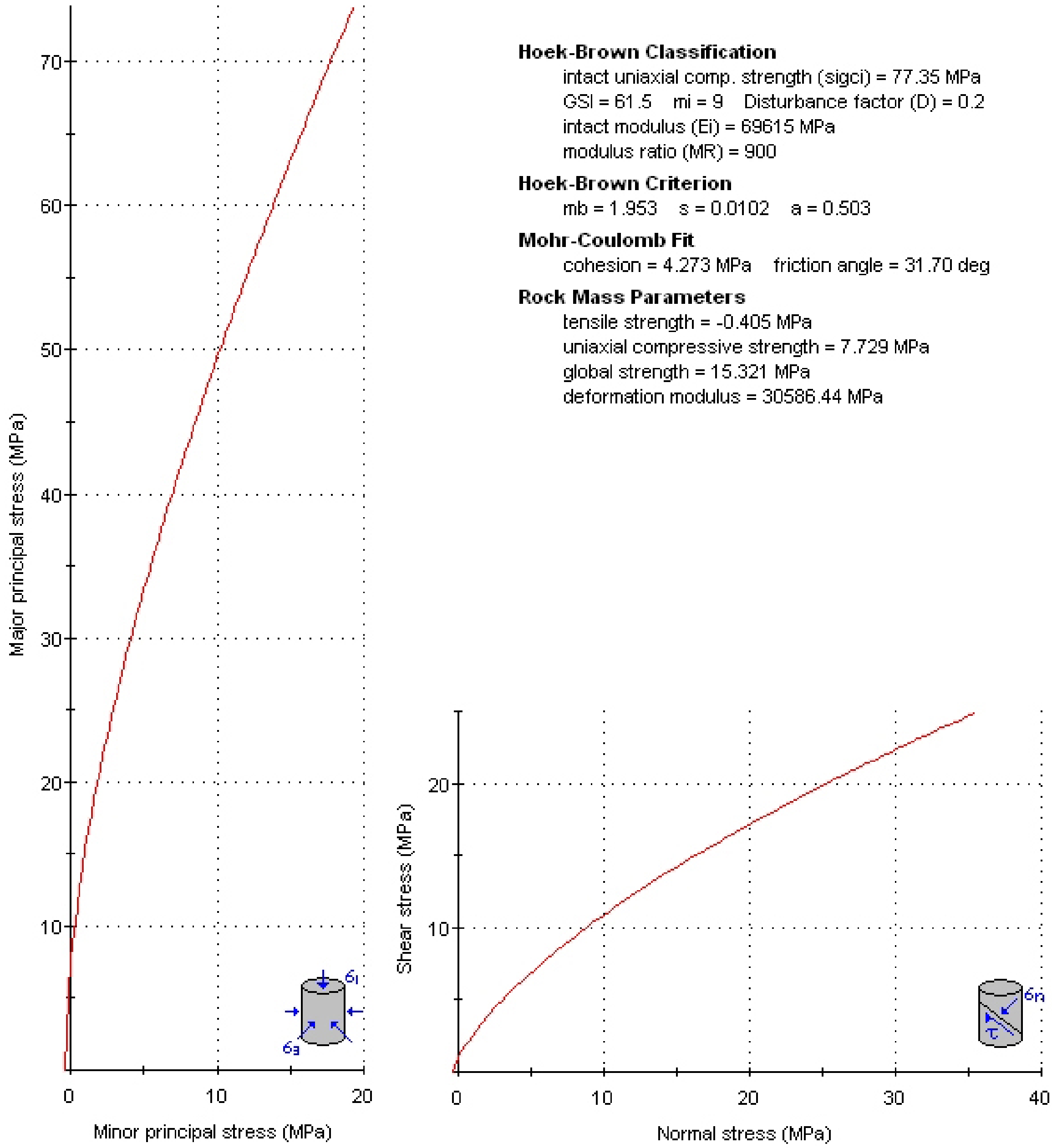


Fig (B - 11) Analysis of rock strength for surface section no.3 - unit no.3 (Sinjar Formation), using RocLab programme

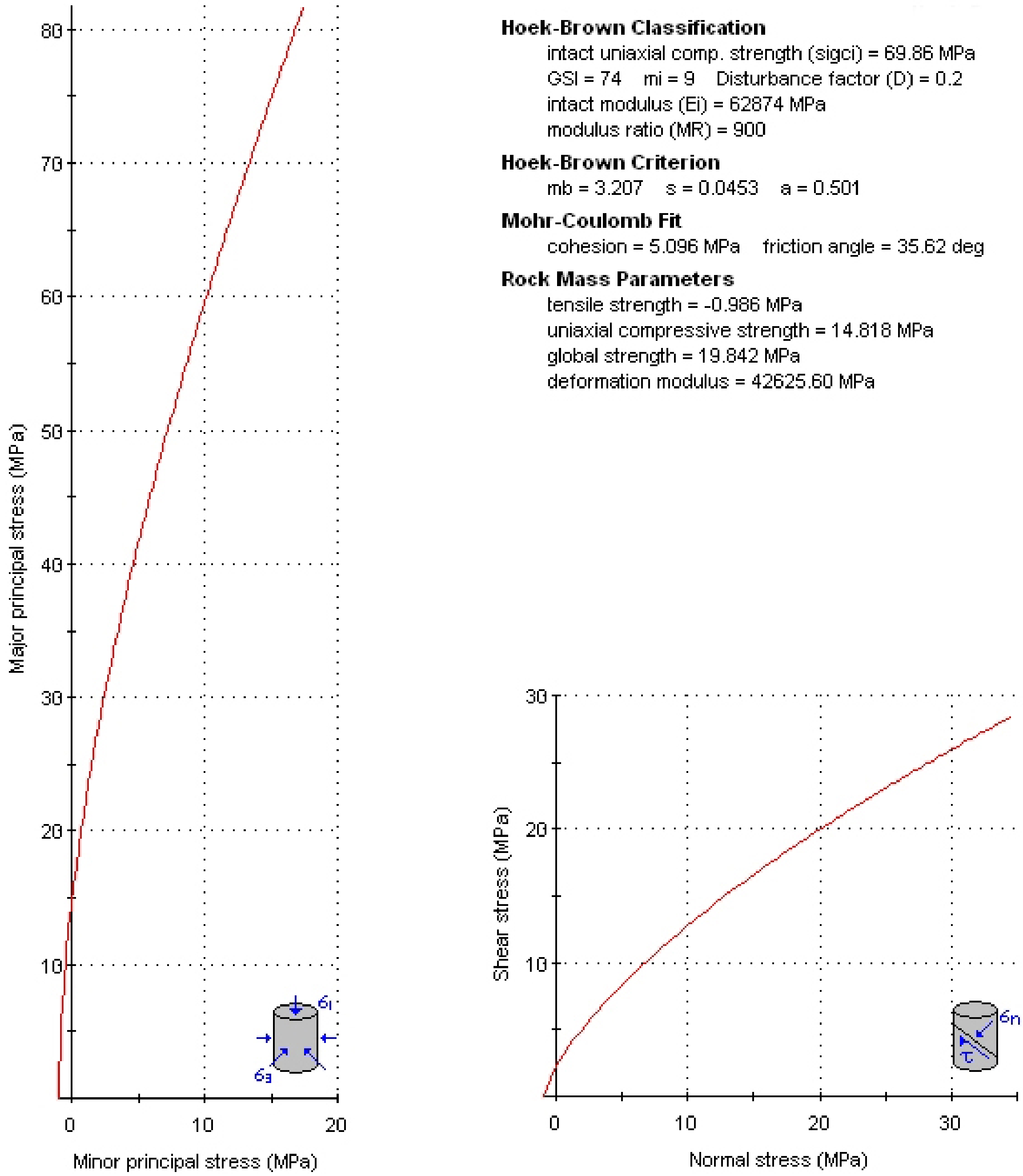


Fig (B -12) Analysis of rock strength for surface section no.3 - unit no.4 (Sinjar Formation), using RocLab programme

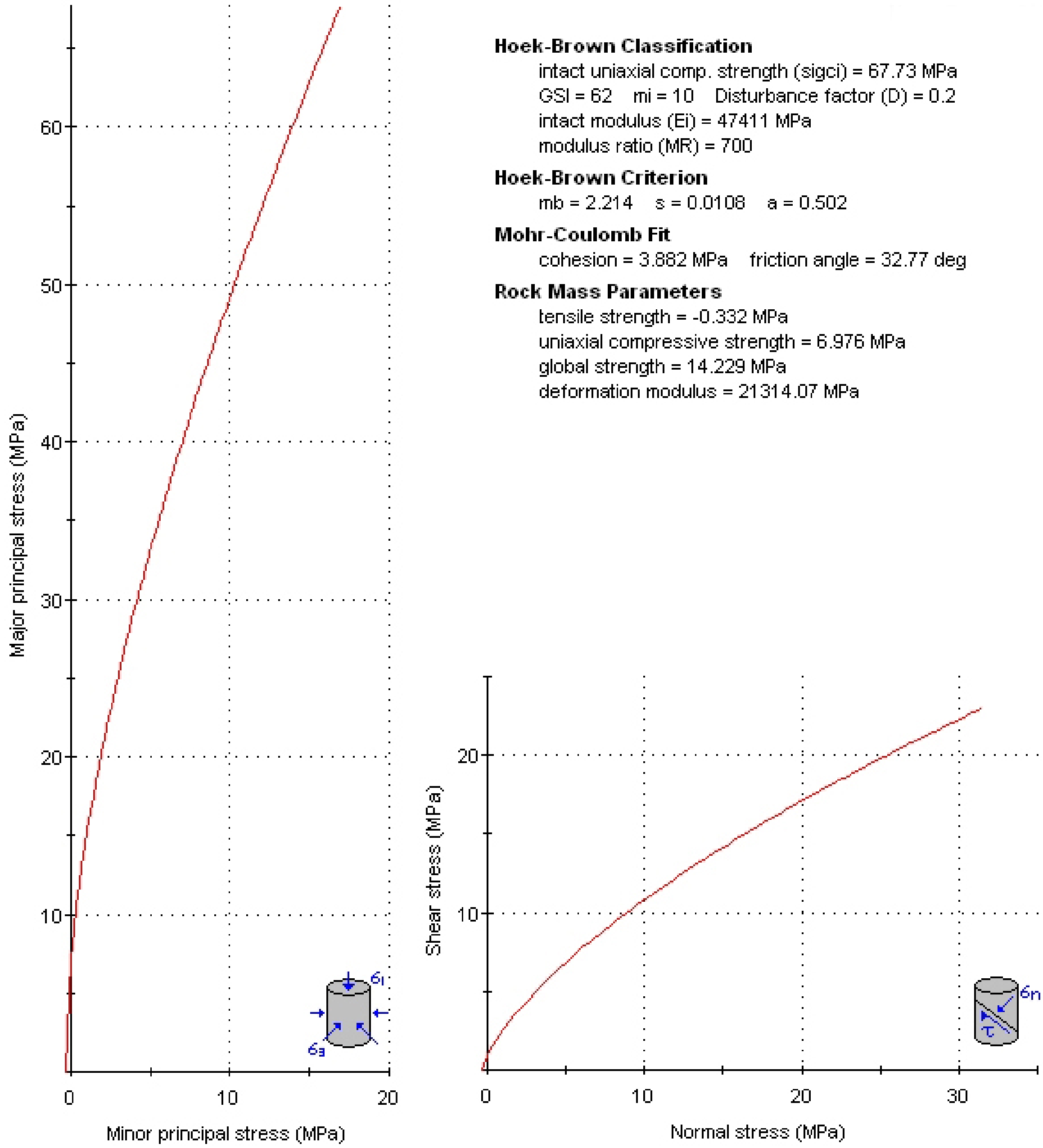


Fig (B - 13) Analysis of rock strength for surface section no.3 - unit no.5 (Unconformity between Sinjar and Kolosh Formations),using RocLab programme

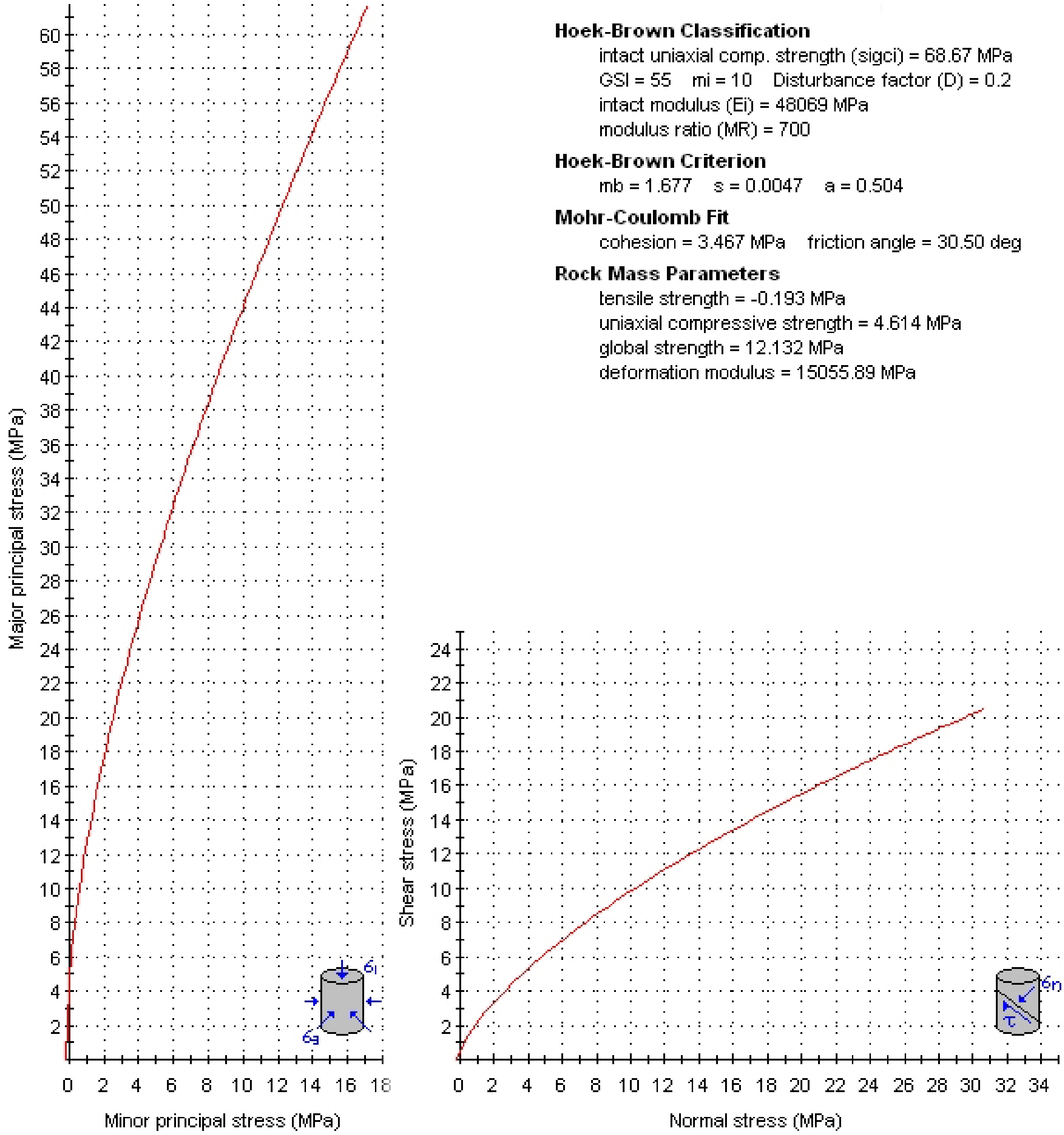


Fig (B - 14) Analysis of rock strength for surface section no.3 - unit no.6 (Unconformity between Sinjar and Kolosh Formations),using RocLab programme

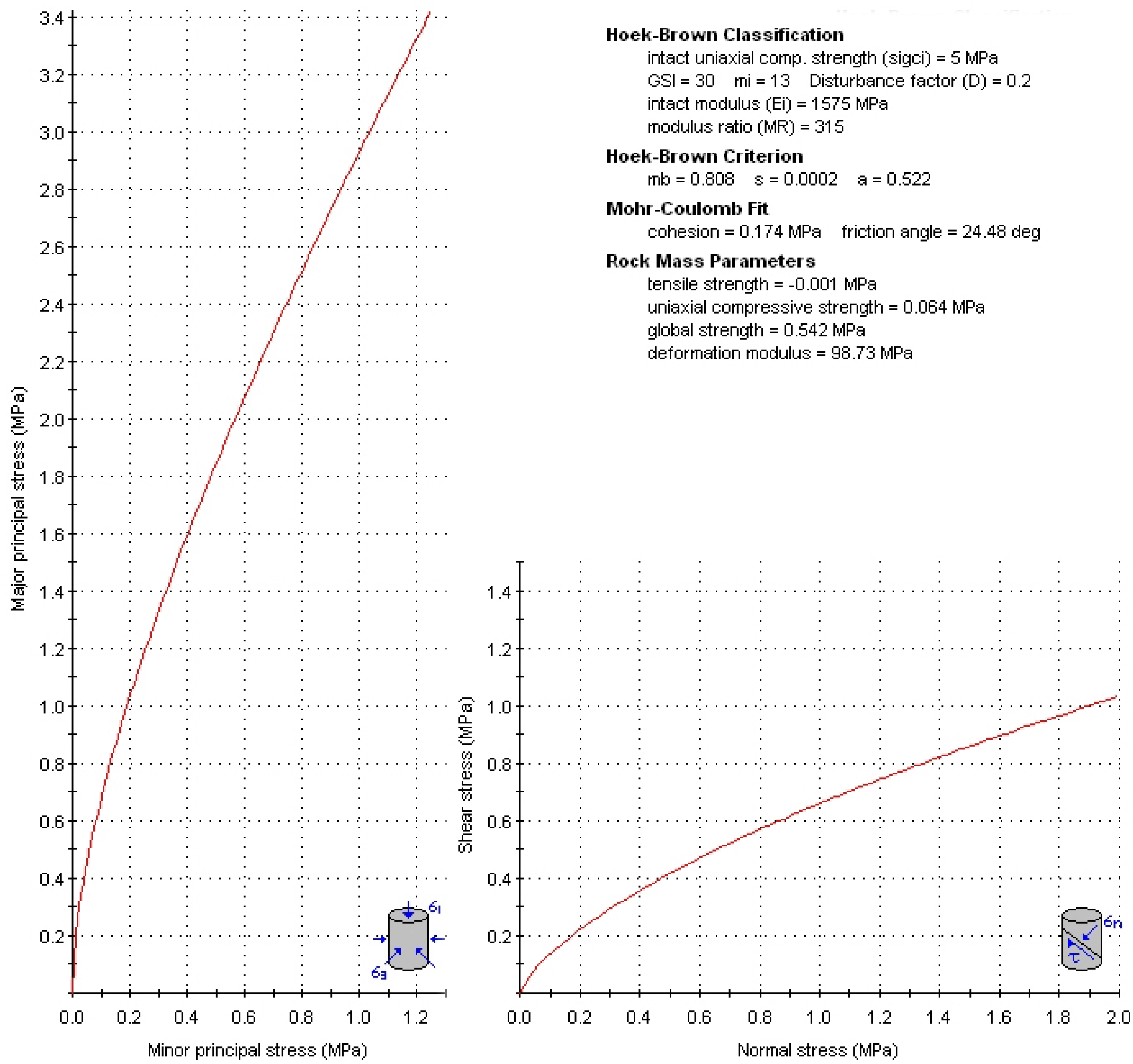


Fig (B - 15) Analysis of rock strength for borehole no.1 - unit no.1 (Kolosh Formation), using RocLab programme

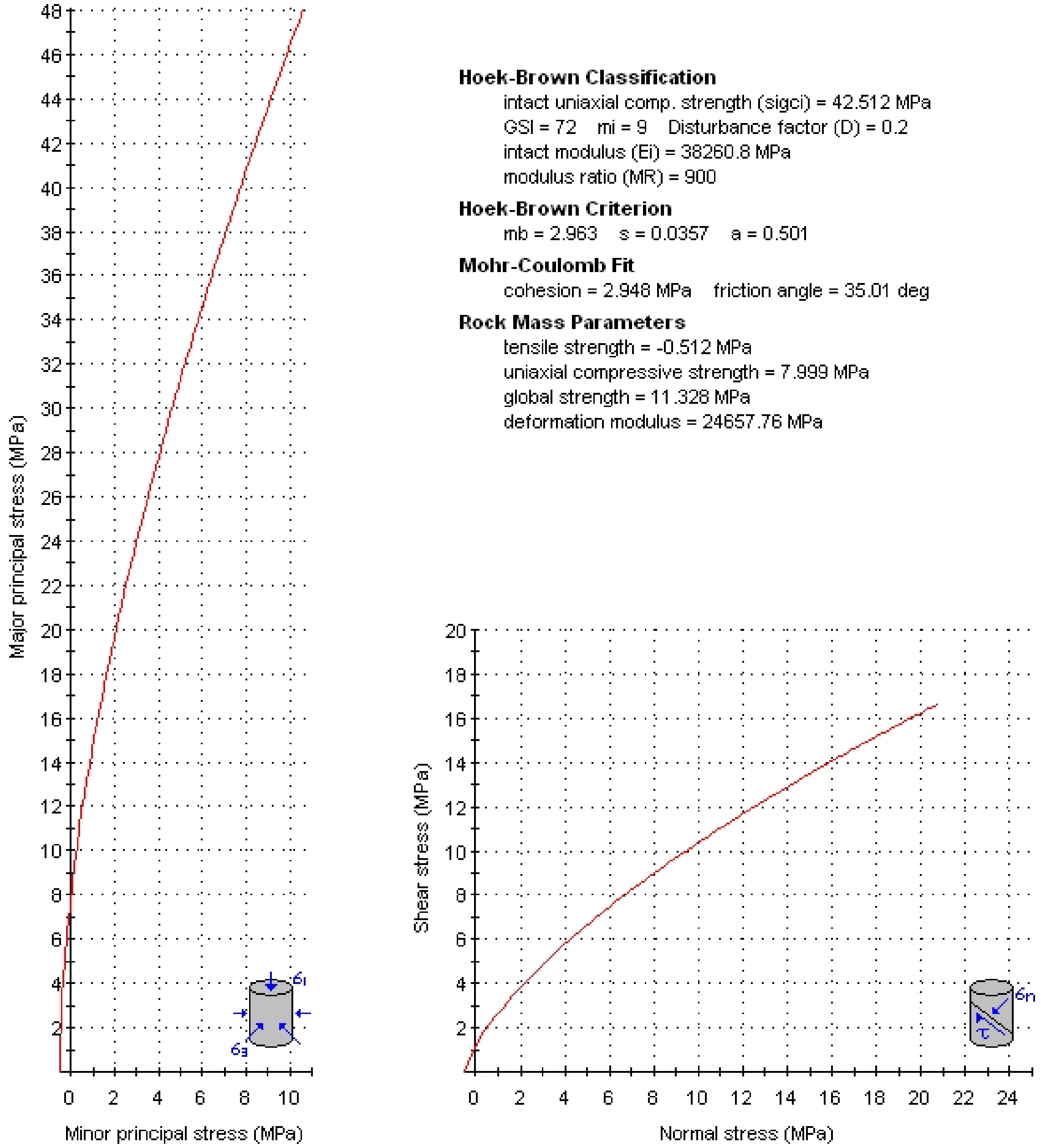
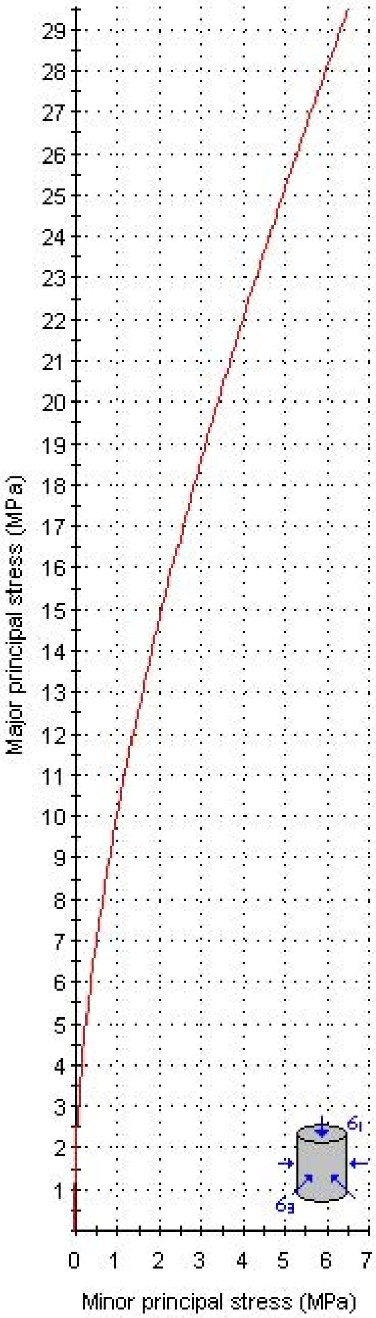


Fig (B - 16)Analysis of rock strength for borehole no.1 - unit no.2 (Kolosh Formation - carbonate rock), using RocLab programme



Hoek-Brown Classification

intact uniaxial comp. strength (sigci) = 26.11 MPa
GSI = 57   mi = 17   Disturbance factor (D) = 0.2
intact modulus (Ei) = 7180.25 MPa
modulus ratio (MR) = 275

Hoek-Brown Criterion

mb = 3.086   s = 0.0060   a = 0.504

Mohr-Coulomb Fit

cohesion = 1.591 MPa   friction angle = 35.73 deg

Rock Mass Parameters

tensile strength = -0.051 MPa
uniaxial compressive strength = 1.983 MPa
global strength = 6.210 MPa
deformation modulus = 2514.72 MPa

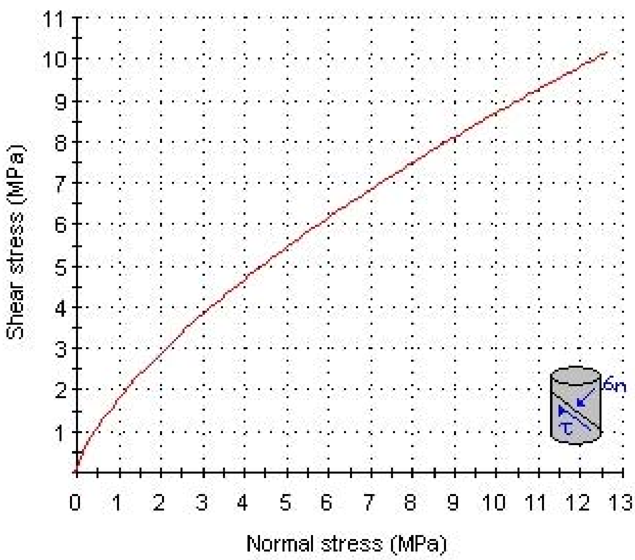


Fig (B - 17) Analysis of rock strength for borehole no.1 - unit no.3 (Kolosh Formation), using RocLab programme

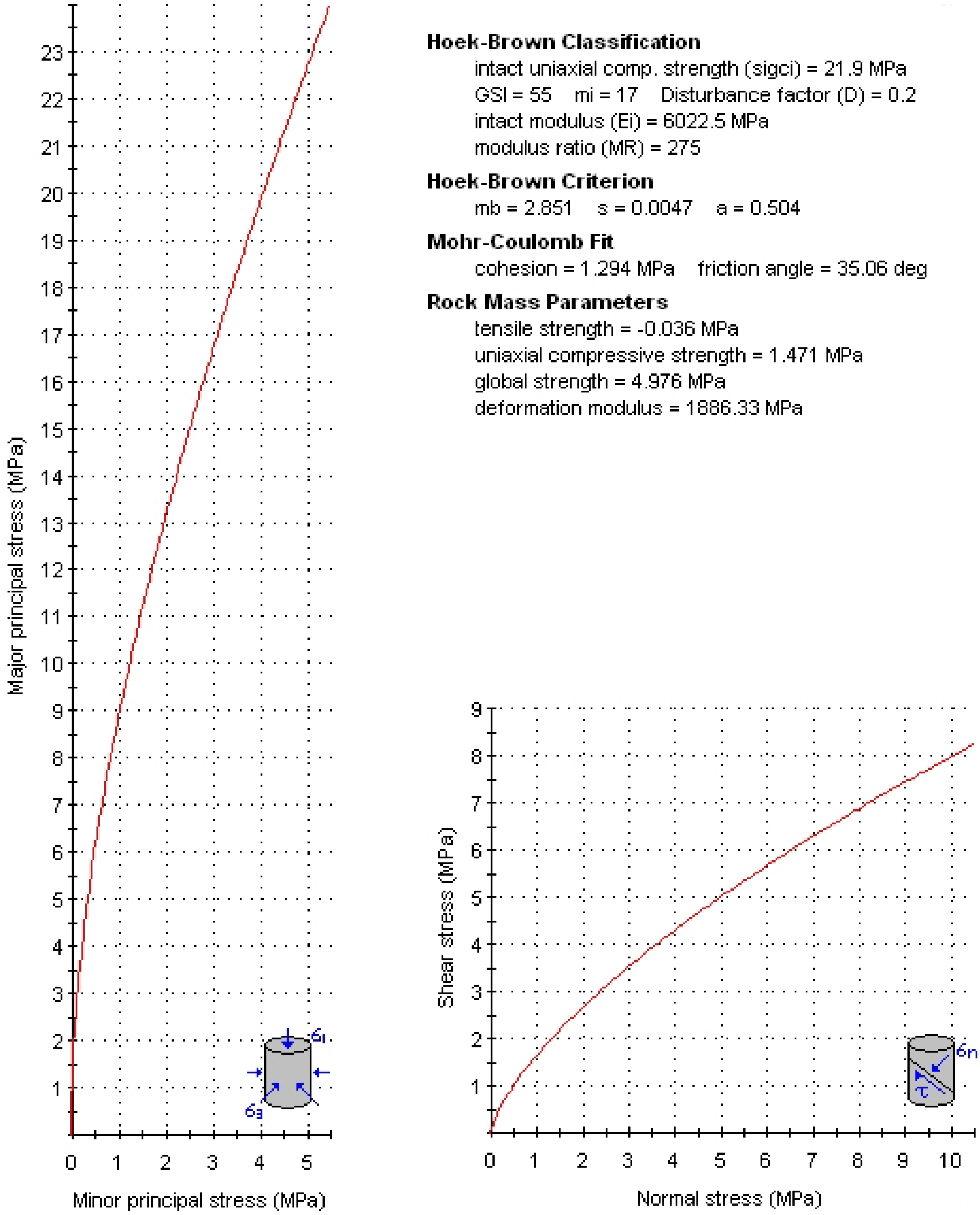


Fig (B - 18) Analysis of rock strength for borehole no.1 - unit no.4 (Kolosh Formation), using RocLab programme

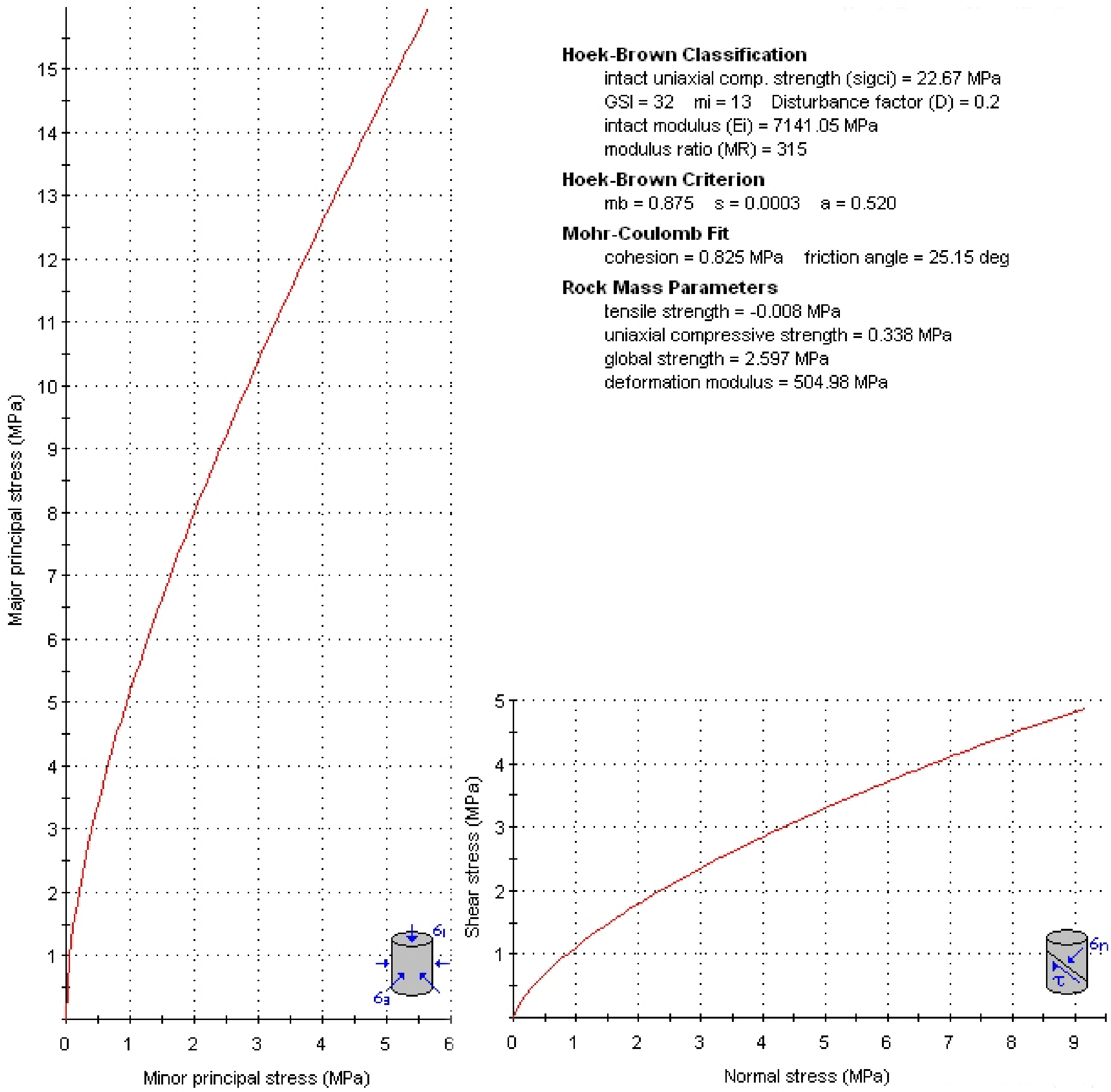


Fig (B - 19) Analysis of rock strength for borehole no.2 - unit no.1 (Kolosh Formation), using RocLab programme

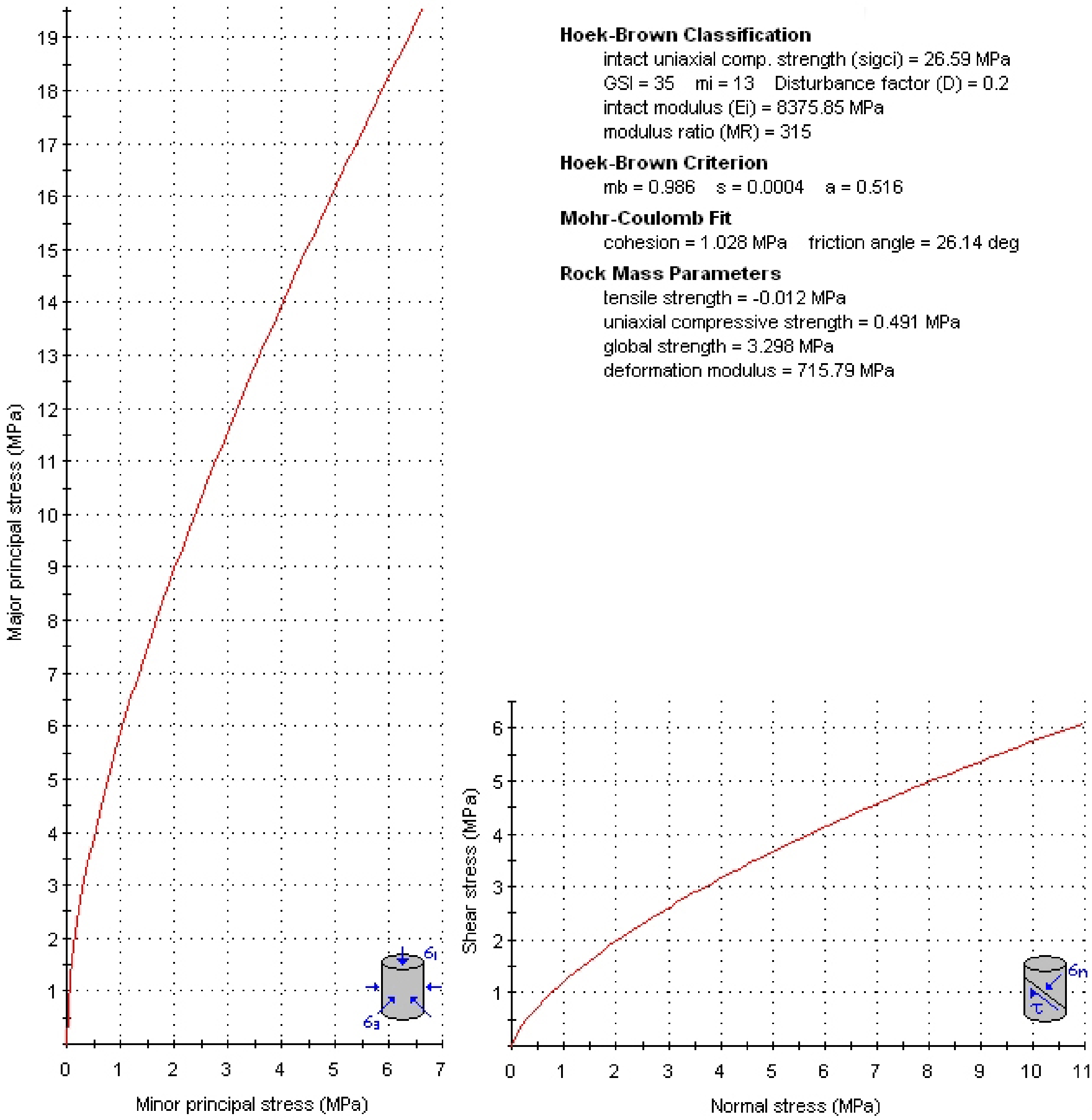


Fig (B - 20) Analysis of rock strength for borehole no.2 - unit no.2 (Kolosh Formation), using RocLab programme

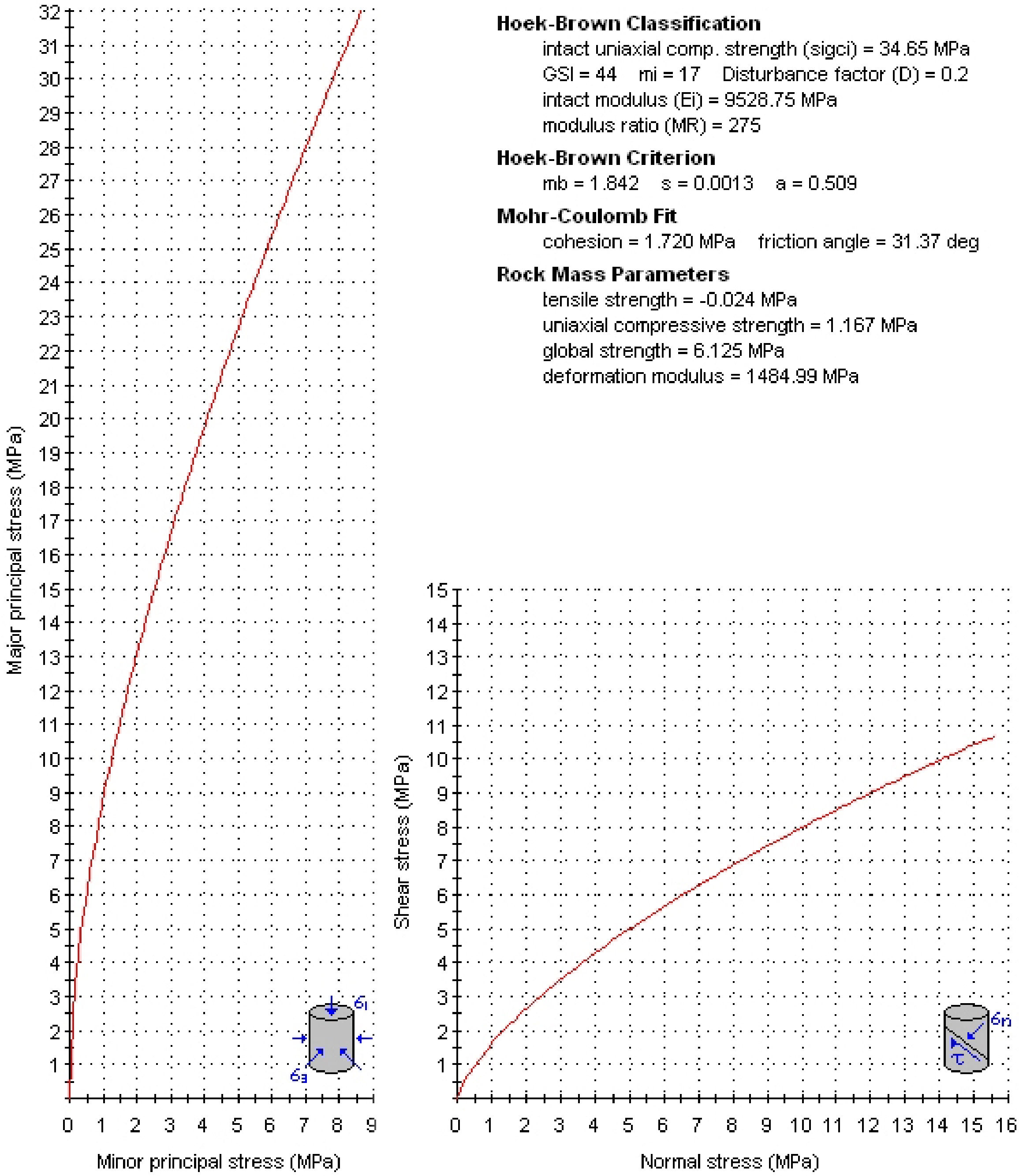


Fig (B - 21) Analysis of rock strength for borehole no.2 - unit no.3 (Kolosh Formation), using RocLab programme

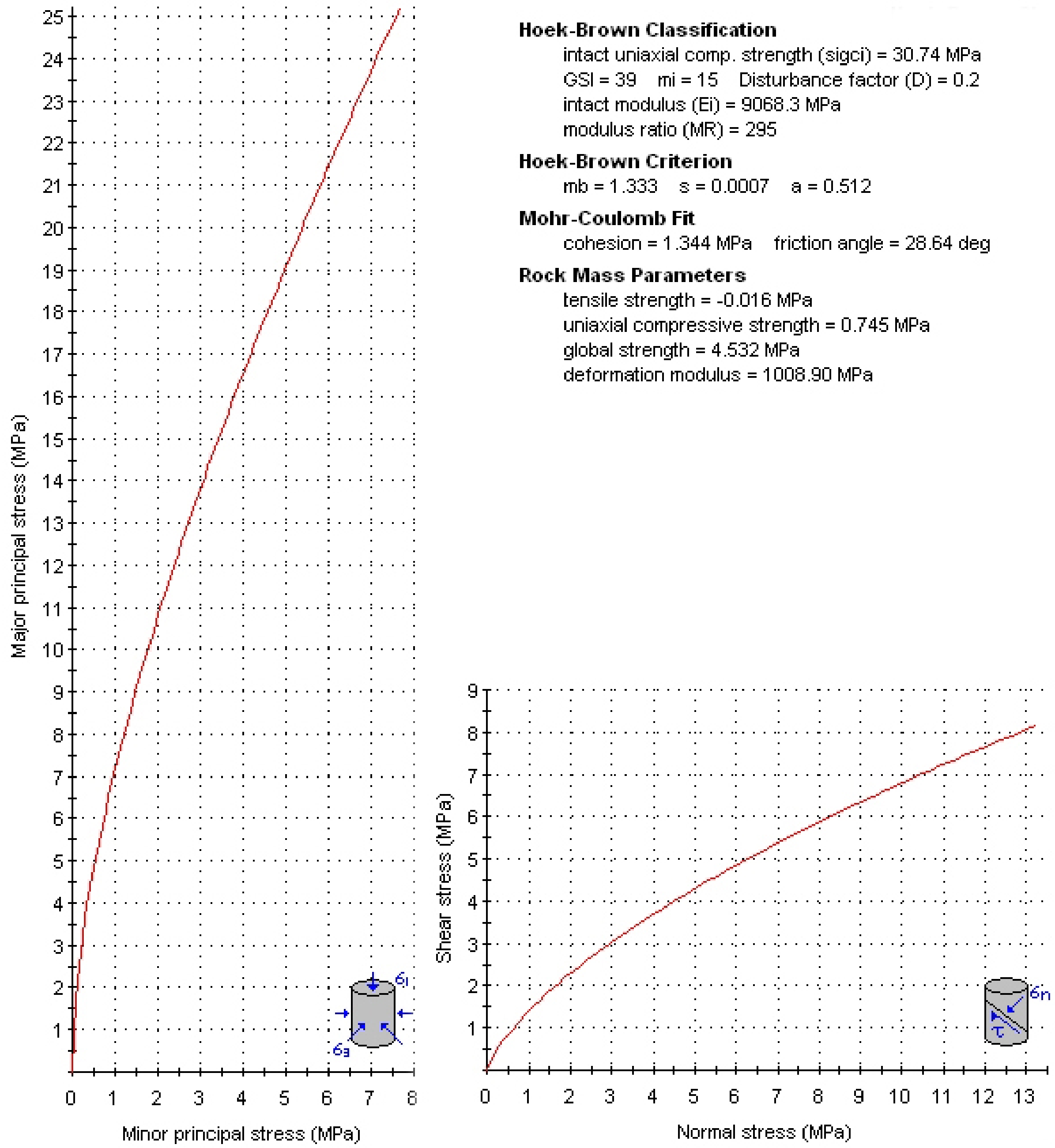


Fig (B - 22) Analysis of rock strength for borehole no.2 - unit no.4
(Kolosh Formation), using RocLab programme

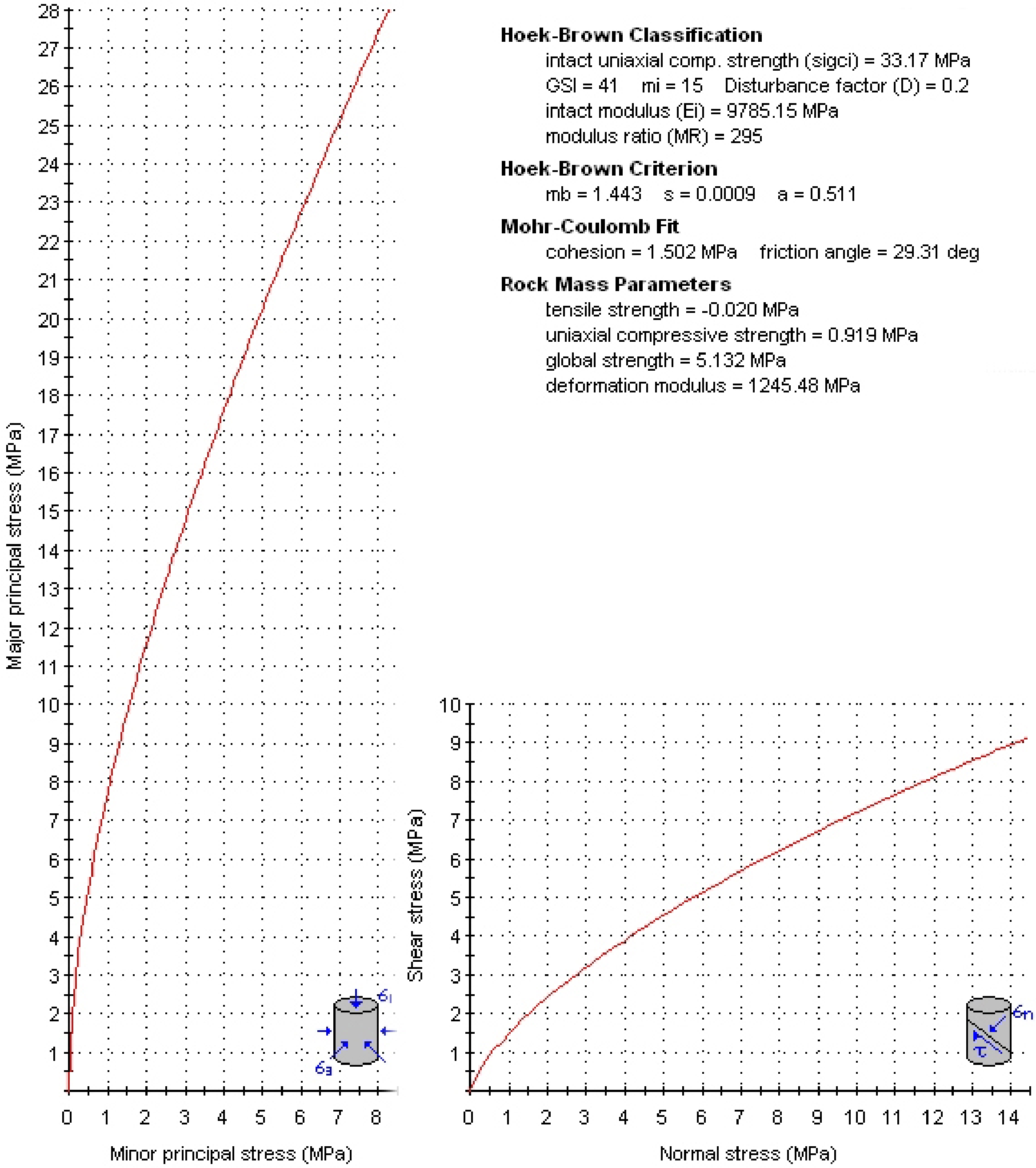


Fig (B - 23) Analysis of rock strength for borehole no.2 - unit no.5 (Kolosh Formation), using RocLab programme

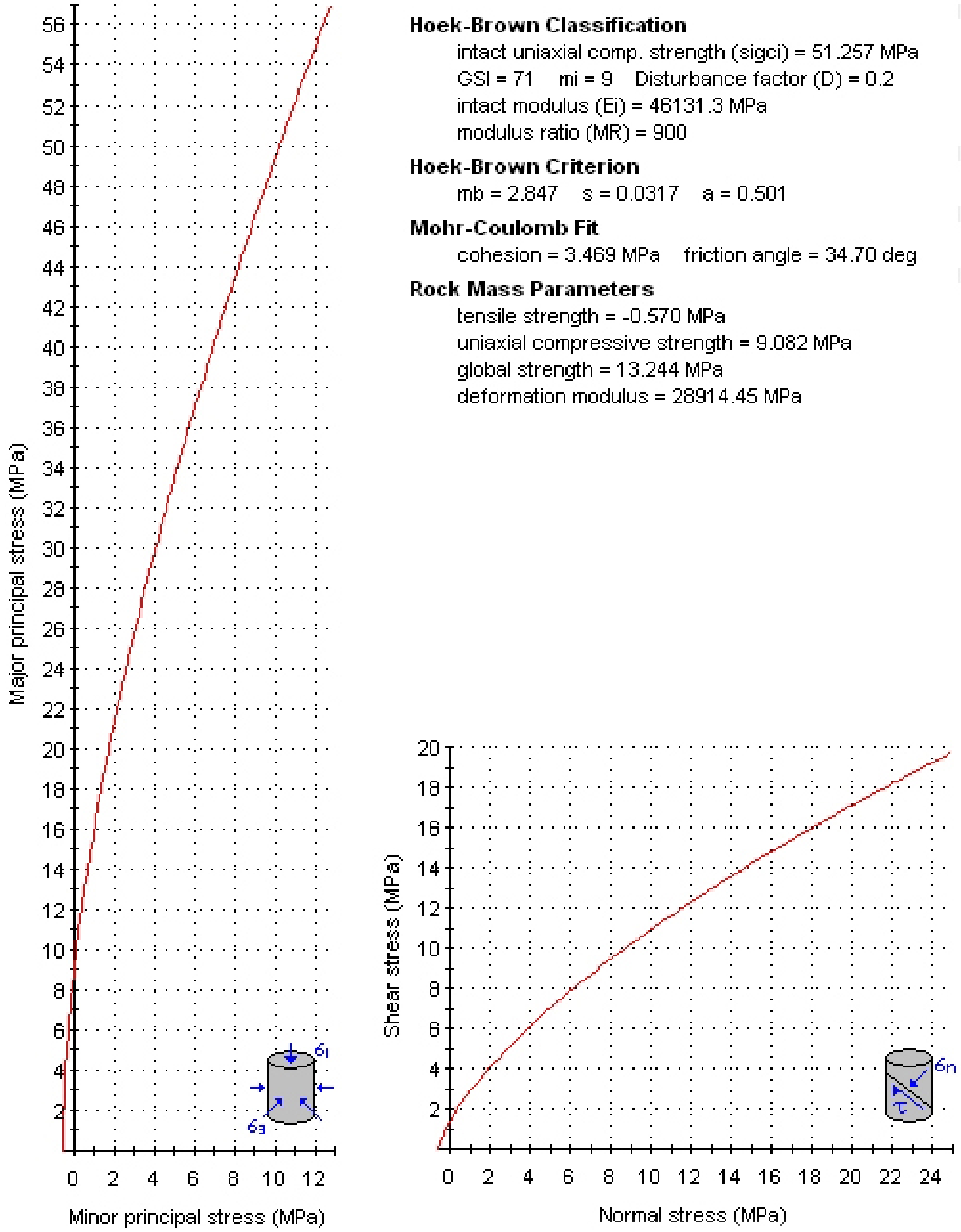


Fig (B - 24) Analysis of rock strength for borehole no.2 - unit no.6 (Kolosh Formation-carbonate rock), using RocLab programme

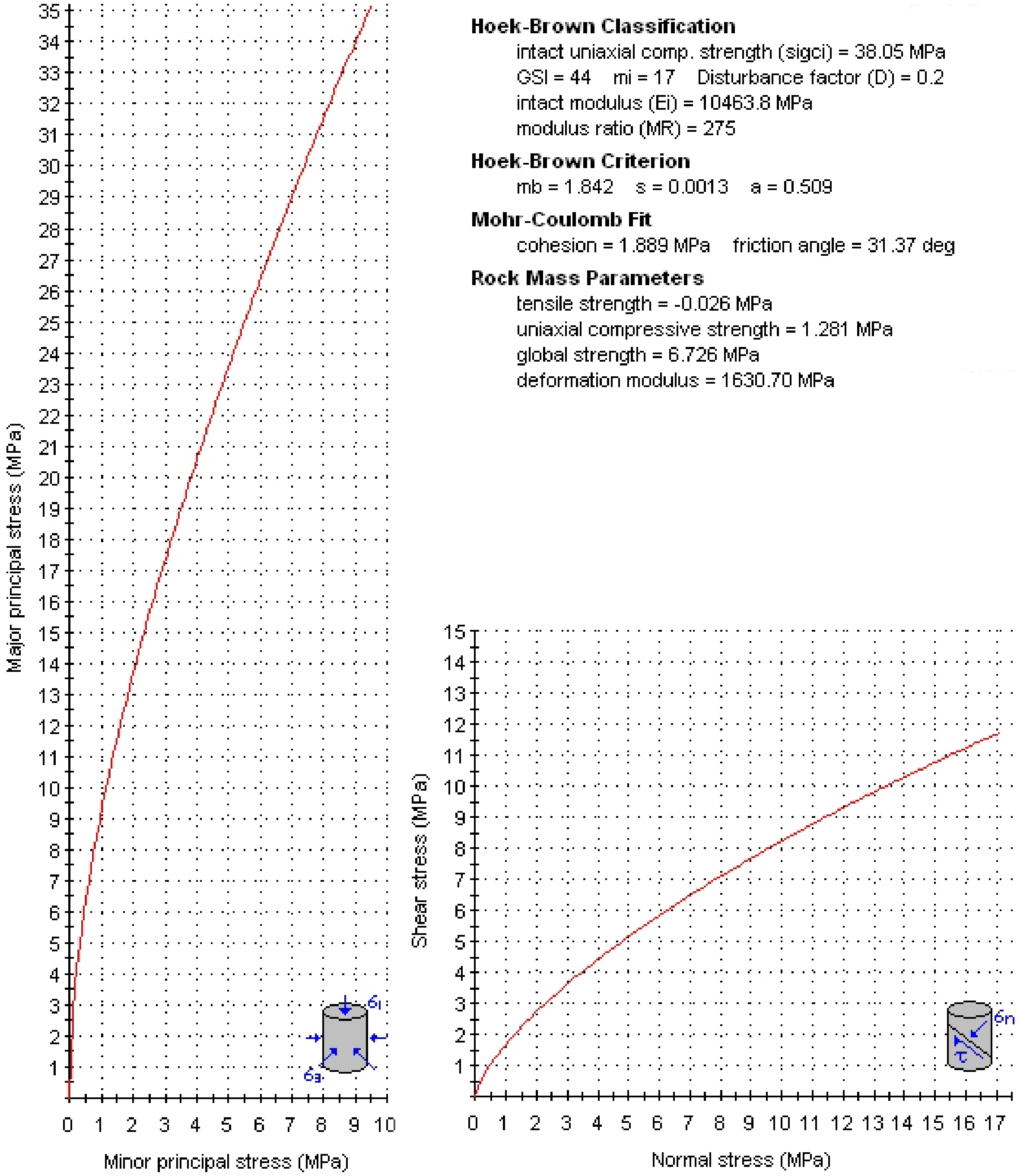


Fig (B - 25) Analysis of rock strength for borehole no.2 - unit no.7 (Kolosh Formation), using RocLab programme

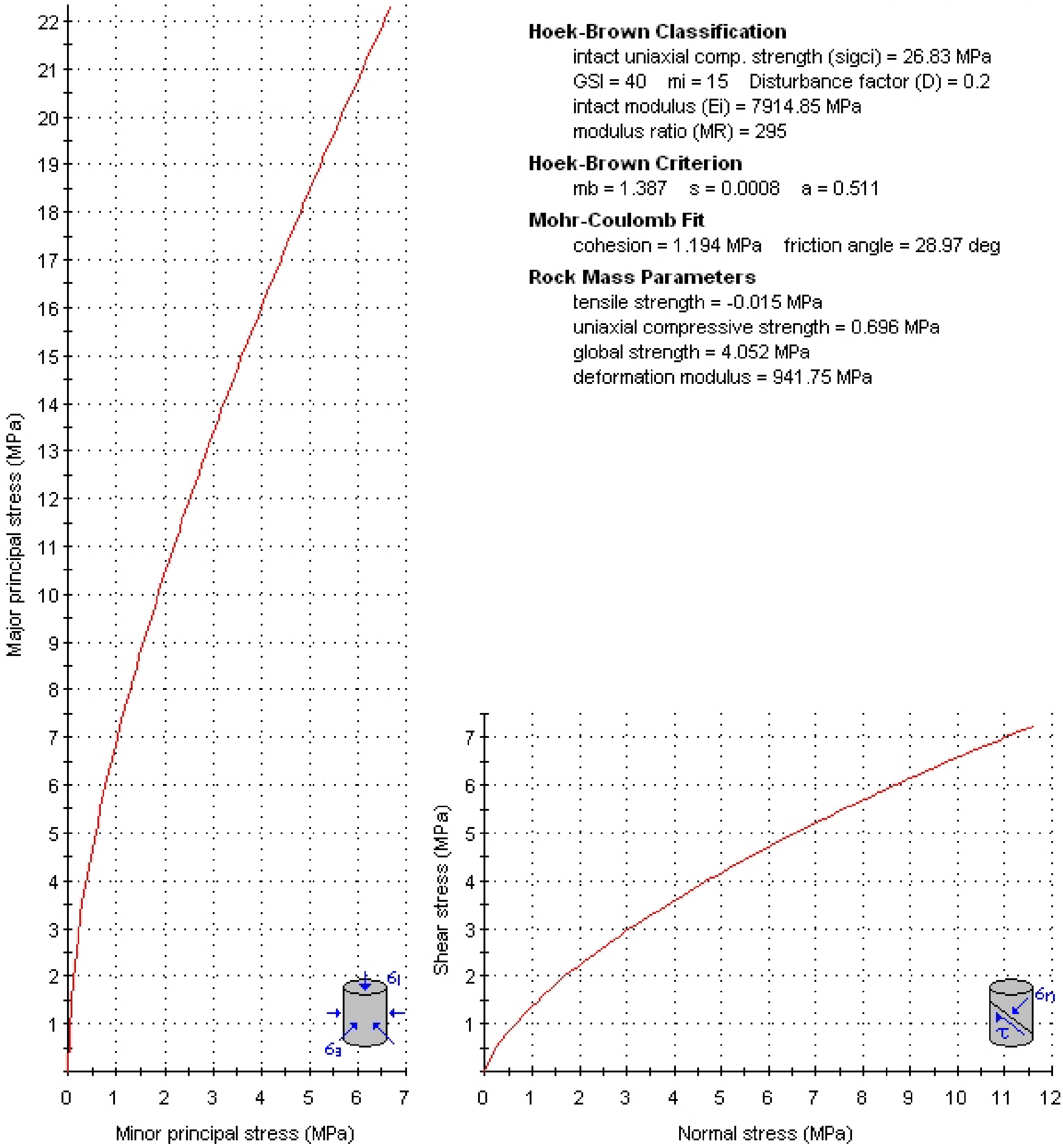


Fig (B - 26) Analysis of rock strength for borehole no.3 - unit no.1
(Kolosh Formation), using RocLab programme

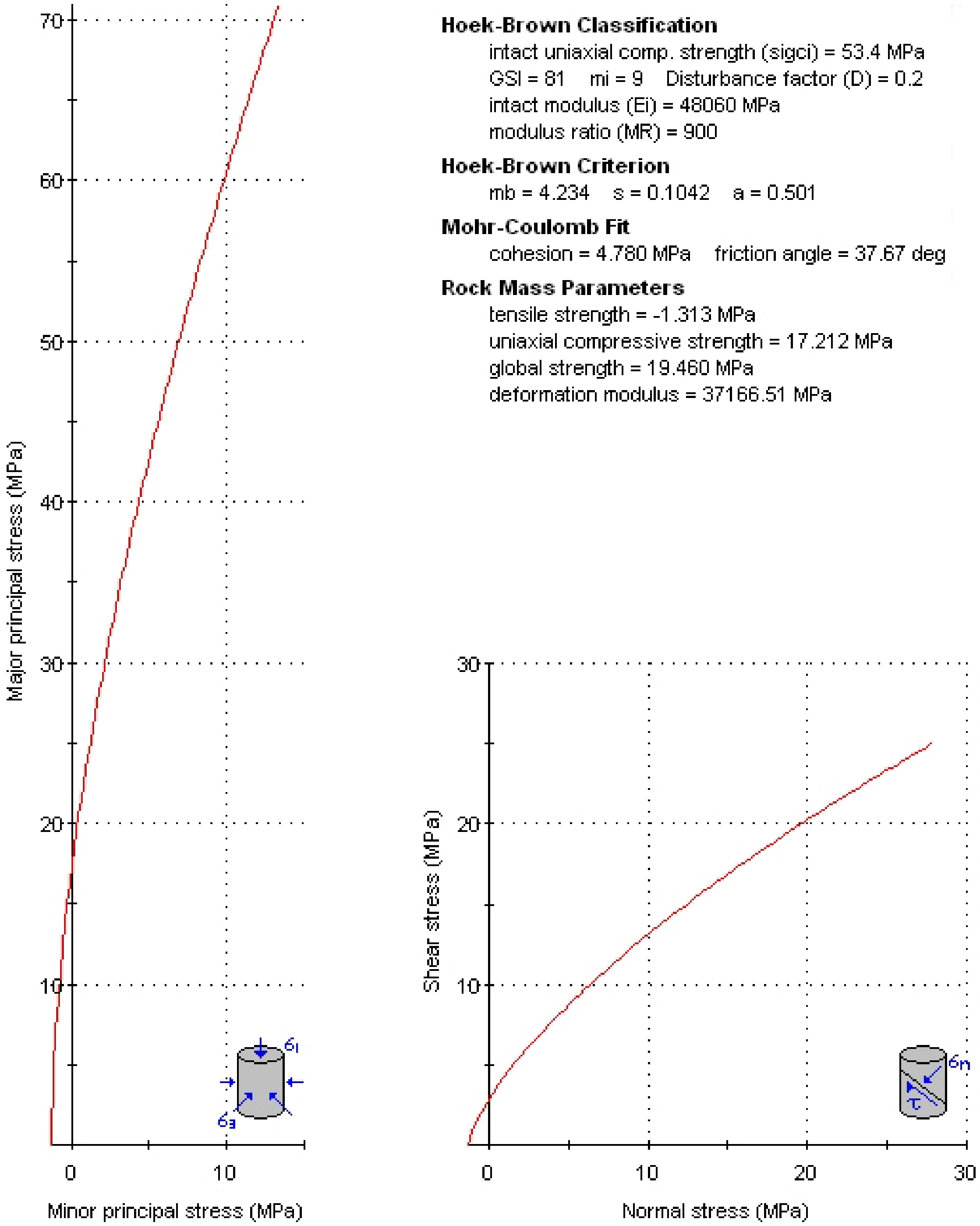


Fig (B - 27) Analysis of rock strength for borehole no.3 - unit no.2 (Kolosh Formation - carbonate rock), using RocLab programme

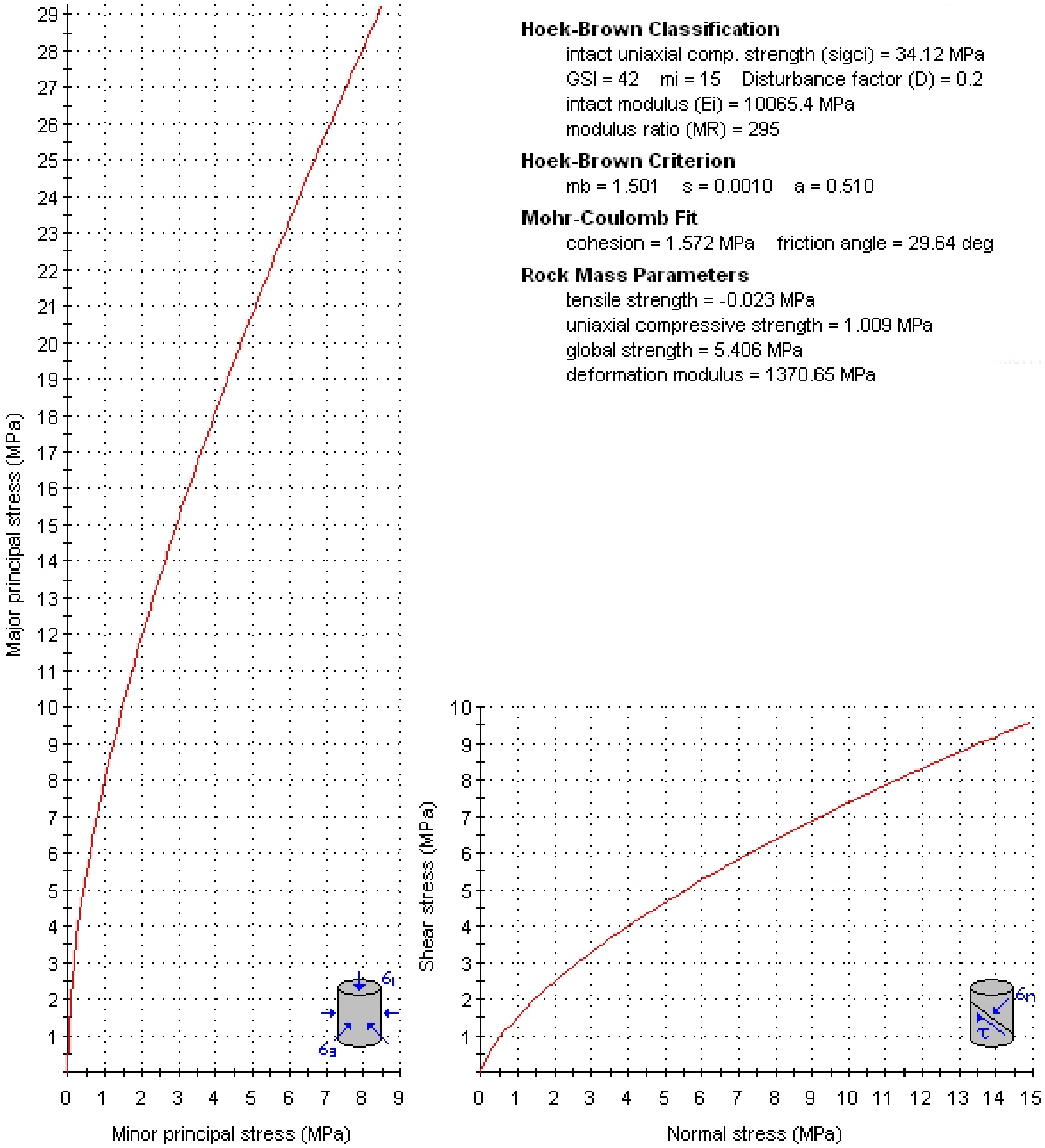
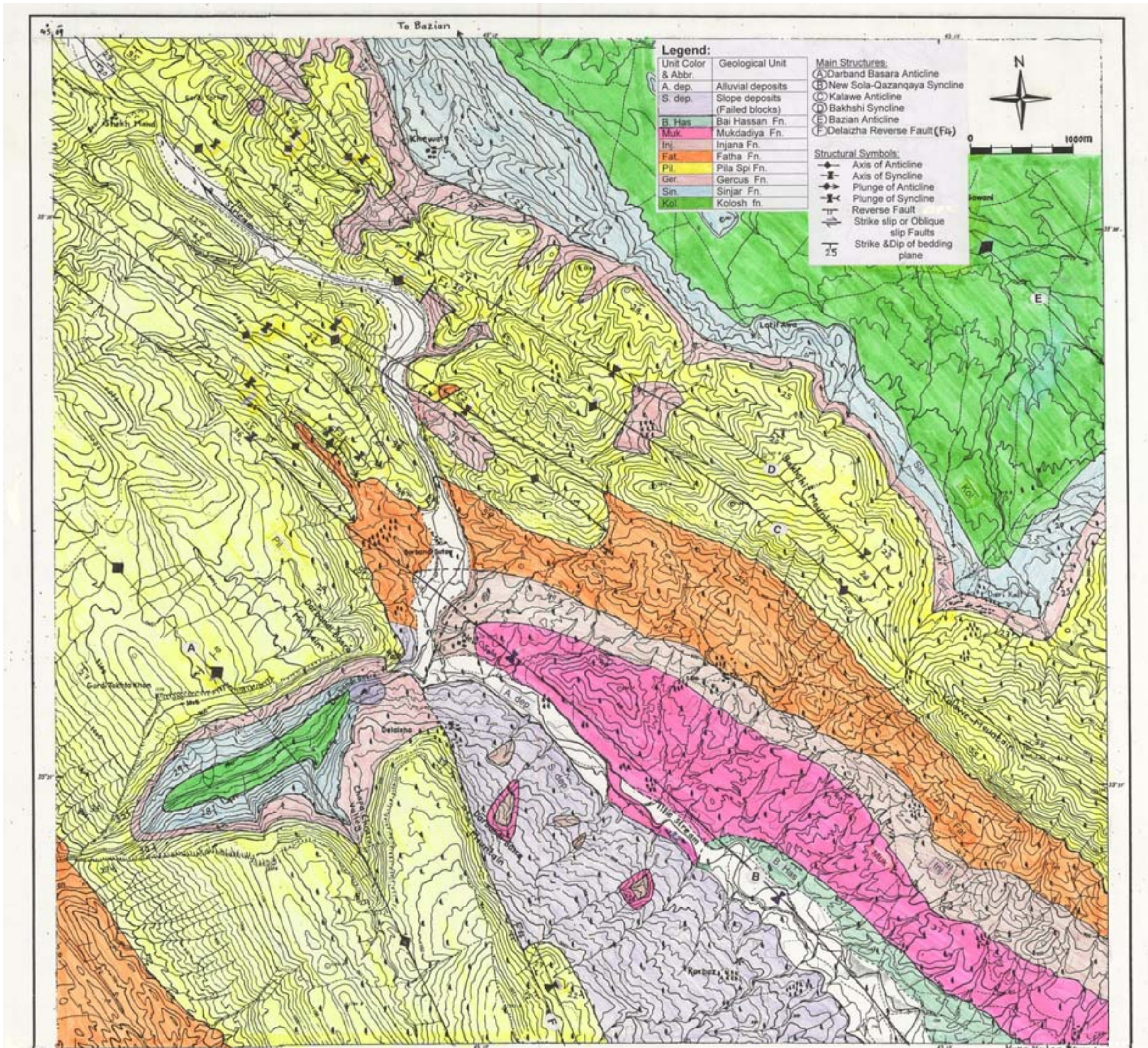


Fig (B - 28) Analysis of rock strength for borehole no.3 - unit no.3 (Kolosh Formation), using RocLab programme



Enclosure: Geological map of the study area (Darband Basara & surrounding areas)

المستخلص

تم إجراء هندسة الكتل الصخريه في موقع سد باسره المقترح، قرب قرية ديليزه- محافظة السليمانية – إقليم كردستان – شمال شرق العراق، حيث تنكشف تكوينات كولوش، سنجاروجركس. تقع أجزاء كثيرة من خزان السد ضمن طية مقعرة (طية سوله الجديدة – قازان قاية المقعره).

تم اعداد خارطة جيولوجية لمنطقة الدراسه و لأول مرة بمقياس 1 : 20000.

اشتملت هذه الدراسه علي ثلاثة أجزاء : عمل حقلى،فحوصات المختبرية وعمل مكتبى. تضمن العمل الحقلى جمع المعلومات من ثلاثة مقاطع سطحيه وثلاثة آبار عموديه (boreholes) عند مضيق باسره، إذ قسمت الكتل الصخرية إلى ثلاثين (30) وحدة صخريه (ستة عشر وحدة فى المقاطع السطحيه وأربعة عشر وحدة فى الآبارالعموديه).

أظهرت الفحوصات المختبريه إن المقاومه الأنضغاطيه غيرالمحصورة (النماذج ذات قطر 50 ملم) للصخور الجيرية تتراوح بين 40.14 – 92.26 ميكاباسكال، أى ذات مقاومة متوسطة – عالية، حيث تتواجد فى الجانب الأيمن لمضيق باسره. المقاومة الأنضغاطية غيرالمحصورة لصخور كولوش الفتاتى تتراوح بين 5 – 38.052 ميكاباسكال، أى ذات مقاومة ضعيفة – متوسطة، حيث تتواجد فى الآبار العمودية فى الجانب الأيسر لمضيق باسره.

تقترح هذه الدراسة مخططا" جديدا" لمؤشرالمقاومة الجيولوجي (GSI) ، مبنى على تحليل كمى لتركيب الكتلة الصخرية (من خلال الحساب الحجمى للفواصل "Jv" أو حجم الكتل "Vb") والظروف السطحية للأنقطاعات. تم تحديد قيم GSI للصخور الجيرية ذات المقاومة الأنضغاطية العالية فى أساس موقع السد بوساطة هذا المخطط الجديد لـ GSI ، أوضحت مقارنة قيم GSI الحاصلة من هذا المخطط مع قيم $RMR_{BD(1976)}$ الدقة العالية لهذا المخطط.

تم إيجاد قيم GSI للصخور الفتاتية لتكوينى كولوش وجركس من خلال مخططات (GSI) للفلش (Flysch) والمولاس (Molasse) على التوالى.

كما تم إيجاد الخواص الميكانيكية لوحدات الكتل الصخرية من خلال دليل الأنهيارلهوك – براون (Hoek – Brown failure criterion) مستخدما برنامج (RocLab) .

تم تقييم وحدات الكتل الصخرية حسب نظام (Dam Mass Rating) لنواحي مختلفة. هذا التقييم يبين إن الصخور الجيرية في الجانب الأيمن وتلك التي في الآبار العمودية في الجانب الأ يسر من مضيق باسرة : (1) ليست لها المشاكل التي تنتج من E_c/E_m (معامل التشويه للسد \ معامل التشويه لصخور الأساس) ، (2) مناسبة للحفر، (3) لا تحتاج التحشية او تحتاجها موضعيا و(4) ليست لها مشاكل التشويه (باستثناء وحدة صخرية واحدة) ، لكن وحدات الكتل الصخرية الفتاتية لتكوين كولوش في الآبار العمودية عند الجانب الأيسر من مضيق باسره: (1) لها بعض المشاكل ذات الأهمية القليلة –المهمة والتي لايمكن اهمالها خصوصا إذا كان نوع السد تثاقلي (CVC – Gravity dam) ، (2) إنها غير مناسبة للحفر عند إنشاء سد تثاقلي ، (3) تحتاج التحشية بشكل نظامي و(4) لها مشاكل تشويه خطيرة في حالة إنشاء سد تثاقلي.

أظهرت نتائج انظمة تصانيف الكتل الصخرية ودليل الأنهيار لهوك – براون إن وحدات الكتل الصخرية الجيرية (limestone) لتكوين سنجار فى الجانب الأيمن ولتكوين كولوش فى الآبار العمودية فى الجانب الأيسر لمضيق باسره تتميز بقيم عالية لـ GSI & DMR, RMR وخواص ميكانيكية جيدة، لكن الكتل الصخرية لتكوين كولوش الفتاتي عند الجانب الأيسر وتكوين جركس عند الجانب الأيمن لمضيق باسره تتميز بقيم واطئة – متوسطة لـ GSI & DMR, RMR و الخواص الميكانيكية تكون نوعا ما رديئة.

اقترحت هذه الدراسة موديلا" جديدا" سمي هنا موديل الكتل الصخرية – مقطع الوادي (Rock mass – valley section model) في هذه الأطروحه و استخدم هنا للمقاطع العرضية الثلاثة (a-b, c-d & e-f) التي تم رسمها لأول مرة ، حيث اسقطت وحدات الكتل الصخرية داخل كل مقطع.

أظهرت المقارنة الأولية ما بين تلك المقاطع العرضية الثلاثة لأختيار الأمثل ، إن مقطع e-f العرضي في نواحي كثيرة هو الأفضل من مقطع c-d العرضى ومقطع c-d العرضي هو أفضل من مقطع a-b العرضي ، لكن أظهرت المقارنة النهائية بين مقطعي c-d و e-f إن مقطع c-d العرضي يكون اكثر ملائمة من مقطع e-f لأنشاء السد وذلك بسبب وجود نطاق قصي ضعيف (weak sheared zone) في الجانب الأيمن من مقطع e-f العرضي، إن هذا النطاق القصي الضعيف له تأثير سلبي كبير على إستقرارية السد بعد إنشاءه و إمتلاء الخزان بالماء ، ويسبب عدم الأستقرارية ، حيث تزداد عدم الأستقرارية هذه مع مرور الزمن.

هندسة الكتل الصخريه في موقع سد باسره المقترح ، السليمانية ،إقليم كردستان شمال شرق العراق

رسالة
مقدمة الى مجلس كلية العلوم - جامعة السليمانية / السليمانية
كجزء من متطلبات درجة دكتوراه فلسفة في علم الأرض

من قبل
غفورأمين حمه سور أمين
ماجستير في الجيولوجيا الهندسية – جامعة صلاح الدين / ١٩٩١

بأشراف
أستاذ د. سعد نعمان السعدي

ذي الحجه - ١٤٣٠هـ   كانون الأول - ٢٠٠٩م

پوختە

نرخاندنی ئەندازەی تاوێرەبەردی یەکانی شوێنی بەنداوی باسەڕەی پێشنیارکراو لەنزیک دێ ی دێلێژە- پارێزگای سلێمانی-هەرێمی کوردستان لە باکوری ڕۆژهەلاتی عێراق جێ بەجێ کرا، لەو شوێنەدا پێکهاتووەکانی کۆلۆش، سنجار و جەرکەس دەردەکەوێت. بەشێکی زۆری عەمباری بەنداوەکە ئەکەوێتە چینە چەماوەیەکی چاڵەوە کە ناسراوە بە چینە چەماوەی سۆڵەی نوێ-قازان قایە.

نەخشەی جیۆلۆجی ناوچەی لێکۆڵینەوە وێنە کێشرا بە پێوەری ١:٢٠٠٠٠ بۆ یەکەم جار.

ئەم لێکۆڵینەوەیە سێ بەش ئەگرێتە خۆی : ئیشی کێلگەیی، تاقیگەیی وە نوسینگەیی. ئیشی کێلگەیی بریتی یە لەکۆکردنەوەی زانیاری لە سێ پانە بڕگەی سەر ڕووی زەوی (Surface sections) وە سێ بیری ستوونی (boreholes) لەدەربەندی باسەڕەدا. بۆ ئەم مەبەستە تاوێرە بەردی یەکان (Rock masses) دابەشکرا بۆ سی (٣٠) یەکەی تاوێرە بەردی (شانزە یەکە لە پانەبڕگەکانی سەر ڕووی زەوی وە چواردە یەکە لەبیرە ستوونی یەکاندا).

پشکنینی تاقیگەیی دەریخست کە بەرگری پەستانی ئازادی (یەك تەوەری) (Unconfined Compressive Strength) (بۆنموونەی بەردی ٥٠ملم تیرەیی) بەردی کلسی (limestone) لەنێوان ٤٠,١٤- ٩٢,٢٦ میگاباسکاڵدایە ، واتە بەرگری یان مام ناوەند-بەرزە ، کەئەمانە لەلای ڕاستی دەربەندی باسەڕەدایە.

بەرگری پەستانی ئازادی بەردی پێکهاتووی کۆلۆشی دەنکۆڵەیی (Clastic) لەنێوان ٥- ٣٠,٠٥٢ میگاباسکاڵدایە ، واتە بەرگری یان بێ هێز-مام ناوەندە ، کەئەمانە لەبیرە ستوونەکانی لای چەپی دەربەندی باسەڕەدایە.

ئەم لێکۆڵینەوەیە پێشنیاری چارتێکی نوێ دەکات بۆ GSI (Geological Strength Index) ، ئەم چارتە لەسەر شیکردنەوەی بڕی پێکهاتووی تاوێرە بەرد (بەهۆی ژمێریاری قەبارەی نێوانە ڕوو (Volumetric Joint Count) یان قەبارەی تاوێرە بەردەوە) و بارودۆخی سەر ڕووەکانی نابەردەوامی (Discontinuities) بنیاد نراوە ، نرخەکانی GSI ی بەردی کلسی خاوەن بەرگری پەستانی ی بەرز لە بناغەی شوێنی بەنداوەکەدا بەهۆی چارتی نوێ ی GSI دۆزرایەوە (دیاریکرا) ،بەراوردکردنی نرخی GSI ی دیاریکراو لەم چارتەوە لەگەڵ نرخی RMR_{BD} (1976) ووردی و دروستی زۆری ئەم چارتەی ڕوونکردەوە.

نرخەکانی GSI بەردی دەنکۆڵەیی هەردوو پێکهاتەی کۆلۆش و جەرکەس بەهۆی چارتی GSI ی فلێش و (Flysch) و مۆلاسەوە (Molasse) یەك لە دوای یەك دۆزرایەوە.

ڕەووشتە میکانیکی یەکانی هەموو یەکە تاوێرە بەردی یەکان بەهۆی ڕێنمایی دارمانی هووك – براونەوە (Hock-Brown failure criterion) دۆزرایەوە بە بەکارهێنانی بەرنامەی (RocLab) .

یەکەکانی تاوێرە بەردی یەکان لەزۆر ڕووەوە بە گوێرەی بنەمای (Dam Mass Rating) وە هەڵسەگێنران. ئەم هەڵسەنگاندنە دەریدەخات کەبەردی کلس لەلای ڕاست و ئەوانەی ناوبیرە ستوونی یەکان لەلای چەپی دەربەندی باسەڕە : (١) گیروکرفتیان نی یە کە بەهۆی Ec/Em (هاوکۆلکەی پشێوی بەنداو \هاوکۆلکەی پشێوی بەردی بنەوەی بناغەی بەنداو) پەیدا ئەبێت ، (٢) گونجاون بۆ هەڵکەندن تیایاندا ، (٣) پێویستی یان بە پڕکردنەوە (Grouting) نی یە یان بەشێوەی پچڕ پچڕ پێویستیان پێ یەتی وە (٤) گیرو گرفتی پشێوی یان نی یە (جگە لە یەك یەکەی جیۆتەکنیکی بەردی نەبێت) ، بەلام یەکەکانی تاوێرە بەردی یە دەنکۆڵەیی یەکانی پێکهاتووی کۆلۆشی ناو بیرە ستوونی یەکانی لای چەپی دەربەندی باسەڕە : (١) هەندێك گیروگرفتی کەم – گرنگیان هەیە کەبەهۆی Ec/Em وە پەیدا ئەبێت کە نابێت پشتگوێ بخرێت بەتایبەتی ئەگەر جۆری بەنداوەکە کێشی بێت (CVC-Gravity dam) ، (٢) نەگونجاون بۆ هەڵکەندن تیایاندا لەکاتی بنیادنانی بەنداوی کێشی ، (٣) پێویستی یان بە پڕکردنەوە هەیە بەشێوەیەکی ڕێك و پێك (Systematic) وە (٤) گیرو گرفتی پشێوی ترسناکیان هەیە لەکاتی بنیادنانی بەنداوی کێشی دا.

ئەنجامی بنەما پۆلێنی یەکانی تاوێرە بەردی یەکان و ڕێنمایی دارمانی هووك – براون دەریانخست کە یەکەکانی تاوێرەبەردی جۆری کلسی پێکهاتووی سنجار لە لای ڕاست وە بەردی کلسی کۆلۆشی ناو بیرە ستوونەکانی لای چەپی دەربەندی باسەرە بە نرخی بەرزی GSI & DMR , RMR وە بە ڕەووشتی میکانیکی باش دەناسرێنەوە ، بەڵام تاوێرە بەردی پێکهاتووی کۆلۆشی دەنکۆڵەیی لەلای چەپ وە پێکهاتووی جەرکەس لەلای ڕاستی دەربەندی باسەرە بە نرخی نزم-مام ناوەندی GSI & DMR , RMR دەناسرێنەوە وە ڕەووشتە میکانیکی یەکانیان تا ڕادەیەك خراپە.

ئەم لێکۆلینەوەیە مۆدێلێکی نوێ پێشنیار دەکات کە لەم نامیلکەیەدا ناسراوە بە مۆدێلی تاوێرە بەرد- پانەبڕگەی شیوو دۆڵ (Rock mass-valley section model) وە لێرەدا بۆ هەر سێ پانە بڕگەی (a-b, c-d & e-f) بەکارهاتووە کە بۆ یەکەم جار وێنە کێشراوە بە جۆرێك کە یەکە جیۆتەکنیکی یەکانی تاوێرە بەردی یەکان ناڕاستەی ناو هەر یەکە لەو پانە بڕگانە کراوە.

هەڵسەنگاندنی سەرەتایی نێوان ئەو سێ پانە بڕگەیە بۆ هەڵبژاردنی باشترینیان بۆ شوێنی بەنداوی پێشنیارکراو ئەوەی دەرخست کە پانە بڕگەی e-f لە زۆر لایەنەوە لە پانە بڕگەی c-d باشترە ، وە پانە بڕگەی c-d لە a-b باشترە ، بەڵام هەڵسەنگاندنی کۆتایی نێوان هەردوو پانەبڕگەی c-d و e-f ئەوەیان دەرخست کە پانەبڕگەی c-d زۆر گونجاوترە لە پانەبڕگەی e-f بۆ بنیادنانی بەنداو ، ئەمەش بەهۆی بوونی شوێنێکی مقەستی بێ هێزەوە (weak sheared zone) لە لای ڕاستی پانەبڕگەی e-f ، ئەم شوێنە مقەستی یە بێ هێزە کاریگەری خراپی لەسەر جێگیری بەنداوەکە هەیە دوای دروست کردنی و پڕبوونی عەمبارەکەی بەئاو ، کە ئەبیتە هۆی ناجێگیری بەجۆرێك کەئەم ناجێگیری یە لە زیادبووندا ئەبێت بەتێپەڕبوونی کات.

ئەندازەی تـاوێرە بەردی یە کان لە شوێنی بەنداوی باسەڕەی پێشنیارکراو، سلێمانی، هەرێمی کوردستـان، بـاکوری ڕۆژهەڵاتـی عێراق

نامەیەکە

پێشکەش کراوە بە کۆلێجی زانست لە زانکۆی سلێمانی وەك بەشێکی تەواوکەر بۆ بەدەست هێنـانـی دکتۆرای فەلسەفە لە زانستی جیۆلۆجی دا

لەلایەن

غـفـورأمین حمە سور أمین

ماستەر لە زانستی جیۆلۆجیای ئەندازەیی – زانکۆی سەلاحەدین / ۱۹۹۱

بەسەرپەرشتی

پڕۆفیسۆر د. سعد نعمان السعدی

سـەرماوەرز– ۲۷۰۹ کوردی **کـانوونی یەکەم –۲۰۰۹ زایینی**