FEASIBILITY STUDY OF AN UNDERGROUND BARITE MINING AT PANDS BARITE MINING Co., Ltd., LOEI PROVINCE

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A Thesis Submitted in Partial Fulfillment of the Requirements for the Degree of Engineering in Geotechnology

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การศึกษาความเป็นไปได้ในการทำเหมืองแบไรท์แบบเหมืองใต้ดิน ที่บริษัท พี แอนด์ เอส แบไรท์ ไมน์นิ่ง จังหวัดเลย

นายมานะ จาตุวรรณ

วิทยานิพนธ์นี้เป็นส่วนหนึ่งของการศึกษาตามหลักสูตรปริญญาวิศวกรรมศาสตรมหาบัณฑิต

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วัตถุประสงค์ของการวิจัยเพื่อศึกษาความเป็นไปได้ของการนำวิธีการทำเหมืองใต้ดินแบบอุโมงค์หลายระดับมาประยุกต์ใช้ในเหมืองแบไรต์ขนาดเล็ก วิธีการศึกษาวิจัยประกอบด้วยการรวบรวมข้อมูลและแปลความหมายของงานสำรวจที่ผ่านมา เก็บข้อมูลทางด้านวิศวกรรมธรณีในภาคสนาม ทดสอบคุณสมบัติทางกลศาสตร์ ประมวลและวิเคราะห์ข้อมูล ออกแบบเหมือง และประเมินค่าใช้จ่ายของการทำเหมือง ข้อมูลที่มีอยู่ระบุว่าแหล่งแร่แบไรต์ในพื้นที่ศึกษามีลักษณะเป็นสายแร่วางตัวอยู่ในแนวเกือบดิ่ง หนาประมาณ 4 ถึง 12 เมตร สายแร่โผล่ตามแนวสันเขาและมีความลึกจากผิวดินลงไปประมาณ 150 เมตร คุณภาพเชิงวิศวกรรมธรณีของสายแร่และมวลหินข้างเคียงจัดอยู่ในระดับปานกลางถึงต่ำ การทำเหมืองใต้ดินแบบอุโมงค์หลายระดับด้วยวิธีเจาะระเบิดได้ถูกเลือกเพื่อพัฒนาและทำเหมืองแร่ การพัฒนาใต้ผิวดินมีอุโมงค์ทางเข้าสำหรับการขนส่งคนงาน เครื่องจักร และอุปกรณ์การทำเหมือง ซึ่งจะถูกขุดเปิดตามแนวราบที่เส้นความสูง 350 เมตรและเดินขนานไปกับสายแร่ อุโมงค์ระดับสำหรับเป็นพื้นที่ปฏิบัติการอยู่ระหว่างเส้นความสูง 350 เมตรถึงความสูง 460 เมตร อุโมงค์จำนวน 7 ชั้นจะถูกขุดเปิดตามแนวราบด้วยความต่างระดับ 20 เมตรเข้าหาสายแร่ ขนาดของสโต๊ปถูกกำหนดให้มีความยาวและความสูงไม่เกิน 80 เมตร มี 9 สโต๊ปที่ได้ถูกออกแบบไว้เพื่อการผลิตแร่ เสาค้ำยันในแนวนอน แนวตั้ง และแนวบนสุดมีความหนา 20 เมตร มวลหินที่ไม่มีเสถียรภาพจะถูกเสริมด้วยการใช้หมุดยึดและลวดตาข่ายเหล็ก แร่ที่ระเบิดย่อยแล้วจะไหลออกจากสโต๊ปด้วยแรงโน้มถ่วงลงมาสู่ระดับการขนส่ง ปริมาณแร่หกแสนสองหมื่นตันจะผลิตด้วยอัตรา 530 ตันต่อวัน อายุการทำเหมืองประมาณ 14 ปี ต้นทุนของการพัฒนาและผลิตแร่รวมประมาณ 346 บาทต่อตันและราคาจำหน่ายในท้องถิ่นประมาณ 550 บาทต่อตัน

สาขาวิชาเทคโนโลยีธรณี ลายมือชื่อนักศึกษา________________________

ปีการศึกษา 2547 ลายมือชื่ออาจารย์ที่ปรึกษา___________________

MANA JATUWAN : FEASIBILITY STUDY OF AN UNDERGROUND BARITE MINING AT PANDS BARITE MINING Co., Ltd., LOEI PROVINCE. THESIS ADVISOR : ASSOC. PROF. KITITEP FUENKAJORN, Ph. D., P.E. 113 PP. ISBN 974-533-369-7

BARITE/UNDERGROUND MINING/STOPING/EXCAVATION/GEOLOGY/MAPPING/DRILLING

The objective of this research is to study the potential of applying the sublevel stoping method for the development and exploitation of a small-scale barite mine. The tasks include literature review, geologic mapping, mechanical laboratory testing, data compilation and analysis, mine design, and mine evaluation. The existing information shows that the barite deposit is a vein type and steeply dipping with the thickness varied from 4 to 12 m. It exposes along the ridge with a depth of 150 m. The geotechnical properties of the barite vein and surrounding rock masses are classified as fair and poor qualities. The sublevel stope opening with drill and blast methods is selected for the mine development and exploitation. The underground development has access tunnel for transporting personal, equipment, and materials. It is horizontally excavated at the elevation of 350 m and run parallel to the barite vein. The sublevel tunnels for working place are at the elevations from 350 m to 460 m. Seven level tunnels are also horizontally opened with a vertical interval ranging of 20 m into the barite vein. The stopes have the maximum length and height of 80 m. Nine stopes are designed for the production. The rib, sill and crown pillars are 20 m thick. The unstable rock masses will be reinforced by the rock bolts and wire mesh.

The fragmentation of barite ore flows through the stope by gravity to haulage level. The 0.62 million tons of ore can be produced with a rate of 530 tons per day. The mine life is fourteen years. The overall mining cost is about 346 Baht/ton and local ore price is 550 Baht/ ton.

School of Geotechnology Student's Signature______________________

Academic year 2004 Advisor's Signature______________________

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Mana Jatuwan

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CHAPTER I

INTRODUCTION

1.1 Background

Surface mining has played an important role in the development and exploitation of several mineral deposits in Thailand. Surface mining activities normally disturb land surface in order to recover the ores, and dispose the wastes. Changes and degradation of the landscape in the mine area cause serious environmental impact. Such environmental problems as erosion, decreasing of vegetation density, poor visualization, are the results of the unreclaimed surface mine areas.

Recognizing the matters of environmental protection and preservation, government agencies have issued several acts, and laws to regulate the mining industry. These include, for example, (1) declaring area classification for conservation of forest, national park, and wildlife sanctuaries, (2) declaring watershed classification for appropriate land-use, (3) issuing environmental law relating to the mining industries, and (4) collecting fees for the land-use and deforestation areas.

For mining investors, these environmental issues are considered to be the most complicated and frustrating problems. Environmental criteria have become decisive factors for starting new development projects in most countries, including Thailand. Many valuable ore deposits could not be exploited, either they are unfeasible to mine or are delayed by granting process due to the environmental reasons (Prakmard Suwanasing, 1992). Decreasing of the land available for mining makes permission to use forestland for mining even more difficult to obtain (Duangjai Intarapravich, 1992). Other problems occurring in the Thai mineral industry, which affect the development of mining technology are the depletion of mineral resources, small-scale of operation, lack of technology and information, economics constraints and marketing problems (Gawee Permpoon et al., 1992).

1.2 Research objectives

The objective of this thesis is to conduct a feasibility study of a small-scale underground barite mine at Loei province. The effort is part of an attempt at extracting the ore at greater depths, and minimizing the environmental impact. If successful, the results can be used as a general guideline for other mines with similar geological conditions. The aspects of the study include mineral exploration, mine planning excavation, support design, and reserve estimation.

1.3 Basic assumptions

The available geological data relevant to the site, as obtained from previous investigation (such as from Department of Mineral Resources) are assumed to be true and reliable. For the economic evaluation, the current market value of the ore will be used in the analysis. Fluctuation of the market value in the future will not be considered. The mining techniques will be selected from the conventional underground mining methods, and using market available mining equipment. These include the drilling and mining methods, opening support, material handling, ventilation, site commissioning and decommissioning, personal training, and safety implementation. Assessment of the environment impact will only be in the aspects of ground surface subsidence and blast vibration. Since the mine is located above the groundwater table, groundwater contamination will be excluded from this study. It is also assumed that the interpolation of the mineralization zone boundaries and of the assay is linear between the exploration locations where known and proven data have been established.

1.4 Scope of the study

Geology of the site will be studied by means of surface mapping and core drilling. The selection criteria for the mining methods will aim at small-scale nonmetallic ore deposit where the characteristics of orebody are massive, tabular shape, steeply dipping, narrow width, and infinite depth. Engineering geology investigation will include the physical, chemical and mechanical properties of the ore and country rocks. The economic assessment will consider the ore value, mine life, and production rate. The environmental concern will consider the ground control to maintain integrity of openings and subsidence or caving effects on the surface.

1.5 Methodology

The proposed study can be divided into 7 tasks

1.5.1 Literature review

The available geological and relevant geotechnical data will be reviewed to reveal the ground conditions and potential problems and to highlight main factors required for underground development and exploitation methods. These include, for example, geological maps, geological reports, and drilling core from "Mineral Investigation in Northeastern Thailand" in 1969, and PANDS's surveyed report

1.5.2 Geological mapping

Geologic data are rock types, rock boundary, bed orientation, ore boundary, ore orientation and structural geology. Such information will be investigated by visual inspection and geological survey. The collected data will be sketched and reported on notebook and plotted on field map with scale 1:2000. All corrected data will be analyzed for mine designation and planing.

1.5.3 Geological data collection

Geotechnical data will be collected in the field. The geological investigation consists of discontinuities, such as joints, fractures and faults. The discontinuities will be collected by measuring the orientations of joints, spacing of joints, continuity of joints, filling and aperture.

1.5.4 Laboratory testing

The barite ore and rock samples collected from the study area will be tested in the laboratory to examine physical and mechanical properties of the ore deposit and the country rocks. The laboratory testing comprises point load test, Brazilian tensile strength test, and uniaxial test.

1.5.5 Compilation and analysis

The maps, detailed core logs, and ore/rock properties will be compiled and analyzed for evaluating the ore shape, ore orientation and ore reservation. They serve as the specific requirements for selecting the most suitable underground method.

1.5.6 Mine design and planning

The basic knowledge obtained from compilation and analysis data will be applied for mine design and planning. The field data will include, but not limit to, rock strength, ore strength, rock quality designation, hardness, specific gravity, and orientation or shape. These will be used to design the underground mine properly. The design will include mine layout, opening shape and size, ore extraction, ore handle, underground support, production rate, etc.

1.5.7 Economic analysis

Production cost and benefits will be calculated and compared to determine the economic feasibility of the mine.

1.6 Expected Results

The expected result could represent a technological improvement of small-scale mines by mean of appropriate underground technology by extracting ore at greater depths while minimizing the environment impact. The results can be used as a general guideline for other mines with similar geological and economic conditions.

CHAPTER II

LITERATURE REIEW

2.1 Previous works

The Bua Hin Khoa Barite deposit is the former name of the barite deposit for the mining lease and mining lease application of PANDS BARITE MINING Co., Ltd. It was first explored and reported by Jacobson et al. (1969). The detailed surface topographic and geologic maps were compiled and eight trenches were excavated, mapped and sampled. Three main lithologic units are found. The first is massive barite vein, the second is dolomite with variable amounts of barite, and the third is shale. The deposit is within the sequence of the Devonian and Lower Carboniferous sediments. The barite mineralization zone is approximately 1.5 kilometers long. It had been reported that the maximum thickness was 9 meters with maximum depth of 200 meters. The deposit was described as a vein type.

During 1992-1993, it is reinvestigated as part of the Bua Hin Khoa Zinc Project. The geology is studied by surface mapping method, trenching, and drilling. The summary of exploration results reported by Mustard (1993) indicates that the host rocks and mineralization zone consist of calcareous black shale, siltstone, sandstone, and limestone beds. These rocks are classified in the Permo-carboniferous era. The general strike is in north-south with varied dips.

Two fault zones are found. The north-south fault is steeply dipping with strike parallel to the strike of the bedding planes. This fault is in the contact zone of rocks in the west of the study area. The northeast striking faults are in the mineralization zone. They are strike-slip faults in a dextral form with horizontal movements of about 2 to 350 meters.

The barite is found parallel to the limestone bed in the west and to the shale with interbeded diabase sill in the east. At least two zones of barite are found. The main zone is parallel to the fault with 100 meters away to the west. It is nearly vertical with over 1.5 kilometers long and 2-7 meters wide. Numerous faults with a strike of 340 degrees are within the mineralized zone and adjacent shales.

The second mineralization zone known as the western zone is 1 to 3 meters thick and inclines 45 degrees to the east toward the main zone. Surface mapping and diamond drilling indicate that the two zones were once horizon and later folded to form a syncline. The hinge of the syncline is typically marked by fault.

2.2 Relevant mining methods

2.2.1 Shrinkage Stoping

Shrinkage stoping is applicable to ore zones with dipping more than an angle of repose with widths from 1 to 30 meters. The enclosing waste rock must be competent. It should not fail so that dilution is kept to a minimum. Another major requirement is that the contact between wall rock and the ore zone should be relatively sharp and regular without any abrupt change in either strike or dip along the stope interval.

The key design parameters are the dimensions of the stope, largely governed by the shape and size of the deposit. For a relatively narrow ore body, the stope is placed longitudinally. For a wide ore body, the stopes are placed transversely. Stope widths normally vary from 1 to 30 m, with length from 45 to 90 m, and heights from 60 to 90 m (Lucas and Haycocks, 1973; Lyman, 1982; Hartman, 1987).

The stope is commonly accessed by crosscuts driven into the ore body at regular intervals from a drift driven in the footwall, or from headings driven along the length of the ore. The stope is mined by horizontal drilling short holes (2 to 3 meters) along the length of the vein and blasting the ore down, or by using vertical drilling with stoppers (Lyman, 1982; Hartman, 1987). Access to the next lift is gained by standing on the broken ore, and repeating the process until the upper level is reached. During the mining phase, only enough muck is drawn out of the bottom of the stope to permit the miner to access the stope, and to drill off the next lift. Typically, during the mining stage of the stope, approximately 30-40% of the total broken muck is drawn off. Productivity within shrinkage stopes is largely dependent on the width of the ore zone, and can vary from 10 to 15 tons per manshift (Hartman, 1987).

Applications of shrinkage stoping find little use in modern mining (Hartman, 1987). The popular recent examples of ore are gold (Homestake, USA), iron (Pea Rigde, USA), lead-silver (Idarado USA), lead-zinc-silver (South Bay, Canada), limestone (Bellefonte, U.S.A.), and nickel (Sudbury, Canada).

The Pea Ridge iron mine owned by St. Joe Minerals Corp., USA is the only iron ore pellet producer in south-central United States. The exploration began in

1953, development in 1958 with $52 million, and exploitation in 1964. The iron deposit occurs as dike or veinlike structure lying nearly vertical in Precambrian rhyolite porphyry at depth from 390 m to 900 m. The ore types are magnatite and hematite with both proven reserves of about 102 million tons and average grade of 56% Fe. The cutoff grade is 45%. The airtracs, fan drills, and downhole percussion drills basthole, and pneumatic cartridge loader or slurry pump explosive is used to break the ore in the stopes. The production rate is about 1.9 million tons per years.

2.2.2 Sublevel Stoping

This type of mining method applies to ore zones with similar geometry to the shrinkage stoping; relatively regular ore body and waste contacts with good ground conditions. The method is often referred to as blast hole stoping or longhole stoping.

The key design parameters in sublevel stoping involve the factors and dimensions similar to the shrinkage stoping. Stope width can be varied from 6 to 30 m with a maximum length of 90 m, and maximum sublevel height of 90 m (Morrison and Russell, 1973; Mann, 1982; Hartman, 1987).

Sublevel stoping methods use a longhole drilling and blasting from sublevels to break the ore. The ore blocks are prepared for mining by opening sublevels in the ore at intervals ranging from 15 to 18 meters. From these sublevels, a nominal 5 mm diameter blastholes is drilled in a ring pattern to the ore limits. Mining usually commences at one end of the stope from a vertical slot raise and continues along the stope, and from one sublevel to the next until the stope is completed. Broken muck is drawn from the stope through draw points located at the bottom of the stope.

The drilling methods, in which holes are drilled between sublevels are the ring drill and the parallel drill with blasting into a vertical slot. The ring drill is original version of stoping (Hartman, 1982, 1987). The drillholes are relatively small (50 to 75 mm), bored with percussion rock drills mounted on a column and bar (drifter) or fandrill rig, and with extension drill steel to a maximum length of 24 to 30 m. The ring drill requires accuracy in hole placement to obtain proper fragmentation. Hole deviation is not more than 2%. For charging and blasting, the holes are loaded heavily with a ring detonated simultaneously. The effect of blasting can be disastrous from inaccuracy of drilling and heavily charged.

A further improvement has been realized in the last decade with the advent of the VCR method. Large parallel holes are used as in the parallel drilling method, but a major innovation has been blasting horizontal slice of ore with near spherical charges into the undercut (Mitchell, 1981; Lang, 1983; Hartman, 1987). Spherical placement of explosives is the most efficient in terms of fragmentation and power consumption. Holes are charged from collar after plugging the opposite end; the size of charge is restricted to a length to diameter ratio of 6:1, which suffices in practice to simulate a spherical charge. All holes in a stope are detonated together. When the broken ore in stope has been drawn sufficiently, then the next slice of ore is blasted. Because drilling is carried out from a sublevel and is usually completed before blasting commence, unit operation with the VCR method can be conducted with high efficiency and productivity (Hartman, 1987). Productivity in sublevel stope varies with the stope width and sublevel interval which ranging between 15 to 30 tons per manshift (Hartman. 1987).

Applications of sublevel stoping, while most commons in metal mines, are frequent and varied in hard rock mines in general (Hartman, 1987). They include copper (Carr Fork, USA), copper-iron-zinc-sulfur (Copperhill, USA), copper-lead (Mt. Tsa, Australia), gold (Homestrake, USA), iron (Pea Ridge, USA), limestone (Bellefonte, USA), and nickel (Sudbury, Canada).

The Copperhill copper-iron-zinc-sulfur mine owned by Tennessee Chemical Company is located in Copper Basin in southern Appalachians, USA. The mining dates to 1847. The mineralized zone consists of pyrrhotite, pyrite, chalcopyrite, and sphalerite. It is the massive sulfide lenses in metamorphosed schists and graywackes. The depths of ore bodies extend to at least 900 m with up to 120 m in thickness. Overall reserves are about 250,000 to 70,000,000 tons and averaged grade of 0.7% copper, 0.5% Zinc, and 20% sulfur. The sublevel stoping are primarily applied to produce the sulfur (as sulfuric acid). The horizontal tunnels with levels about 30 to 60 m apart are opened. A stope is drilled upward from a drift and broadened into a V-shape pattern with the drillhole bored in a pattern covering the area from 5.5 to 7.3 m^2. The material is blasted and falls down into the drift for hauling. Production rate is 2.2 million tons per year.

CHAPTER III

GEOLOGICAL SETTING OF BARITE DEPOSIT

The geologic conditions of the barite deposit and wall rocks are studied and used as preliminary data for selection of the most suitable underground mining method, mine layout, and support design. Such information includes the geometry and depth of the ore body as well as the characteristics of the wall rock mass.

3.1 Source of information

The source of information that is used to compile the geology in the study area includes as follows.

1) Topographic map with a scale of 1:50,000, series L7017 and sheet no. 5344 II prepared by the Royal Thai Survey Department, Bangkok, Thailand, 1993.

2) Geological map of Thailand with a scale of 250,000, sheet no. E47-12 (Changwat Loei) published by the Geological Survey Division, Department of Mineral Resources, Thailand, 1984 (Figure 3.1).

3) Local geologic map with a scale of 1:2,000 compiled by Mustard (1993) (Figure 3.2).

4) Borehole data comprise eight reverse circulation (RC) holes and three diamond drillholes (DDH). The lithological data are logged and shown as the stratigrahpic columns in Figures A 1 to A 11 of Appendix A.

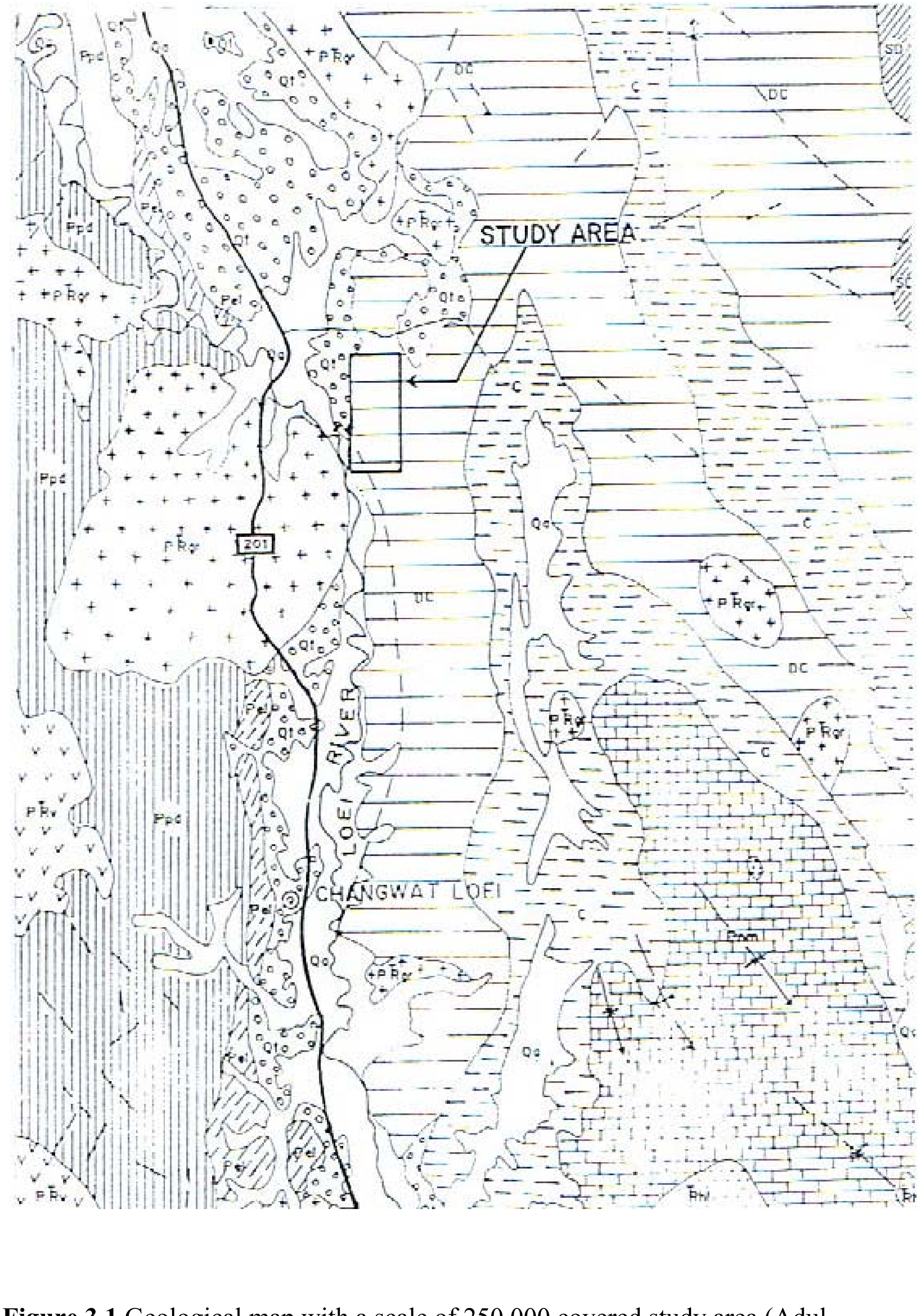

Figure 3.1 Geological map with a scale of 250,000 covered study area (Adul charoenpravat et. al., 1976)

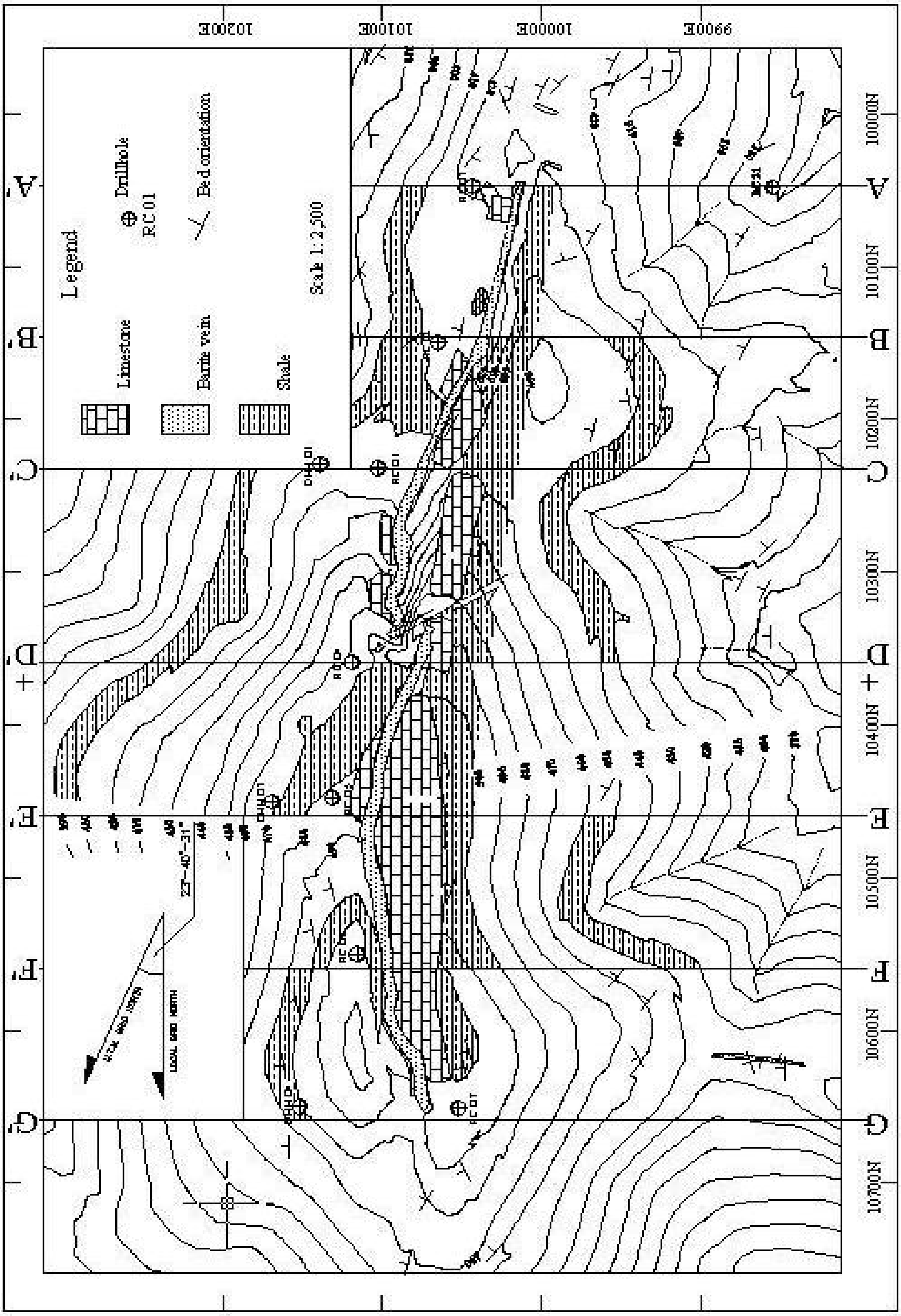

Figure 3.2 Geologic map compiled from geological data in the study area.

3.2 Compilation

The existing geological information is interpreted and compiled to construct the cross-sections to illustrate three-dimensional view of the barite deposit and nearby rock formations.

Seven cross-sections have been drawn along east-west direction, parallel to the dip direction of the drill holes. These sections are shown as AA' to GG' lines in Figure 3.2). The spacing between two adjacent cross-sections is 100 m. Similar stratigraphic units are illustrated in all sections (Figures 3.3 to 3.9), whereby, in the eastern block from approximate 10000E line, they comprise beds of lower shale, limestone, barite, limestone, diabase sill and upper shale. The exception is along section AA', where the section barite bed was found in the western block west of line 10000E. The similar sequence of wall rocks, including lower shale, limestone and upper shale was also found. Barite crops out at different elevations along north-south trending ridge. The highest altitude is at 520 m in the north part of the area (Figure 3.9) and declines to 490 m in the southern part of the area (Figure 3.3). Barite bed extends from the surface to the elevation of 350 m. Its thickness varies between 4 and 12 meters. Tabular barite ore body bulges in the central part and pinched out at the northern and southern ends.

3.3 Geology of barite deposit

The results from the study of the existing information suggest that the rocks in the study area comprise the Middle Devonian to Lower Carboniferous age sediments of the Nong Dok Bua Formation. They are predominantly grayish black shale and thick-bedded limestone with some siltstone and sandstone interbedding. The barite vein, that typically crops out along the ridges with over 800 kilometers long, is hosted in limestone bed with a strike of 340° (Figure 3.2). It is tabular in shape with variable dips from 74° to 88° to the east and variable widths from 4 to 12 meters. The average depth of barite can be estimated as 150 meters from the surface. The lower limestone bed on the western side of barite vein exposes on surface with a thickness up to 70 meters. The eastern barite vein is bound by spotty and thin limestone bed, diabase sill and shale. The outer sediments predominantly consist of brown and grayish black shale on both sides of the vein. The western shale bed underlying barite and limestone beds shows dipping angles between 20° and 60°. The eastern shale bed overlies the barite and limestone with steeply dipping to the east. The barite vein has the tabular shape with the dimension of 4 meters wide, 800 meters long, and 155 meters deep. The depth of the ore is defined at the lowest point where the exploration hole is intersected. Its volume is estimated as 0.89 million tons.

3.4 Discussions

The depth of barite vein is uncertain. Most of drillholes with dipping between 50 and 60 degrees intersect barite vein about 100 meters below the surface. Further extension of barite from the intersection is possible. For this study the depth of barite vein at the level of 350 meters is conservatively considered. Due to the limited number of exploration holes, the precise shape and grade of the barite deposit can not be precisely determined. Judging from the existing information, it is believed that the estimated tonnage of ore is sufficient to justify the further study in the feasibility level.

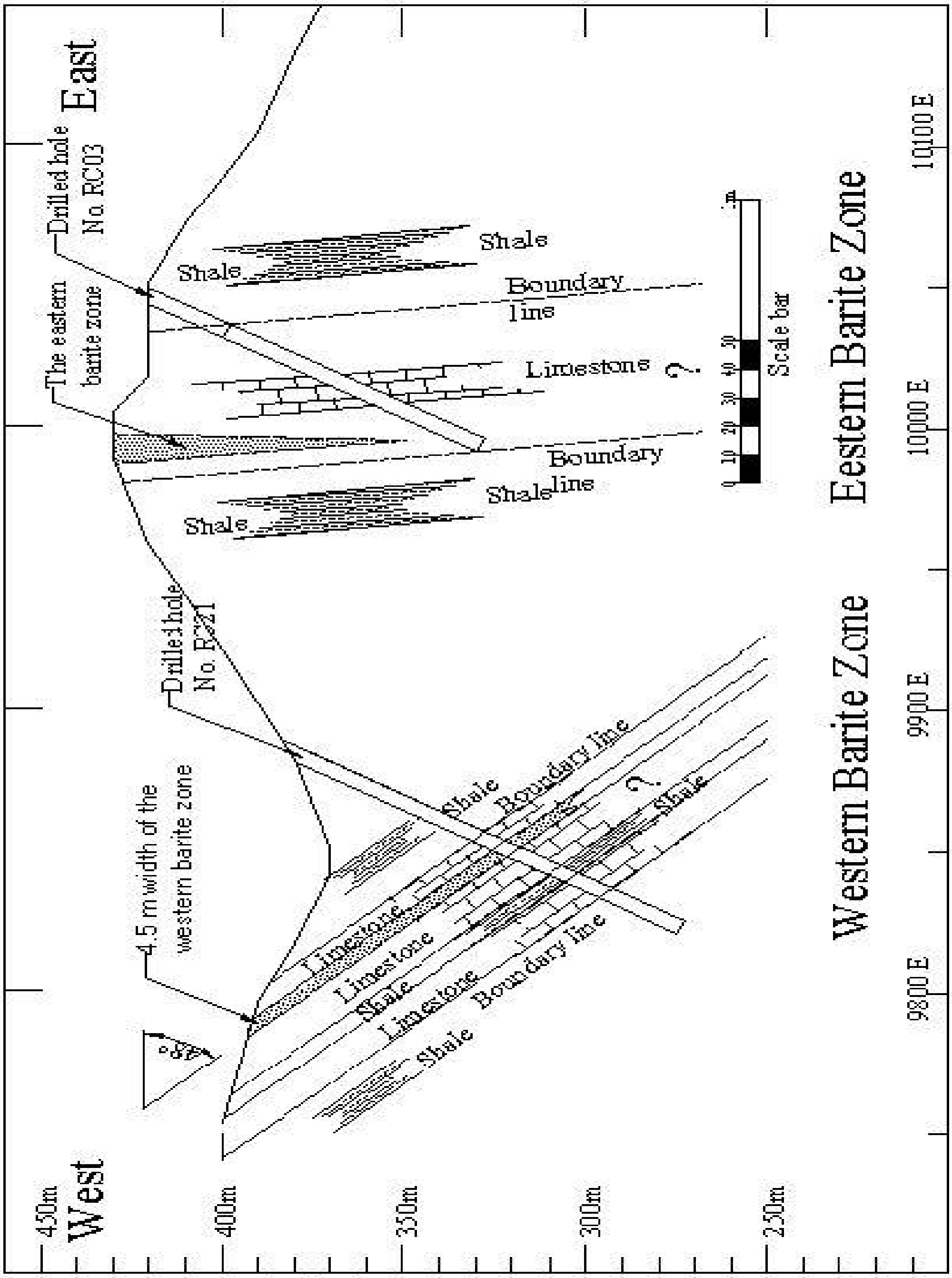

Figure 3.3 Cross section AA' along 10050N gridline showing the stratigraphy of barite and hosted rocks across the study area

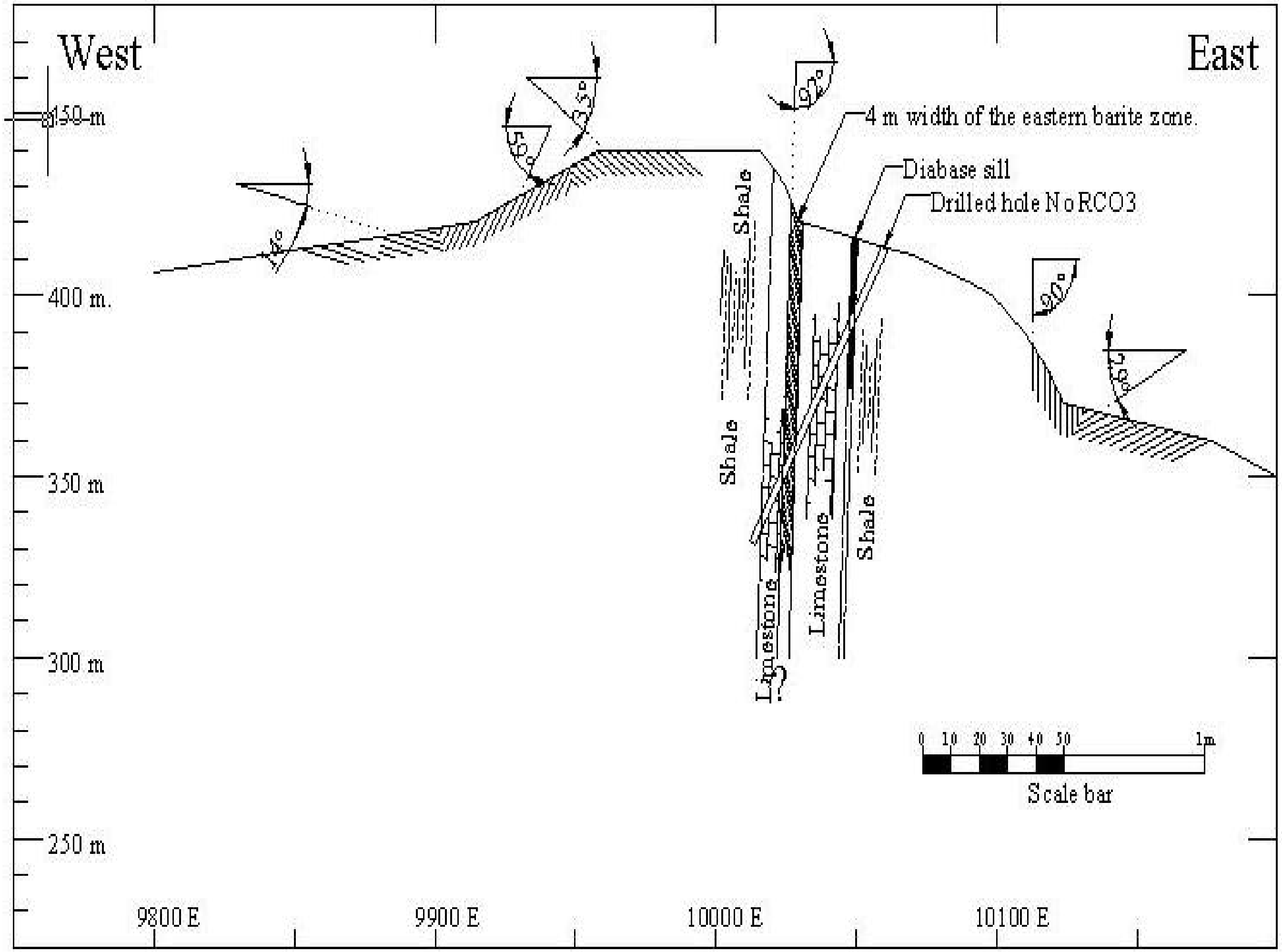

Figure 3.4 Cross section BB' along 10150N grid line showing the stratigraphy of barite and hosted rocks across the study area.

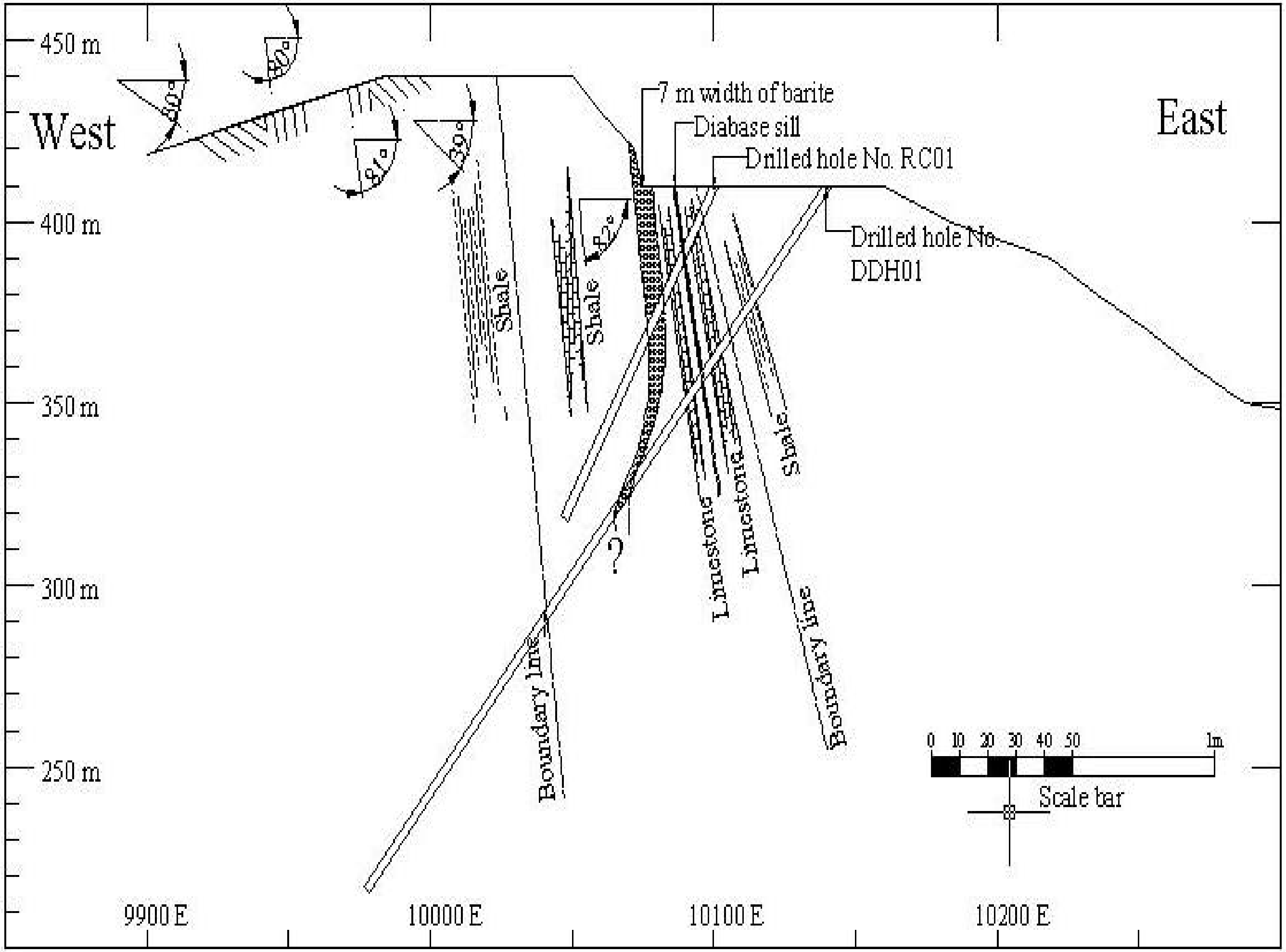

Figure 3.5 Cross section CC' along 10250N grid line showing the stratigraphy of barite and hosted rocks across the study area.

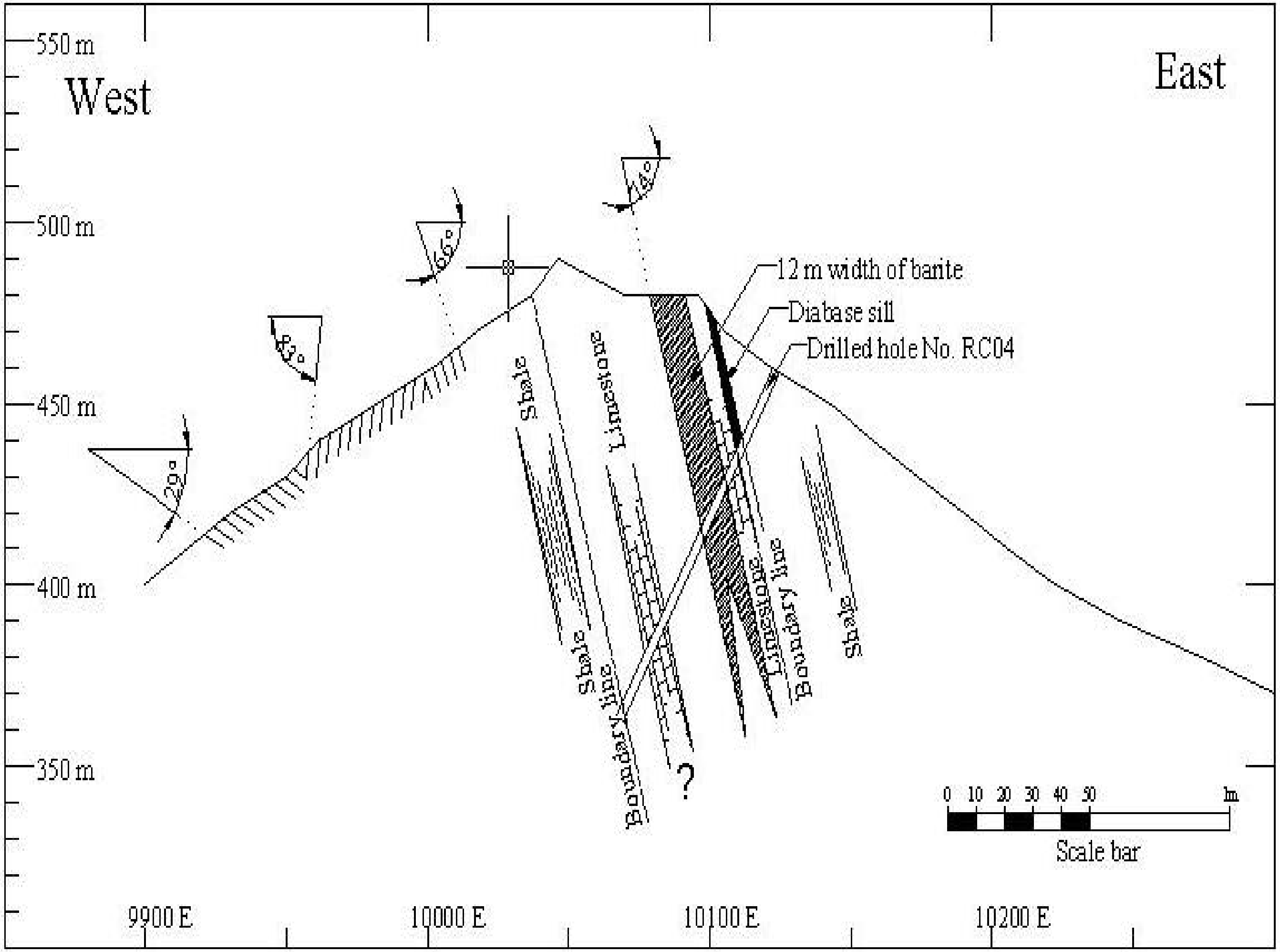

Figure 3.6 Cross section DD' along 10350N grid line showing the stratigraphy of barite and hosted rocks across the study area.

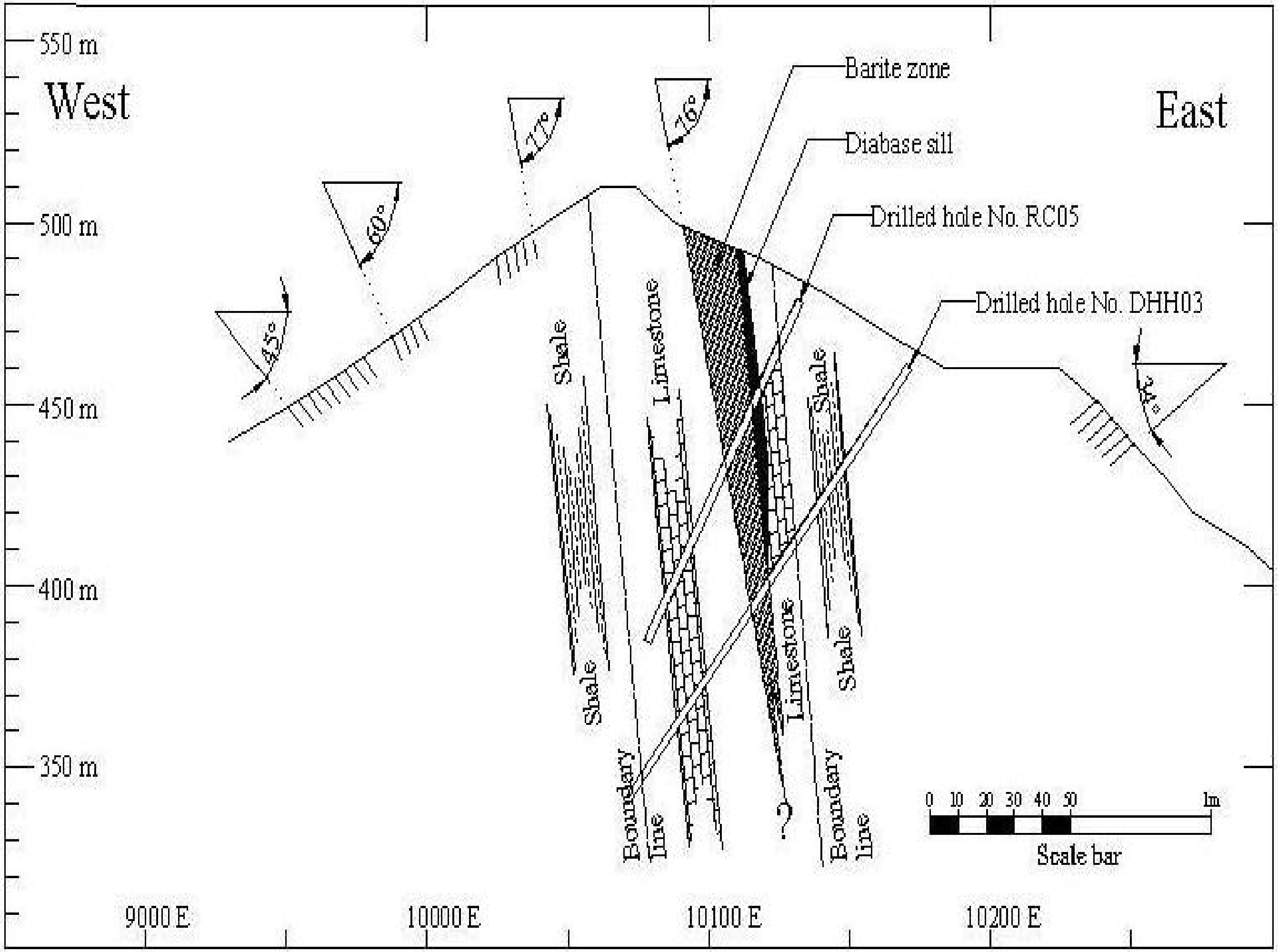

Figure 3.7 Cross section EE' along 10450N grid line showing the stratigraphy of barite and hosted rocks across the study area.

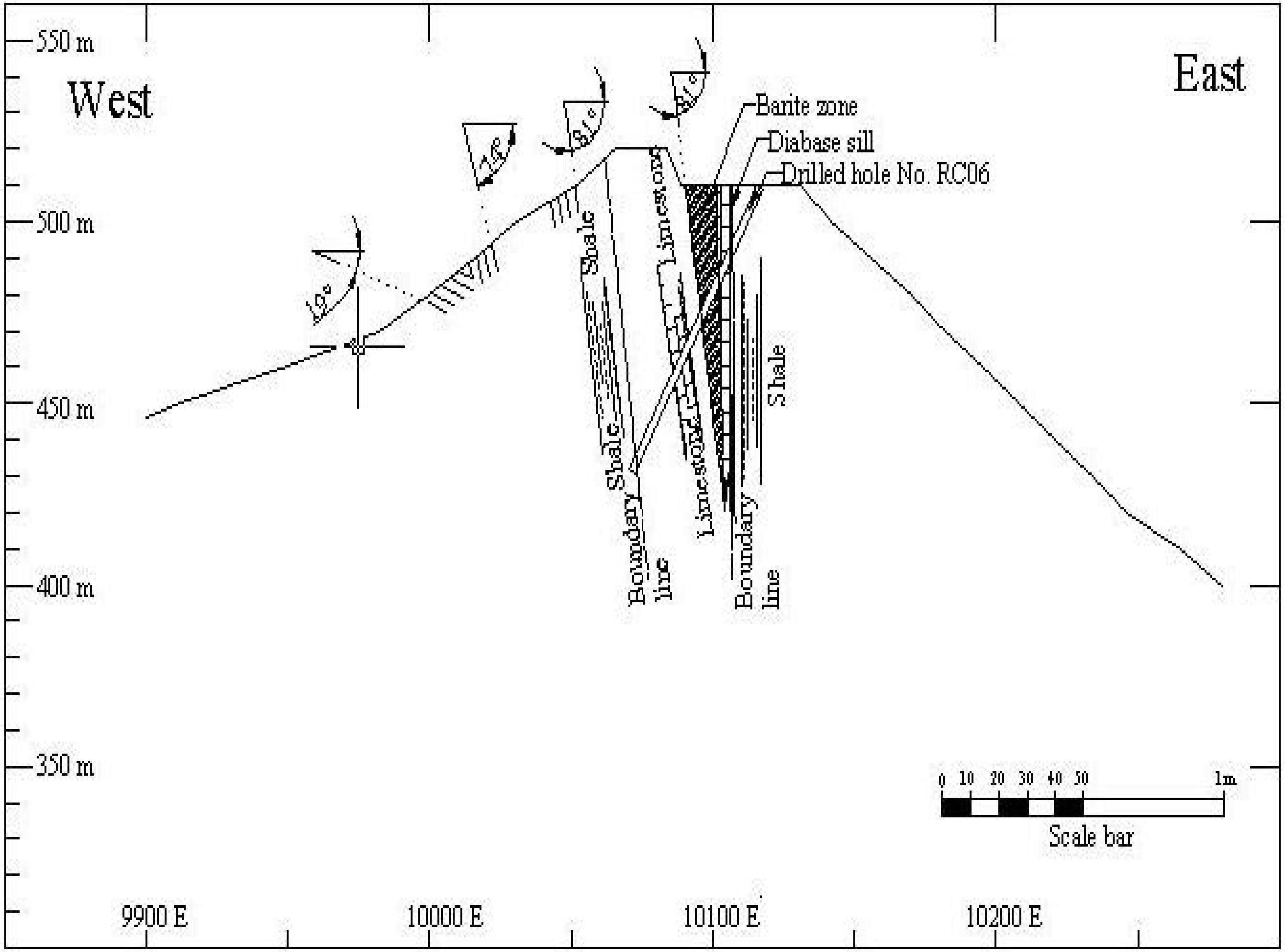

Figure 3.8 Cross section FF' along 10550N grid line showing the stratigraphy of barite and hosted rocks across the study area.

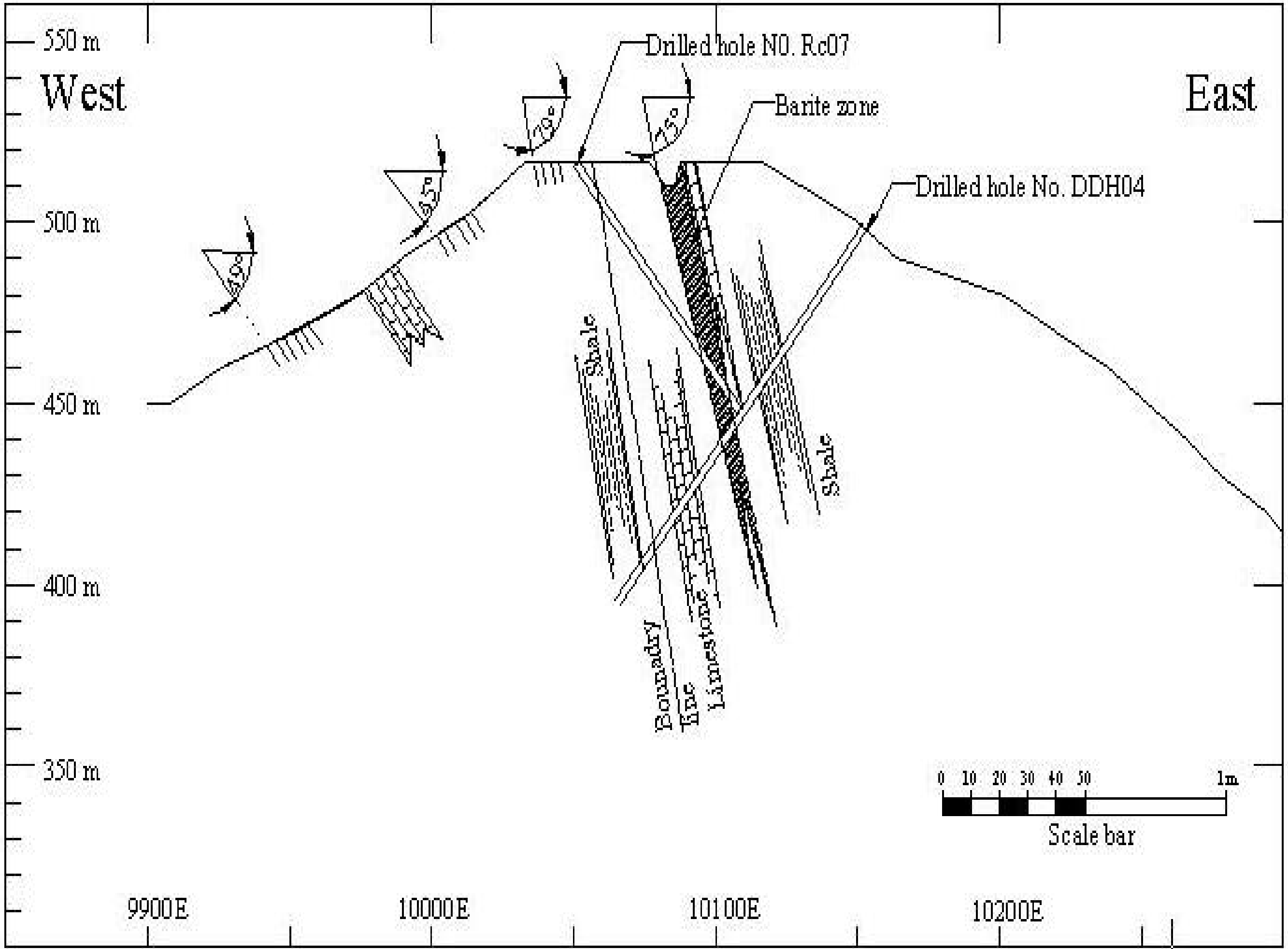

Figure 3.9 Cross section GG' along 10650N grid line showing the stratigraphy of barite and hosted rocks across the study area.

CHAPTER IV

GEOTECHNICAL INVESTIGATION

The objective of the geotechnical investigation is to collect and compile discontinuity data in the study area for use in the rockmass classification. The discontinuity data here include bedding planes, faults, joints, folds and fractures. Their characteristics to be identified and recorded are orientation, spacing, persistence, roughness, aperture, and filling materials.

4.1 Method

The geotechnical investigation includes (1) review of the geology of the previous geological map with a scale of 1:2,500, and relevant cross sections, and (2) reconnaissance survey of the rock masses. The rock masses in each classified zone are investigated and recorded for their geotechnical properties from the measurements, observations, and identification. Field estimation of the uniaxial compressive strength of the intact rocks from the surface exposure and road cut are performed. The classification used the criteria given by the International Society of Rock Mechanics (ISRM, Brown, 1981).

4.2 Classification of rock zones

The existing geology from the previous geological map and reconnaissance survey shows that the rock masses in the study area contain different rock types, properties, and structures. They have been subjected to different post tectonic activities. They can be classified into five rock zones (Figure 4.1).

4.2.1 Barite-bearing zone (BBZ)

The barite-bearing zone mostly consists of the barite vein that deposits between the footwall limestone in the west and the hanging wall shale in the east. It is 4 to 12 m wide and about 800 m long. The barite is milky white, fine-grained and massive. Some shows contamination of iron oxide, weathered clay minerals, and oxidized limestone. Three discontinuity sets are found in this zone. One is parallel to the bedding plane of barite, and two are transverse joints. These discontinuities have 20 to 50 cm of spacing and 0.1 to 0.5 cm of joint aperture. The strength is field-determined as R4. The estimated strength ranges between 50 and 100 MPa.

4.2.2 Footwall limestone zone (FLZ)

The footwall limestone zone is grayish black and fine-grained showing thick-bedded limestone with spotty barite along the western contact zone. There are 3 discontinuity sets with spacing varying from 0.50 m to 0.90 m. The discontinuities with about 0.1 cm aperture are filled with iron oxide and clay minerals. The strength is classified as R4 in accordance with ISRM criteria. The estimated strength ranges between 50 and 100 MPa.

4.2.3 Footwall low fracturing shale (FLS)

The footwall low fracturing shale zone is brown and grayish black shale. It lies below the footwall limestone and in the north area of fault line. This rock zone shows the loosen block on the surface with the size of 0.1 to 0.2 m. Three joint sets are found. The strength is classified as R3 in accordance with ISRM criteria. The estimated strength ranges between 25 and 50 MPa.

4.2.4 Footwall high fracturing shale zone (FHS)

The footwall high fracturing shale zone is brown and grayish black shale. It also lies below the footwall limestone and in the south area of fault line. This rock shows polygon shapes of loosen rock on the surface with small pieces. Four joint sets are found with narrow spacing. The strength is classified as R3 in accordance with ISRM criteria. The estimated strength ranges between 25 and 50 MPa.

4.2.5 Hanging wall high fracturing shale zone (HWS)

The hanging wall high fracturing shale zone is brown and grayish black shale with some limestone lens interbeded. It lies on the barite-bearing zone and it shows polygon shapes of loosen rock with small piece. Four joint sets are found with narrow spacing (0.1 to 0.15 m). The strength is classified as R3 in accordance with ISRM criteria. The estimated strength ranges between 25 and 50 MPa.

4.3 Geotechnical data collection

A total of 852 measurements of discontinuity properties are recorded from the five rock zones in the study area. The discontinuity orientations are analyzed to determine the clusters of joint sets (Figure 4.2). The other parameters include joint spacing, joint aperture, persistent of joint, and filling materials.

4.3.1 Barite-bearing and footwall limestone zones

There are three discontinuity sets in the barite-bearing and footwall limestone zones as follows.

The discontinuity set #1 has an average orientation of 143º/29º (strike/dip angle). The average spacing is about 0.50 m. The average aperture is 0.1 cm. It has low persistence (1 to 3 m). The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is the iron oxide and clay minerals.

The discontinuity set #2 has an average orientation of 282º/83º (strike/dip angle). The average spacing is about 0.50 m. The average aperture is 0.20 cm. It has low to high persistence (1 to 20 m). The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #3 has an average orientation of 356º/71º (strike/ dip angle). The average spacing is about 0.20 m. The average aperture is 0.1 cm. It has high persistence (10 to 20 m). The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

4.3.2 Footwall low fracturing shale zone

The footwall low fracturing shale zone has three discontinuity sets as follows.

The discontinuity set #1 has an average orientation of 023º/38º (strike/dip angle). The average spacing is about 0.18 m. It has high persistence (10 to 20 m). The average aperture is 0.25 cm. The roughness is in class 1 of the ISRM (rough or irregular, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #2 has an average orientation of 163º/81º (strike/dip angle). The average spacing is about 0.20 m. It has high persistence (10 to

20 m). The average aperture is 0.25 cm. The roughness is in class 1 of the ISRM (rough or irregular, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #3 has an average orientation of 272º/77º (strike/dip angle). The average spacing is about 0.20 m. It has high persistence (10 to 20 m). The average aperture is 0.25 cm. The roughness is in class 1 of the ISRM (rough or irregular, stepped). The filled material is iron oxide and clay minerals.

4.3.3 Footwall high fracturing shale zone

The footwall high fracturing shale zone has four discontinuity sets as follows.

The discontinuity set #1 has an average orientation of 143º/29º (strike/dip angle). The average spacing is about 0.15 m. It has high persistence (10 to 20 m). The average aperture is 0.5 cm. The roughness is in class 1 of the ISRM (rough or irregular, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #2 has an average orientation of 225º/48º (strike/dip angle). The average spacing is about 0.13 m. It has high persistence (10 to 20 m). The average aperture is 0.25 cm. The roughness is in class 1 of the ISRM (rough or irregular, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #3 has an average orientation of 282º/83º (strike/dip angle). The average spacing is about 0.19 m. It has high persistence (10 to 20 m). The average aperture is 0.5 cm. The roughness is in class 1 of the ISRM (rough or irregular, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #4 has an average orientation of 356º/71º (strike/dip angle). The average spacing is about 0.19 m. It has high persistence (10 to

20 m). The average aperture is 0.5 cm. The roughness is in class 1 of the ISRM (rough or irregular, stepped). The filled material is iron oxide and clay minerals.

4.3.4 Hanging wall high fracturing shale zone

The hangingwall high fracturing shale zone shows different discontinuity characteristics in the north and south areas of the fault line as follows.

The north hanging wall shale zone has four discontinuity sets as follows.

The discontinuity set #1 has an average orientation of 204º/22º (strike/dip angle). The average spacing is about 0.10 m. It has high persistence (10 to 20 m). The average aperture is 0.1 cm. The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #2 has an average orientation of 142º/47º (strike/dip angle). The average spacing is about 0.15 m. It has high persistence (10 to 20 m). The average aperture is 0.5 cm. The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #3 has an average orientation of 341º/63º (strike/dip angle). The average spacing is about 0.13 m. It has high persistence (10 to 20 m). The average aperture is 0.5 cm. The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #4 has an average orientation of 356º/71º (strike/dip angle). The average spacing is about 0.10 m. It has high persistence (10 to 20 m). The average aperture is 0.5 cm. The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

The south hanging wall shale zone has four discontinuity sets as follows.

The discontinuity set #1 has an average orientation of 201º/31º (strike/dip angle). The average spacing is about 0.20 m. It has high persistence (10 to 20 m). The average aperture is 0.20 cm. The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #2 has an average orientation of 354º/81º (strike/dip angle). The average spacing is about 0.20 m. It has high persistence (10 to 20 m). The average aperture is 0.30 cm. The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #3 has an average orientation of 178º/86º (strike/dip angle). The average spacing is about 0.10 m. It has high persistence (10 to 20 m). The average aperture is 0.50 cm. The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

The discontinuity set #4 has an average orientation of 076º/87º (strike/dip angle). The average spacing is about 0.10 m. It has high persistence (10 to 20 m). The average aperture is 0.30 cm. The roughness is in class 3 of the ISRM (slickensided, stepped). The filled material is iron oxide and clay minerals.

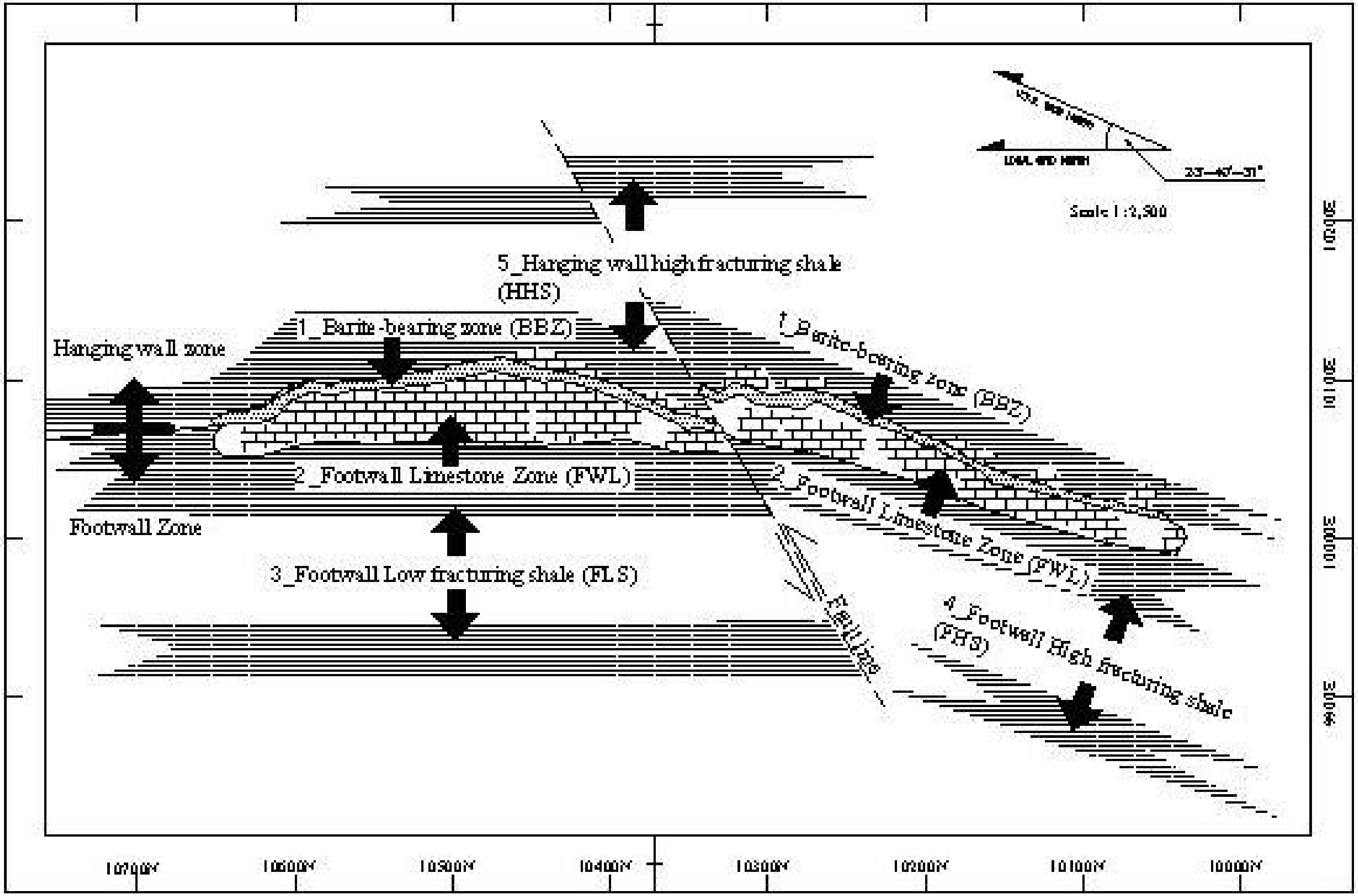

Figure 4.1 Boundary of classified the rock mass by three geological characters

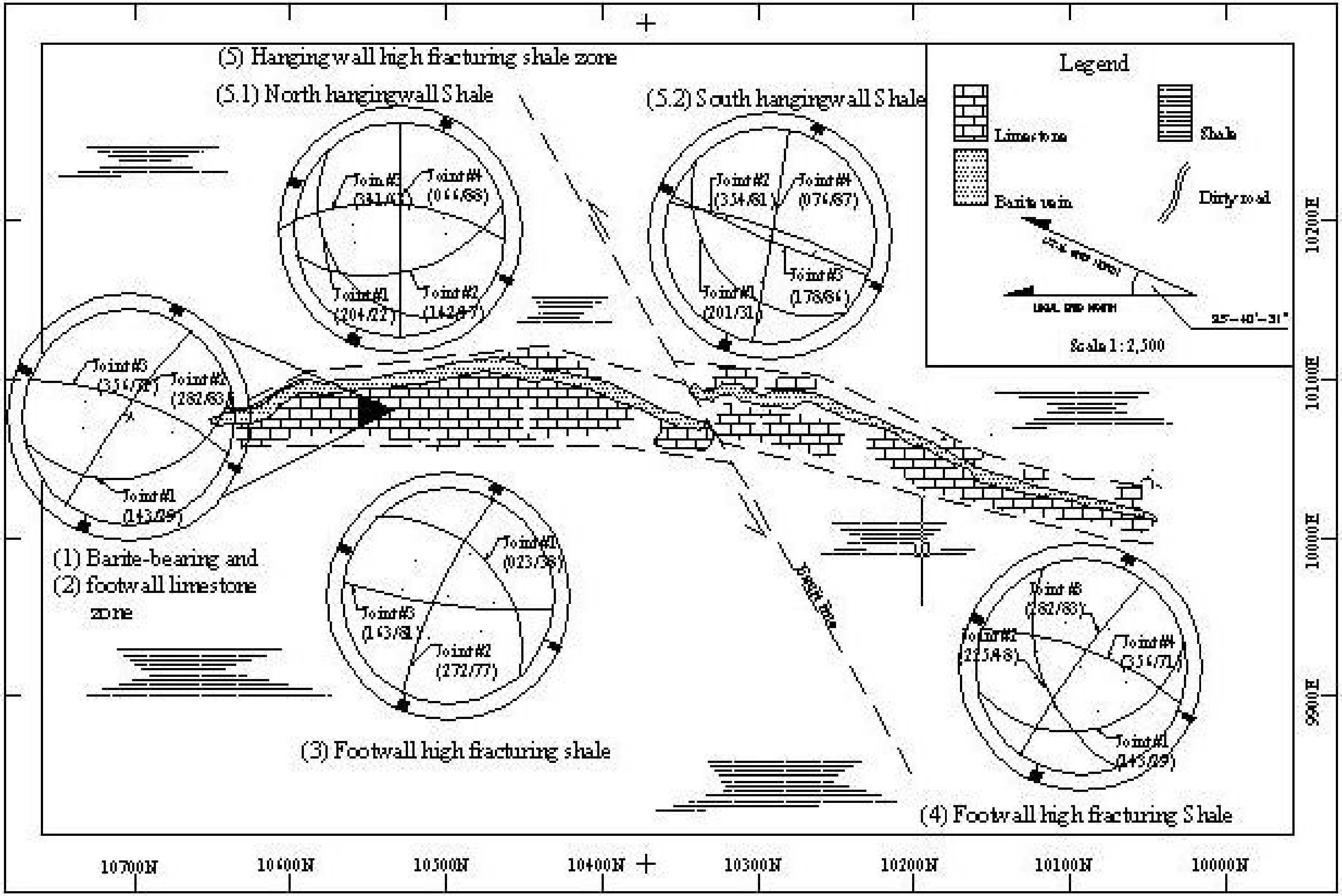

Figure 4.2 Plan view showing boundary and joint orientation of five rock zones.

CHAPTER V

MECHANICAL LABORATORY TESTS

The objective of laboratory testing is to determine the strengths of rocks in the study area. The uniaxial compressive strength (σ_C) and Brazilian tensile strength (σ_B) are required for use in the rock mass classification. This chapter describes the sample preparation test procedures and results. Limestone and barite block samples collected from the study area are prepared and tested in the laboratory.

5.1 Rock sample collection

Twelve block samples have been collected from the study area for use in the mechanical laboratory testing. They are collected from the outcrops in the mineralization zone and in footwall. Six blocks are from the barite-bearing zone and the other six blocks are from the limestone footwall. Figure 5.1 shows the locations from which these block samples are obtained. Care has been taken to ensure that each rock block is fresh and can be used to represent the rock conditions in each location. The average size of each block sample is approximately one cubic foot. Each block is numbered and examined to identify the mineralogical characteristics. Table 5.1 summarizes the rock characteristics.

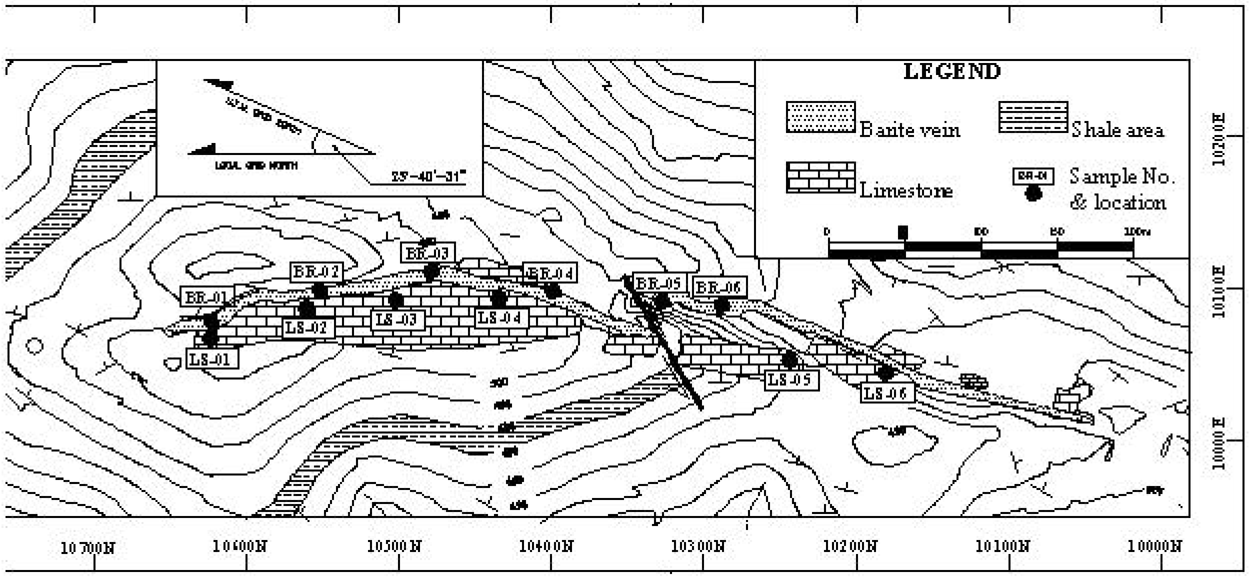

Figure 5.1 Location where the rock samples are collected.

Table 5.1 Rock description

Block No.	Grid location	Descriptions
BR-01	10625N/10083E	Light gray massive limestone with spotty and white barite.
BR-02	10552N/10098E	Reddish white massive barite with minor pyrite, hematite, and calcite.
BR-03	10479N/10111E	
BR-04	10400N/10097E	
BR-05	10328N/10091E	White laminated barite with minor pyrite, hematite, and calcite.
BR-06	10288N/10088E	
LS-01	10624N/10067E	Light gray massive limestone with some barite, pyrite, and hematite.
LS-02	10558N/10087E	
LS-03	10502N/10092E	
LS-04	10434N/10094E	
LS-05	10244N/10053E	
LS-06	10181N/10044E	

5.2 Laboratory Tests

All rock samples are prepared and tested at the Suranaree University of Technology to obtain the uniaxial compressive strength (σ_C) and Brazilian tensile strength (σ_B).

5.2.1 Uniaxial compressive strength tests

The uniaxial compressive strength test is carried out on six specimens from the barite-bearing zone and six specimens from the limestone footwall. The sample preparation and test procedures follow the methods given by the International Society for Rock Mechanics (ISRM) Commission on Standardization of Laboratory and field Tests (Brown, 1981), and the applicable ASTM standard (ASTM D2938-86). The cylindrical test specimens are drilled, cut, and ground to have a length-to-diameter ratio of 2.5 with a diameter of 54 mm. All specimens are loaded by the compression machine model ELE-ARD2000 with capacity of 2000 kN (Figure 5.2). The specimens are loaded axially to failure under a constant stress rate of 0.5 to 1.0 MPa/s. The post-failure characteristics are observed and recorded (Figures 5.3 through 5.7). The specimens are failed under three modes of failure. There are extension failure mode (Figures 5.3 and 5.4), the shear failure (Figures 5.5 and 5.6), and the compressive shear failure mode (Figure 5.7). The uniaxial compressive strength (σ_C) is calculated by dividing the failure load (P) by the original cross-section area (A),

$$\sigma_C = P/A \qquad (5.1)$$

5.2.2 Brazilian tensile strength tests

A total of twenty specimens are prepared for the Brazilian tensile strength testing. Ten specimens are from the blocks collected from the barite-bearing zone and ten are from the limestone footwall. The sample preparation and test procedures follow the applicable ASTM standards (ASTM D3967-81) and ISRM suggested method (Brown, 1981). The specimens have a thickness-to-diameter ratio of 0.5 with a diameter of 54 mm (Figure 5.8). They are loaded diametrically by the compression machine model ELE-ARD2000 with capacity of 2000 kN. Each disk is diametrically loaded to failure with the loading rate about 200 N/s. The post-failure characteristics are observed and recorded (Figures 5.9 and 5.10). All specimens are failed along the loading diameter. The tensile strength is calculated by

$$\sigma_B = 2P/\pi Dt \tag{5.2}$$

where σ_B is the Brazilian tensile strength, P is the failure load, D is the disk diameter, and t is the disk thickness.

5.3 Testing results

The compressive strength of the barite-bearing specimens varies from 39.4 MPa to 87.6 MPa (Table 5.2). The average value is 61.3 MPa. The standard derivation is 18.1 MPa. The compressive strength of limestone specimens varies from 56.8 MPa to 100.7 MPa (Table 5.3). The average value is 82.5 MPa . The standard derivation is 16 MPa.

The tensile strength of the barite-bearing specimens varies from 3.7 MPa to 9.7 MPa (Table 5.4). The average value is 5.8 MPa. The standard derivation is 1.9 MPa. The tensile strength of the limestone specimens varies from 6 MPa to 11.1 MPa. The average value is 8.6 MPa (Table 5.5). The standard derivation is 1.9 MPa.

Deere and Miller classification of intact rock strength (Hoek and Brown, 1980) shows that such rock types as limestone, sandstone, slate, shale normally have the uniaxial compressive strength ranging between 50 and 100 MPa (medium strength). The uniaxial compressive strength of the limestone specimens tested here agrees well with most limestone found elsewhere.

There is no correlation between the specimen densities and strengths. This means that higher density rock specimens (higher barite content) do not necessary have greater compressive or tensile strengths. Post-tested observations indicate that micro-cracks or fractures seem to have influence on the failure loads and on the characteristics of failure modes.

The high standard derivation of the strengths for both rock types may be caused by the intrinsic variability of the barite-bearing specimens and limestone specimens. This is also evidenced by the variation of rock density due to the non-uniform distribution of the barite ore in the host limestone and of micro-cracks and fractures in both rock types.

Table 5.2 Results of uniaxial compressive strength test of the barite-bearing zone.

Specimen No.	Average Diameter (mm)	Average Length (mm)	Density (g/cc)	Fail load (kN)	σ_c (MPa)
BR-1-UN-1	54.00	134.00	3.90	101	43.9
BR-1-UN-2	53.92	135.17	3.50	200	87.6
BR-1-UN-3	53.98	138.33	3.42	140	61.2
BR-1-UN-4	53.98	138.17	3.74	170	74.3
BR-2-UN-5	53.88	136.00	4.40	140	61.6
BR-4-UN-6	53.92	135.83	4.35	90	39.4
		Maximum	4.40	200	87.6
		Minimum	3.42	90	39.4
		Mean	3.89	140	61.3
		S.D.	0.42	41.4	18.13

Table 5.3 Results of uniaxial compressive strength test of the limestone.

Specimen No.	Average Diameter (mm)	Average Length (mm)	Density (g/cc)	Fail load (kN)	σ_c (MPa)
LS-2-UN-1	53.93	137.33	3.14	220	96.3
LS-4-UN-2	54.00	134.83	3.08	170	74.2
LS-3-UN-3	53.93	136.00	3.00	230	100.7
LS-3-UN-4	53.97	136.00	3.00	200	87.7
LS-2-UN-5	53.87	134.83	3.16	180	79.2
LS-4-UN-6	53.97	135.33	3.20	130	56.8
		Maximum	3.20	230	100.7
		Minimum	3.00	130	56.8
		Mean	3.10	189	82.5
		S.D.	0.08	36.6	16.04

Table 5.4 Results of Brazilian tensile strength test of the barite-bearing zone.

Specimen No.	Average Diameter (mm)	Average Length (mm)	Density (g/cc)	Fail load (kN)	σ_B (MPa)
BR-3-BZ-1	53.90	26.85	4.23	12	5.1
BR-3-BZ-2	53.85	27.80	4.28	10	4.3
BR-3-BZ-3	53.95	26.85	4.27	9	3.7
BR-3-BZ-4	53.95	26.60	4.32	13	5.6
BR-3-BZ-5	53.88	27.38	4.30	12	5.2
BR-3-BZ-6	54.00	27.43	3.23	20	8.4
BR-3-BZ-7	53.83	26.88	3.73	14	5.9
BR-3-BZ-8	53.90	26.97	3.42	11	4.6
BR-3-BZ-9	53.98	26.87	3.26	22	9.7
BR-3-BZ-10	53.97	27.27	3.54	14	5.8
		Maximum	4.32	22	9.7
		Minimum	3.23	9	3.7
		Mean	3.86	13.4	5.8
		S.D.	0.47	4.2	1.85

Table 5.5 Results of Brazilian tensile strength test of the limestone.

Specimen No.	Average Diameter (mm)	Average Length (mm)	Density (g/cc)	Fail load (kN)	σ_B (MPa)
LS-3-BZ-1	53.95	23.78	2.98	21	10.2
LS-3-BZ-2	53.90	24.77	3.04	14	6
LS-3-BZ-3	53.88	24.60	2.99	20	9.6
LS-3-BZ-4	53.87	23.18	2.98	19	9.4
LS-3-BZ-5	53.87	23.55	2.96	17	8.5
LS-3-BZ-6	53.93	24.82	3.01	14	6.4
LS-3-BZ-7	53.93	25.25	3.00	21	9.6
LS-3-BZ-8	53.93	24.55	3.01	21	9.9
LS-3-BZ-9	53.92	24.95	2.93	24	11.1
LS-3-BZ-10	53.93	26.42	2.88	13	5.6
		Maximum	3.04	24	11.1
		Minimum	2.88	13	5.6
		Mean	2.98	17.9	8.6
		S.D.	0.04	3.9	1.94

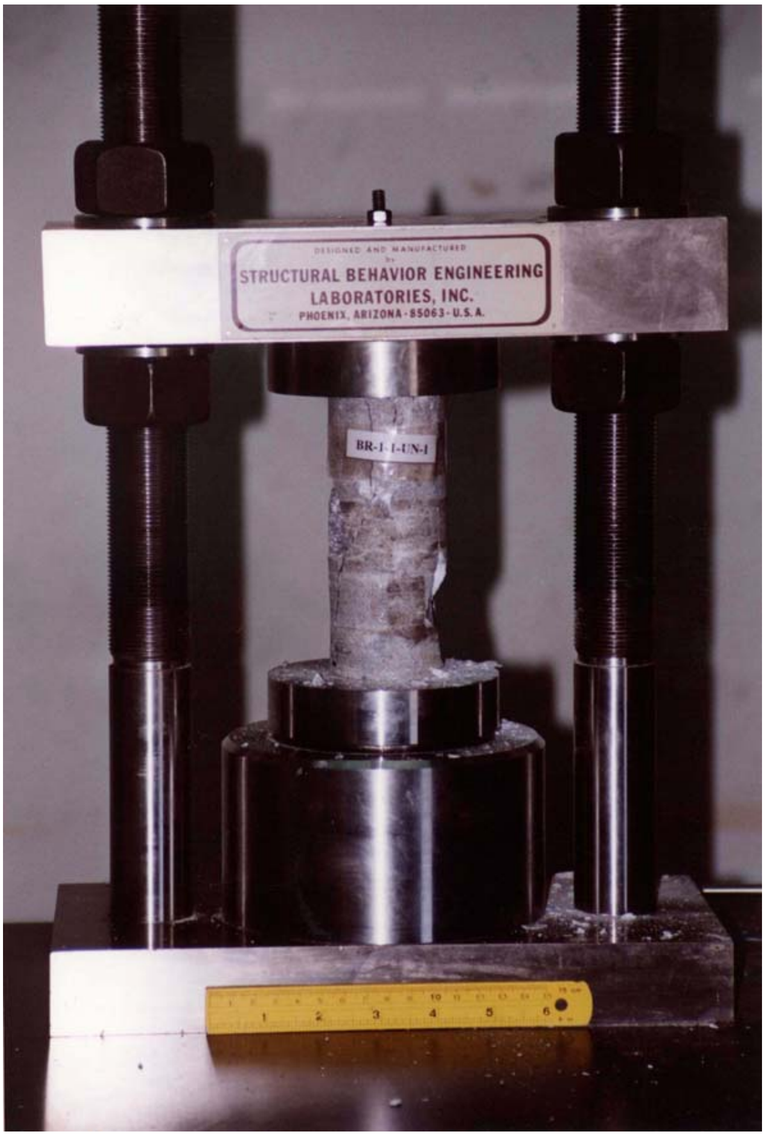

Figure 5.2 Uniaxial compressive strength testing of 54 mm diameter of cylinder barite specimen. The specimen is axially loaded in compression machine model ELE-ADR200.

Figure 5.3 Extension fractures occur along axis of the barite specimen in the uniaxial compressive strength test.

Figure 5.4 Extension fractures occur along axis of the limestone specimen in the uniaxial compressive strength test.

Figure 5.5 Shear failure plane with an angle about 22º from specimen axis occurs in the barite specimen after the uniaxial compressive strength testing.

Figure 5.6 Shear failure plane with an angle about 32º from specimen axis occurs in the limestone specimen after the uniaxial compressive strength testing.

Figure 5.7 Compressive shear failure occurs in the limestone specimen after the uniaxial compressive strength testing.

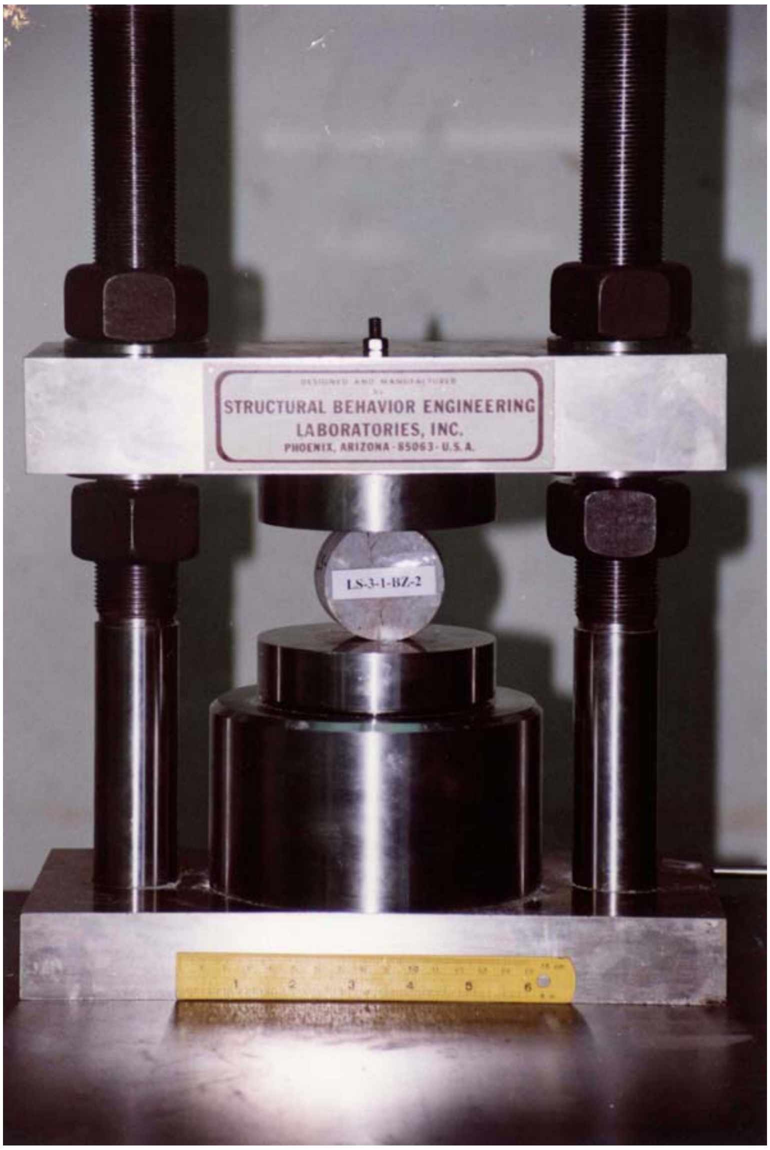

Figure 5.8 Brazilian tensile strength testing of 54 mm diameter disc barite specimen. It is diametrically loaded in compression machine model ELE-ADR200.

Figure 5.9 Diametrical failure of the barite specimens in Brazilian tensile strength testing.

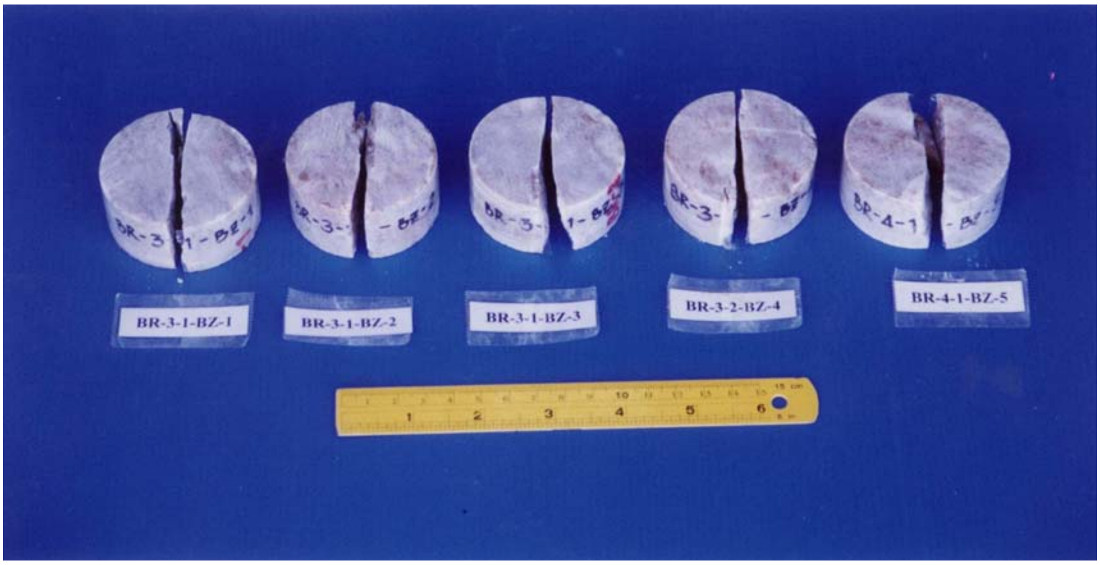

Figure 5.10 Diametrical failure of the limestone specimens in Brazilian tensile strength testing.

CHAPTER VI

ROCK MASS CHARACTERIZATION

The objective of rock mass characterization is to classify the rock in the study area in terms of the geotechnical characteristics. The results are used for analyzing the stability of the underground excavations in the preliminary mine design. The criteria for rock mass classification used here are the South African Council for Scientific and Industrial Research (CSIR) geomechanics classification (Beinaiwski 1976) and the tunneling quality index, Q, proposed by Baton, Lien and Lunde (Hoek and Brown, 1980).

6.1 Rock Quality Designation (RQD)

Rock quality designation (RQD) is the one of the parameters used for rockmass classification. The RQD is defined as the percentage of core recovered in intact pieces of 100 mm or more length in the total length of a core run. If the borehole core is unavailable, the RQD can be estimated by using the distribution of joint spacing as proposed by Priest and Hudson (Brady and Brown, 1985). The relationship is proposed as follows.

$$RQD = 100 \exp.^{(-0.1S)} (1+0.1S) \tag{6.1}$$

where S is the discontinuity frequency per meter.

Table 6.1 shows the results of RQD calculated from equation (6.1) for each rock zone in the study area.

Table 6.1 Estimating RQD of the rock masses for each rock zone in the study area.

Rock Zone	Joint orientation (strike/dip)	Joint spacing (m)	Discontinuities frequency		RQD (%)
			Joints/m	Sum	
(1) Barite_bearing and (2) Footwall limestone	143/29 282/83 356/71	0.50 0.50 0.20	3 3 6	12	66
(3)Footwall low fracturing shale	023/38 163/81 272/77	0.18 0.20 0.20	6 6 6	18	46
(4) Footwall high fracturing shale	143/29 225/48 282/83 356/71	0.15 0.13 0.19 0.15	7 8 6 7	28	23
(5)North Hangingwall high fracturing shale	204/22 142/47 341/63 066/88	0.10 0.15 0.13 0.10	11 7 8 11	37	11
(6)South Hangingwall high fracturing shale	201/31 354/81 178/86 176/87	0.20 0.20 0.10 0.10	6 6 11 11	34	14

6.2 Rock mass classification

The rock mass classification to be performed here is based on two commonly used systems of rock mass classification; CSIR geomechanics classification and Q system.

6.2.1 CSIR Geomechanics classification

The CSIR classification of jointed rock masses uses Rock Mass Rating (RMR) that is calculated from the rating values of five parameters (Hoek and Brown, 1980). The five basic classification parameters are uniaxial compressive strength, RQD, spacing of joints, joint conditions and groundwater condition.

The uniaxial compressive strength of the intact rocks in the barite-bearing and footwall limestone zones is obtained from laboratory testing (Tables 5.2 and 5.3 chapter V). The uniaxial compressive strength of the other rock zones is obtained from field estimation as described in chapter IV. The groundwater condition is examined from the local hydrological map and artesian wells. The groundwater has no effect on the study area. However, the rock masses contain joints into which the rainwater can infiltrate. The groundwater condition is therefore considered in the rock mass classification scheme.

The calculated RMR for each rock zone is given in Table 6.2. The RMR of barite-bearing and footwall limestone zones is 53 indicating the rock mass in range of class III with a description of fair. The RMR of footwall low fracturing shale zone is 35 indicating class IV with a description of poor. The RMR of footwall high fracturing shale zone and hanging wall high fracturing shale zone is equal 33 also indicating class IV with a description of poor.

6.2.2 NGI tunneling quality index

The NGI tunneling quality index is proposed by Baton et al. (1974). The value of quality index "Q" is defined as:

$$Q \quad = \quad \frac{RQD}{J_n} * \frac{J_r}{J_a} * \frac{J_w}{SRF} \qquad (6.2)$$

where RQD is Rock Quality Designation, J_n is the joint set number, J_r is the joint roughness number, J_a is the joint alteration number, J_w is the joint water reduction factor, and SRF is a stress reduction factor.

The calculated Q index for each rock zone is given in Table 6.3. The Quality Index "Q" of the barite-bearing zone and footwall limestone zone is 0.55. The "Q" for the footwall low fracturing shale zone is 0.39. The "Q" for the footwall high fracturing shale zone is 0.12. The "Q" for the north hanging wall high fracturing shale zone is 0.06. The "Q" for the south hangingwall high fracturing shale zone is 0.07.

6.3 Discussions

The CSIR geomechanics classification shows that the rock masses in the study area are classified as fair to poor quality. The geomechanical properties of the rock masses can be utilized in terms of stand-up times for a given unsupported span and rock strength parameters. The barite-bearing and footwall limestone zones can have 3 m of unsupported span for 1 week of stand-up time. The cohesion is 159-200 kPa and the friction angle is from 35° to 40°. The low fracturing and high fracturing shale can have the average stand-up time up to 10 minutes for 0.5 m span. The cohesion is less than 100 kPa, and the friction angle is less than 30°.

The tunneling quality index Q shows that the barite-bearing and the footwall limestone zones are classified as fair quality. The footwall low fracturing shale is poor quality. The footwall high fracturing shale is very poor quality. The hanging wall high fracturing shale zone is extremely poor quality. The maximum unsupported span can also be calculated by the Index Q values.

The index Q can be related to the stability and support requirements of underground excavations. Barton et al. (1974) define an additional parameter called the equivalent dimension (D_e) of excavation. This dimension is obtained by dividing the span, diameter or wall height of excavation by a quantity called the excavation support ratio (ESR).

$$D_e = \frac{\text{Excavation span, diamerter or height (m)}}{\text{Excavation Support Ratio } ESR} \qquad (6.3)$$

ESR is related to the intended use of the excavation and to the degree of safety that is required for support system installed to maintain the stability of excavation. The ESR values of excavation category are suggested by Barton et al. (1974).

The D_e plotted against the value of index Q and used to define the maximum unsupported span of excavation in a chart published by Barton et al. (1974).

Table 6.2 CSIR Geomechanical classification of the rock masses in the study area.

	Rock zones	Classification parameters					Rating Score	Rock mass classification
		Rock Strength (MPa)	RQD (%)	Joint Spacing (m)	Condition of joints	Ground water		
1	Barite-bearing	61.3	66	0.20 to 0.50	Slickensided surface, joint open 1 to 5 mm, continuous joint	Moist only		Class No. III Fair rock
	Rating	7	13	20	6	7	53	
2	Footwall limestone	82.5	66	0.20 to 0.5	Slickensided surface, joint open 1 to 5 mm, continuous joint	Moist only		Class No. III Fair rock
	Rating	7	13	20	6	7	53	
3	Footwall low fracturing shale	25 to 50	46	0.18 to 0.20	Irregular surface, joint open 1 to 5 mm, continuous joint	Moist only		Class No. IV Poor rock
	Rating	4	8	10	6	7	35	
4	Footwall high fracturing shale	25 to 50	23	0.13 to 0.19	Irregular surface, joint open 1 to 5 mm, continuous joint	Moist only		Class No. IV Poor rock
	Rating	4	8	10	6	7	35	
5	Hanging wall high fracturing shale (North)	25 to 50	11	0.10 to 0.15	Slickensided surface, joint open 1 to 5 mm, continuous joint	Moist only		Class No. IV Poor rock
	Rating	4	3	10	6	7	30	
6	Hanging wall high fracturing shale (South)	25 to 50	14	0.10 to o.20	Slickensided surface, joint open 1 to 5 mm, continuous joint	Moist only		Class No. IV Poor rock
	Rating	4	3	10	6	7	30	

Table 6.3 NGI Tunneling Quality Index of the rock mass in the study area.

Rock Zone		The description parameters						Index Q
		RQD%	Num. of joints (Jn)	Joint roughness (Jr)	Joint alteration (Ja)	Joint water (Jw)	SRF	
1	Barite-bearing zone	Fair	3 sets	Slickensided and undulating	Low friction clay coating	Minor inflow	Loose open joint, heavily joint	
	Value	66	9	1.5	4	1	5	0.55
2	Footwall limestone	Fair	3 sets	Slickensided and undulating	Low friction clay coating	Minor inflow	Loose open joint, heavily joint	
	Value	66	9	1.5	4	1	5	0.55
3	Footwall low fracturing shale	Fair	3 sets	Rough or irregular, planar	Low friction clay coating	Minor inflow	Loose open joint, heavily joint	
	Value	46	9	1.5	4	1	5	0.39
4	Footwall high fracturing shale	Very poor	4 sets	Rough or irregular, planar	Low friction clay coating	Minor inflow	Loose open joint, heavily joint	
	Value	23	15	1.5	4	1	5	0.12
5	North hanging wall high fracturing shale	Very poor	4 sets	Slickensided and undulating	Low friction clay coating	Minor inflow	Loose open joint, heavily joint	
	Value	11	15	1.5	4	1	5	0.06
6	South hanging wall high fracturing shale	Very poor	4 sets	Slickensided and undulating	Low friction clay coating	Minor inflow	Loose open joint, heavily joint	
	Value	14	15	1.5	4	1	5	0.07

CHAPTER VII

EXCAVATION SUPPORT AND DESIGN

The objective of excavation support design is to select a suitable underground mining method for the geological and geotechnical conditions of barite deposit and related host rocks. The support components are selected for the unique characteristics of the rock masses in which their are placed, within the limits imposed by safety, technology, and economic, to yield the lowest cost of mine production.

7.1 Mining method selection

The narrow width with steeply inclined barite deposit at the site favors to the underground mining method, such class as the shrinkage stoping and sublevel open stoping (Peters, 1978, Brady and Brown, 1985, Hartman, 1987). Both underground mining methods are similar in mine layout, development stage and operation, except the ore breaking method for exploitation stage. The ore breaking in shrinkage stope is an overhand method in which the ore is mined in horizontal slices and remains in the stopes as temporary support to the wall and provide a working platform for miners, which is unsafe for non-skilled labors. For sublevel stoping method, drill and blast crews work in the protective cover of sublevel tunnel or crosscut. Therefore, the sublevel open stoping method is selected for developing the barite deposit in the study area because of the safety reason.

The components for sublevel open stoping method to be considered here are the main entry or access tunnels, sub-entries or sublevel, stope panels, and drawing points with crosscut at the haulage level.

7.2 Excavation design tasks

All the mine components will be planned and designed for their location, direction, shape, size, excavation method, and rock supports to mine out the barite deposit with the elevation ranging between 300 m and 550 m from the mean sea level. Such excavations as access tunnel, sublevel and crosscut will be horizontally run into the rock masses. All excavation components will be set in the barite vein and footwall zone. The unsupported excavation spans of these rock masses assigned by NGI Tunneling Quality Index Q are 2.3 m for the barite-bearing and footwall limestone, 2 m for the low fracturing shale, and 1.3 m for the high fracturing shale.

The access tunnel, sublevel, and crosscut are placed between contour intervals of 350 m and 550 m and lie parallel to the strike of the barite vein. The tunnel geometry will be designed by considering such factors as its function and facility for working and transportation. The unsupported span will be analyzed using the stereographic projection methods to assess the potential fall or slide of wedge blocks from the surrounding rock masses.

The proposed excavation method will be drill-and-blast method. The blastholes are drilled with the 38-mm diameter bit. The drillhole pattern will follow empirical rules for tunnel blasting developed by Langefore and Kihlstrom (Persson et al. 1994). Ammonium nitrate with fuel oil (AN-FO) ratio of 94:6 (by weight) is used and Dynamite with millimeters caps (MS) and half-second caps (HS) are used for ingredient of the primer for firing. The specific charges are determined from the related graph between tunnel area and specific charge for 38 mm diameters of drillhole (Olofsson, 1988).

The rock masses in which the tunnels are driven have joint sets with narrow spacing. The joints in surrounding rock masses may potentially fall or slide into the excavation. Rock bolts will be selected to apply reinforcement for the unstable wedge, and wire mesh will be installed to protect the unraveling of small piece of rocks (Hoek and Brown, 1980). The techniques that are used for the design of the rock bolts will be the empirical rules suggested by Lang (Hoek and Brown, 1980, Douglas and Arthur, 1983).

7.3 Main entry or access tunnel

The access tunnel will be excavated in the footwall zone that consists of low fracturing shale, limestone, and high fracturing shale zone. The portal position will be placed at an elevation of 350 m with a grid location of 11000N/10116E. The tunnel drives horizontally from the north to south along the direction line from 336º to 156º and from 357º to 177º with 1023 m long. The tunnel axis will be parallel to the strike of barite vein with an offset distance of 15 m apart.

The horseshoe shaped tunnel has a width of 5 m, abutment height of 2.5 m, and radial curvature at roof of 2.5 m. The tunnel cross-sectional area is 22.3 m^2. The blasting pattern has an advance face of 2 m, blasting factor of 0.33 m^2/hole, and specific charge of 1.5 kg/m^3 (Figure 7.1).

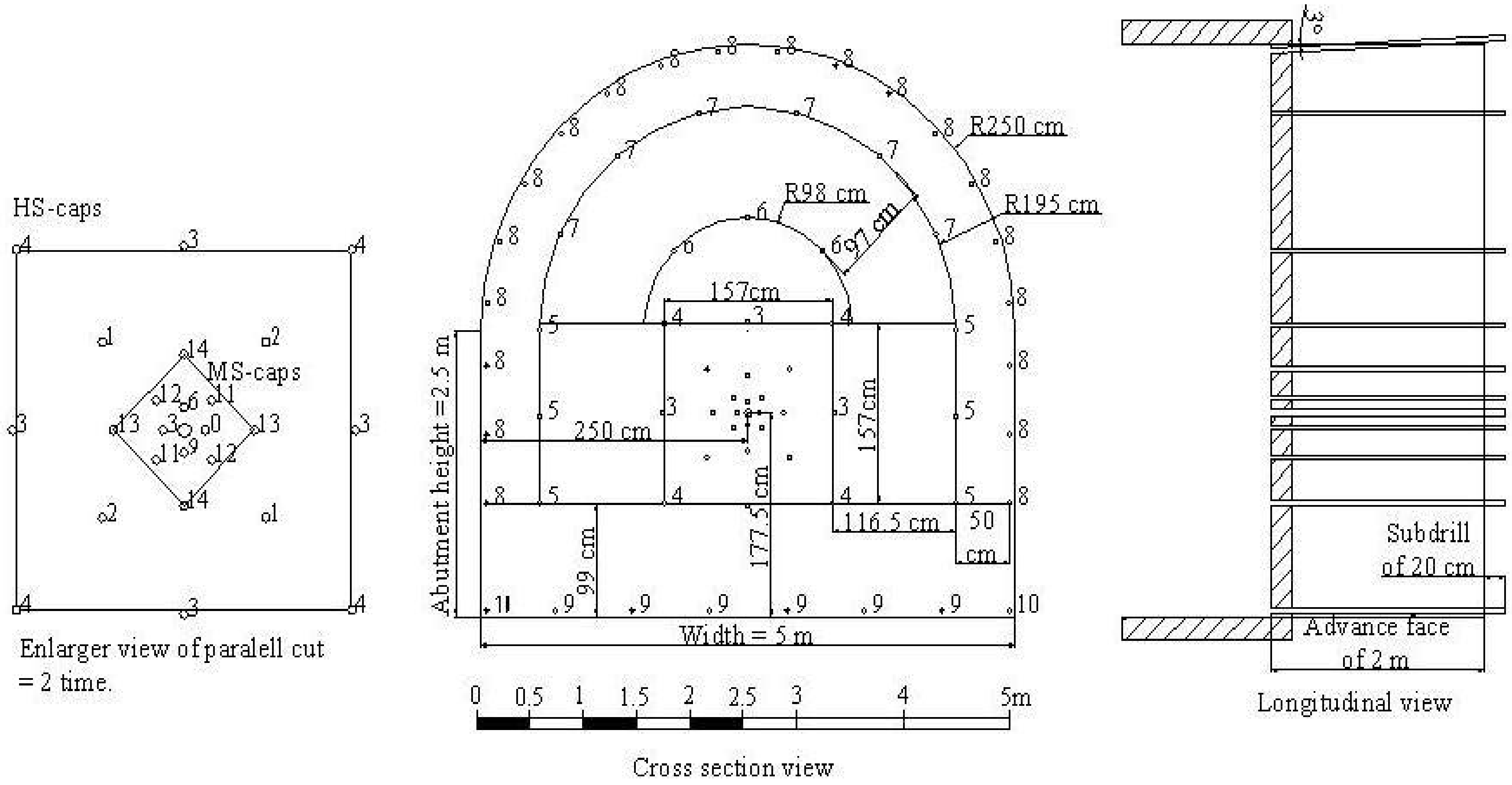

Figure 7.1 Drilling pattern for an access tunnel; MS stands for msec caps (no. 4 =100 ms) and HS stands for half-sec caps (no. 1 = 0.5 s)

The behavior of surrounding rock masses suggests potentially fall and slide of rock blocks into the excavation. The tunnel width of 5 m is reinforced by the 25 mm diameter and 2.5 m long grouted rockbolts. They are installed with spacing varied from 0.6 to 1.2 m. Wire mesh with the opening of 15 x 15 cm is installed at the arch roof to prevent the falling of small pieces of loosen rocks.

In the footwall low fracturing shale zone, it is found that rock wedges potentially fall into the excavation from the roof and slides, along joint #3 at the northeastern wall and along the intersection line of joint #1 and #2 at the southwestern wall (Figure 7.2). These joints have spacing of 0.18 m and 0.20 m. Ten grouted rockbolts are designed to install with 0.75 m grid spacing to support the excavated tunnel (Figure 7.3).

In the footwall limestone zone, the rock wedges potentially fall from the roof and the northeastern side wall. The sliding occurs along the intersection line of joint #2 and #3 (Figures 7.4). These joints have spacing of 0.5 m. Seven grouted rockbolts are installed with 1.20 m grid spacing to support the excavated tunnel (Figures 7.5).

In the footwall high fracturing shale zone, rock wedges potentially fall from the roof and the sliding occurs along joint #1 at the northeastern wall, and along the intersection line of joint #3 and #4 at the southwestern wall (Figure 7.6). These joints have spacing of 0.15 m. Eleven grouted rockbolts are installed with 0.6 m grid spacing to support the excavated tunnel (Figures 7.7).

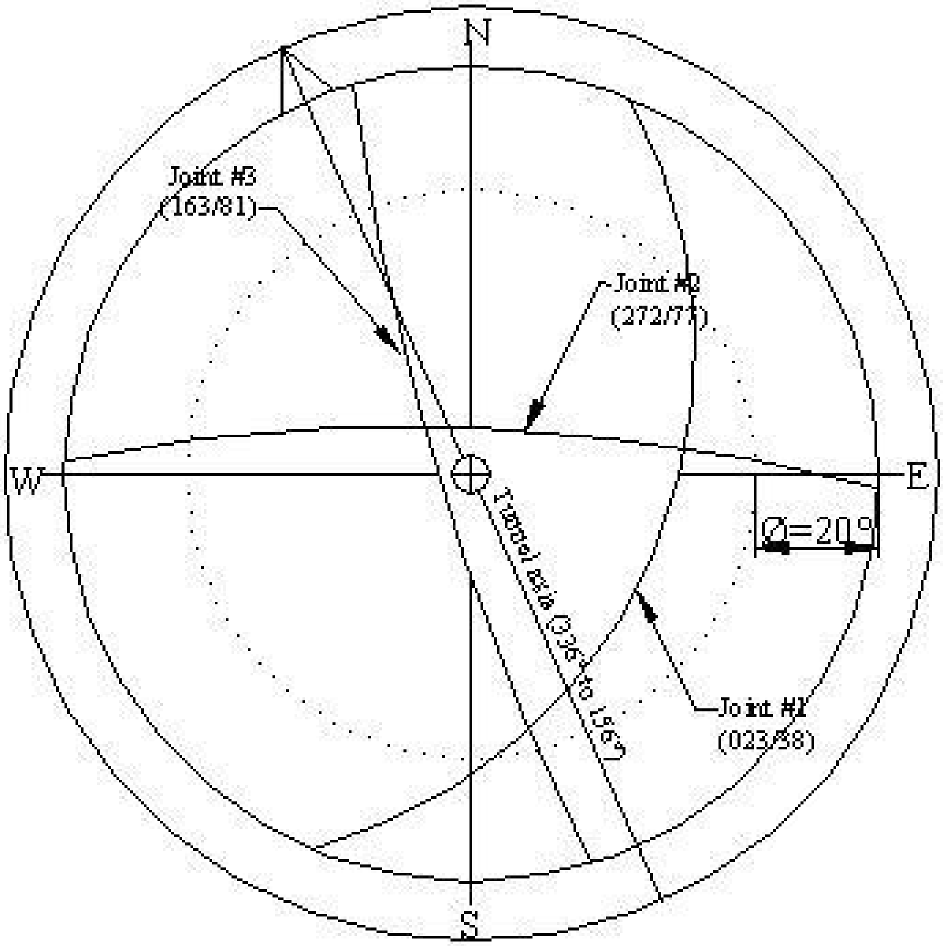

Figure 7.2 Stereographic projection of footwall low fracturing shale zone showing the tetrahedral wedge falling from the roof, sliding along joint #3 at the northeastern wall and along the intersection line of joint #1 and #2 at the southwestern wall.

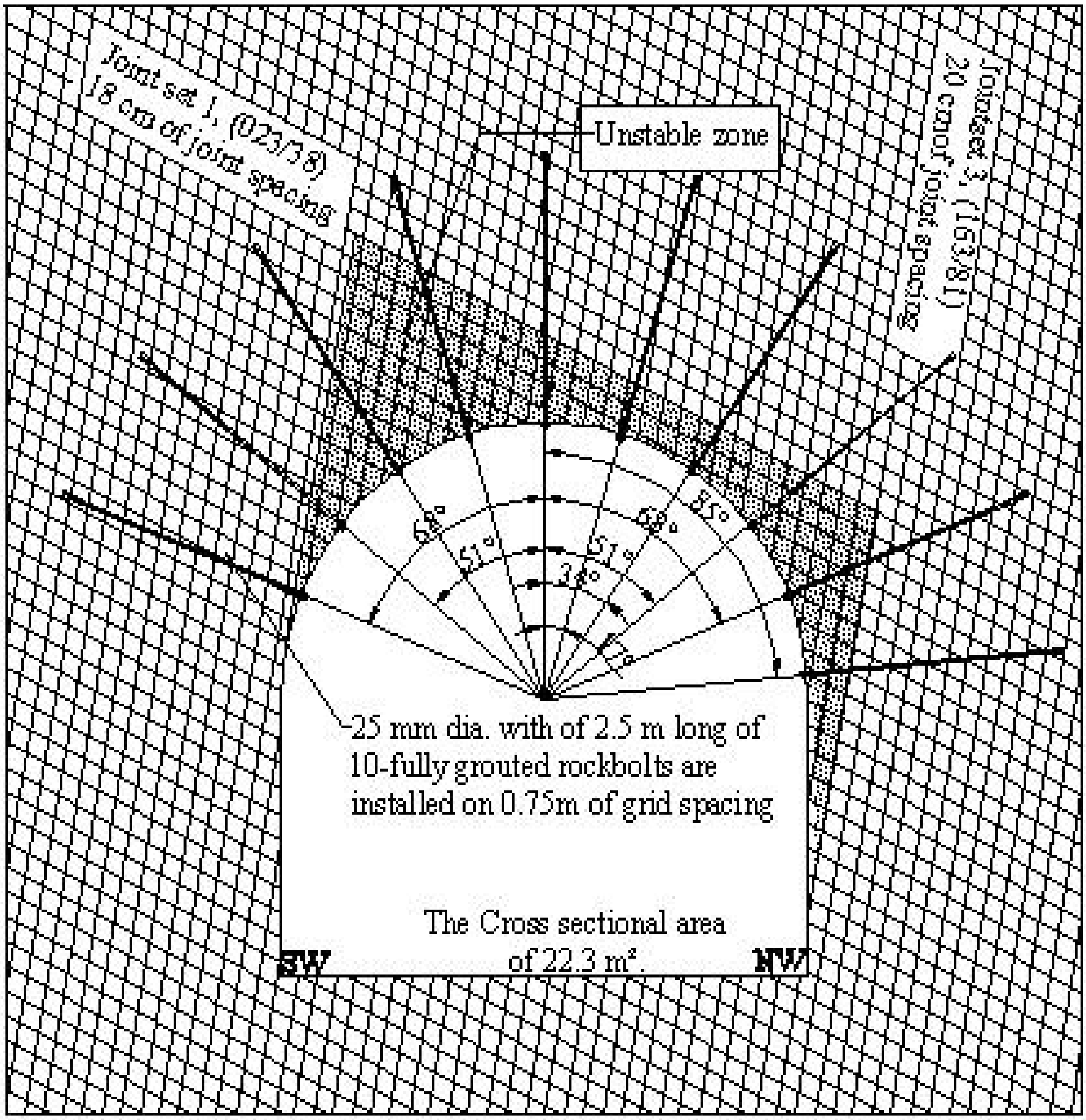

Figure 7.3 Preliminary layout of rockbolt pattern in access tunnel in the footwall low fracturing shale zone.

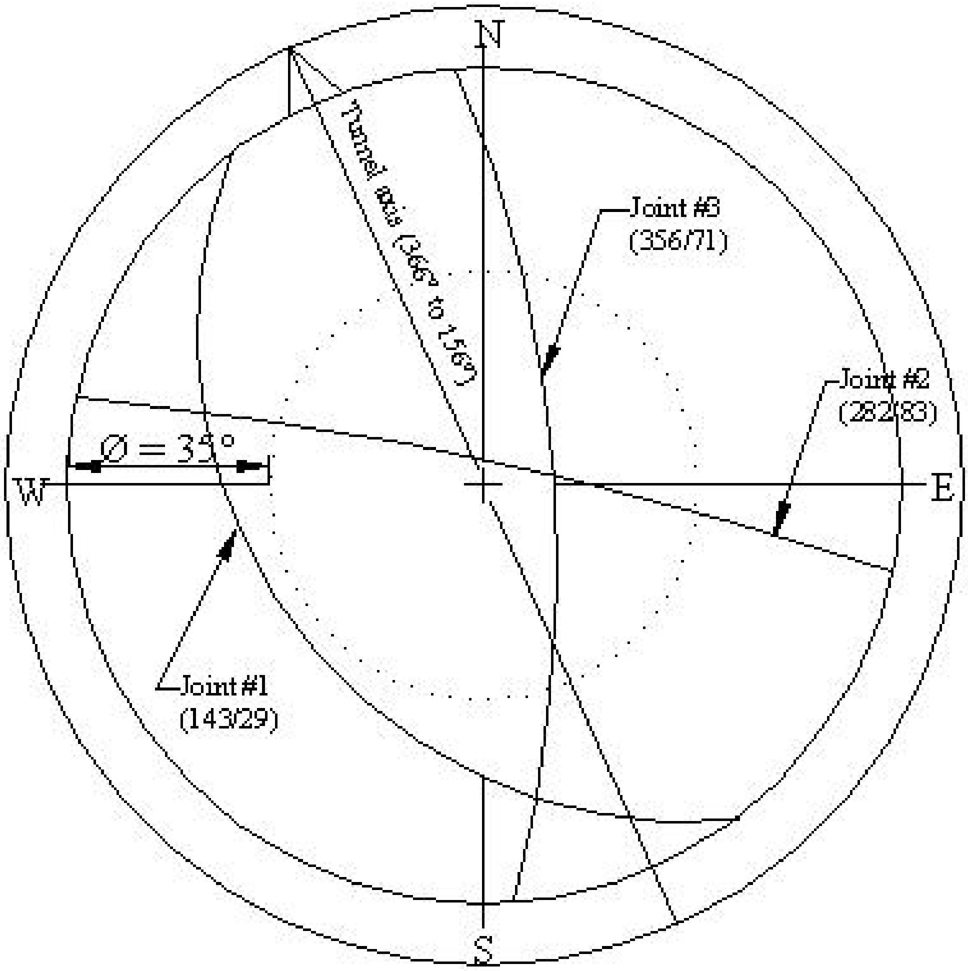

Figure 7.4 Stereographic projection of barite-bearing and footwall limestone zones showing the tetrahedral wagde falling from the root, stable at the northeastern wall and sliding along the intersection line of joint #2 and #3 at the southwestern wall.

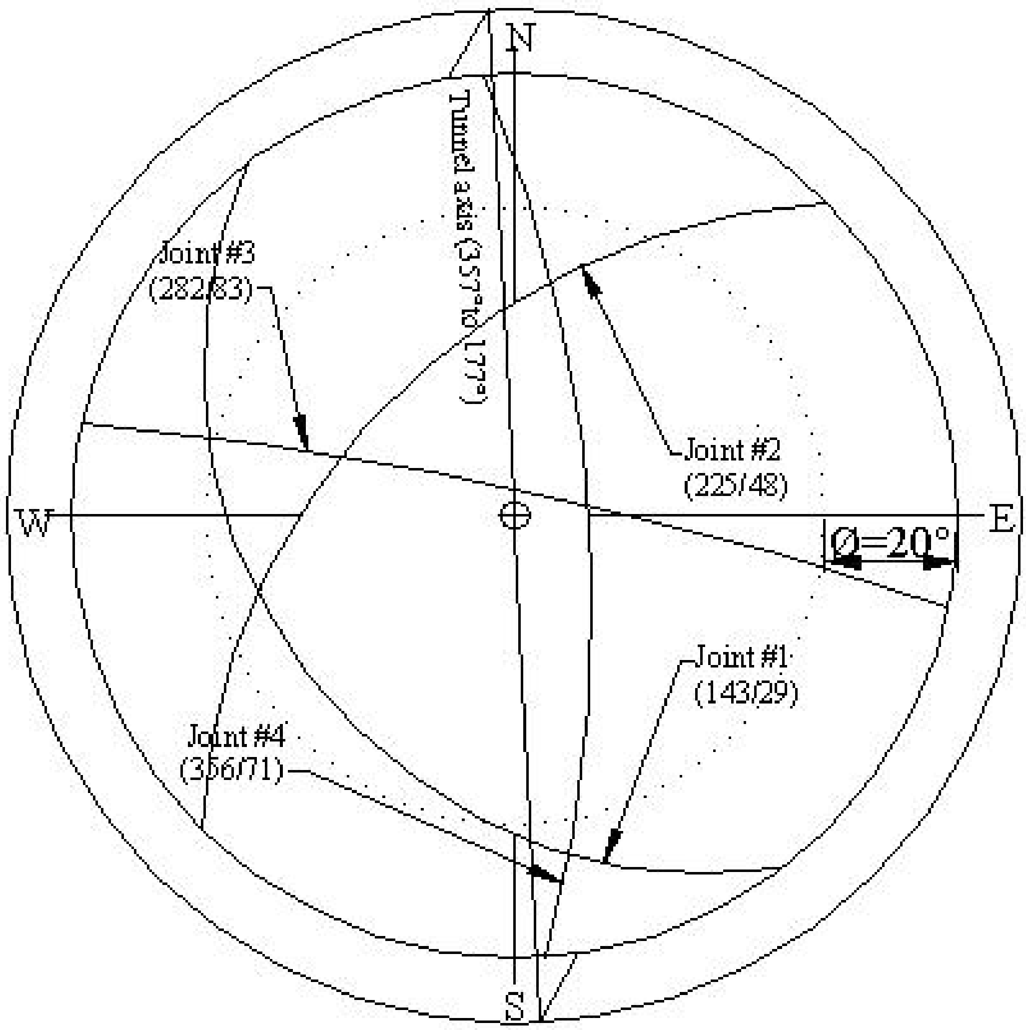

Figure 7.5 Stereographic projection of footwall high fracturing shale zone showing the tetrahedral wedge falling from the roof, sliding long joint #1 at the northeastern wall and along the intersection line of joint #3 and #4 at the southwestern wall.

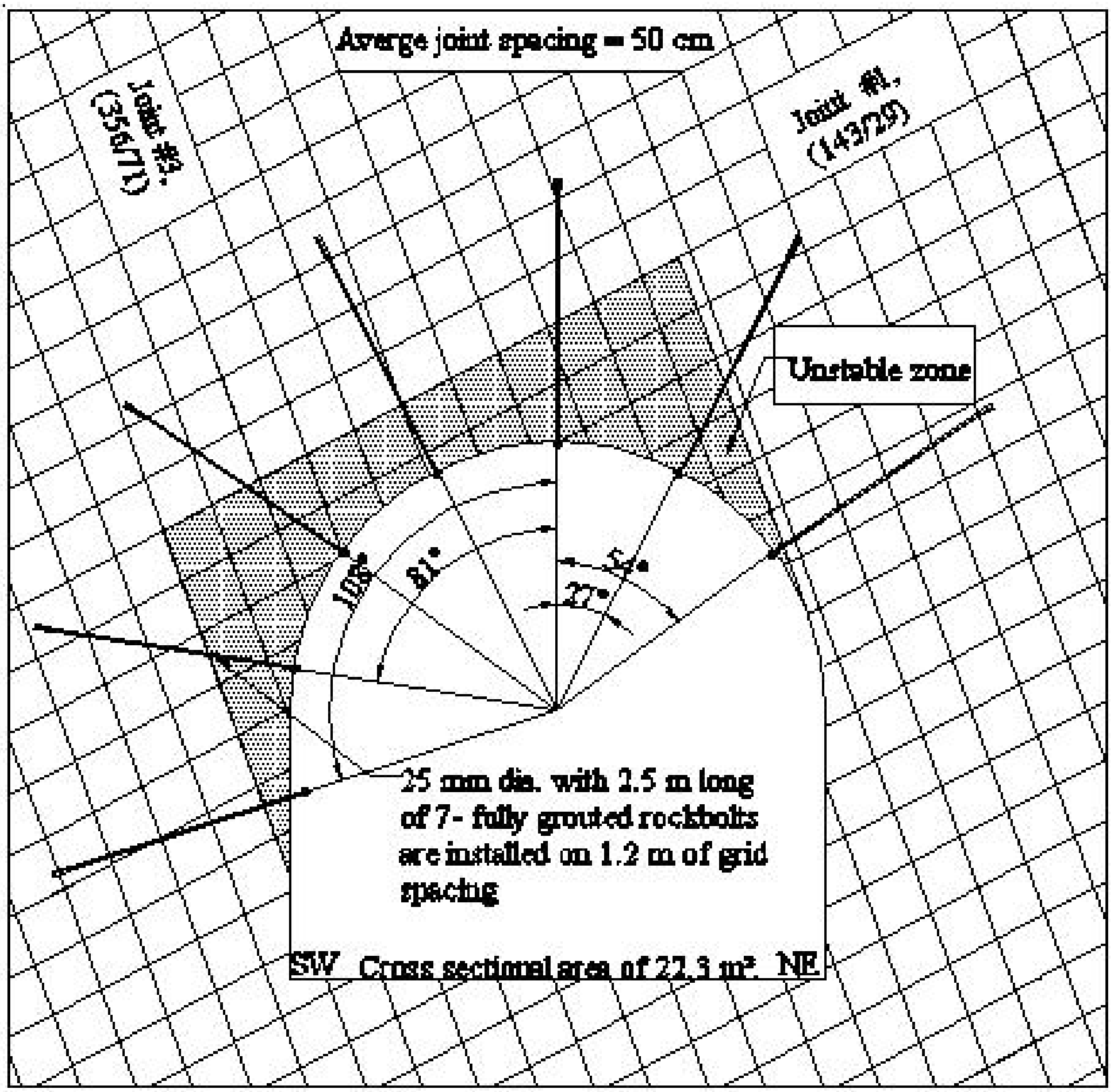

Figure 7.6 Preliminary layout of rockbolt pattern in access tunnel in footwall limestone zone.

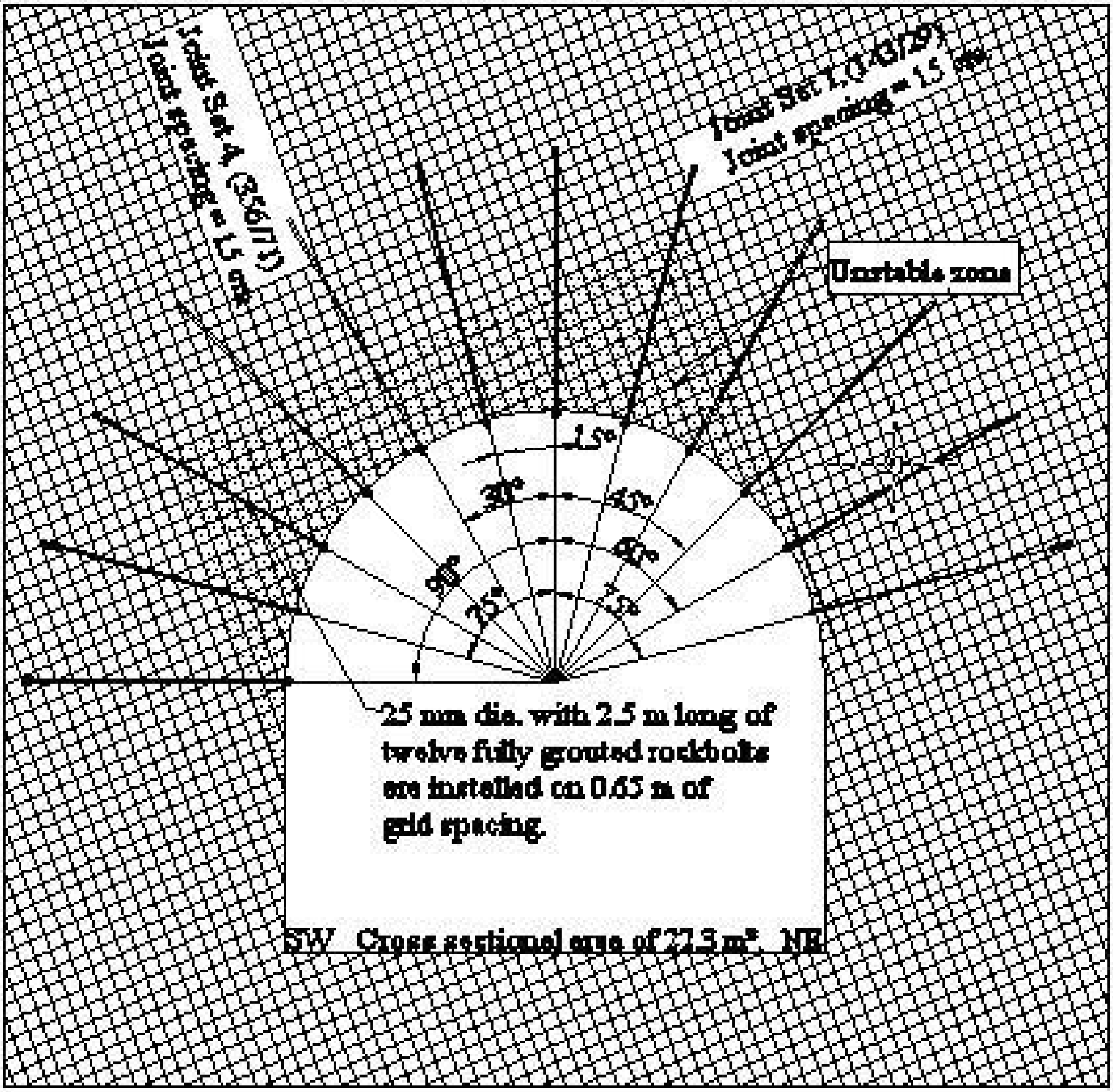

Figure 7.7 Preliminary layout of rockbolt pattern in access tunnel in footwall high fracturing shale zone.

7.4 Sub-entry or sublevel tunnel

The sublevel tunnels will be excavated in the footwall low fracturing shale and in the barite-bearing zone. The seven sublevel tunnels with a vertical offset distance of 20 m are horizontally excavated from the north to south along the direction line from 336º to 156º and from 357º to 177º. The positions and length of tunnels are as follows.

For the first tunnel, the portal is placed at an elevation of 350 m at local grid of 11000N/10134E. The length of the tunnel is 1030 m.

For the second tunnel, the portal is placed at an elevation of 370 m at local grid of 10963N/10130.5E. The length of the tunnel is 984 m.

For the third tunnel, the portal is placed at an elevation of 390 m at local grid of 10938N/10127E. The length of the tunnel is 950 m.

For the fourth tunnel, the portal is placed at an elevation of 410 m at local grid of 10891N/10123.5E. The length of the tunnel is 913 m.

For the fifth tunnel, the portal is placed at an elevation of 430 m at local grid of 10856N/10120E. The length of the tunnel is 877 m.

For the sixth tunnel, the portal is placed at an elevation of 450 m at local grid of 10860N/10116.5E. The length of the tunnel is 834 m.

For the seventh tunnel, the portal is placed at an elevation of 470 m at local grid of 10714N/10113E. The length of the tunnel is 813 m.

The horseshoe shaped tunnel has a width of 3.5 m, abutment height of 1.75 m, and radial curvature at the roof of 1.75 m. The tunnel cross-sectional area is 11 m^2. The blasting pattern has advance face of 2 m, blasting factor of 0.24 m^2/hole, and specific charge of 2.2 kg/m^3 (Figure 7.8).

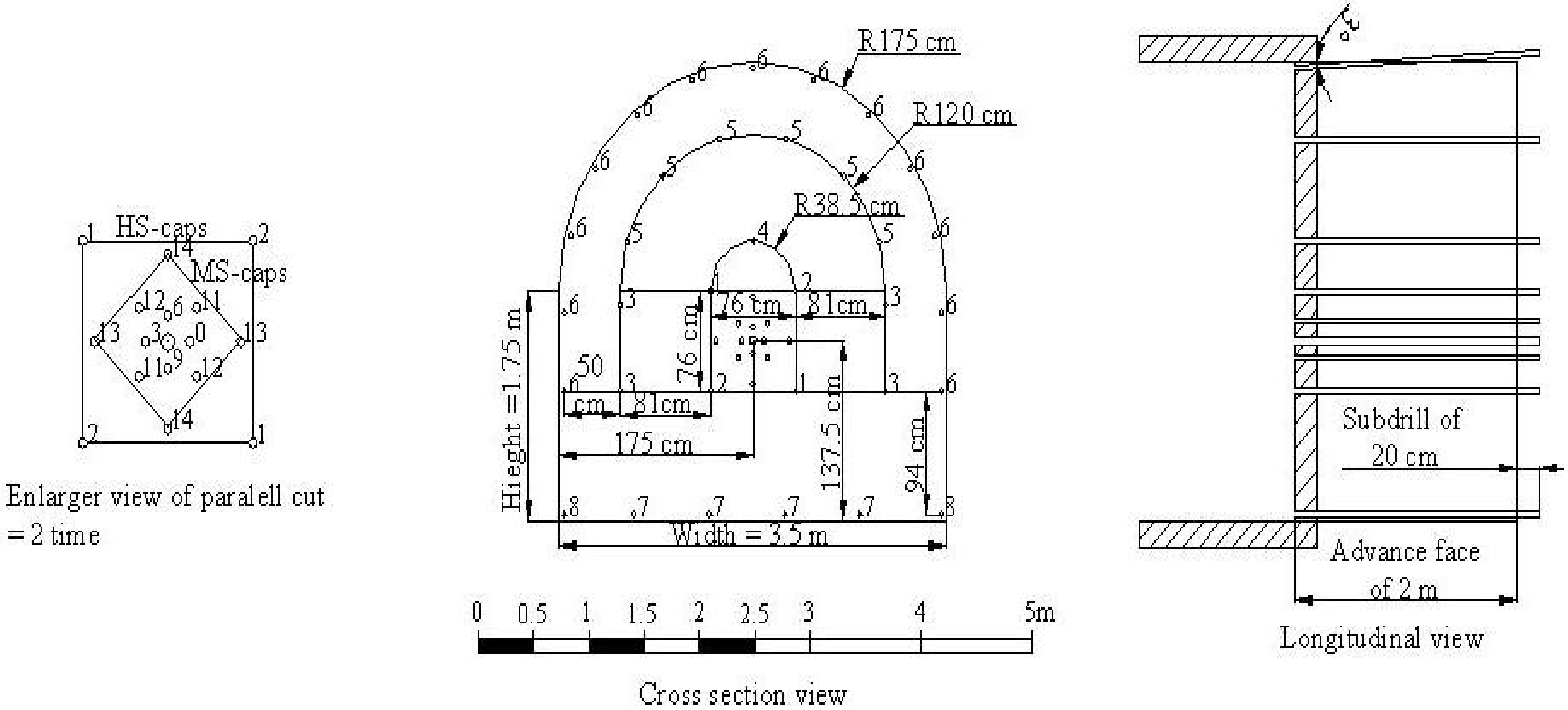

Figure 7.8 Drilling pattern for the sublevel; MS stands for msec caps (no. 4 =100 ms) and HS stands for half-sec caps (no.1 = 0.5 s)

The behavior of surrounding rock masses suggests potentially fall and slide of rock blocks into the excavation. The tunnel width of 3.5 m is reinforced by the 25 mm diameter and 2 m long grouted rockbolts. They are installed with spacing varied from 0.75 to 1 m. Chainlink mesh with the opening of 15 x 15 cm is installed at the arch roof to prevent the falling of small pieces of loosen rocks.

In the footwall low fracturing shale zone, seven grouted rockbolts are designed to install with 0.75 m grid spacing to support the excavated tunnel (Figure 7.9).

In the barite-bearing zone, six grouted rockbolts are installed with 1 m grid spacing to support the excavated tunnel (Figures 7.10).

7.5 Crosscut

The crosscuts are excavated in the footwall limestone zone for connecting between the access tunnel and the drawing points. Twenty-four crosscuts with 15 m long are constructed with an offset distance of 20 m.

The horseshoe shaped tunnel has a width of 4 m, abutment height of 2 m, and radial curvature at the roof of 2 m. The tunnel cross-sectional area is 14.3 m^2. The blasting pattern has advance face of 2 m, blasting factor of 0.33 m^2/hole, and specific charge of 1.8 kg/m^3 (Figure 7.11).

The behavior of surrounding rock masses suggests potentially fall from the roof. The sliding occurs along joint #2 at the southwestern wall (Figure 7.12). The tunnel width of 4 m is reinforced by the 25 mm diameter and 2 m long grouted rockbolts. Seven grouted rockbolts are installed with 1 m grid spacing to support the excavated tunnel (Figures 7.13). Wire mesh with the opening of 15 x 15 cm is installed at the arch roof to prevent the falling of small pieces of loosen rocks

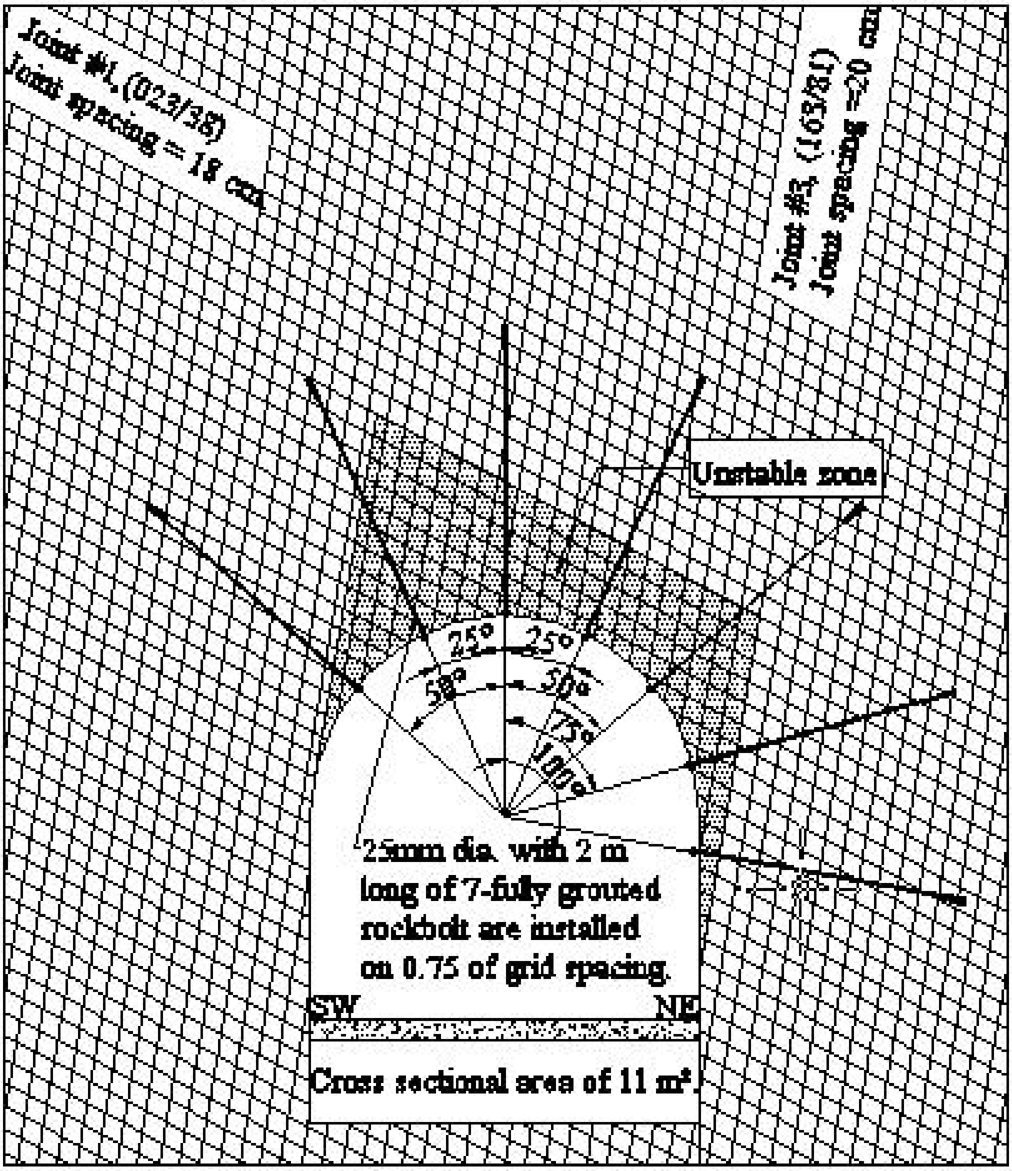

Figure 7.9 Preliminary layout of rockbolt pattern in sublevels in footwall

low fracturing shale zone.

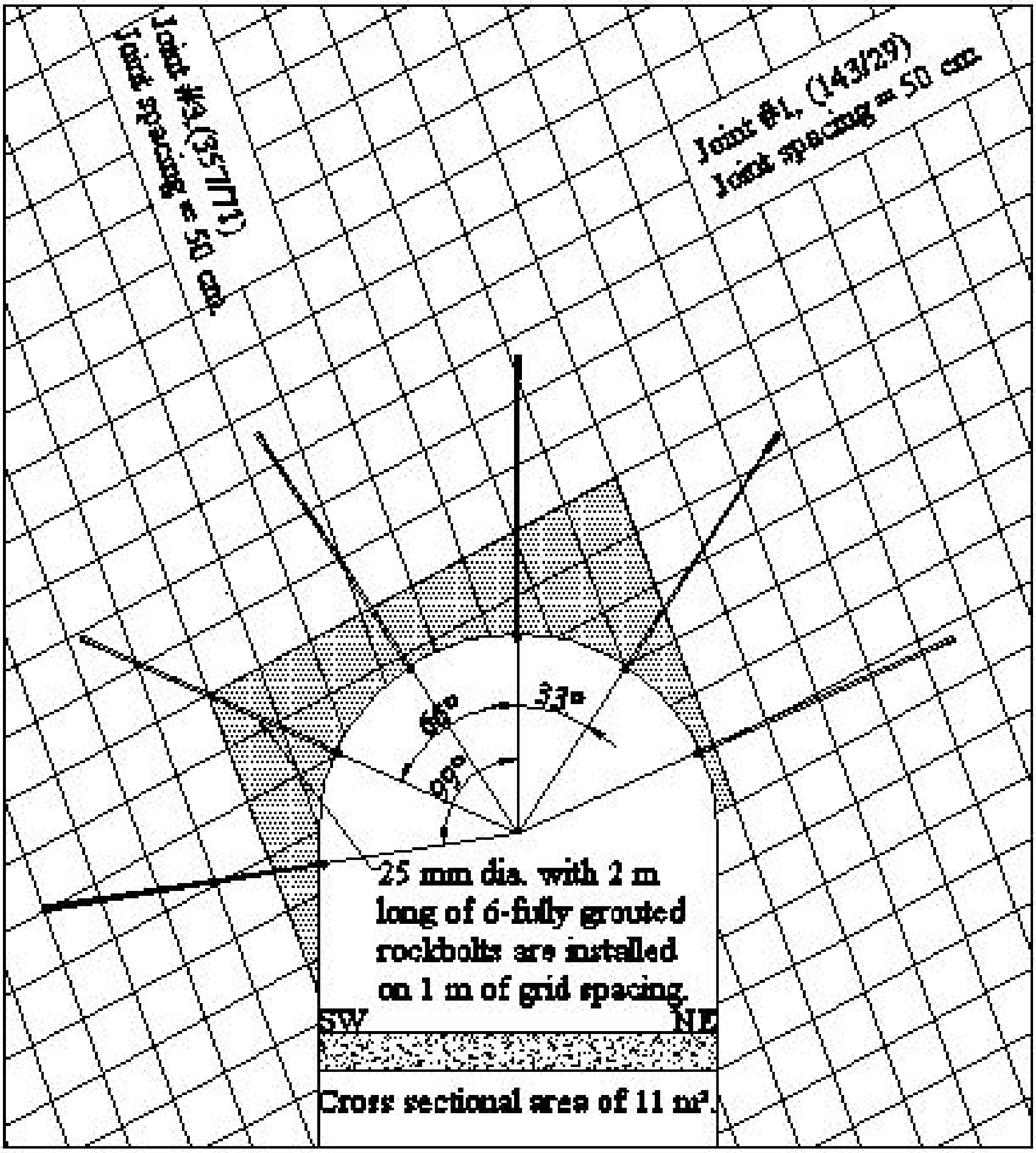

Figure 7.10 Preliminary layout of rockbolt pattern in sublevels in barite bearing zone.

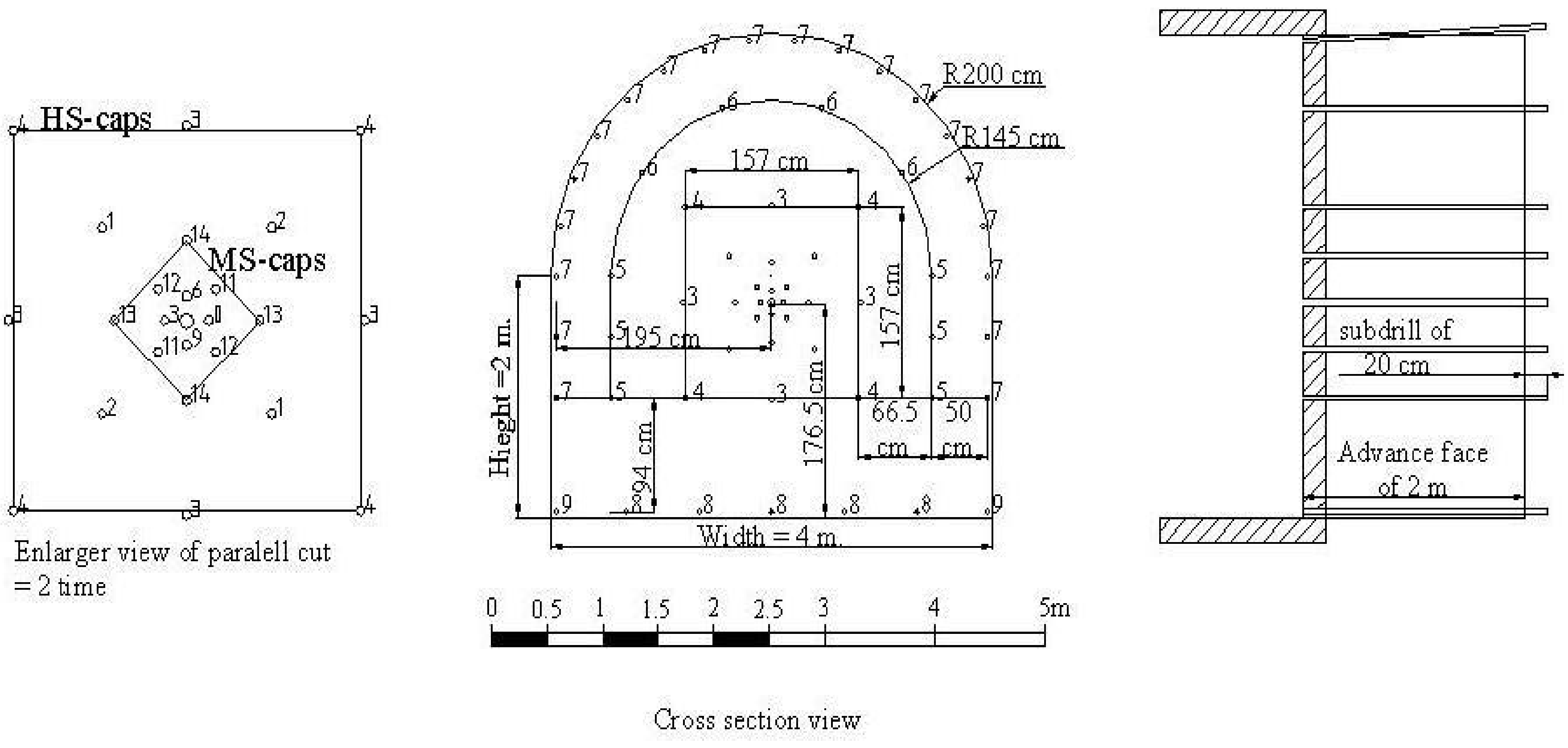

Figure 7.11 Drilling pattern for the crosscut; MS stands for msec caps (no. 4 =100 ms) and HS stands for half-sec caps (no.1 = 0.5 s)

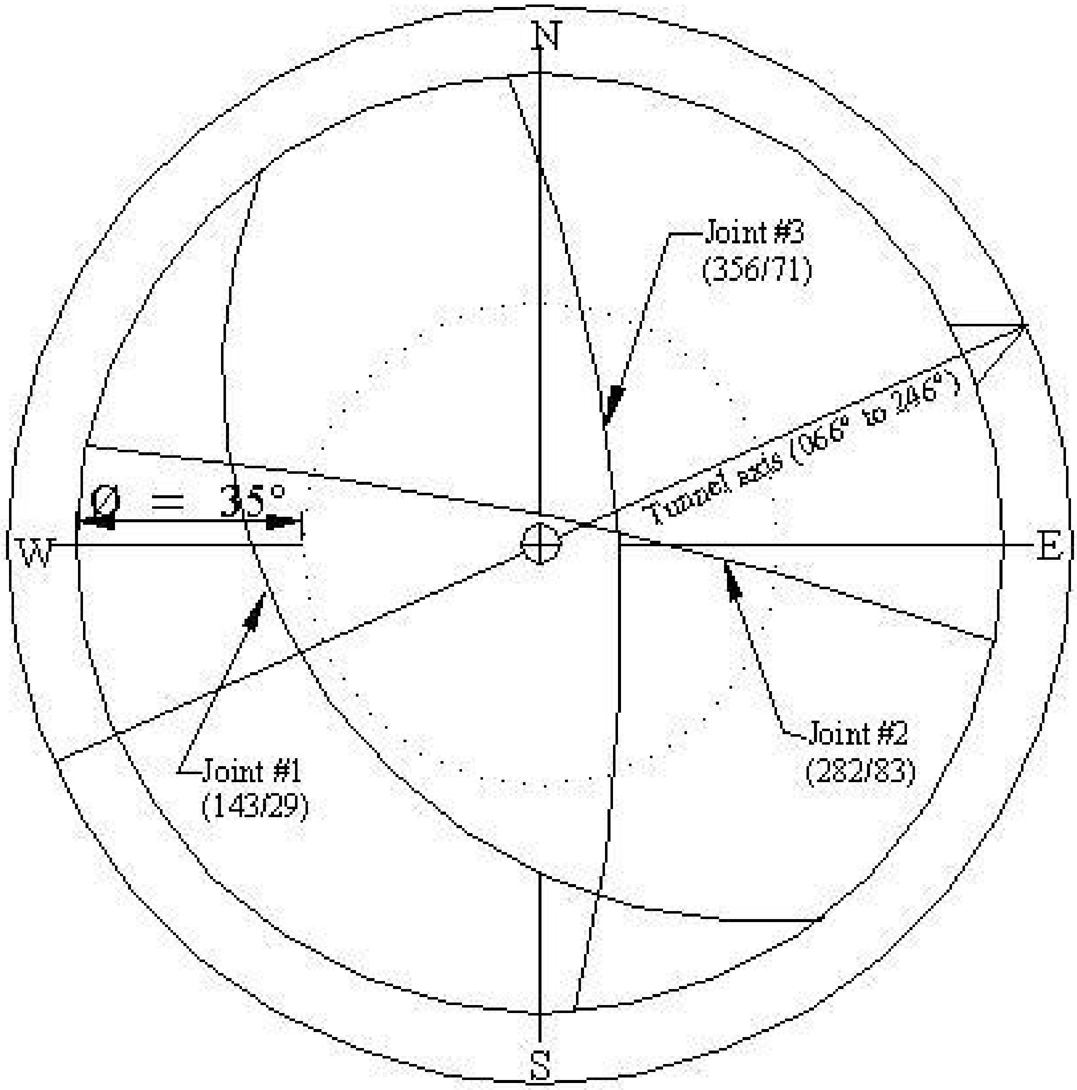

Figure 7.12 Stereographic projection of footwall limestone zone in crosscut showing tetrahedral wedge falling from the roof, stable at the northwestern wall and sliding along joint #2 at the southwestern wall.

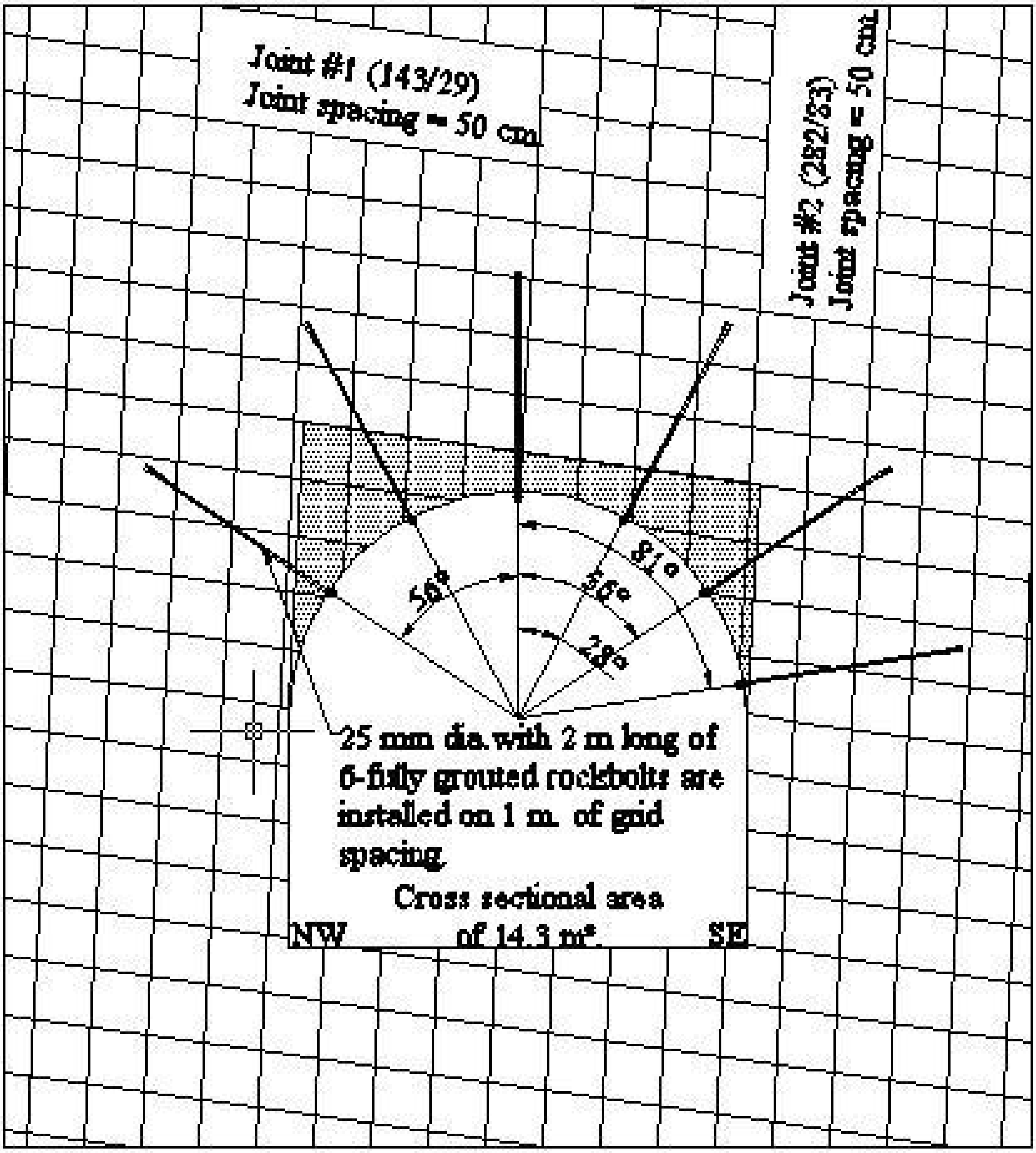

Figure 7.13 Preliminary layout of rockbolt in crosscuts in footwall limestone zone.

7.6 Stopes and natural support

The stope geometry is rectangular shape with the maximum length and height of 80 m. Nine stopes are placed in barite vein. The production working is naturally supported by pillars. The top barite over the stopes are left about 15 to 20 m as crown pillar. The barite is left between stopes for rip pillars, and sill pillars (Figure 7.14).

The square shaped shaft with 20 m deep has a surface area of 16 m^2. The blasting pattern has blasting factor of 0.26 m^2/hole and specific charge of 2.0 kg/m^3. The charging is done from the upper level. Each 2 m charged is fired (Figure 7.15).

Vertical slot blasting is similar to that of high bench blasting method. Blasting pattern calculated using the same principles as benching blasting method (Olofsson, 1988) has 1 m burden distance, 1.1 m spacing, 1 m stemming, and 0.29 kg/m^3 specific charge.

7.7 Drawing point and ore pass

The hopper shaped drawing points has an upper area of 80 m^2 (20 m x 4 m) a bottom area of 16 m^2 (4 m x 4 m), and a height of 16 m. Three or four sets are constructed at the bottom of each stope. The barite beam in drawing point shape is also excavated with shaft opening and vertical slot blasting.

7.8 Mine layout

The sublevel stope opening here consists of nine vertical production stopes. Six stopes are in the lower row and three are in the upper row. Seven sublevel tunnels with cross sectional area of 11 m^2 that are set between the elevation from 350 m and 470 m are the entries and production working. Twenty-three drawing points are constructed at the lower row of the stopes and eleven ore passes are in the upper row of the stopes. A tunnel with cross sectional area of 22.3 m^2 and 1023 long is the main entry for transportation of personal, equipment, and materials. Twenty-three cross cut tunnels with cross sectional area of 14.3 m^2 and about 15 m long are excavated for connecting between access tunnel and drawing points. Pillars of barite are left 20 m in place at the top of stopes (crown pillar and sill pillar) to support the next major level and at the ends of stope (rib pillar) for stability (Figure 7.16).

7.9 Mining sequence

The mine development has the purposes to access the ore body. The access tunnel and sublevel tunnels are the primary excavations to the barite deposit. All crosscuts are secondary excavations to connect the main access tunnel and the drawing points. The drawing points and ore pass will be developed later at the bottom boundary of the stopes.

The sublevel open stoping in steeply dipping barite vein starts with the development of free face with a 16 m^2 area and 20 m deep shaft block, and later fragmentation of the barite beam with long parallel blast holes. The barite is drilled and blasted in the beams so that each beam retreats slightly ahead of the upper level. This pattern allows the broken barite to fall directly to the bottom of the stopes. The broken barite will be left for temporary support. The broken barite will be moved to the haulage level and to a stockpile after completing the stope.

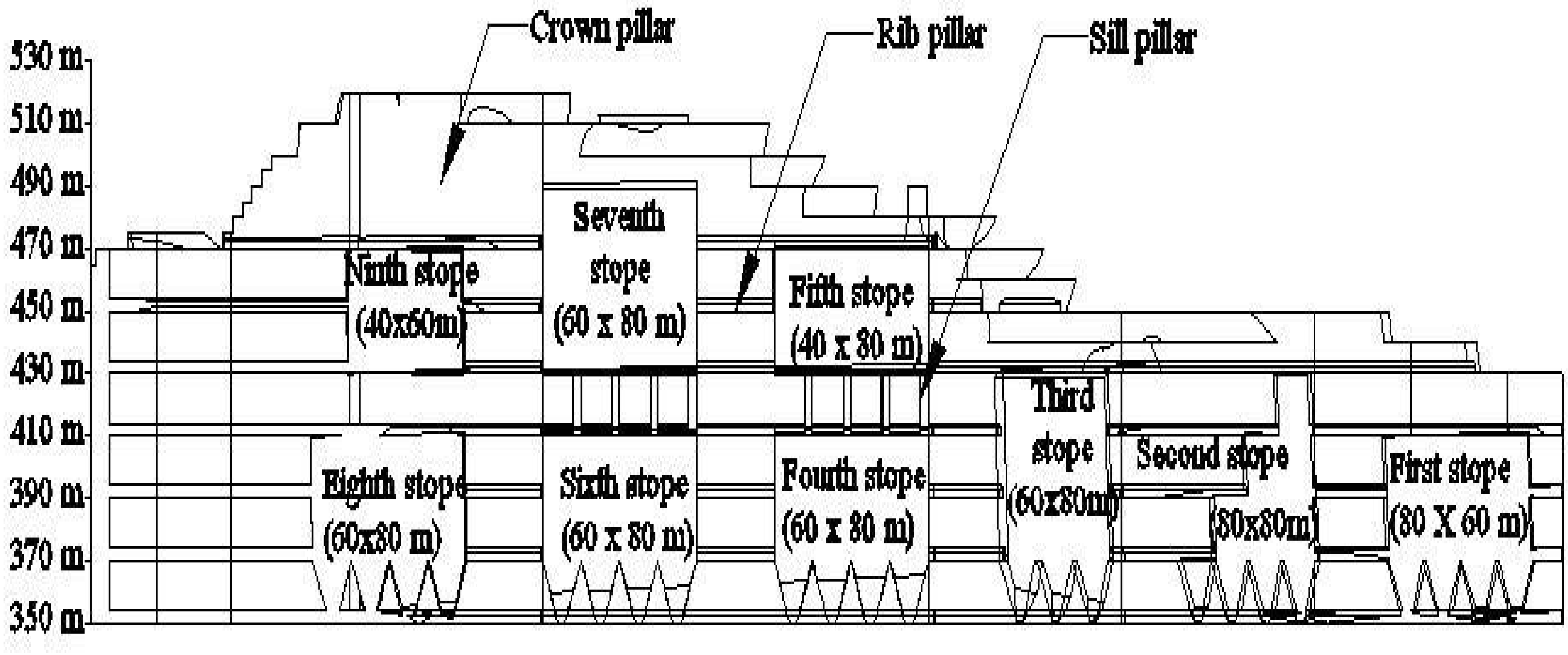

Figure 7.14 Stope dimension with sequence of opening number.

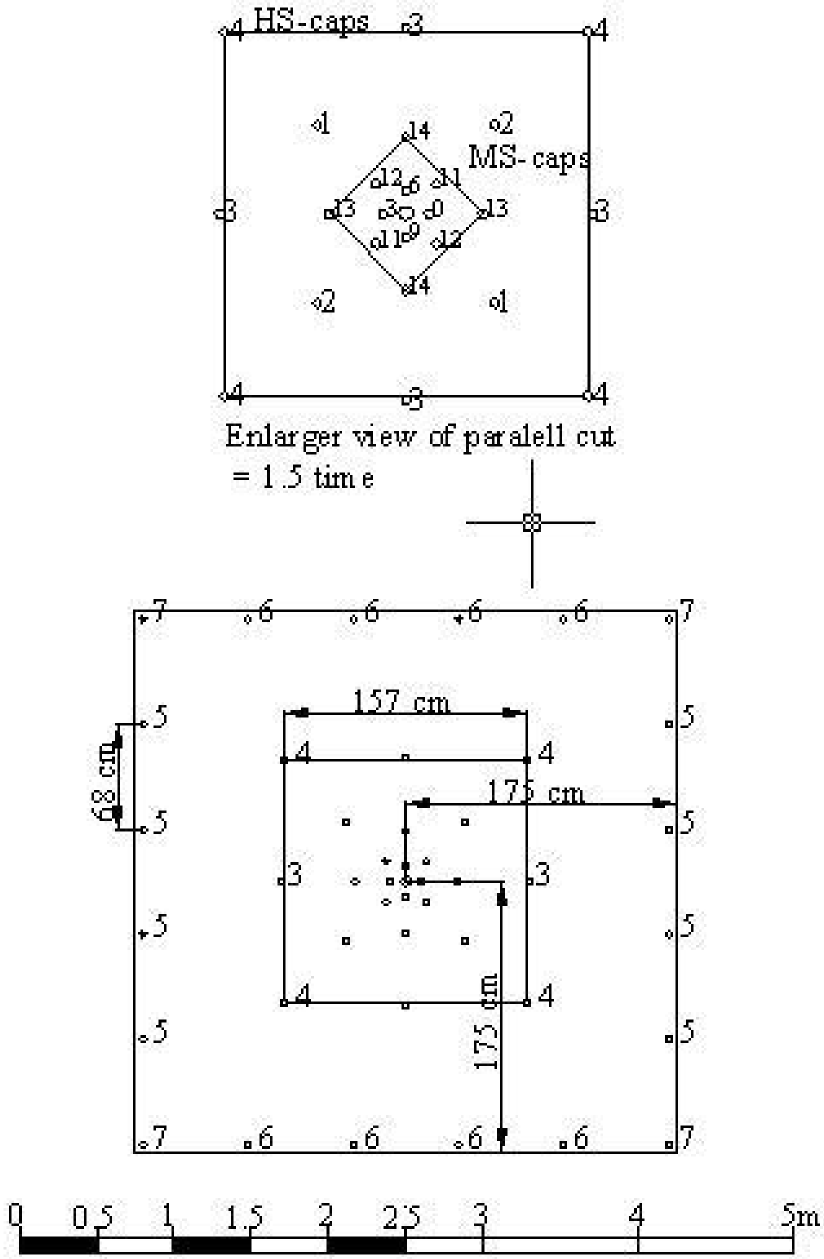

Figure 7.15 Drilling pattern for vertical shaft raising

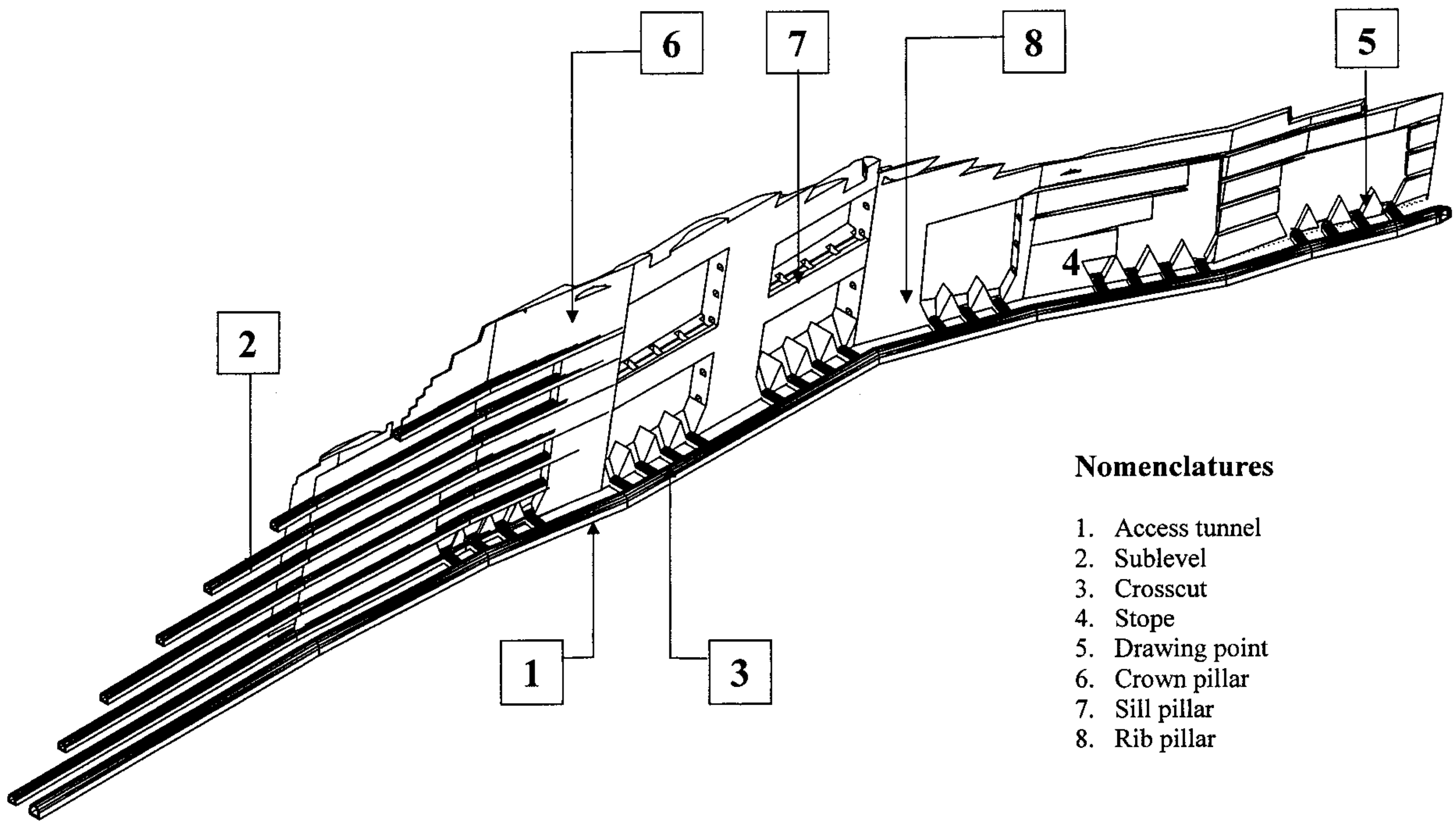

Figure 7.16 Mine layout of the sublevel stoping method.

CHAPTER VIII

EXCAVATION SCHEME

The objective of this chapter is to describe the sequence of excavation and mine development. The underground excavation has a basic cycle of mine operation including drilling, blasting, loading, hauling, and support.

The mechanization used to drill, load and haulage will be a single boom hydraulic percussion mounted on a mobile rig for drilling, the self-loading transport type as scooptram load-haul-dump (LHD) for loading, and short distance haulage, and the lower dump truck for long distance haulage. The mine equipment capacity is calculated from their specifications in which drill equipment is defined. It has a penetration rate of 24 m/hour (Table 8.1). The LHD with the two drum tracks provides a load capacity of 60 tons/hour (Table 8.2).

All tunnel excavations (an access tunnel, seven sublevels, and twenty-four crosscuts) are advanced 2 meters per round. Drilled blastholes will be charged and blasted in full face. Each blasting round will move broken rock mass about 44.6 m^3, 22 m^3, and 28.6 m^3 for access tunnel, sublevel, and crosscut. The rock bolt will be immediately installed after broken rocks are removed.

A working cycle for tunnel excavation comprises drilling, charging, blasting, haulage, scaling, and support. Stoping, drawing point and vertical shaft opening involve long hole drilling, charging, blasting and haulage. The drilling and haulage time can be calculated from machine capacity. The working times activated by men are obtained from experience crews. The charging is about 3 minutes/hole for 2.2 m long of drillhole, and 30 minutes/hole for 20 m long drillhole, scaling is bout 30 minutes, and supporting is about 4 hours/round. A cycle time can be described as follows.

1) Access tunnel requires about 17 hours to complete a round of excavation, which includes 6.2 hours for 67 drillholes, 4.3 hours for charging (add 1 hour for blasting and ventilation), 2 hours for 44.6 m^3 haulage, 0.5 hour for scaling, and 4 hours for support.

2) Sublevel requires about 13.2 hours to complete a round of excavation, which includes 4.3 hours for 47 drillholes, 3.4 hours for charging (add 1 hour for blasting and ventilation), 1 hour 22 m^3 for haulage, 0.5 hour for scaling, and 4 hours for support.

3) Crosscut requires 15.5 hours to complete a round of excavation, which includes 5.7 hours for 61 drillholes, 4 hours for charging (add 1 hour for blasting and ventilation), 1.3 hours for 28.6 m^3 haulage, 0.5 hour for scaling, and 4 hours for support.

4) A vertical shaft requires 6 days to complete, which includes 37 hours for 44 m long drillholes, and 27 hours for charging and blasting.

8.1 Mine development

8.1.1 Portal

The portal is located in loose shale. The shale can be removed by using backhoe. The portal will be excavated as multiple benches with retained slope angle of 45º. The total removal of overburden that is about 206,500 m^3(loose) is directly hauled to the adjacent area and leveled them for use as stockpile, dumping area, etc. The 1.8 m^3 bucket sized backhoe and three 12 m^3 capacity dump trucks can move 850 m^3 (loose) of overburden per day. A complete portal excavation requires about 9 months.

8.1.2 Tunnel excavation

The underground tunnels are excavated in seven levels ranging between 350 m at lower level (first) and 470 m at the upper level. The 350 m level comprises three types of excavation, including access tunnel, sublevel, and crosscut. All these tunnels will be simultaneously excavated with two working area. The upper level is 370 m and 390 m levels that are the sublevels will also be excavated. The remaining levels will be excavated in pair. The broken waste rocks are about 97500 m^3. They are moved to a temporary disposal area. The limestone and shale will be used for developing facility area, such as disposal area, stockpile area, road, water treatment pond, etc. The barite will be moved to a stockpile area. The excavated tunnel will be advanced 4 m per day. The total time is 1932 days or 6.7 years to complete all excavating tunnels.

8.1.3 Drawing point and ore pass excavation

The drawing points are excavated in the barite beam between the 350 m and 370 m elevations. The ore passes are between 410 m and 430 m levels. Three or four units of drawing points or ore passes can be set in the bottom of the stopes. A hopper shaped drawing point begins developing the free face at the middle point that is opposite to the roof of the cross cut tunnel. The remaining barite on the two sides is drilled and blasted as the same blasting pattern as the vertical slot blasting, provided 1 m of burden distance, 1.1 m of spacing, and 0.29 kg/m^3 of blasting factor for 38 mm diameter bits. A complete drawing point requires 14 days, which includes 6 days for free face opening, 4 days for drilling and blasting, and 4 days for haulage. The broken ore from the thirty-five opened drawing points and ore passes is about 768 m^3. They consume 1.7 years to complete all developed drawing points.

8.2 Production stope

The total production is about of 0.62 million tons of barite (Table 8.3) from nine stopes. The free face is developed at the south end of beam using vertical shaft method. The remaining barite beam is fragmented by vertical slot blasting. The blasthole pattern is similar to bench blasting method, given 1 m burden distance and 1.1 m spacing for 38 mm diameter bits. Four drillholes can be placed in 3.5 m wide barite vein. The drilled holes use ammonium nitrate with fuel oil ratio of 94:6 by weight with 1 m stemming. Two meters of barite will be retreated in each blast cycle until reaching about ten meters, the upper barite beam will be opened using the same procedure as described above. The barite beams will be drilled and blasted with 10 m offset distance retreated ahead of the upper level. The broken ore is moved to a stockpile area. A complete cycle requires 12 hours, which includes 7 hours for drilling, and 5 hours four blasting. A complete production stope consumes about 4 years.

8.3 Summary

The production is about 0.62 million tons with 2 shifts per day; about 530 tons per day or 38 tons per hour. The ore value evaluated by using the local price imposed by PANDS Barite Mining Co., Ltd. with 550 Bath per ton is about 341 millions Bath. The mine life is fourteen years.

Tables 8.1 Calculation of estimating drill capacity.

A. Design and performance data

1) Drill type	A single boom with hydraulic percussion on mobile rig.
2) Bit type	Carbide and cross bit with a diameter 38 mm.
3) Rod length	2.5 meters.
4) Time delay	3 minutes for positioning, collaring, rod charging, etc.
5) Depth of drilling	2.2 m for excavation tunnel 20 m for stoping
6) Rock type	Limestone, shale, and barite
7) Drillability	1.19 (Hartman, 1987)
8) Penetration rate (Barre Granite)	11.9 mm/s (use as norm value for finding penetration rate of other rocks, Hartman, 1987)

B. Calculation

$$\text{The penetration rate of the rocks} = \frac{11.9 \text{x} 60}{1000} \text{ x } 1.19 = 0.85 \text{ m/min.}$$

2.1 drill capacity for opening excavation

$$\text{Net drill time per hole} = \frac{2.2}{0.85} = 2.59 \text{ min/hole}$$

$$\text{Gross drill time per hole} = 2.59 + 3 = 5.59 \text{ min/hole}$$

$$\text{Drill capacity} = \frac{2.2 \text{x} 60}{5.59} = 23.61 \text{ m/hour or } 24 \text{ m/hour}$$

2.2 drill capacity for stope

$$\text{Net drill time per hole} = \frac{20}{0.85} = 23.53 \text{ min/hole}$$

$$\text{Gross drill time per hole} = 23.53 + (3 \text{ x } 9) = 50.53 \text{ min/hole}$$

$$\text{Drill capacity} = \frac{20 \text{x} 60}{50.53} = 23.74 \text{ m/hour or } 24 \text{ m/hour}$$

Table 8.2 Calculation of load and haulage capacity.

A. Design and performance data

1) Loader type	Self-loading transport, load-haul-dump (LHD)with 3.4 m^3 bucket size or loaded capacity about 6 tons.
2) Loading time	About 1 minute per round (experience working).
3) Haulage type	2 dump trucks with a capacity of 6 tons
4) Average travel	Average loaded and unloaded travel is 9500 mph.
5) Travel distance	Average 1200 meters.
6) Dumping time	About 1 minute per round (experience working).
7) Correction factor	80% for favorable condition.

B. Calculation

1. The overall time for load and travel cycle = load + travel +dump

$$= 1+\frac{60x1200}{9500} + 1 = 9.60 \text{ minute}$$

2. Load and haulage capacity $= \frac{6x60}{9.60}$ x 0.80 x 2 = 60 tons per hour

CHAPTER IX

MINING COST ESTIMATION

The overall mining costs are estimated from direct mining costs and indirect mining cost. The direct costs are the sum of all mining costs associated with the four stages of prospecting, exploration, development and exploitation. The indirect costs are related to administration, engineering, and other nonitemized services that usually includes allowance of 5 to 10% of direct cost (Hartman, 1987).

9.1 Design and performance data for mine excavation calculation

The cost estimation of mine excavations related to the cost of mine equipment and supplied materials, labor, conditions of the excavations, and working time. In order to simplify the calculation, all costs involved in the mining process will be determined as unit costs.

The mine equipment included the drill machine, the LHD loader, and the dump trucks here are determined the unit cost in Baht per hour by using calculation procedure developed by the U.S. manufacturers of earth moving and materials handling equipment (Hartman, 1987).

The unit prices of explosive, rockbolt and chainlink mesh for rock excavation and support are obtained from the manufacturer price list. Table 9.1 lists the unit costs of mine equipment and unit prices of materials.

The conditions of excavation used in calculation are size of excavation, depth of drillhole, drilling factor, and specific charge which are summarized in Table 9.2.

Labors costs here are 1200 Bahts/shift, which includes wage of 4 men for explosive charge and 4 men for support installation.

The working time is 2 shifts/day for tunnel excavation and stope. The complete vertical shaft takes 6 days. The complete drawing point takes 4 days.

9.2 Excavation costs

The costs of elemental excavations here use simple calculation for cost estimation. Drilling cost is the multiplication of the total times of drilling and unit cost of drilling. Blasting cost is the sum of product between the total consumption of both explosive and electrical caps and their unit prices. Load and haulage cost is product of the total load, and haulage times, and the sum of unit cost of both load and haulage.

Table 9.3 shows overall cost per round of drilling, blasting, load and haulage, labor, and unit cost in Baht/tons for the mining excavation.

The support cost is the sum of the support material prices used in rock support (Table 9.4).

9.3 Prospecting and exploration costs

The costs of prospecting and exploration as well as mining least permitting have already been paid. The barite deposit was acquired by bidding from the former ministry of national development in 1969 with the cost of 300,000 Bahts. During

1991-1993, the mine is reinvestigated to evaluate the ore reserve with the costs related to surface and subsurface mapping. The investigation cost is about 2.2 millions Baht.

9.4 Development cost

The excavation costs in the stage of the mine development include the cost of the portal excavation, the tunnel excavation, and the drawing point construction.

The surface excavation cost generally is 25 Bahts/m^3 (at Loei province). The overall cost to complete moving 206,500 m^3 of overburden out is 5,162,500 Bahts or about 5.2 millions Bahts.

The excavating volume of rock masses in cross sectional area of 22.3 m^2 with 1,023 m long of an access tunnel is 22843 m^3or 63,962 tons. The overall cost is the sum of the costs for drilling, blasting, and loading and haulage, labor, and support; (63,962 x 121) + 15,571,900 = 23,311,308 or about 23.3 millions Bahts.

The excavating volume of rock masses in cross sectional area of 11 m^2 with 6339 m of all sublevels length is 69,729 m^3 or 264,970 tons. The overall cost is (264970 x 124) + 49,517,300 = 82,375,605 or about 82.37 millions Bahts.

The excavating volume of rock masses in cross sectional area of 14.3 m^2 with 15 m long for the twenty-three crosscut is 4,934 m^3 or 13,815 tons. The overall cost is (13,815 x 165) + 2,227,725 = 4,506,870 or about 4.50 millions Bahts.

The excavation cost of a drawing point includes the cost of a vertical shaft opening and fragmentation of two shoulder sides. The excavating weight of rock masses in the two shoulder sides is about 2,918 tons. The overall cost is (1,216 x 68) + (2,918 x 62) = 263,604 Bahts. The overall cost for developing thirty-five of both drawing points and ore passes are = 9,226,140 Bahts or about 9.23 millions Baht.

9.5 Exploitation cost

The cost of exploitation includes the development of free face cost and fragmentation cost. Nineteen barite sills need the free face for producing in nine stopes.

The overall cost of free face development is (19 x 1216) x 68 = 1571072 Baht.

The overall cost of barite production is 620,000 x 62 = 38,444,000 Baht.

The total cost of exploitation = 40,011,071 Baht.

or = 40 millions Baht.

9.6 Mining cost

The direct costs for the four stages of mine life are as follows.

Both prospecting and exploitation cost	=	2.50 millions Baht.
Development cost	=	124.60 millions Baht.
Exploitation cost	=	40.00 millions Baht.
Total direct mining cost	=	167.10 millions Baht.

Indirect cost estimated 10% of direct cost is 16.71 millions Baht.

The overall mining cost includes direct cost and indirect cost is 183.81 millions Baht or about 296 Bahts per ton.

9.7 Discussions

The total cost of both prospecting and exploitation stages is 2.5 millions Baht. It is in the range of cost estimation of mineral investigation plan for mining lease application. The costs of investigation are between 0.7 and 3 millions Baht for non-metallic mineral.

The costs of both development and exploitation stages here are estimated only for the costs that related to mining activities in the excavation scheme. The estimation procedures for mining costs are carried out from currency data sources. It is believed that the estimated costs of the excavations are sufficient for feasibility level. Some of the mining costs are not included here such as the costs of mining least permit, approval of an environmental impact statement, and auxiliary operation. The former approval of the mining least did not require the statement of environmental impact. Nowadays, the cost of preparation and obtaining approval of an environmental impact statement is between 500 to 700 thousands Baht. The total costs of the auxiliary operation such as ventilation, power supply, maintenance, waste disposal, environmental control, land reclamation etc. generally are about $1.2/ton or about 50 Bahts per ton (Hartman, 1987). The total mining cost increases by 50 Bahts per ton to become 346 Bahts per ton.

Table 9.1 Unit costs and prices of mine equipment and supplied materials.

Items	Specifications	Unit cost	Remarked
1) Drilling cost	Single boom hydraulic percussion type mounted on mobile rig.	1017 ฿/h	Drilling capacity is 24 m/hour.
2) Load cost	3.14 m^3 bucket size Scooptram LHD.	1699 ฿/h	Load and haulage capacity is 60 tons per hour.
3) Haulage cost	6 tons lower drums tuck.	370 ฿/h	
4) Explosive	ANFO	13.40฿/unit	PandS_Group
5) Electrical caps	Half-second types.	18.50฿/unit	PandS_Group
	Millisecond types.	24.50฿/unit	PandS_Group
5) Rockbolt	25-mm dia. with 2 m longs.	1450 ฿/unit	Right tunneling Co., Ltd.
	25-mm dia. with 1.5 m longs.	1050 ฿/unit	Right tunneling Co., Ltd.
6) Wire mesh	6.2 mm wire size with square 15 x 15 cm^2 opening.	25 ฿/m^2	Right tunneling Co., Ltd.

Table 9.2 Conditions of the excavation.

Excavation types	Conditions of the excavations				Fragmented rocks/round	
	Opened area (m^2)	Drillhole depth (m)	Drilling factor (m^2/hole)	Specific charge (kg/m^3)	(m^3)	tons
Access tunnel	22.3	2.2	0.33	1.5	44.6	125[1]
Sublevel tunnel	11	2.2	0.24	2.2	22	84[2]
Cross cut tunnel	14.3	2.2	0.22	1.8	28.6	80[1]
Vertical shaft	16	20	0.26	2	320	1216[2]
Vertical slot blasting for stope and drawing point	7	20	0.89	0.29	140	532[2]

<u>Remark</u> [1] volume converted to tons by multiplying by the tonnage factor of 2.8 ton/m^3.
[2] volume converted to tons by multiplying by the tonnage factor of 3.8 ton/m^3.

Table 9.3 Overall cost estimation for one round of excavation.

Excavation types	Fragmented rocks/round (tons)	Drilling cost (฿)	Blasting cost (฿)	Load and haulage cost (฿)	Labor cost (฿)	Unit cost (฿/tons)
Access tunnel	125	6246	2226	4310	2400	121
Sublevel	84	4288	1660	2138	2400	124
Crosscut	80	6060	1964	2759	2400	165
Vertical shaft	1216	52545	20766	41932	14400	68
Vertical slot blasting	532	6780	2024	21626	2400	56

Table 9.4 Overall costs of the supplied materials for tunnel supports.

Excavation types	Length of tunnels (m)	Length of rockbolts (m)	Number of rockbolts (unit)	Required chainlink mesh (m^2)	Overall cost (฿)
Access tunnel	1023	2	10566	10048	15,571,900
Sublevel	6339	1.5	46433	30506	49,517,300
Crosscut	345	1.5	2070	2169	2,227,725

CHAPTER X

DISCUSSIONS AND CONCLUSIONS

10.1 Discussions

The objective of the study of an underground barite mining at PANDS Barite Mining Co, Ltd., Loei province is to assess the feasibility for extracting barite ore at greater depths, while minimizing the landuse and the environment impact. The study requires the geological information of the barite deposit and the geotechnical properties of barite-bearing zone and surrounding rocks for the mine selection and excavation design. The geology of barite deposit has been interpreted from the existing geological maps and borehole data showing that the barite zone is a continuous, tabular and narrow vein with steeply dipping and interbedded between the footwall limestone and the hanging wall shale. The ore depth is uncertain, due to limited number of exploration holes. The characteristics of rock masses in the study area are determined from the CSIR Geomechanics classification and NGI tunneling index Q systems. Both classification systems require similar geotechnical data that are carried out by means of field investigation using basic field instruments. The geology of barite deposit is revealed from surface and subsurface geological work. It indicates enough geological parameters for preliminary mine selection and design. All of geotechnical information are collected from surface outcrops that mostly are disturbed by weathering, unloaded land, and mine actives. The fair and poor quality of the classified rock masses is accepted for conservative design reason.

10.2 Conclusions

A feasibility study of applying an underground mining method to a small-scale barite deposit is carried out. The existing geological and geotechnical information in the study area is analyzed for mining methods selection and excavation design. The ore reserve is enough to pursue further investment. The geology of barite deposit is appropriate for the underground mining method. The high relief terrain of ore deposit suggests the mine excavation in the horizontal level. The rock masses are classified as fair to poor quality which are considered in the rock support design. The sublevel stoping method has been adopted for mining the steeply dipping barite vein. The main entry and services tunnels are proposed between the elevations from 350 m and 550 m and in the footwall rock masses. Seven sublevels are excavated between the elevations from 350 m and 460 m into the barite vein to access the stope. Nine stopes have maximum length and height about 80 m. The productions are initially excavated at the end point of the barite sill and retreat to the access point. The barite sill levels below all bottoms of the stopes are excavated for drawing points or ore passes. The main access tunnel at the elevation of 350 m is driven parallel to barite vein for loading and hauling. The crosscut are constructed at haulage level to connect between the access tunnel and the drawing points. The drilling and blasting are used to fragment the rock. The tunnels are advanced 2 m for each complete round. The excavation of stopes also use the drilling-and-blasting method to break the barite for free face. The barite sills are developed for the free face by vertical shaft opening. The remaining barite sill is broken by vertical slot blasting method. The rock masses at the stope boundary are left about 20 m to form pillars for stope stability. The unstable areas are reinforced by rockbolts and wire mesh attached at the roof of excavation for preventing the falling of small pieces of rock. The total barite of about 0.62 million tons is produced at 530 tons/day. The mine life is 14 years. The total mining cost is about 346 Bahts per ton and the local ore value is 550 Bahts per tons.

10.3 Recommendations

The uncertainties and remaining questions for this feasibility study involve the subsurface geology, depth, uniformity, shape, grade, and total reserve of the barite deposit. Additional exploratory holes and pilot adit, and ore grade evaluation are required.

The exploratory holes may be required in the north of the study area, such as the location grids of 10600N/10200E, 10500N/10200E, and 10300N/10200E. They should be inclined to the west. The cored barite samples can be used for examining the ore grade by both physical and chemical tests.

The pilot adit should start at an elevation of 430 m at the local grid of 10275N/10095E and driving to the north. Eleven square meters with horseshoe shaped of adit may be selected for excavating into the barite-bearing zone. The crosscuts should also be excavated into the adjacent footwall limestone zone and footwall shale. The geological and geotechnical information from the pilot adit can be used to confirm the results of the feasibility study presented in this thesis.

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APPENDIX A

STRATIGRAPHIC COLUMNS OF ROCKS

IN DRILLHOLE

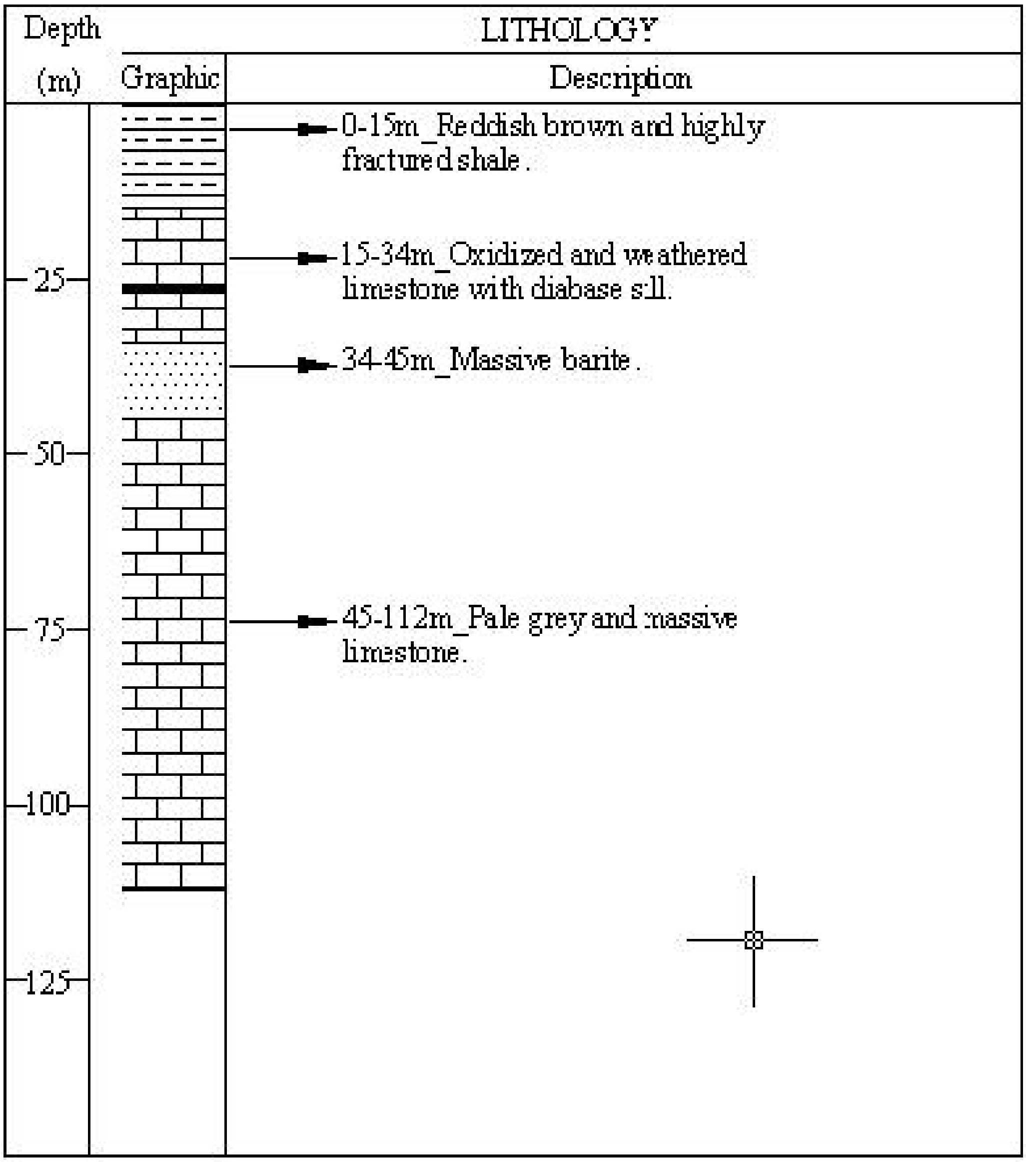

Figure A1 Stratigraphic column of rocks chip in drill hole No. RC01 (modified from Mustard, 1993).

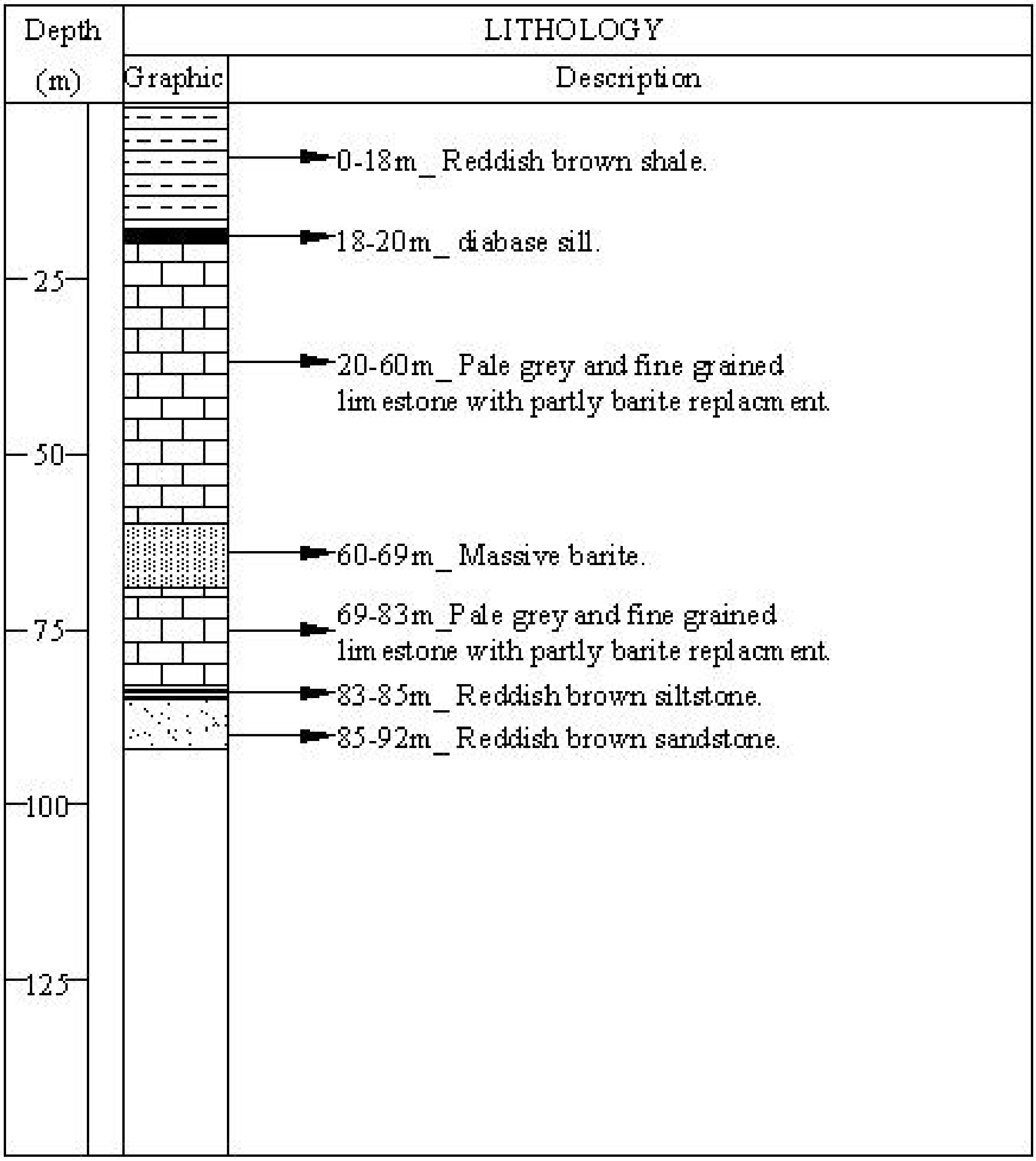

Figure A2 Stratigraphic column of rocks chip in drillhole No. RC02 (modified from Mudstard, 1993).

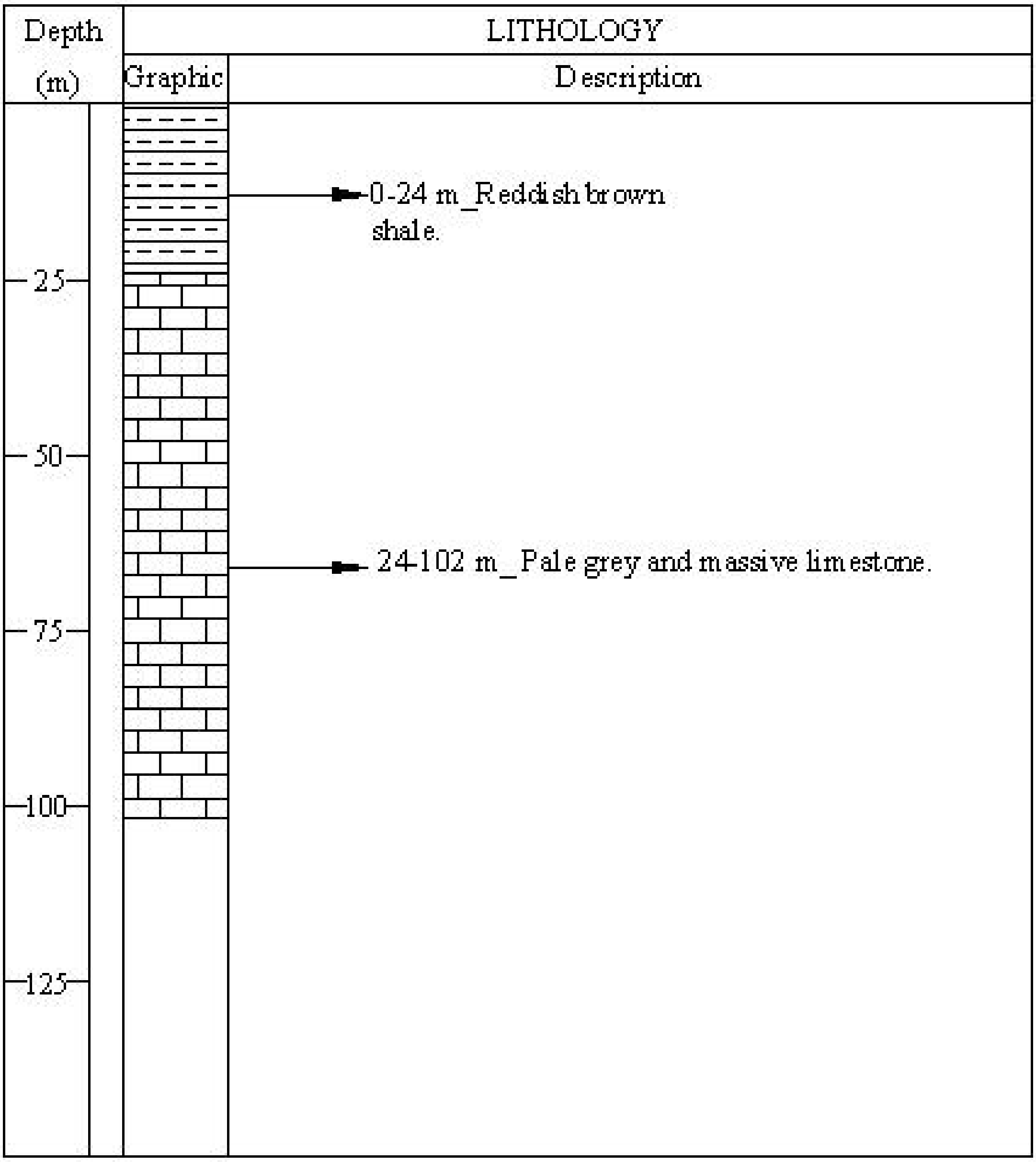

Figure A3 Stratigraphic column of rocks chip in drillhole No. RC03 (modified from Mustard, 1993).

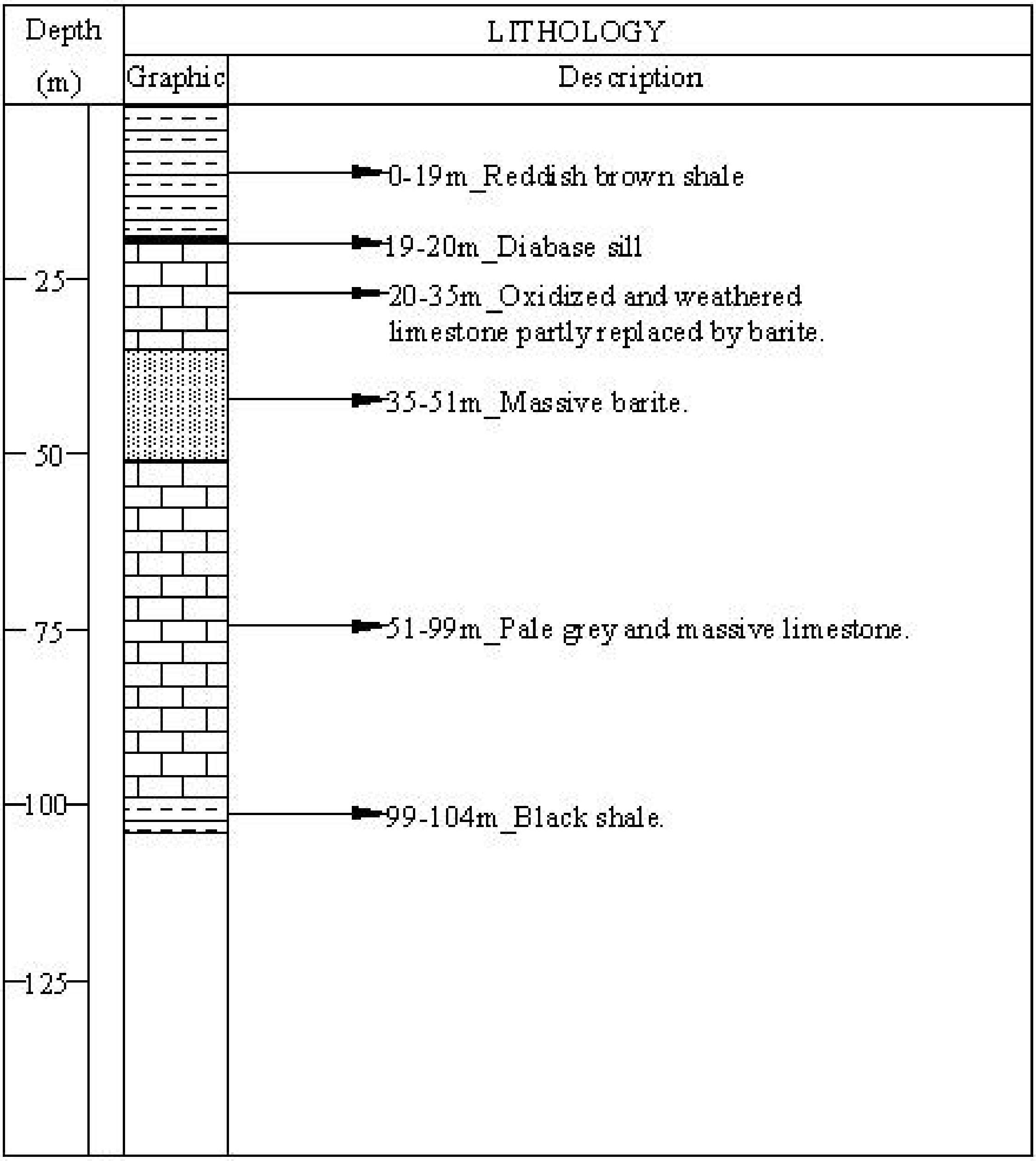

Figure A4 Stratigraphic column of rocks chip in drillhole No. RC04 (modified from Mustard, 1993).

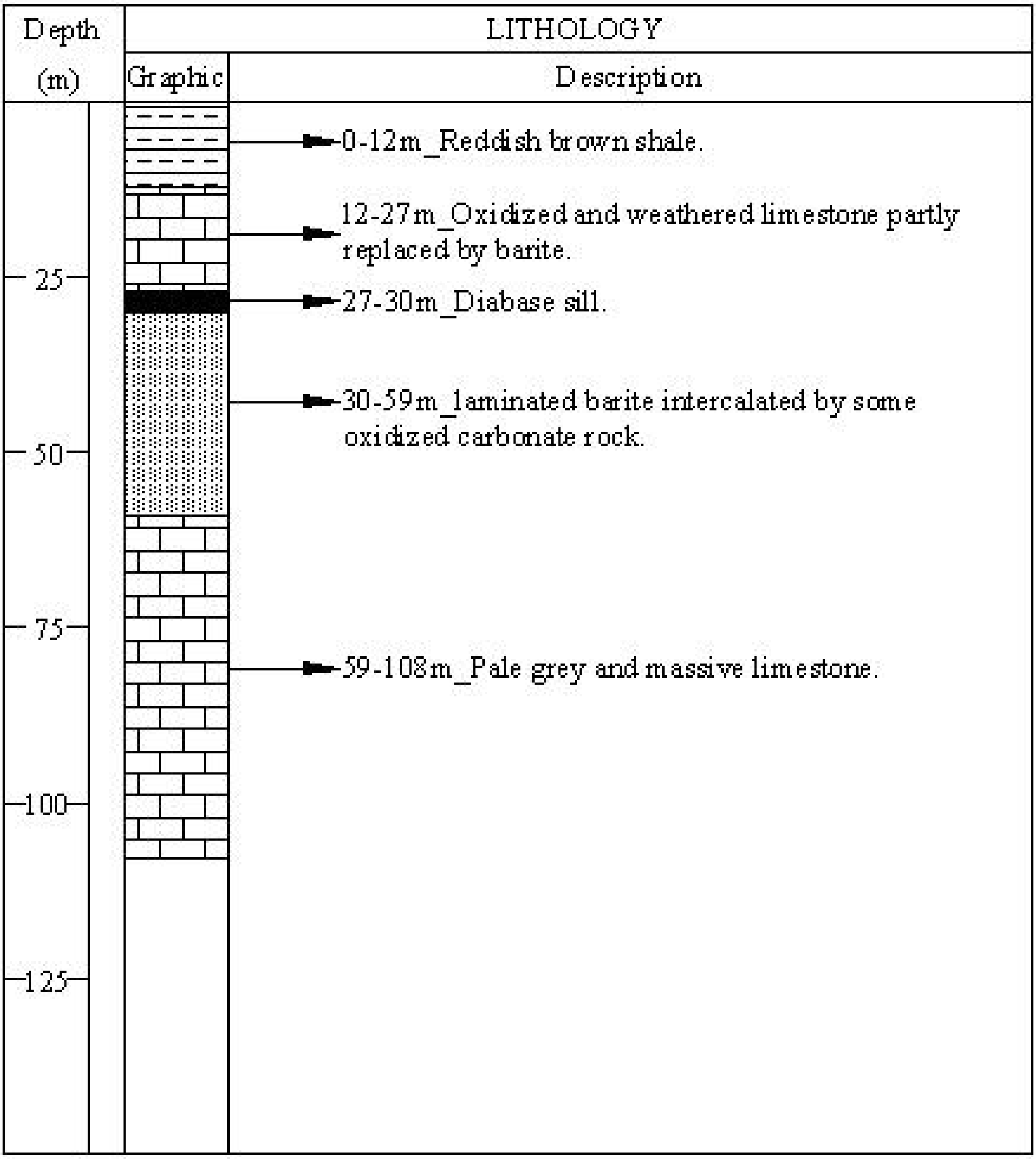

Figure A5 Stratigraphic column of rocks chip in drillhole No. RC05 (modified from Mustard, 1993).

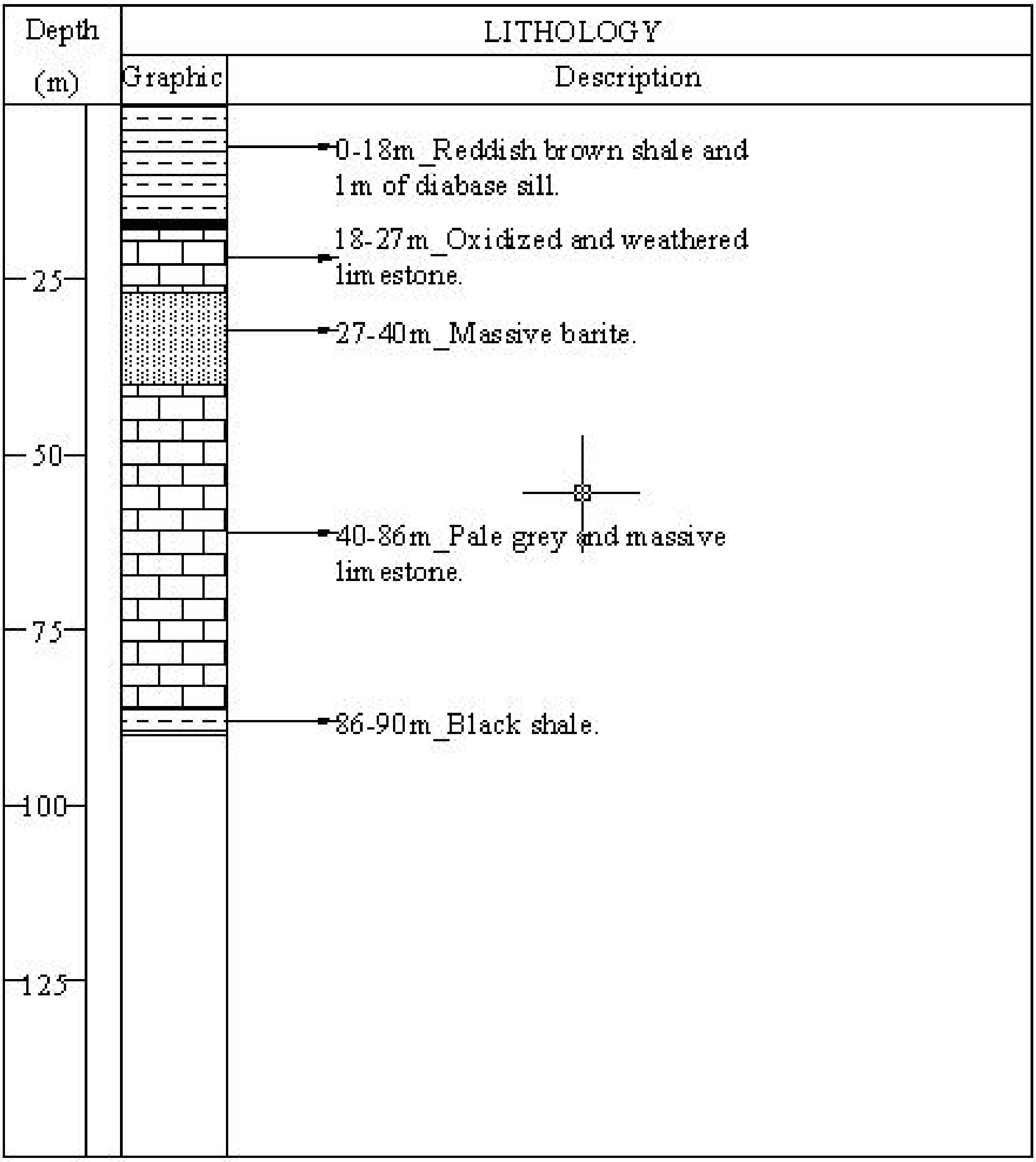

Figure A6 Stratigraphic column of rocks chip in drillhole No. RC06 (modified from Mudstard, 1993).

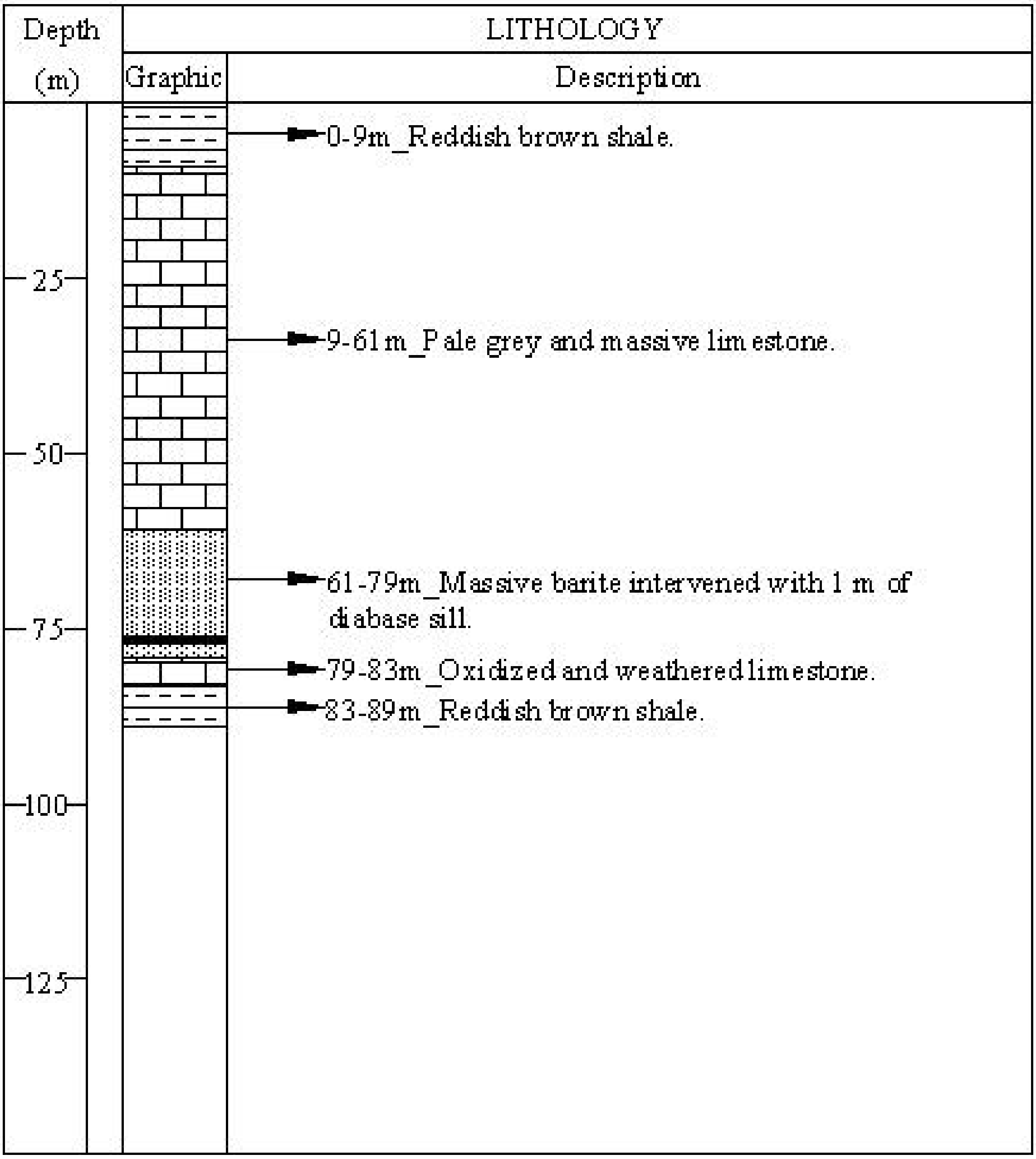

Figure A7 Stratigraphic column of rocks chip in drillhole No. RC07 (modified from Mustard, 1993).

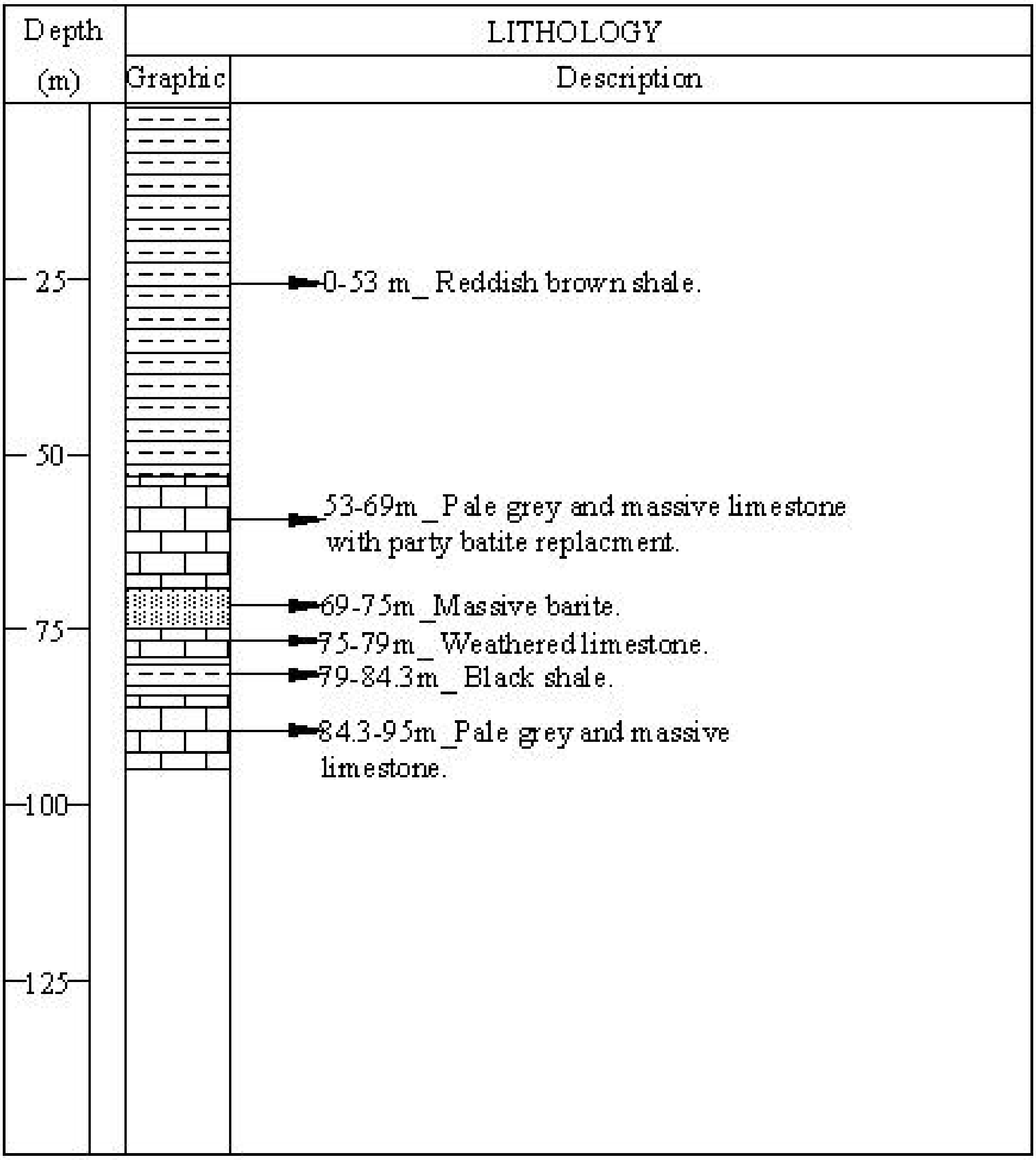

Figure A8 Stratigraphic column of rocks chip in drillhole No. RC21(modified from Mustard, 1993).

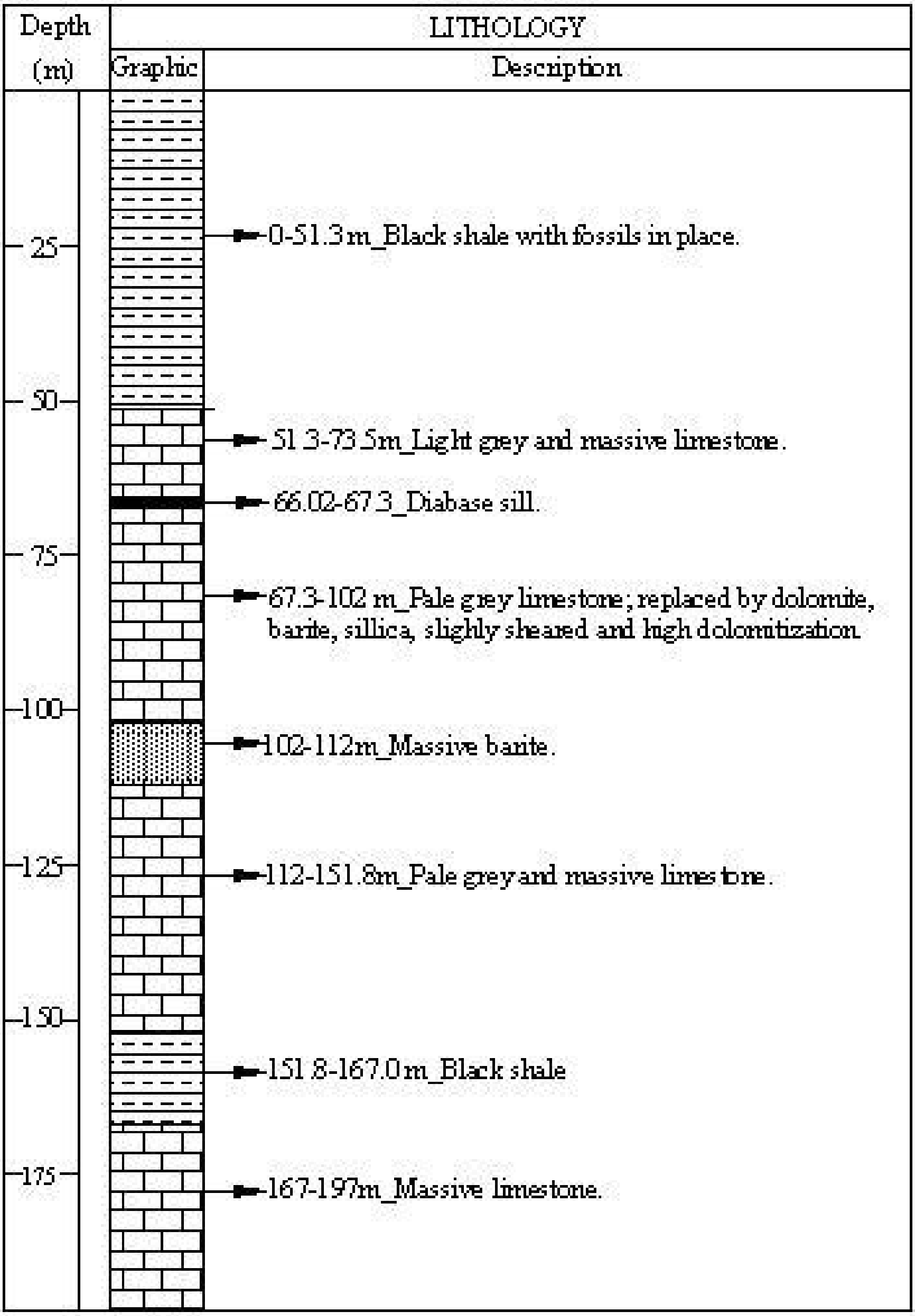

Figure A9 Stratigraphic column in drillhole No. DDH01 (modified from Mustard, 1993).

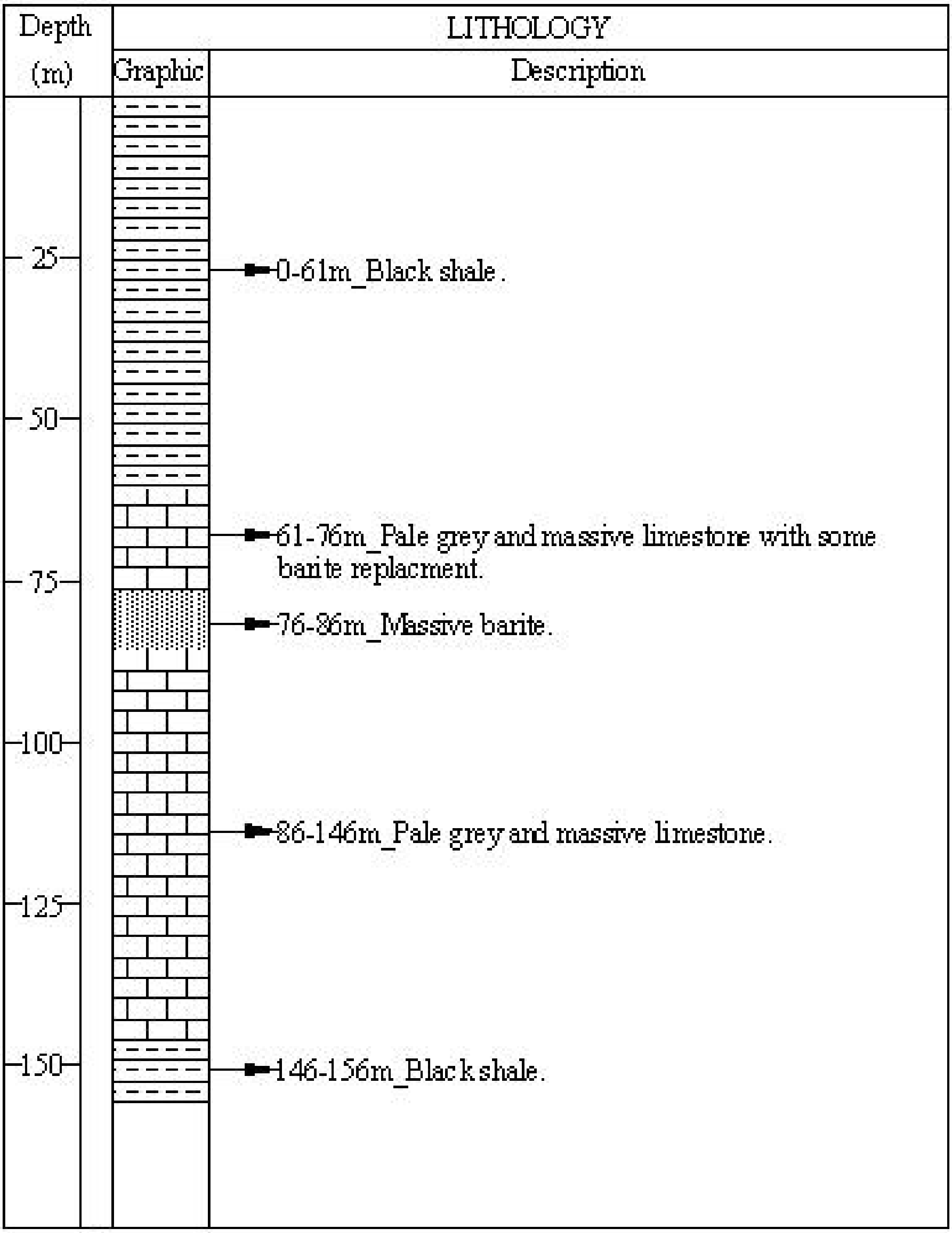

Figure A10 Stratigraphic column in drillhole No. DDH03 (modified from Mustard, 1993).

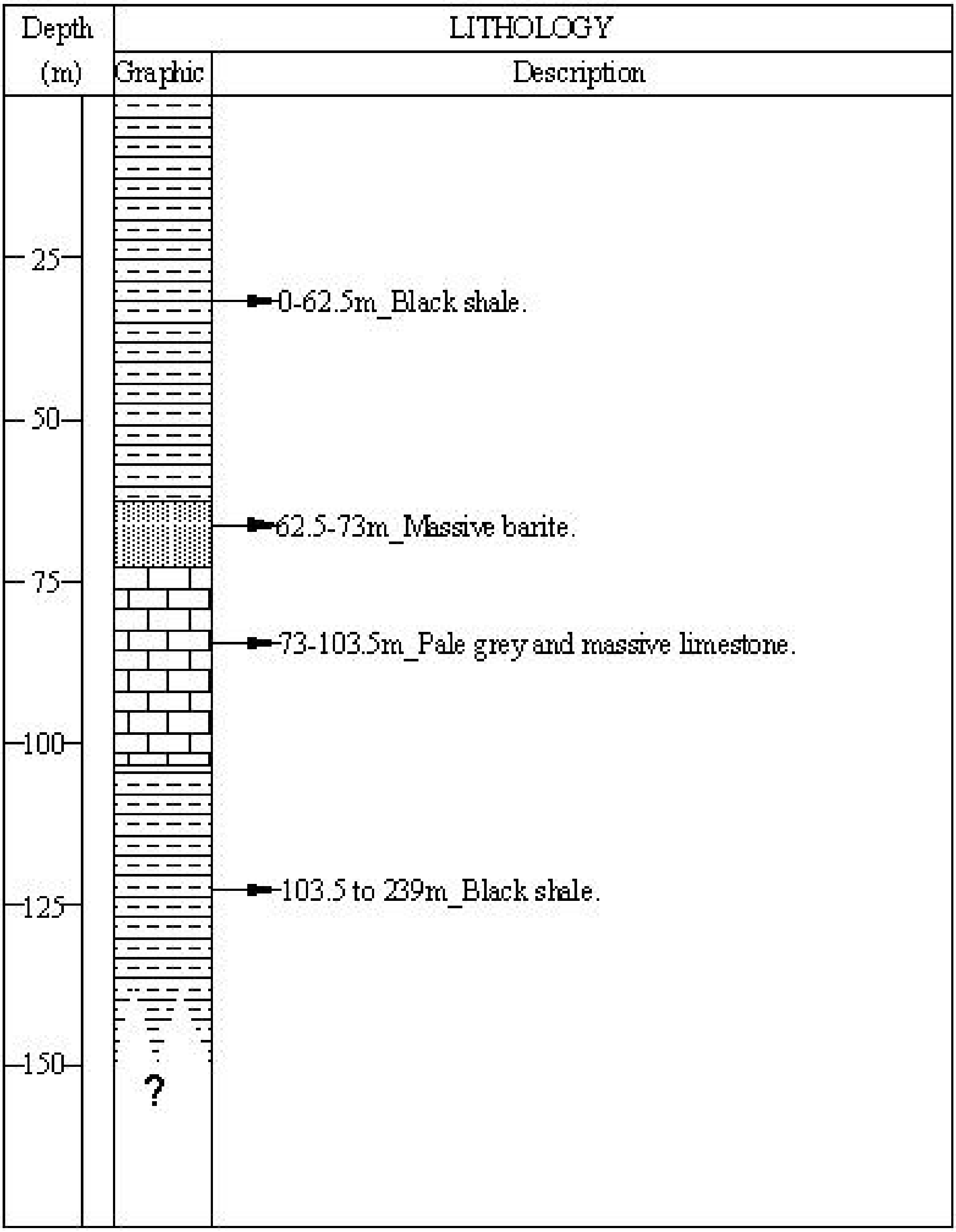

Figure A11 Stratigraphic column in drillhole No. DDH04 (modified from Mustard, 1993).

BIOGRAPHY

Mr. Mana Jatuwan was born on February 25, 1958 in Pranakorn Sri-Ayuttaya province, Thailand. He graduated in Bachelor of Science (engineering geology) from Khon Khaen University in 1981. He has been worked for industrial mining business in PANDS Barite mining Co., Ltd., since 1982. At present, he has taken senior geologist position for overseeing activities related mining, including investigation, mine development, production, and government regulations. Previously, he had occupied geologist position for barite production and investigation at Loei province during 1983 to 1985. Mining manager positions for lead mining at Puttalung province, dolomite mine at Kanchanaburi province, and limestone mining at Tak province during 1986 to 1992. Branch manager positions for barite mining and processing at Loei province and Nakorn Sri Thummarat province during 1993 to 2003.