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Geological and Geotechnical Considerations in Open Pit Planning and Design; A case of Okote Gold Deposit, Southern Ethiopia

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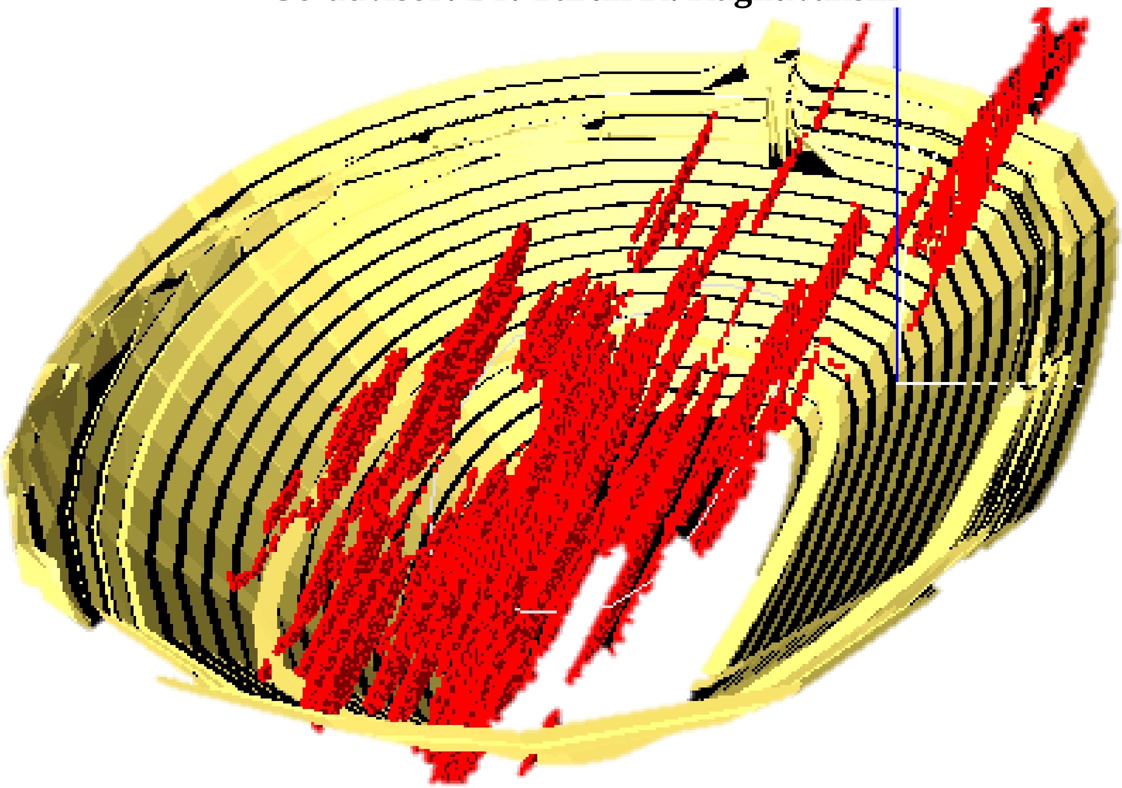
A Thesis Submitted to School of Earth Sciences of Addis Ababa University in Partial Fulfilment of the requirements for the Degree of Master of Science in Mining Geology

By Shibiru Merga Nemara

Advisor: Dr. Worash Getaneh
Co-advisor: Dr. Tarun K. Raghuvanshi



May, 2019
ADDIS ABABA UNIVERSITY
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Abstract

This manuscript particularly presents the geological geotechnical factors and parameters to be considered for the designing and planning of Okote gold project which is located in Borena, southern Ethiopia. The deposit is currently under feasibility study by National Mining Corporation (NMiC). Geologically, the area lies within the southern extension of the Megado meta-volcano-sedimentary terrain. Carbonate amphibole schist, chlorite schist/ carbonate chlorite schist, metagabbro, metagranodiorite, metadiorite, are identified as the main lithological units. They are all aligned to the NNE-SSW direction and dipping to the west at about ~65-78^0. There are two distinctly identified mineralization types: chlorite schist hosted gold mineralization and metagranodiorite hosted mineralization. The resource estimation using ordinary kriging reveals the Okote gold deposit has 77 million tons of ore at 1.18g/ton average grade and 0.49 cut-off grade. Thus, the contained value is calculated to be 23,062Kg of gold. Geotechnical study shows that the rock mass in the area is strong enough for hard rock mining. According to the RMR_{89} classification system, waste and ore body are rated 65 and 65.5 respectively. All the factor of safety calculated for bench, inter-ramp, and overall slope angles for the proposed open pit range between 1.12 to 1.30 for dynamic condition and 1.27 to 1.43 for static condition. The bench face angle should not exceed 70^0. Bench height is 14m and has width of 7m. The selected ramp width is 18m (without ditch on both side of road, 2m each) depending on the preferred size of dump track. Depth of the pit is ~190m, that means starting from 1100m to 1290m. The calculated stripping ratio for the proposed open pit is 7.43 with 13 years of mine life. The pit has total enclosed volume of $89x10^6$ m^3 with longest diameter of 1.19Km.

Key words: Okote, open pit, RMR_{89}, inter-ramp slopes, overall slopes, factor of safety

Acknowledgement

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I am deeply thankful to my families for their continuous help and encouragement throughout my thesis work. I also like to thank all my friends who are contributed to this work directly or indirectly. Finally, I would like to thank Wollega University for sponsoring my study.

Declaration

I, hereby declare that this thesis is my original work and has not been presented for a degree in any other university, and that all sources of material used for the thesis have been duly acknowledged.

Shibiru Merga	___________________	_________________
Student	Signature	Date

This thesis has been submitted for examination with my approval as university advisor.

Dr. Worash Getaneh	___________________	_________________
Advisor	Signature	Date

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List of acronyms

AD	Aplite Dike
AGB	Adola Gold Belt
CAS	Carbonate Amphibole Schist
CCS	Carbonate Chlorite Schist
CS	Chlorite Schist
DH	Drill Hole
Ei	Elastic Moduli
FF	Fracture Frequency
GSE	Geological Survey of Ethiopia
GW	Groundwater
LDGM	Lega Dembi Gold Mine
MD	Metadiorite
MG	Metagabbro
MGD	Metagranodiorite
Mi	Intact Material Constant
MPa	Megapascal
MRMR	Mining Rock Mass Rating
NMiC	National Mining Corporation
Pli	Point Load Index
PPL	Plane Polarized Light
QV	Quartz Veins
RF	Rainfall
RMR	Rock Mass Rating
RQD	Rock Quality Designation
SCR	Solid Core Recovery
SH	Strongly Healed
SS	Slicken Sided
TAS	Talc Tremolite Schist
TCR	Total Core Recovery
Ti	Intact Tensile Strength
UCS	Uniaxial Compressive Strength
UNDP	United Nations Development Program
UTM	Universal Transverse Mercator
WH	Weakly Healed
XPL	Cross Polarized Light

CHAPTER ONE

1. INTRODUCTION

1.1. Background of the Study

Mining sector is ranked as the second human endeavor after agriculture. The two industries considered together as the primary or basic industries of early civilization in the world. Compared to other sectors agriculture and mining sectors are continuously supplying basic resources for human beings. From the pre-historic times to the present, mining has been playing a vital role in modern civilization (Hartman, 1992).

The term mining in its very broadest context, is extraction of any naturally occurring mineral substances, liquid, solid and gas, from earth's surface and subsurface. Extraction of these resources from the earth's subsurface is not as such simple because, most of the time target commodities found under the earth's surface are enclosed by unwanted materials that may even make them worthless from economic point of view. Both geological and non-geological factors affect economics of mineral resources extraction. The most important and primary geological factors affecting economics of mineral extraction are location, extent or tonnage, grade, morphology, attitude and depth of orebodies. An open pit mine may be excavated within relatively uniform material types (for example, clayey paleochannel deposits) or combination of materials. In many open pits, the wall profile may take the form of a completely weathered rock (with essentially soil like engineering properties near the surface), grading through highly to slightly weathered rock (with both soil and rock engineering properties) to hard fresh rock materials at depth.

Subsequently, pit wall design is a significantly challenging task, and the mine operator must ensure that, through the diligent application of sound geotechnical engineering practice, safe open pit mine slopes are maintained in any geological environment.

Different approaches are used to design an open pit. These are geological, geotechnical, structural and hydrogeological modeling approaches (Abzalov, 2016; Hustrulid *et al*., 2013).

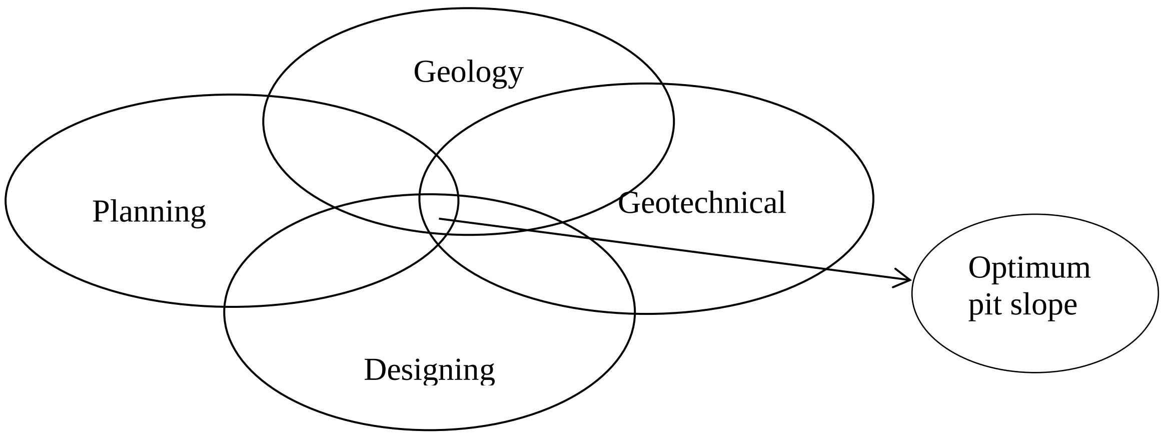


Fig. 1.1 Idealized interaction between geology, geotechnical, planning and designing groups in pit slope creation (modified after Hustrulid, *et al*. 2001)

This research work focuses on the geological and geotechnical considerations used in open pit planning and designing of Okote gold deposit.

1.2. Previous Works

Adola Greenstone Belt (AGB) has been under several stages of studies mainly targeting the placer gold depost by different scholars since early 20th century. In 1930s, geological studies and placer gold explorations in Adola area resulted in discovering of placer occurrence and even mining started by prospectors in Bedakessa valley (Worash Getaneh, 1994).

Jelenc (1966) has identified nickel and placer gold occurrences in the Adola region, particularly in Bore, Upper and Lower Mormora, Shakisso, Awata and Dawa basins. He was also the first to classify rocks of Adola into Gariboro gneissic terrain and Adola series comprising the Megado volcano sedimentary terrain.

UNDP (1972) classified the rocks of Adola region into Upper, Middle and Lower complexes that are interpreted to be separated by both lithological and structural unconformities. Kazmin (1976) later categorized the complexes into several subdivisions and applied to all metamorphic rocks of Ethiopia.

The Megado and Kenticha greenstone belts (the two well-known greenstone belts in Adola) were interpreted as mafic-ultramafic-sedimentary assemblages of ophiolites obducted during collision of continental blocks (Kazmin 1976). They are known for their distinct type of mineralization.

Megado terrain comprising low-grade meta-volcano-sedimentary and mafic-ultramafic associations is characterized by gold mineralization and the Kenticha terrain composed of high-grade gneisses, schists and associated ultramafic complexes is known for its rare metal mineralization.

The Adola Gold Belt (AGB) or Goldfield of Southern Ethiopia has undergone exploration for an extended period of time (Tadesse, 1999). While a known deposit is undergoing active exploitation, the LDGM (Lega Dembi Gold Mine) several previous workers have inferred that the remainder of the Adola Belt of southern Ethiopia remains prospective for Au and even base metals (e.g. Getaneh, 1994; Worku, 1996).

Geophysical investigations consisting of radiometric, resistivity and magnetic methods were conducted in Okote area by Bulbul – Ageremariam mineral exploration project in 1988 and 1989. In 1993/1994, an airborne geophysical survey consisting of electromagnetic, radiometric and magnetic methods was carried out in Borena region including Okote area by Geological Survey of Ethiopia. The aim of the survey was to provide information about geology and structure of the area.

Based on the agreement between Geological Survey of Ethiopia (GSE) and National Mining Corporation (NMiC), resistivity survey was also carried out in Okote area in 1997 and enabled to delineated high chargeability corresponding to high apparent resistivity having considerable depth and lateral extensions.

Debele (2004), identified ore minerals of Okote area as pyrite, chalcopyrite, pyrrhotite, gold, rarely chalcocite, covellite, galena and melonite are the ore minerals of the vein and wall rocks and the most common alterations as silicification, sulphidation, carbonatization, epidotization and chloritization.

Abu (2005) described Okote gold mineralization as sulfide rich auriferous quartz veins of mesothermal mineralization occurring along highly fractured contact zone between massive amphibolites and amphibole schist and to a minor extent along the schistosity planes of the chlorite carbonates schist.

The Okote gold deposit is found in low grade meta-volcano-sedimentary and mafic-ultramafic associations of southern Megado terrain. According to recent studies by National Mining Corporation, Okote deposit has two forms; shear hosted vein type gold mineralization and metagranodiorite hosted gold mineralization.

Solomon (2015) studied granodiorite hosted gold mineralization and gave account to its economic evaluation as the second type of primary mineralization in the area in addition to the early identified vein type mineralization.

Okote gold deposit is currently under definitive feasibility study by National Mining Corporation (NMiC).

1.3. Objectives of the Present Work

The overall objective of pit slope design is to determine the steepest practical slope angles for the open pit mine so that the operator can maximize the extraction of the gold ore. The general objective of this research is concerned with characterization of Okote ore body and enclosing materials to plan and design the open pit, giving emphasis to optimal exploitation of gold ore. Specific objectives are:

- To study physical property and extent of the deposit
- To study geotechnical characteristics of both ore and waste materials
- To propose an open pit slope design

1.4. Description of Methodology

1.4.1. Geological methods

Literature survey and secondary data collection on the selected geo-scientific problem was carried out. Secondary geological data of Okote gold deposit was collected, prepared and rearranged in a database file for further analysis using Datamine Studio3 software. Geological samples were collected both from outcrop and drill cores, and analysed to describe lithological units present in the area. 8 thin sections and 6 polished sections selected samples were prepared at Geological Survey of Ethiopia (GSE). This was used to identify mineralogical make-up of the rocks, types rocks available, condition of weathering and composition of the ore and waste materials around.

1.4.2. Geotechnical methods

Core sample description and geotechnical logging, basically rock quality designation (RQD) and fracture frequency per meter (FF) from selected core samples was conducted. Geotechnical samples collected from representative drill holes (DH 800 N/1, DH 1300 N/1, DH 1650 N/2, and DH 1700 N/3) at several depths considering weathering profiles. These samples were analyzed for strength parameters such as compressive strength and unit weight at laboratory of

Geological Survey of Ethiopia. The rest parameters were calculated using RocLab software. While geotechnical logging the following activities were carefully carried out:

- ✓ Recording of the basic drill hole identification data on the log,
- ✓ Geotechnical logging of the solid core, including the rock type, the degree of weathering and/or alteration, the strength of the intact rock, total core recovery (TCR), solid core recovery (SCR) and the rock quality designation (RQD)
- ✓ Geotechnical logging of the fractures that intersect the solid core, including details of fracture, foliation and shear orientation relative to the core axis (figure 5.4) and fracture spacing and infillings.

Fig. 1.2 Core samples and geotechnical evaluation

Rock mass classification was carried out to study the overall quality of the rock mass within pit area. The Benaiwski's RMR_{89} (Rock Mass Rating), Laubscher's MRMR (Mining Rock Mass Rating) and The GSI (geological Strength Index) were used to classify the rock masses. Several geological and geotechnical data were collected both from the site and from the drill holes. This are, discontinuity data collected from field as well as calculated from drill cores using alpha angle method (fig 6.4).

1.4.3. Pit Slope Design

To conduct the geotechnical design for open pit slopes, the following strategic approaches were followed.

- ✓ Site investigation and formulation of a geotechnical model for the pit area.
- ✓ Slope design and stability assessment

The following flow chart shows methodology and general workflow.

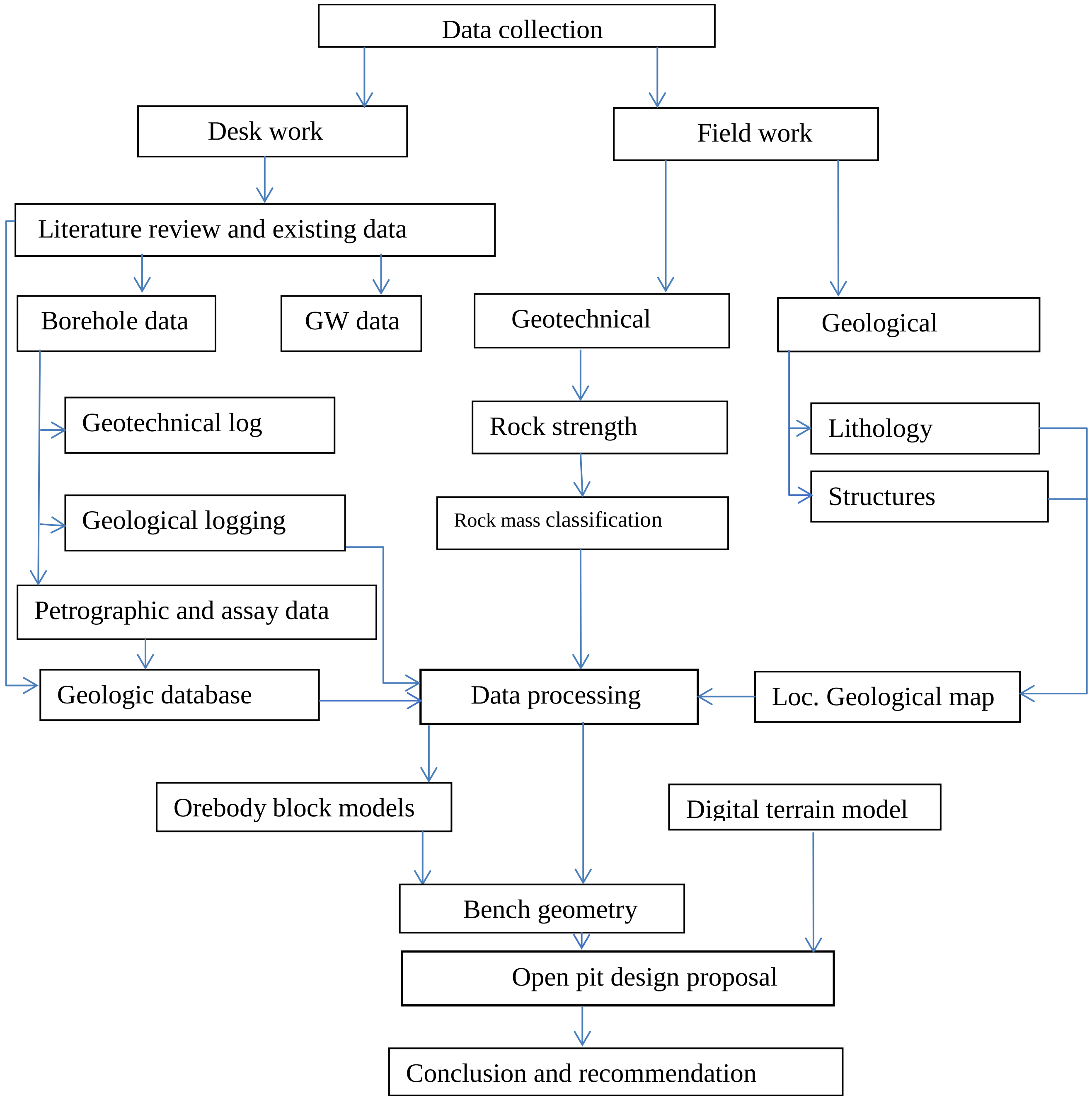


Fig. 1.3 Methodological flowchart

CHAPTER TWO

2. THE STUDY AREA

2.1. Location and Accessibility

The survey area is geographically bounded between 38° 46' 15"- 38° 46' 40" E longitude and 5°07'00" – 5° 07'27" N latitude within Hirmaye sub sheet (D4-0538). The area can be accessed along Addis Ababa-Moyale asphalt road up to Fincha-Woha 560km from Addis. Fincha-Woha to Okote is about 90km along the Fincha-Woha-Shakiso dry weather road.

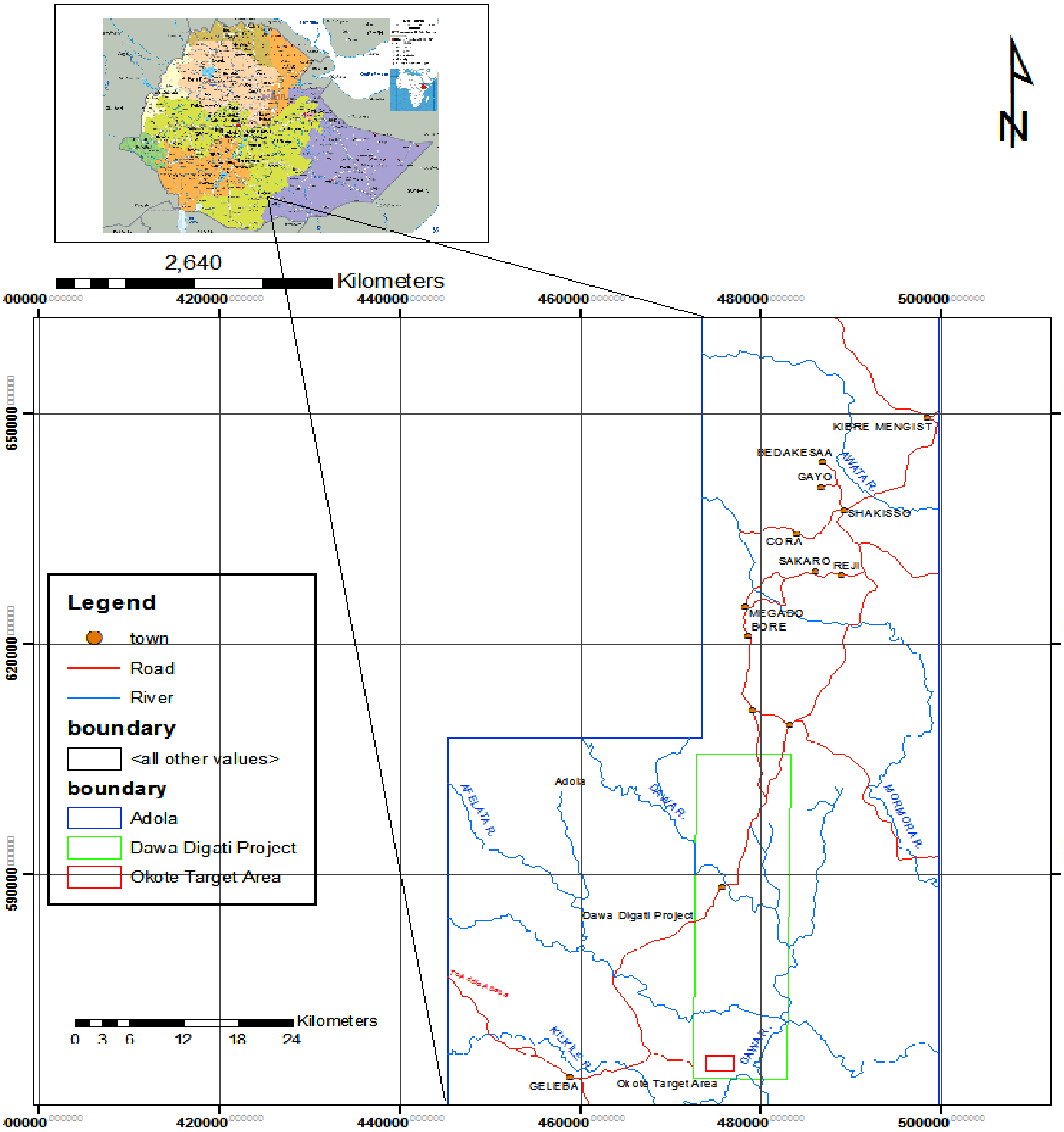


Fig. 2.1 Location map of the study area

2.2. Drainage and topography

The area is characterized by rugged topography with unevenly distributed steep sloping ridges and small hills. Okote valley drains from West to East with its North-South and Northeast dendritic pattern first order tributaries. The Okote Mining area is characterized by very rugged and rolled surfaces with numerous ridges, spurs, vales, deep gorges and valleys bounding the area to the south, east and north west direction. Small intermittent streams and run off courses from north and south of the mining area flows to the valley center to join small intermittent Okote stream. The stream and its tributaries drain the mining area. Okote stream flows from west to east and drains into Okote Guda or Kilkile. All the stream flows down slope forming a pattern like a tree with a trunk and branches commonly known as dendrite pattern, this type of drainage pattern shows that the sub surface geology is characterized by hard rock, mainly Precambrian basement complex which is resistant to erosion. The patterns of the streams are the result of the relief and distribution of hard rocks.

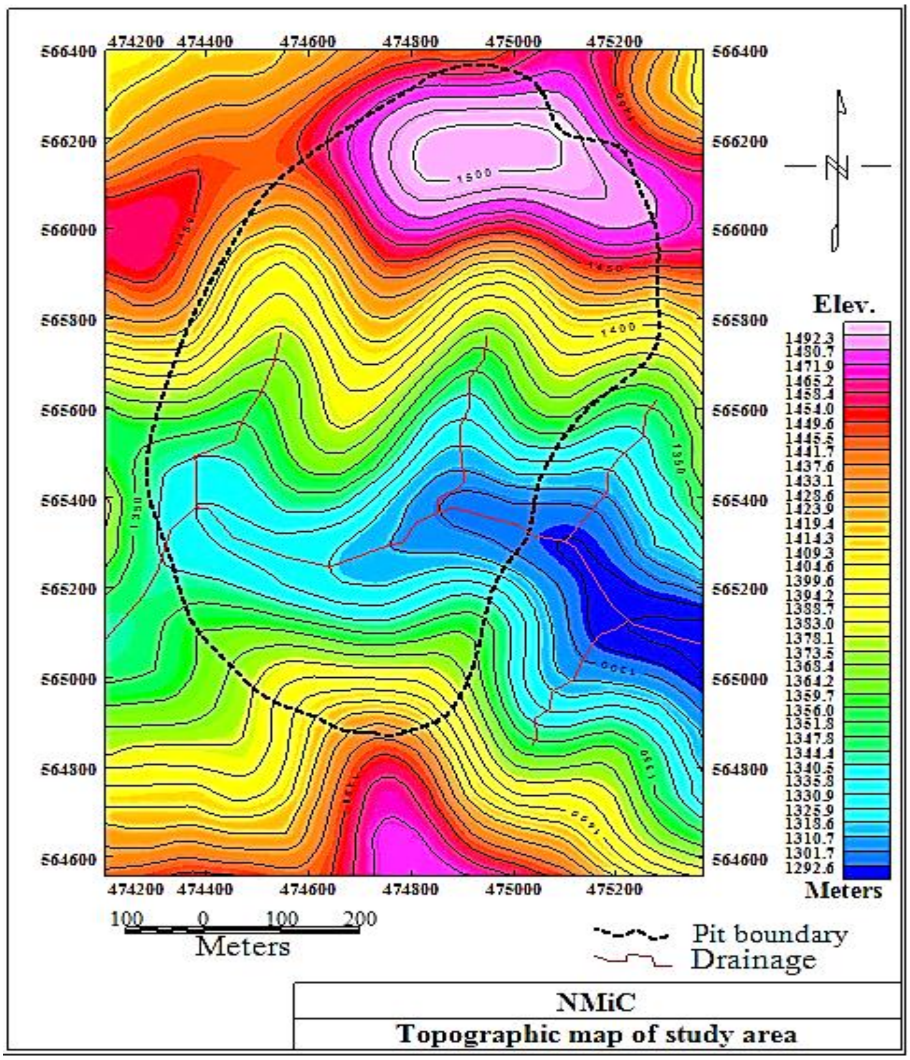


Fig. 2.2 Topography of Okote area

2.3. Climate and Vegetation

The region has a tropical climate with alternate dry and wet seasons, the annual rainfall being less than 800mm. There are two rainy seasons from April to June and from October to November which otherwise remains hot and dry. Daily temperature varies from 20-38^{o}c depending on seasons. The vegetation of the area is sparse to dense. Generally, the ridge tops are covered by grasses and short shrubs whereas the low-lying area and hill slopes are covered by a relatively thicker shrubs and thorny bushes.

Rainfall data of nearby meteorological stations have been obtained from the project office (NMiC, 2016) The stations are Finchwuha (381611 UTM East and 523350 UTM North), Dadime (380330 UTM East and 522285 UTM North), Afraide (371900 UTM East and 525000 UTM North), and Hageremaryam or Bule Hora (381400 UTM East and 539000 UTM North). These stations are situated about 100 kilometers west and southwest of Okote. The amount of rainfall in Ethiopia reported to have a positive correlation with elevation.

Table 2.1 Recorded annual rainfall of the Adola region

Station	Elevation (m)	Annual RF (mm)
Bule Hora	1861	783
Dadime	1579	554
Afraide	1587	763
Finchwuha	1634	754

The annual rainfall of Okote area is approximately estimated to be 200 mm, according to Genale-Dawa River Basin Integrated Master Plan Study (NMiC, 2016) The analysis of meteorological data of the nearby stations, and their corresponding elevations shows that an annual rainfall of 350 to 500 mm is a reasonable and realistic estimation for Okote area.

Table 2.2 Monthly rainfall data from Bule Hora station (National Meteorological Agency, 2015)

	Months											
Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
2003	26.4	3.2	33.6	253	107	nr	nr	61.4	nr	nr	nr	47.3
2004	nr	26.7	83.3	136	87.7	14.5	26.2	17.3	97.3	75.7	106	7.1
2005	18.2	8	107	120	493	73.1	35	11.8	76	139	49.3	0
2006	0	30.7	74.4	142	86.7	12.8	7.4	100.8	18.8	213	152	22.2
2007	8.5	14.4	53.5	195	99.8	71.5	51	122.8	202	151	26.8	0.1
2008	0	0.6	104	96.9	127	34.7	37.1	17.9	85	244	141	0.2
2009	22.8	0.2	40.8	146	155	22.4	6.8	1	49.2	98.9	nr	nr
2010	nr	nr	Nr	163	414	nr	nr	88.6	nr	nr	nr	nr
2011	nr	16.4	4.3	49.8	253	62.3	nr	nr	108	106	247	34.3
2012	nr	0	Nr	nr	nr	30.8	57.4	37	201	nr	61.4	17.4

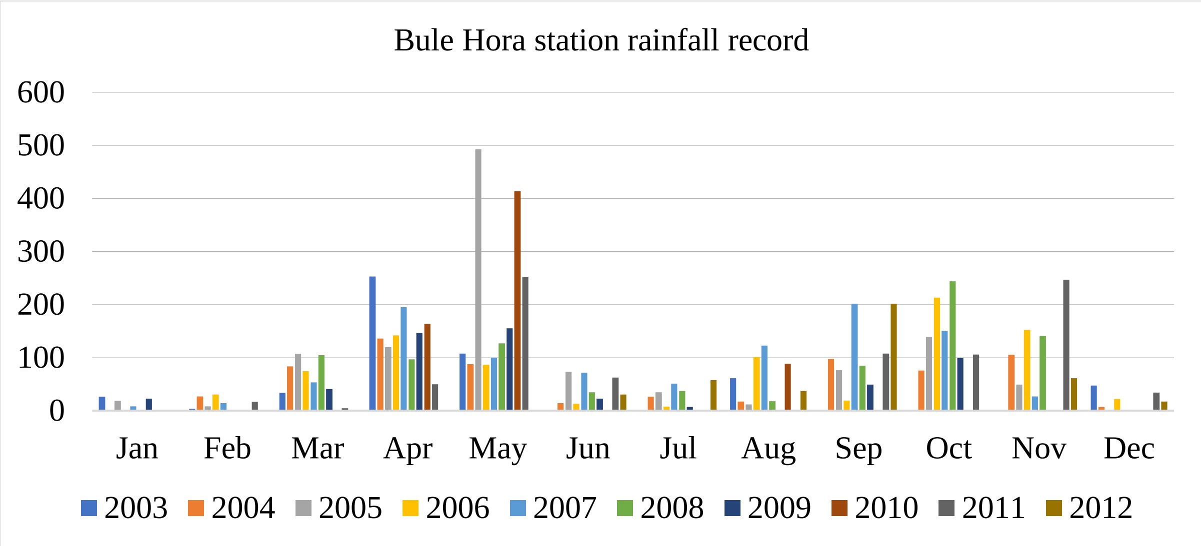


Fig. 2.3 Monthly rainfall at Bule Hora station

CHAPTER THREE

3. GEOLOGICAL SETTING AND MINERALIZATION OF ADOLA REGION

3.1. Regional Geology of Adola Gold Belt

The Adola area is the northern extension of the Mozambique orogenic belt forming a portion of the late Proterozoic intracratonic trough initiated on the ancient consolidated basement (Kazmin, 1972). The rocks in the area include a wide variety of the volcano-sedimentary, terrigenous, and intrusive units formed under conditions of intensive tectonic activity and various degrees of metamorphism.

Adola region has been under several stages of geological studies for extended period of time. The area was first studied by Italian geologists in early 20th century. Even though their main target is exploring for potential areas with gold and other metallic minerals, they have paved a way for other geologists interested in geological studies.

First, Kazmin (1971, 1972, 1975) divided the Ethiopian basement rocks into three-fold classes mainly based on lithological and structural properties of the rocks.

The Lower Complex (Archean age) is primarily composed of gneisses' and migmatites of granitic composition are the oldest rocks of the country. They are mainly composed of amphibole and biotite bearing gneisses with quartzofeldspathic granitic gneisses, calc-silicates and amphibolites. They are crystallized to amphibolite and locally granulite facies.

The Middle Complex (Lower-Middle Proterozoic) are politic and psammitic metasediments (mainly, biotite and amphibole schists), marble and quartzite. They are exposed covering the underlying granite-gneisses complexes. They are known by a well preserved primary sedimentary structure such as cross-bedding. Metamorphism of the middle complex rocks not exceed amphibolite facies.

The Upper Complex (Upper Proterozoic- Lower Paleozoic) is considered youngest succession of the basement rocks in the country. It is thicker than both lower and upper complexes and composed of ophiolite assemblages (amphibolites, actinolite-chlorite schists), metavolcanic, politic, clastic and carbonate sediments. They are mainly exposed in South (Adola group and Kadjimiti beds), West (Birbir group), East (Soka group), and in the North (Tsaliet and Tambien groups).

The Adola belt, trending NS, is characterized by Precambrian rocks comprising basic-ultra-basic and other older basement gneisses which form crustal scale shear zones extending for about 150km along its strike (Yibas *et al.,* 2001; Worku *et al.,* 1989).

Two broad sub-parallel litho-stratigraphical domains are recognized in Adola belt (Worku, 1996). These are: High-grade gneisses and schists and associated ultramafic complexes that can be subdivided into four lithotectonic terrains separated by major tectonic boundaries (Sodda, Shakisso, Kenticha and Zembaba Terrains), and 5-16 km wide low-grade metavolcanosedimentary and mafic-ultramafic association of the Megado Terrain. The domains are surrounded and separated by gneissic rocks of Middle Complex, Kazmin (1972) which is intercalated with marble unit.

Woldehaimanot and Behrmann (1995) also classified metamorphic rock of Adola belt into four major groups:

- The temporally and spatially related Megado and Kenticha meta-igneous-sedimentary rocks, Reji amphibolite and Bursano gneiss-amphibolite assemblage;
- The Daba meta-igneous complex;
- The Awata gneiss; and
- The gneissic complexes.

Woldehaimanot and Behrmann (1995) identified major thrust zones, the Megado and Kenticha thrust zones, considered the most prominent deformation zones that are resulted in the formation of 500m wide mylonitic zone.

Kazmin (1976) suggested that the Megado and Kenticha greenstone belts with their mafic-ultramafic-sedimentary assemblages represent an ophiolite in a suture zone emplaced by the collision of two continents. He interpreted the Kenticha belt as a thrust sheet and the Megado belt as the root. Adola region has led to proliferation of the division of the rocks of the Adola granite-greenstone terrain (Schmerold 1988; Worku and Yifa 1989; Ghebreab 1989). Schmerold (1988) divided the rocks into litho-structural units, namely the Eastern Basement, the metavolcano-sedimentary Belt, the Central Basement, the Ultramafic Belt and the Western Basement, in which gneisses represent basements. Worku and Yifa, (1989) re-named Schmerold's basements as Domains, Eastern Domain, the metavolcano-sedimentary Belt, the Central Domain, the Ultramafic Belt, the Western Domain, in which gneisses represent Domains.

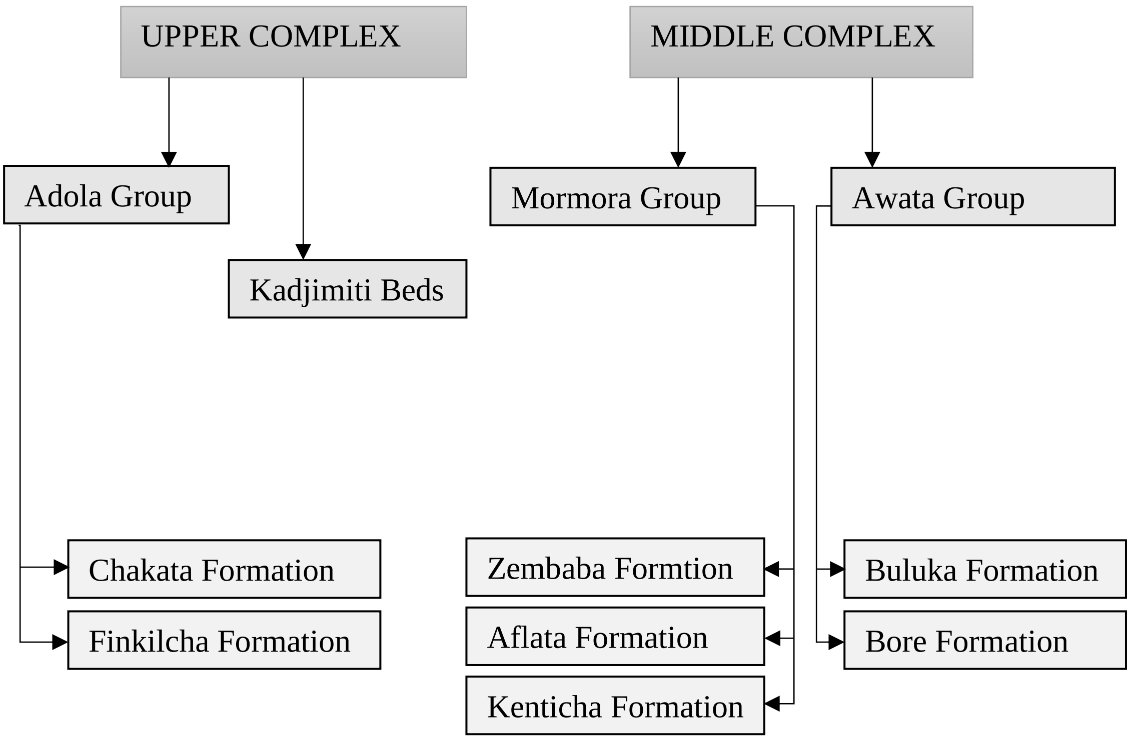


Fig. 3.1 Upper and Middle complex groups of Adola area (after Kozyrev *et al.*, 1985)

3.2. Characteristic Deformation, Alteration & Mineralization Events

Worku and Schandelmeier (1996) propose that the geochemical signatures and structural features of the rock associations of the Adola Belt represent a Wilson Cycle. The formation of the Adola Belt therefore involved the evolution of a passive continental margin and formation of ocean floor in the Kenticha Terrain (fig. 3.2), W-directed subduction, arc development in the Megado Terrain (fig. 3.2), closure of an oceanic basin of unknown size and the collision of crustal blocks. Oblique plate convergence led to a generally-recognized sequence of continuous deformation events, namely (i) subduction-related folding and thrusting (D1), which culminated in the obduction of mafic-ultramafic assemblages onto the passive continental margin sediments, ii) collision of crustal blocks (D2), leading to re-folding of D1 structures, development of upright N-S-trending folds and the generation of reverse faults and shear zones, and finally (iii) evolution of sinistral strike-slip shear zones (D3) with N and NW orientations, the latter being interpreted as antithetic Riedel Shears. Therefore, in terms of classic structural geology, the N-S trending sinistral shears, aligned parallel to the overall N-S trend of lithologies, represent the major shear planes under the NW-SE directed maximum principal compression. Most of the major structures of the Adola Belt are compatible with a NW-oriented stress regime and hence can be interpreted to reflect sinistral transpression. In a larger geodynamic framework, structures related to this collision event and the stress regime that produced them are consistent with the position of East and West Gondwana during the Neoproterozoic.

The number of deformation events recognized in the Adola Gold Belt varies considerably from three (e.g. Worku, 1996; Worku and Schandelmeier, 1996) to six (Bogliotti, 1989). Clearly, there will be some commonality between these two chosen structural schemes despite their origin in terms of locally-derived measurements. Note that the two structural schemes, of which there are several, emphasize different features. Worku (1996) emphasizes the timing and nature of shearing with a focus on the Lega Dembi Gold Mine, while Bogliotti (1989) emphasizes the sequence of folding events in the Adola Gold Belt.

These workers agree however, on several points. The dominant structural fabric of the Adola Belt is N-S in orientation (this can also be clearly observed from geological maps and DTM images of the area), resulting from D2 folding (Worku, 1996) or D4 folding (Bogliotti, 1989). What Worku (1996) and other workers emphasize is that initiation of N-S to NNE-SSW trending, sinistral shears occurred near or after the peak of metamorphism. Mylonites along these shears contain intense mineralized veining at Lega Dembi, with internal, en-echelon sub-zones of enhanced shearing. Mylonitic zones also contain deformed and fragmented mineralized and non-mineralized metamorphic rocks, which are severely affected by shearing and grain-size reduction. Au-bearing quartz (±carbonate) veins show en-echelon patterns, indicating or supporting an overall sinistral transpressional model (see section on Okote), although the issue of transpression will be addressed later in this report. Breccia and intense stockwork zones, which suggest a degree of very late-tectonic movement (i.e. faulting), also host some of the high-grade Au mineralization. These stockworks consist of abundant random quartz veinlets which contain Au and associated sulphides that are disseminated in breccia clasts and inter-vein material (i.e. “matrix”). The latter is altered to albite, minor sericite and quartz (Gebreab *et al*., 1992; Worku, 1996). It will be seen later that this mode of mineralization is very similar to that found at Ejersa South.

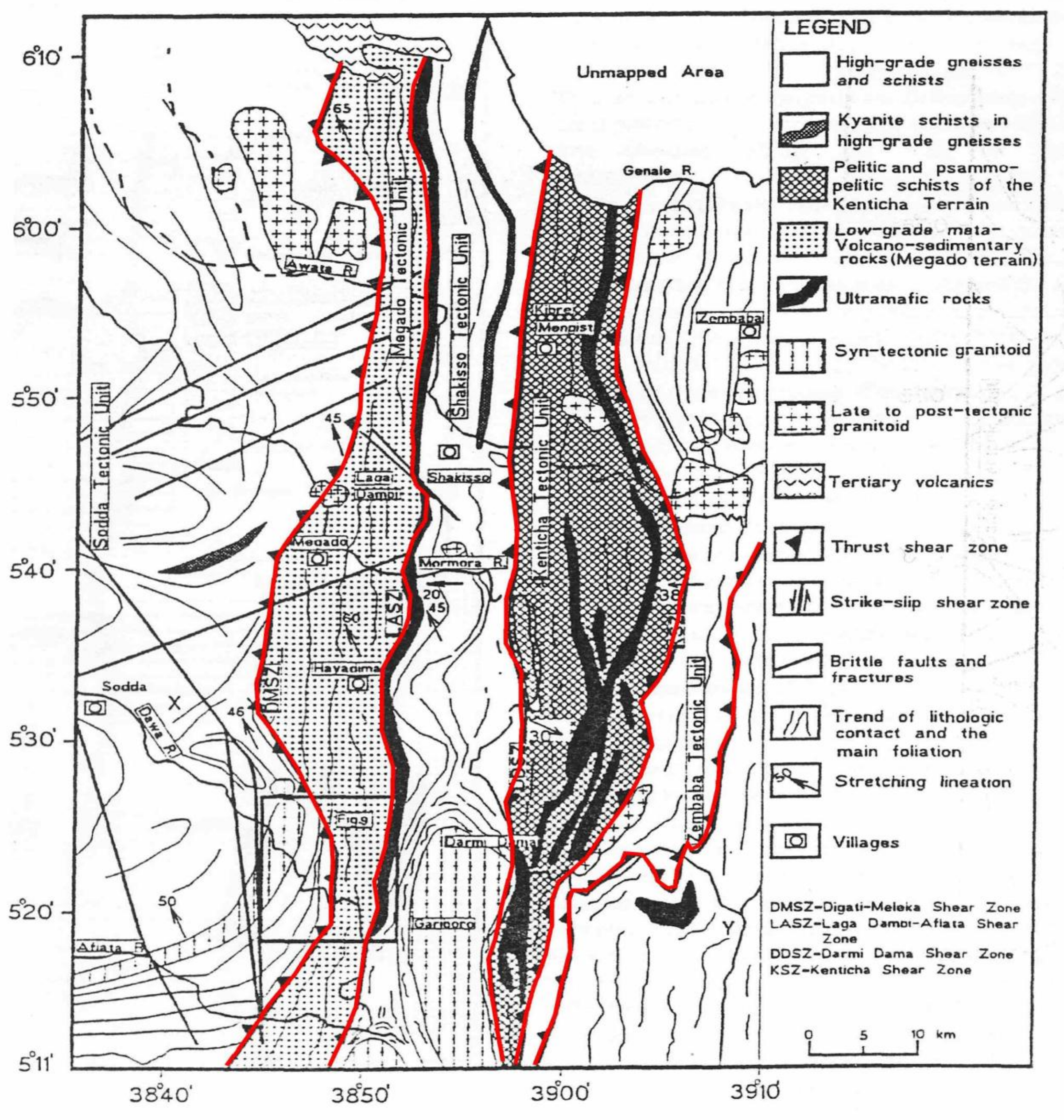


Fig. 3.2 Division of the Adola Gold Belt into three tectonometamorphic terrains.

(a) metamorphosed passive continental margin sediments, mafic-ultramafic rocks and associated pelitic metasediments (Kenticha Terrain); (b) high-grade gneisses and schists, intruded by syn-tectonic calc-alkaline magmatic rocks in the central and western part of the Adola Belt; and (c) low-grade metavolcano-sedimentary and mafic-ultramafic rocks, and associated granitoids (Megado Terrain). Adapted from Worku and Schandelmeier (1996).

CHAPTER FOUR

4. GEOLOGY AND MINERALIZATION OF OKOTE

4.1. Introduction

The development of a mineral inventory involves substantial judgement, assumptions being made regarding sample and assay quality, and the interpretation and projection of geologic features based upon very limited data. The geologic data base, properly gathered and interpreted, should remain useful for many years. It forms the basis for current and future feasibility studies, mine planning and financial analyses. The success or failure of a project can thus be directly linked to the quality of its recorded data base, the drill logs and the maps.

Mineralization at Okote is hosted within a relatively narrow shear zone or zones (estimates of its width range from 10 to 20 metres) which cross-cut(s) the trend of the dominant foliation and relict lithological contacts at a very low horizontal angle (8° - i.e. at a strike of 204°). This zone consists of en-echelon, clear/glassy quartz veins which contain tourmaline, pyrite (and other sulphides) that are notably mineralized with Au. Veining and mineralization within this zone are inherently discontinuous, as in similar shear-zone-hosted vein Au deposits worldwide. In effect, one must quantify the discontinuity and distribution of grade within these mineralized zones, rather than trying to avoid it. Such discontinuity, while being unavoidable due to the genesis of these deposit types, can be quantified, using statistical methods. Mineralized veins, which formed in NNE-trending Late-D_2 to D_3 left-lateral shears, are distinct from D_1 to Early-D_2, milky white, highly deformed, segmented and unmineralized quartz veins (NMiC, 2016). Sub-vertical, ductile shear zones are typically very deep; consequently, the minimum depth extent of the mineralized zone at Okote should be determined by deep drilling. Okote is possibly the most promising target of the sub-areas within the Dawa Digati License area.

The Ejersa South sub-area straddles the contact between an amphibolite to the west and a metasedimentary sequence to the east. This contact, which has been delineated for about 1600 metres, defined by graphitic- or graphite-quartz-mica schist, is highly sheared. The contact contains an “annealed breccia” consisting of mineralized quartz vein fragments with lenses of amphibolite. Au mineralization at the contact is much more evenly spread and regular than that at Okote. The abundance of “free gold” (that is, liberated by brecciation) is evident from the widespread alluvial Au mining in the near vicinity. As such, the Ejersa-South type of mineralization is described as “modified Okote”, in that it appears to be a brittlely-reactivated (originally ductile) shear zone, wherein mineralized quartz veins and shear zone material have become brecciated.

4.2. Geology of Okote Mineralized Zone

4.2.1. Lithologies

Okote gold deposit is identified as shear hosted mineralization in multiply deformed low-grade meta-volcano-sedimentary units. The geology of Okote prospect is part of the southern extension of the meta-volcano-sedimentary rocks of the Megado Belt of Adola region. The study area broadly comprises felsic and mafic rocks with intermixing near their contacts. metagranodiorite and aplitic dykes are the felsic rocks while metagabbro/amphibolite, metadiorite, chlorite-amphibole schist, chlorite schist and talc schist are the intermediate-mafic-ultramafic rock units which are outcropping in the area.

These units are:

- Chlorite Amphibole Schist (CAS),
- Chlorite Carbonate Schist or Chlorite Schist (CCS/CS),
- Weakly foliated Metagabbro (MG),
- Talc Trimolite Actinolite Schist (TAS),
- Meta-granodiorite (MGD),
- Meta-diorite (MD).
- Aplitic dikes (AD) and
- Quartz veins (QV)

4.2.1.1. Chlorite Amphibole Schist

This unit covers largest part of the area (~38%). It shows elongated chlorite minerals defining foliation of the unit. Several smaller rock units such as amphibole schist, gabbroic lenses and chlorite schist are found within Chlorite Amphibole schist unit. All the rocks within this unit are generally fine to medium grained and highly schistose.

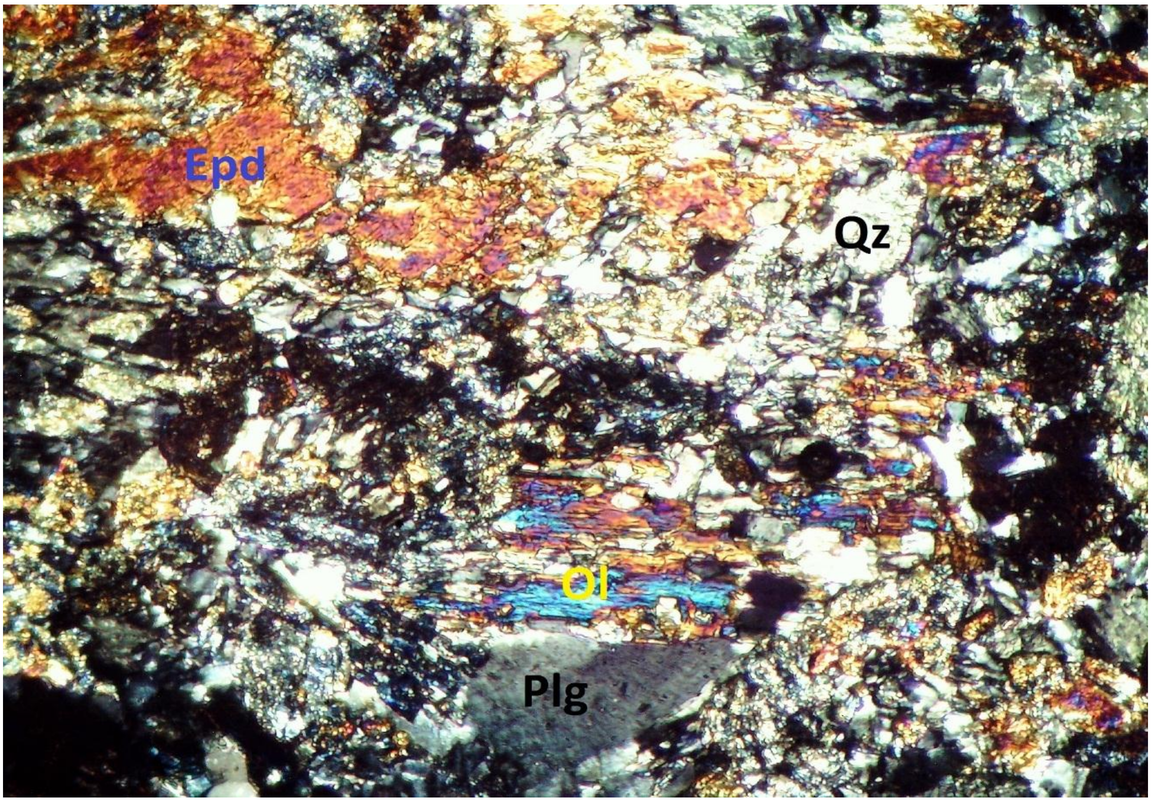


Fig. 4.1 Micro picture of chlorite amphibole schist.

Epidote (Epd), plagioclase (Plg), chlorite (Chl) and quartz (Qz) minerals with opaques. XPL with 20X10 magnifier.

4.2.1.2. Carbonate Chlorite Schist or Chlorite Schist (CCS/CS)

Carbonate chlorite schist dominates eastern and southwestern part of North Okote project area. Distinct feature of the rocks in this unit is that calcite and chlorite minerals are abundant (~50 and 45% respectively) and the rest is covered by chlorite, calcite, quartz, plagioclase, and opaques.

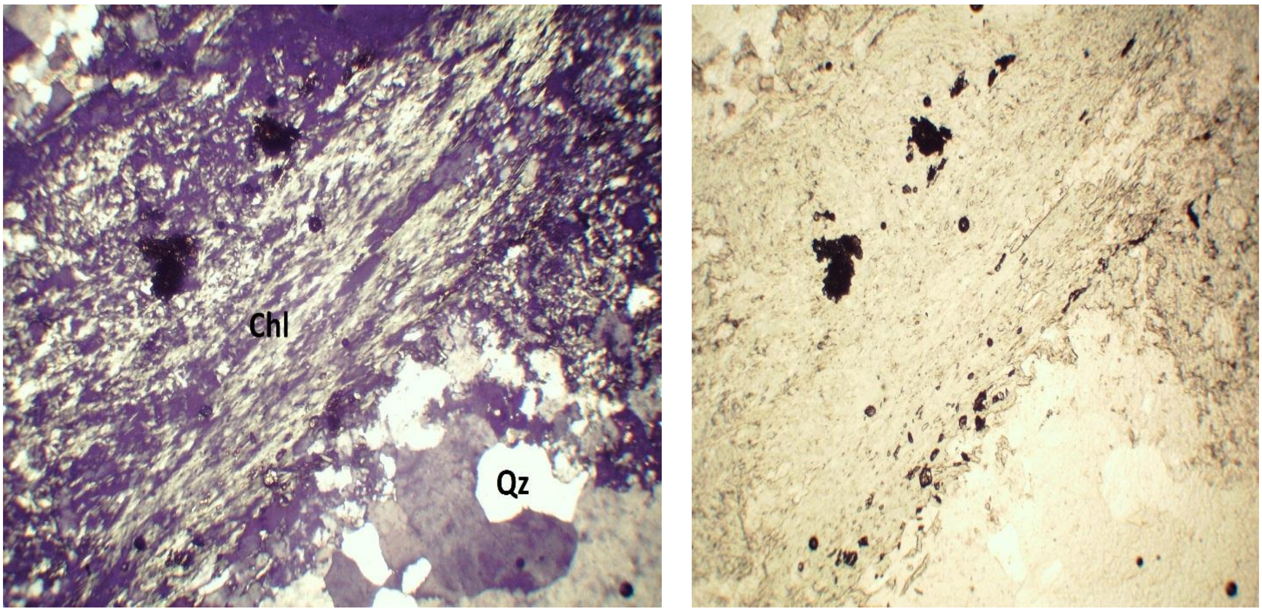


Fig. 4.2 Photomicrograph capture of carbonate chlorite schist: showing flaky chlorite defining foliation, calcite (Cct), quartz (Qz)with few chlorite (Chl) and biotite Bt) minerals. Black dots are opaques (Fe-oxides and/or sulphide minerals). Magnifier 20X10 both under ppl and xpl.

4.2.1.3. Metagabbro (MG)

The metagabbro unit in the area is massive and weakly foliated. It is widely exposed as lenses within chlorite carbonate rocks. Olivine, orthopyroxene, clinopyroxene, hornblende and plagioclase are the main primary minerals. The rock shows some traces of metamorphic minerals such as tremolite and actinolite.

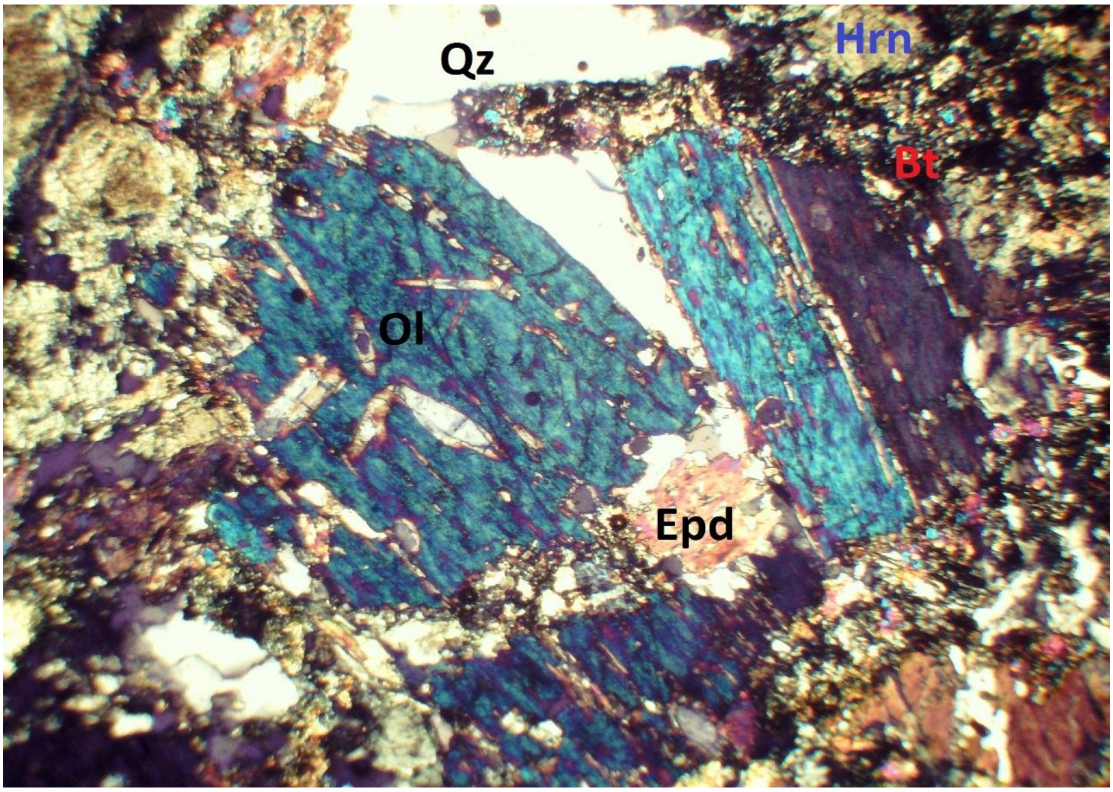


Fig. 4.3 photomicrograph of metagabbro. Phenocryst of olivine (Ol), epidote (Epd), hornblende (Hrn), biotite (Bt) and recrystallized quartz (Qz). XPL with 20X10 magnification power.

4.2.1.4. Talc Trimolite Actinolite Schist (TAS)

Talc Trimolite Actinolite Schist occurs as discontinues patchworks along the strike of the main shear zone in north Okote area. Talc, tremolite, actinolite and quartz are main minerals found in this rock unit.

4.2.1.5. Meta-granodiorite (MGD)

This unit is exposed in some parts of Okote area as dyke with various shape and size, having intrusive contact with the surrounding rocks. The general strike of the unit varies between N40º – 60º E. MGD unit mostly found in northern part of the mapped area and extends to central part along strike of the main shear zone. It is comprised of quartz, muscovite, sericite and feldspar minerals.

A) XPL B) PPL

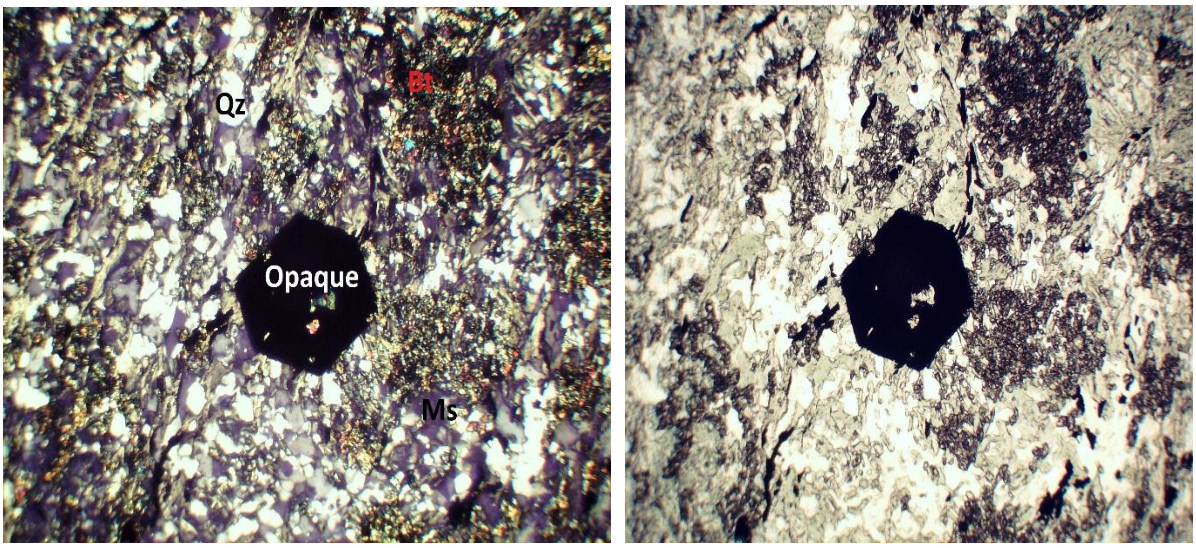


Fig. 4.4 Thin section photo of metagranodiorite: Sericitized muscovite, quartz and some chlorite mineral. The black one in the center of the picture is opaque (pyrite?). 20X10 magnification power for both XPL and PPL.

4.2.1.6. Meta-diorite (MD)

Meta-diorite unit is rarely exposed in western part of North Okote area within Chlorite Amphibole Schist.

4.2.1.7. Aplitic dikes (AD)

Aplitic dykes are the felsic rocks exposed stretched within the Chlorite Amphibole schists and metagabbro. It is clear brown to deep grey in the field. Lenses of aplitic dykes are widespread, intruding the dominant mafic rocks.

4.2.1.8. Quartz veins (QV)

Quartz veins are hosted by both felsic and mafic rocks. The veins are aligned along the shear zones and cut through the metavolcanics, metagabbros and metagranodiorites. Most of the veins are commonly deformed, fractured, faulted, locally folded, dragged and are exposed in an en-chelon array. In some vein's mineral lineation parallel to the vein margins is obvious, and could be traced by the presence of chlorite, magnetite and pyrites.

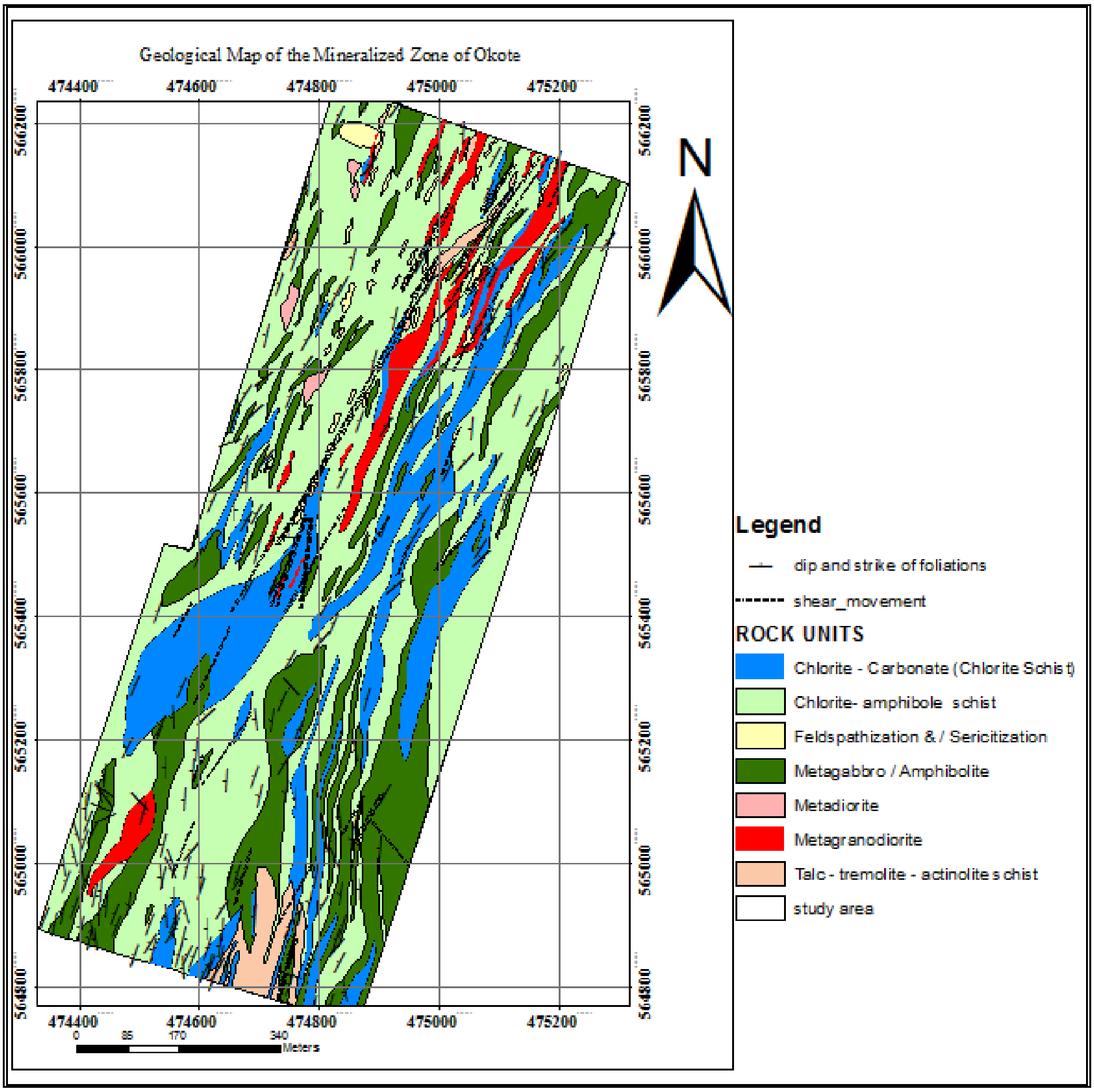


Fig. 4.5 Geological map of Okote mineralized zone (modified from NMiC, 2016)

4.3. Mineralization of Okote

Recent studies indicated that Okote gold mineralization is assumed to be originated primarily from Juvenile porphyry type metagranodiorite gold mineralization latter on followed by shear hosted mesothermal lode carbonate-chlorite hosted gold depositions. Mineralizations in Okote are related to wall rock alteration like chloritization, potassic, phyllic (sericitization), argillic (kaolinization), epidotization, pyritization, tourmalinization, carbonation and silicification. The intrusive related mineralization is far away from major geological structure while the lode gold is confined by or found adjacent to the brittle-ductile shear zones. These results deduct that the gold occurrence of entire Okote is controlled by both lithologies and structures. There are two well identified types of gold mineralizations (NMiC, 2016).

- Carbonate-Chlorite related mineralization -TYPE-I
- Metagranodiorite hosted mineralization-TYPE-II

A detail structural reveals that, the only structural control in favor of gold mineralization is the shear foliation striking 018-198 and dipping towards west. This narrow shear zone coincides with the shear hosted mesothermal lode carbonate-chlorite hosted gold mineralization rather than the juvenile porphyry type metagranodiorite hosted gold mineralization, since the metagranodiorite unit exhibits an alternating foliation dipping both towards west and east. This suggests that major structural controls have a lesser relevance to metagranodiorite gold mineralization.

4.4. Ore Compositions

The main metallic mineral is chalcopyrite, pyrite, pyrrhotite, followed by less magnetite, and ilmenite. There are also a little of limonite, sphalerite, hematite and rutile, and trace amount of galena, arsenopyrite, sheelite, molybdenite (NMiC, 2016). The gold minerals are mostly native gold (95. and trace amount of electrum and petzite. The main gangue mineral are plagioclase and quartz, followed by less chlorite. There are also a little biotite, dolomite, calcite, and trace amount of muscovite and albite.

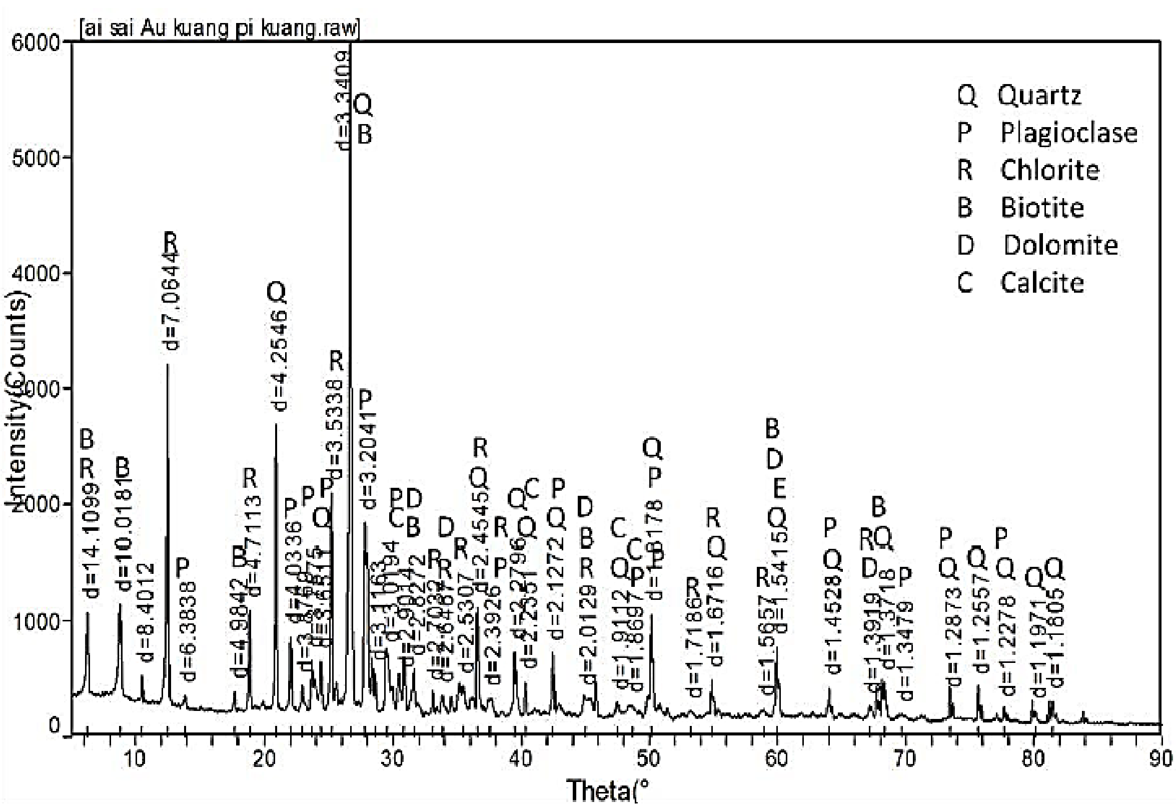


Fig. 4.6 X-ray diffraction analysis image of the ore (NMiC, 2016)

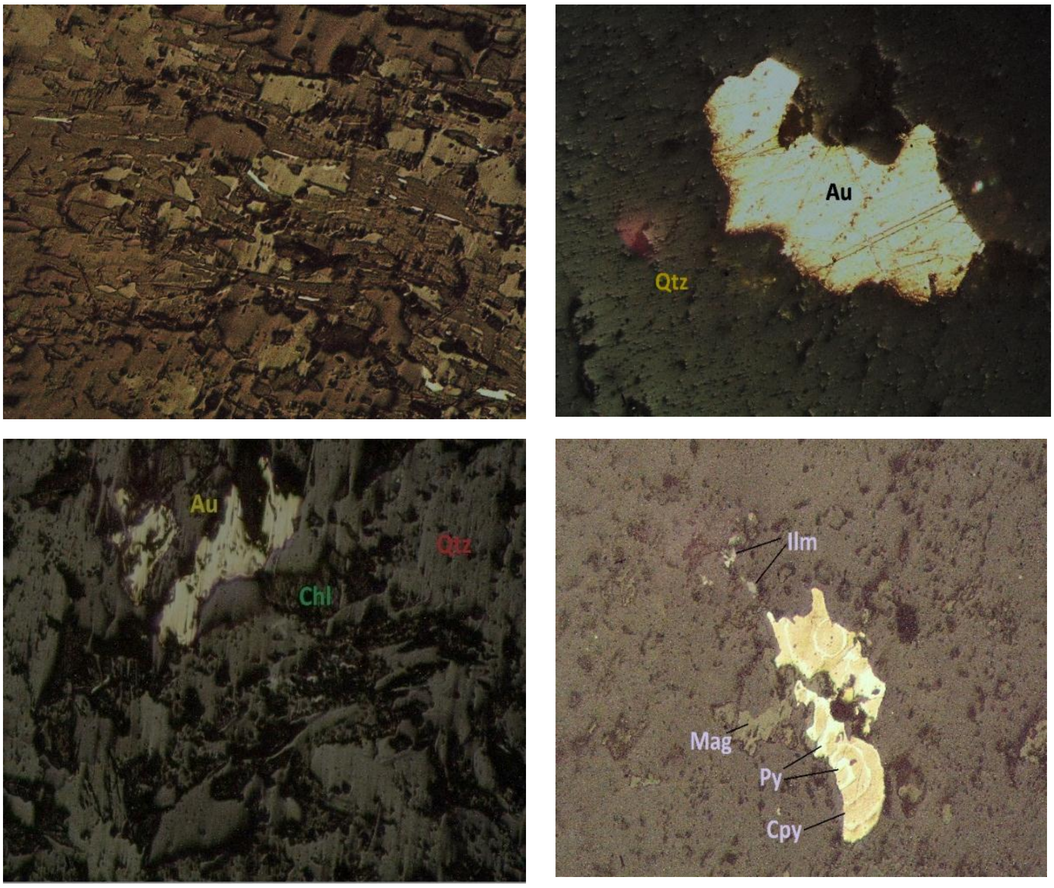


Fig. 4.7 Polished section photos showing different ore minerals. Pyrrhotite and rutile needles in Carbonate Amphibole Schist (top left), Euhedral free gold (Au) grain in quartz vein (top right), Gold leaflets in chlorite (Chl) mineral and Quartz (bottom left), Chalcopyrite (Cpy) and pyrite (Py) paragenetic with magnetite (Mag) and ilmenite (Ilm) (bottom right).

4.5. Gold Distribution in the ore

The result of optical microscope observation and scanning electron microscope (SEM) analysis carried out by NMiC shows that, Au element in the ore exists as independent mineral. Most of the gold minerals are native gold, and there is only trace amount of electrum, petzite etc. The Au distribution rate in native gold is 99.51%, 0.48% in electrum, only 0.01% in petzite. In general, native gold in the ore is characteristic of locally concentrated feature (NMiC, 2016).

4.6. Resource Estimation of Okote

An early task in mine management is the establishment of an accurate model for the deposit. Though a number of models are available, the regular 3D fixed-block model is the most commonly used and is the best suited to the application of computerized optimization techniques.

The information in a block model is gathered from a series of drill holes. Typically, many long, narrow holes are drilled into the ground in the vicinity of the orebody, and their cores are extracted and analysed for mineral concentrations. For simplicity, in the sequel we will assume that the only relevant information contained in the block model is the total tonnage of each block, and the Au grade. Thus, one knows precisely the density of the rock in the drill hole core and the grade along the core. The drill hole cores provide a sparse set of data from which we must construct a full three-dimensional model of rock tonnage and percentage by mass of the metal element in each block. This construction is commonly performed using a process known as Kriging. The kriged estimate of the block model is derived as a local linear interpolation of the measured drill hole grades.

If one assumes that the linear correlation of the grades of pairs of blocks depends only on the distance between the blocks and the direction in 3D from one block to the other, then the Kriged estimate of blocks grades is the best linear estimator of the block grades ("best" in the sense of minimum variance). The estimates have been performed by diamond drilling and surface trenching as well as a digital terrain model (DTM) of the surface.

The Okote resource model includes dilution and mining criteria incorporated in the resource interpretation process (Arvidsson 2001). Further processing is completed to convert the resource model into a format suitable for mine design and reserve calculation. Blocks are created using Datamine Studio 3 plan perimeters and in some cases wireframed interpretations. Although wireframes may provide some advantages over perimeters, the scale and complexity of the deposit makes generation and maintenance of wireframes time expensive. Component models are created separately and include a geological model (density, oxidation and lithology), date model (current and past topographical surfaces), exploitation model (stopping and surface fill), dump model and an ore model. The combined block model for each local map sheet area is used to update the total Okote resource model.

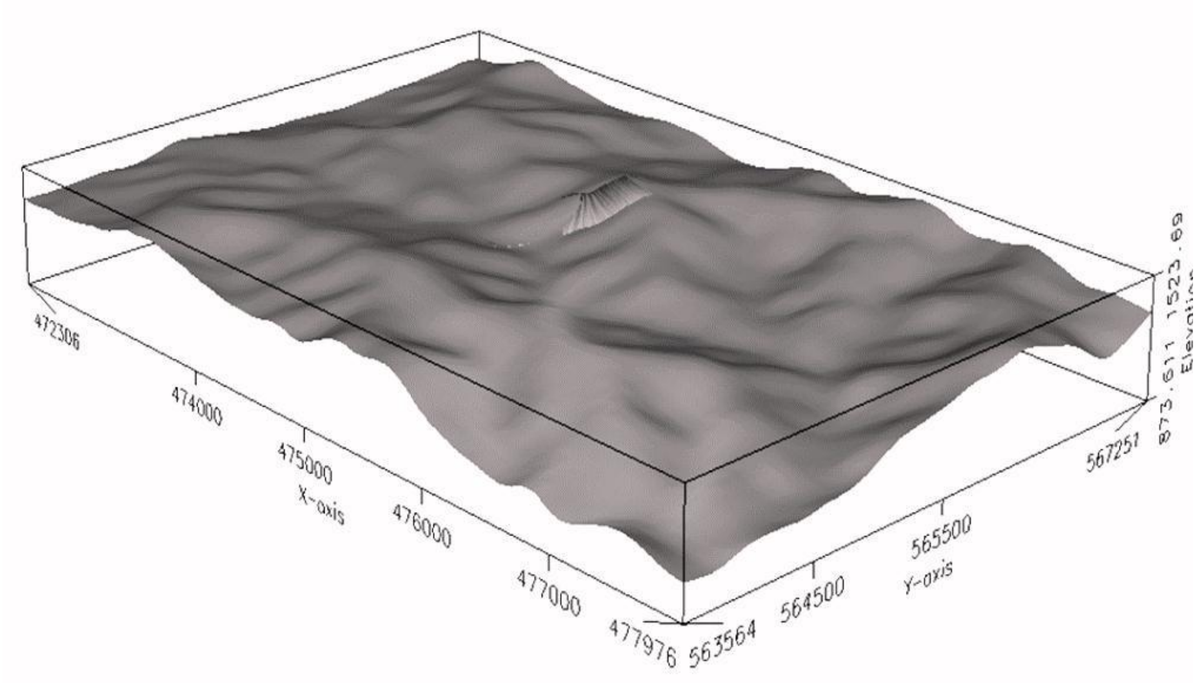


Fig. 4.8 Digital terrain model of Okote deposit area derived by Datamine Studio3.

Ordinary kriging method is used to produce block models for Okote gold deposit using Datamine studio3 package. Attempts to define the orebody by geological controls were successful because, a clear relationship between rock types and mineralized zones was established.

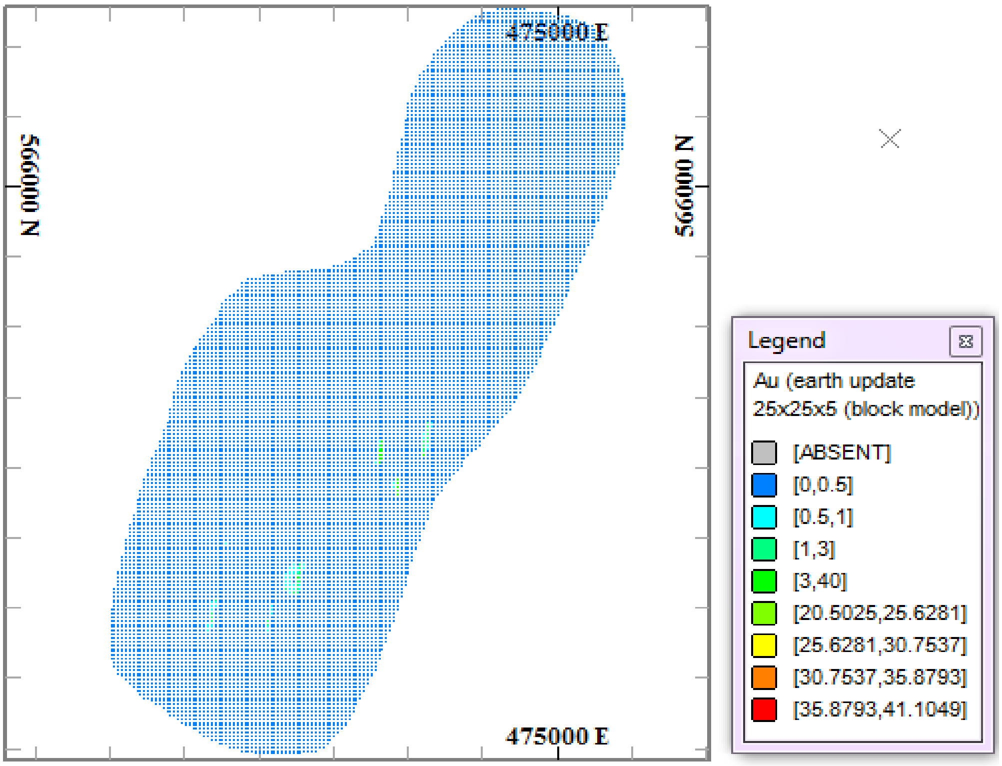


Fig. 4.9 Planar view of the deposit

The delineated deposit for the currently proposed open pit contains 77 million tons of ore at reserve level with average grade of 1.18 g/ton and cutoff grade 0.49 Au g/ton. The contained value (total amount of Au in the deposit in kilogram) of the deposit is 23,062 Kg. the calculated stripping ratio for the proposed pit is 7.43:1.

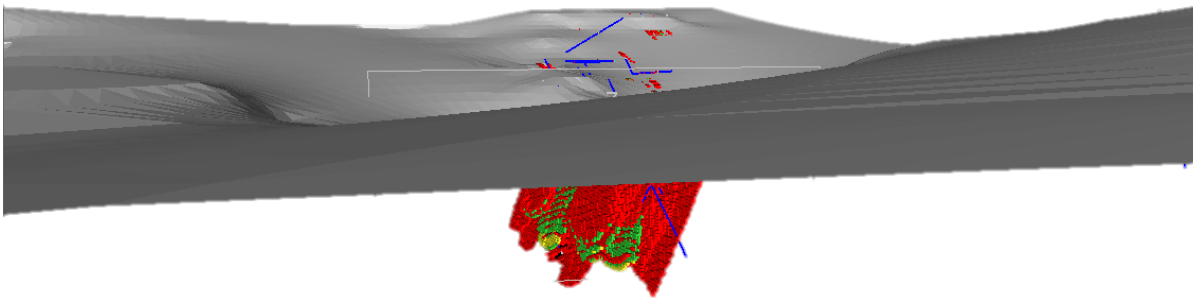


Fig. 4.10 Simulated 3D visualization of Okote gold deposit using datamine software (studio3)

4.7. Attitude, Shape and Extent of the Deposit

According to Hustrulid *et al*., (2013) modeling in mines starts with detail description and quantification of the ore body, including location, tonnage, grade, extent, attitude and morphology. Okote gold deposit is NNE-SSW aligned with maximum length of about 900 meters and maximum width of 196 meters. This is measured after the orebody is displayed by the Datamine studio3 software package.

CHAPTER FIVE

5. LITERATURE REVIEW

5.1. Geotechnical Analysis

The stability of rock slopes has traditionally been evaluated by limiting equilibrium methods (Hoek and Bray 1981), although probabilistic-based approaches are increasingly more commonly applied, because they acknowledge the implicit uncertainties of limit equilibrium methods. Limit equilibrium models fall into two main categories:

Models that deal with structurally controlled ***planar*** or ***wedge slides*** and those that deal with ***circular*** or ***near circular*** failure surfaces in homogenous materials.

Many of these models have been available for more than 25 years and can be considered reliable slope design tools. Assembling basic geological data, rock strength information, and groundwater observations, and integrating this with engineering rules in the form of design charts and graphical methods to permit a non-specialist engineer to obtain approximate answers suitable for assessing open-pit alternatives. The geotechnical analysis is the fundamental basis for all slope designs and is compiled from four component models (Peter and Stacy, 2009). These models are:

- the geological model;
- the structural model;
- the rock mass model (material properties);
- the hydrogeological model.

These models also have applications for other aspects of the mining operation, for example in ore reserves and mining operations. However, particular aspects of each are critical for the slope design process. Peter and Stacy (2009) also described the probability-based slope design methodology the need to define the reliability of the data in the geotechnical model has increased significantly. Consistency and reliability of data is increased with the project development stages as illustrated in diagram below.

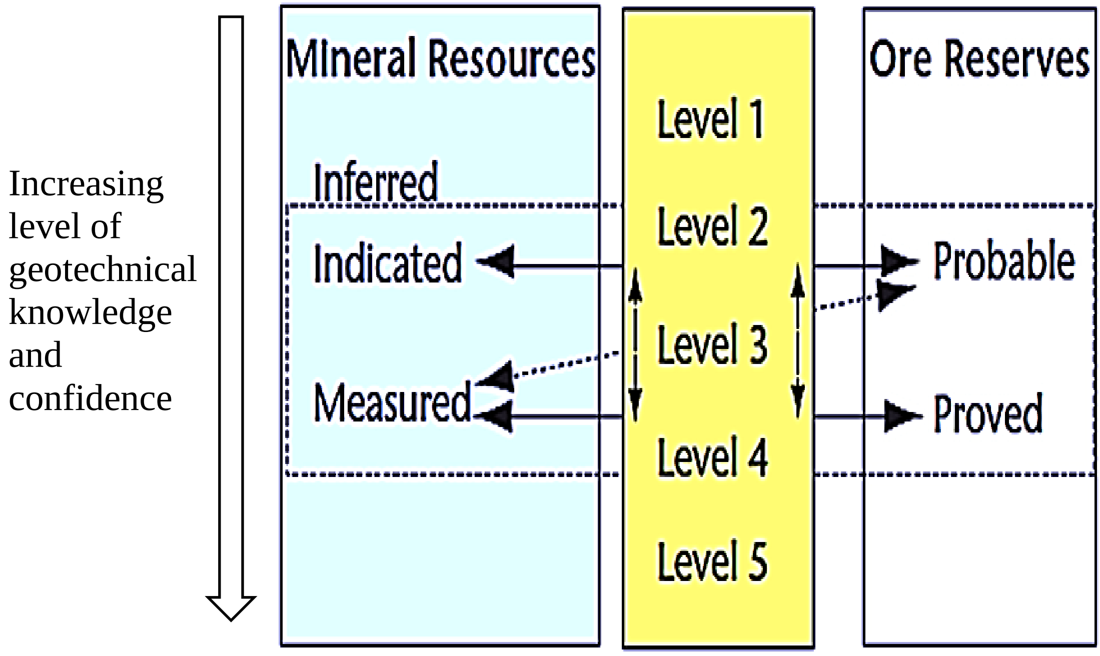


Fig. 5.1 Geotechnical levels of confidence relative to the JORC code (modified from Peter and Stacy, 2009)

5.2. Open pit mining and slope stability

5.2.1. Introduction

Open pit mining can be defined as the process of excavating any near-surface ore deposit by means of an excavation or cut made at the surface, using one or more horizontal benches to extract the ore while dumping overburden and tailings at a dedicated disposal site outside the final pit boundary (Hartman, 1992). Open pits account for the major part of the world's mineral production due to being large scale, high productivity and high effectiveness. The occupied areas of open pits differ between a few hectares and hundreds of hectares with respect to the grade of ore deposits and mining depths. Along with an increase in mining operations, the depth of open pit mines is getting deeper causing slope stability problems and safety issues. Therefore, open pit slope stability has become significant for long term sustainability. On the other hand, designing optimum overall slope angle such that the mining operations are carried out under safe conditions is a crucial work to minimize the amount of stripped waste rock and to reduce the production cost. Thus, technical, economic, environmental and safety conditions must be considered for conducting open pit slope design.

5.2.2. Pit slope terminologies and configurations

Terminologies used to define open pit design vary from place to place. The following are some examples (Hustrulid, 2013)

Bench face (North America) = ***batter*** (Australia): inclined surface found between crest and toe of a single working bench.

Bench (North America) = ***berm*** (Australia): The flat area between bench faces used for rock fall catchment. The adjective 'catch' or 'safety' is often added in front of the term in either area.

Berm (North America) **=** ***windrow*** (Australia): Rock piles placed along the toe of a bench face to increase rock fall catchment.

Bench stack (inter-ramp) = A group of benches between wider horizontal areas, e.g. ramps or wider berms left for geotechnical purposes.

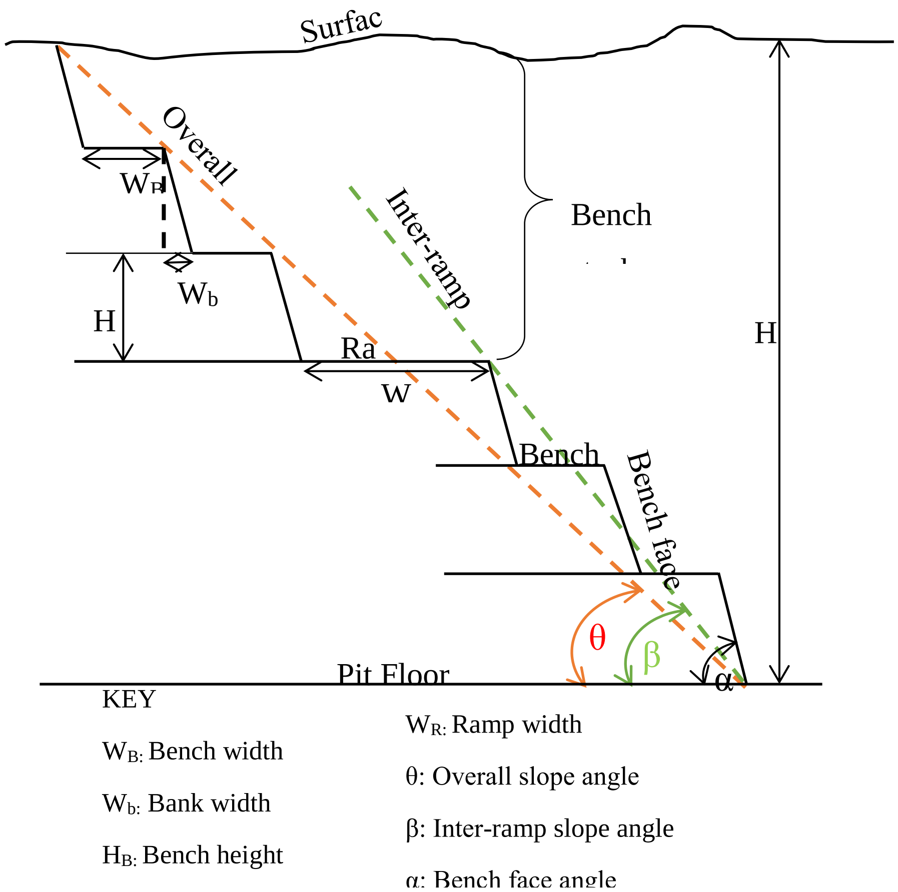


Fig. 5.2 Open pit bench sections and their configurations.

5.3. Key Factors for Pit Slope Design

The stability of pit slopes in rock is typically controlled by the following key geotechnical and mining factors (Hustrulid, 2013).

a. **Lithology and Alteration** – The rock types intersected by the final pit walls and level of alteration are key factors that impact eventual stability of the pit. Geological domains are created by grouping rock masses with similar geomechanical characteristics.

b. **Structural Geology** – The orientation and strength of major, continuous geological features such as faults, shear planes, weak bedding planes, structural fabric, and/or persistent planar joints will strongly influence the overall stability of the pit walls.

c. **Rock Mass Structure** – The orientation, strength, and persistence of smaller scale structural features such as joints will control the stability of individual benches and may ultimately restrict the inter-ramp slope angles.

d. **Rock Mass Strength** – Rock mass strengths are typically estimated via intact rock strength and rock mass classification schemes such as the rock mass rating (RMR) system. Lower rock mass quality typically results in flatter overall slope angles.

e. **Groundwater Conditions** – High groundwater pressures and water pressure in tension cracks will reduce rock mass shear strength and may adversely impact slope stability. Depressurization programs can reduce water pressure behind the pit walls and allow steeper pit slopes to be developed.

f. **Blasting Practices** – Production blasting can cause considerable damage to interim and final pit walls. This increased disturbance is typically accounted for with a reduction in the effective strength of the rock mass. Controlled blasting programs near the final wall can be implemented to reduce blasting induced disturbances and allow steeper slopes. Scaling of blast induced fracturing is essential.

g. **Stress Conditions** – Mining induces stress changes due to lateral unloading within the vicinity of the pit. Stress release can lead to effective reductions in the quality of the rock mass and increases in slope displacements. Localized stress decrease can reduce confinement and result in an increased incidence of raveling type failures in the walls. Modifying the mining arrangement and sequence can sometimes manage these stress changes to enhance the integrity of the final pit walls.

5.4. Open pit slope optimization

Currently, most of the mineral extraction processes in the world are taking place through open pit mining method. Open pit slope designing is gaining special attention because of its risks both from safety and economic viability point of view.

Deepening the open pit mines has revealed the necessity of designing optimized slopes with regard to the economic viability. Steepening the ultimate slope of an open pit as much as possible minimizes the amount of waste rock which results in reduced production cost under a prerequisite of ensuring the mining safety conditions. This is why we always consider slope optimization as part of resource feasibility study.

Optimization of the geometry is the key objective in open pit slope designing. That means, *"maximizing the project profit at a considerably minimized slope failure"* (Read and Stacy, 2009)

The optimizing procedure starts at the surface level and proceeds downwards level by level, to obtain the optimum pit fit each level. Open pit design can be categorized in to two:

Pit evaluation from a given base: This type of approach generally facilitates the incorporation of such sophisticated facilities as infinitely variable slope angles so that pits can be constructed very accurately. By using the maximum slope for the pit sides at all times when waste is being mined, the pit profit is kept high. as only the minimum amount of waste will be mined in the design.

This method has to be used repeatedly, however, in an attempt to arrive at the optimum. The level, size and horizontal positioning of the base are modified manually.

An optimum pit design: This approach which is used to find the optimum pit is used only within a strict set of conditions. (For example, only certain limited pit slopes may be used).

The concept of optimum open pit design was clearly presented in Lerchs and Grossmann (1965). From the information obtained through the exploration program and the preliminary mining feasibility study, and with a knowledge of mining equipment, the following factors can be determined: 1) cut-off grade, 2) stripping ratio, 3) pit slope angle, 4) pit dimensions, 5) total ore reserves, 6) mine life, 7) production rate, 8) bench height, 9) road grades, and 10) size limitations on mining equipment. Once the first three factors are established, the other seven can be determined. These factors are dependent on both economics and the physical nature of the orebody. Their determination is as important as, and probably more vital than, the selection of the mining method. The rest of the Mine Model Development Report is based on these initial figures, but once actual mining has begun, changes can be made in any or all of these factors as more is learned about the orebody or as economic conditions change.

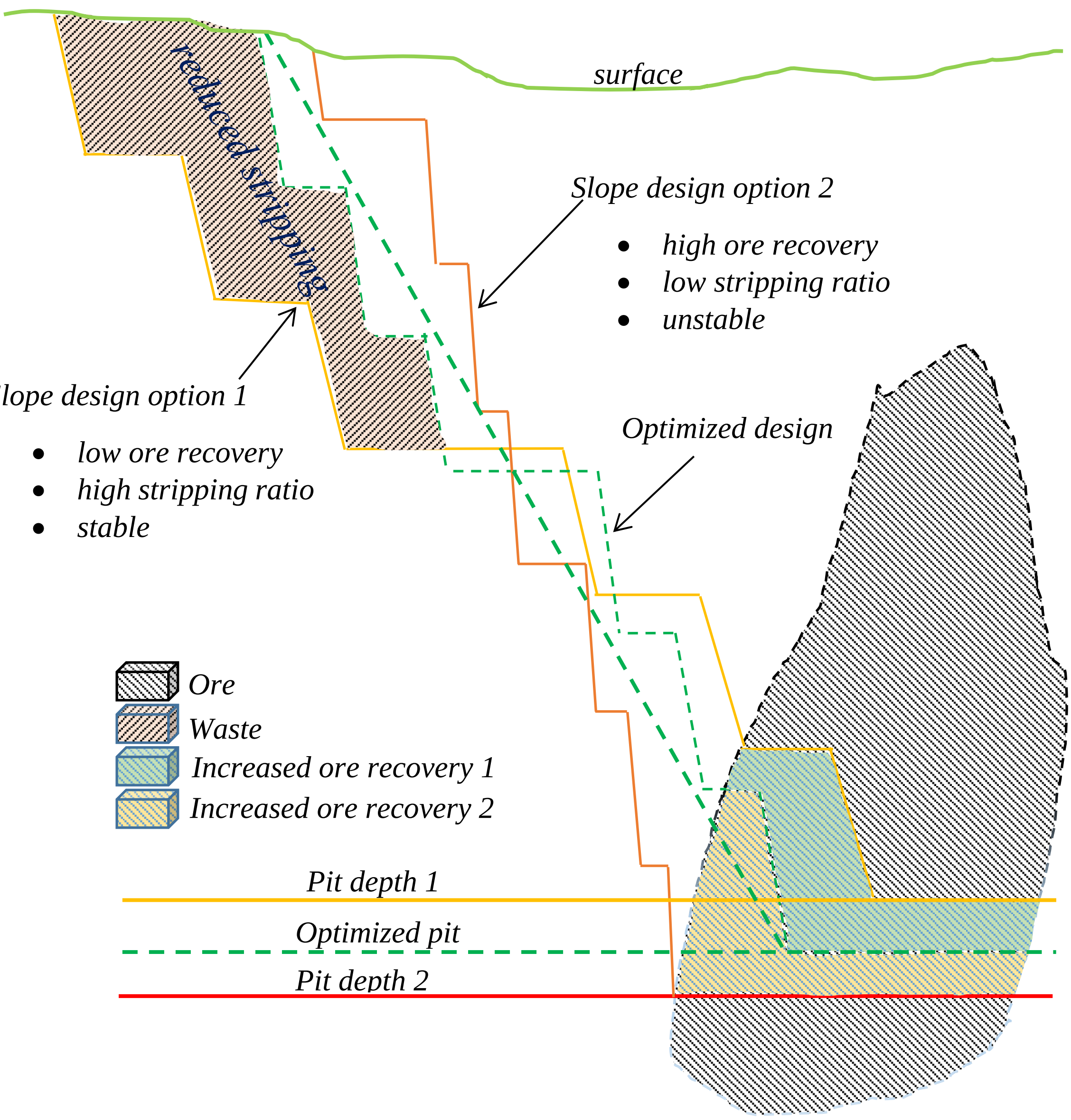


Fig. 5.3 Schematic diagram showing the concept of open pit slope designing with two design options considering safety and economic viability into account.

5.5. Slope Failure Mechanisms

In large scale slopes, various modes of slope failure occur depending on geological structure and the stress conditions of the rock mass. Field data and the failure surface are significant features to gain more and exact failure mechanism. Since determination of complete mode of slope failure is difficult, successive field observations are required to predict the appropriate failure mechanism for slope stability analysis. In the following, commonly governing failure mechanisms are described in detail.

5.5.1. Structurally Controlled Failure Mechanisms

Failures primarily rely on the orientation, shear strength and water pressure conditions of the discontinuities in the rock mass and can be accurately determined by means of proper site investigations and relevant field data. Therefore, it is recommended to gain adequate information about the kinematic constraints. Planar failure, wedge failure and toppling failure are the most widely observed structural failure modes (Simmons & Simpson, 2006).

Planar Failure

Planar failure occurs when a rock block slides along a discontinuity plane which dips out of the slope face. Most of the failure take place by the tension crack formed at the slope crest. General conditions for plane failure are;

- ✓ The plane on which sliding occurs must strike within approximately ±20° of the slope face,
- ✓ The failure plane must “daylight” in the slope face which means that its dip must be less than the dip of the slope face, expressed as $\Psi p < \Psi f$, shown in Figure 5.3 (a),
- ✓ The dip of the failure plane must be greater than the angle of friction of this plane, expressed as $\Psi p > \varphi$, illustrated in Figure 5.3 (a),
- ✓ Failure surface intersects the slope with tension crack in slope face or the slope with tension crack in upper slope surface,

Release surfaces which provide negligible resistance to sliding must present in the rock mass to define the lateral boundaries of the slide. Alternatively, failure can occur on a failure plane passing through the convex portion of a slope (Wyllie & Mah, 2004), presented in Figure 5.3 (b).

where,

Ψp = Dip of the sliding plane,

Ψf = Slope face angle,

φ = Angle of friction of the sliding plane.

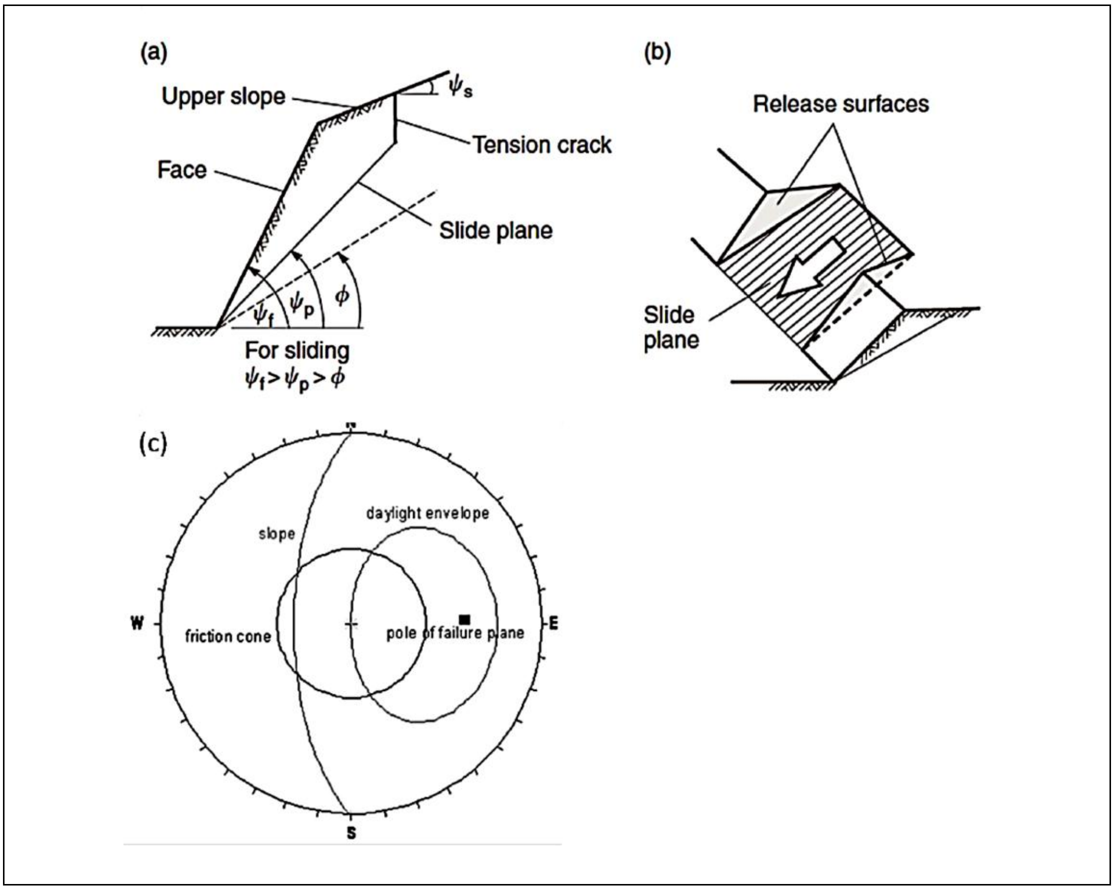


Fig. 5.4 Plane failure with tension crack (b) Required lateral-release surfaces (c) Stereographic analysis for kinematic condition of plane failure (Wyllie & Mah, 2004)

Wedge Failure

Wedge failure occurs in which at least two discontinuities intersecting each other and daylights in the slope face. Due to the geological and geometrical aspects, wedge failure is more frequently seen than planar failures in rock slopes. Kinematic conditions for the occurrence of wedge failure are:

The line of intersection (Ψi) must be less than the dip of the slope face (Ψfi) but also steeper than the average friction angle (φ) of the two sliding planes, which can be presented as, $\Psi fi > \Psi i > \varphi$ (Wyllie & Mah, 2004),

- ✓ For kinematical analysis, discontinuities must strike at angles greater than 20° to the strike of the slope face (Read and Stacey, 2009).

Geometrical and stereographical conditions related to wedge failure are presented in Figure 5.4

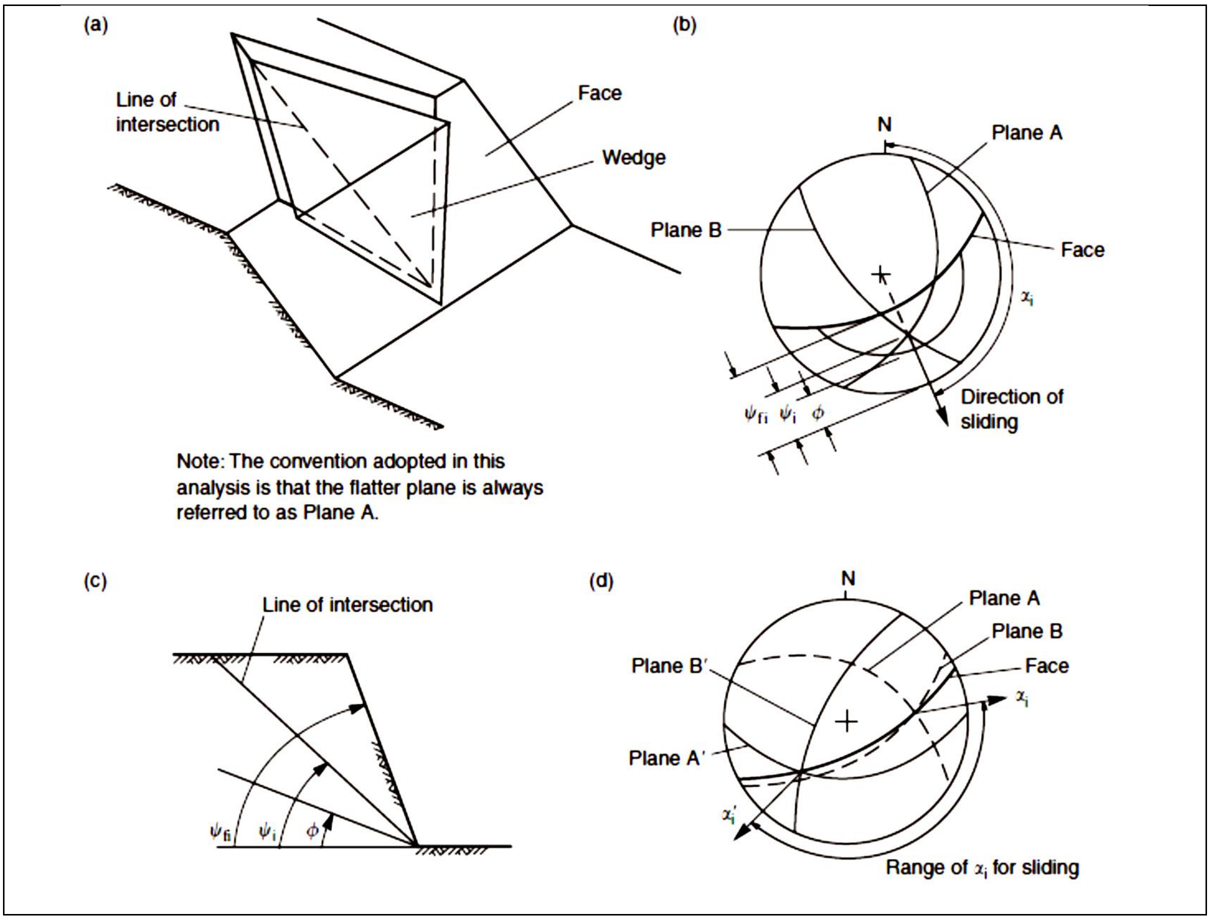


Fig. 5.5 (a) Wedge failure geometry (b) Stereoplot of wedge failure (c) Section view of kinematical condition of wedge failure (d) Stereonet illustration of the limit range with respect to orientation (Wyllie & Mah, 2004)

Toppling Failure

Toppling failure can be described as the failure mode of overturning the rock columns formed by steeply dipping and sub horizontal discontinuities. Several kinds of toppling failures were described by Goodman and Bray (1976).

i. Block Toppling: Block toppling occurs in which individual columns are divided by a set of discontinuities. Load caused by the longer overturning columns pushes forward the short columns which compose the toe of the slope and eventually the sliding of the toe leads the toppling progress through the higher up the slope. Existence of bedded sandstone and columnar basalt with orthogonal jointing make possible to occur this type of failure.

ii. Flexural Toppling: This type of failure occurs in slopes with a steeply dipping discontinuity set. Bedded shale and slate with not well-developed orthogonal jointing are the triggering geological conditions to occur flexural toppling failure.

iii. Block-Flexural Toppling: Pseudo-continuous flexure along long columns divided by sets of cross joints characterize this type of failure. Accumulated displacements on the cross-joints causes the toppling of columns.

iv. Secondary toppling modes: Unlike the primary toppling modes which occur under the action of gravity and in situ stresses, secondary toppling is caused by natural mechanisms such as weathering or by human activities. Undercutting of the toe of the slope by these independent events initiates this type of toppling failure mode. Horizontally bedded sandstone and shale are examples of geological conditions to occur the failure type.

Common types of toppling failures and stereographic representation of the kinematic conditions for toppling failure are presented in Figure 5.5.

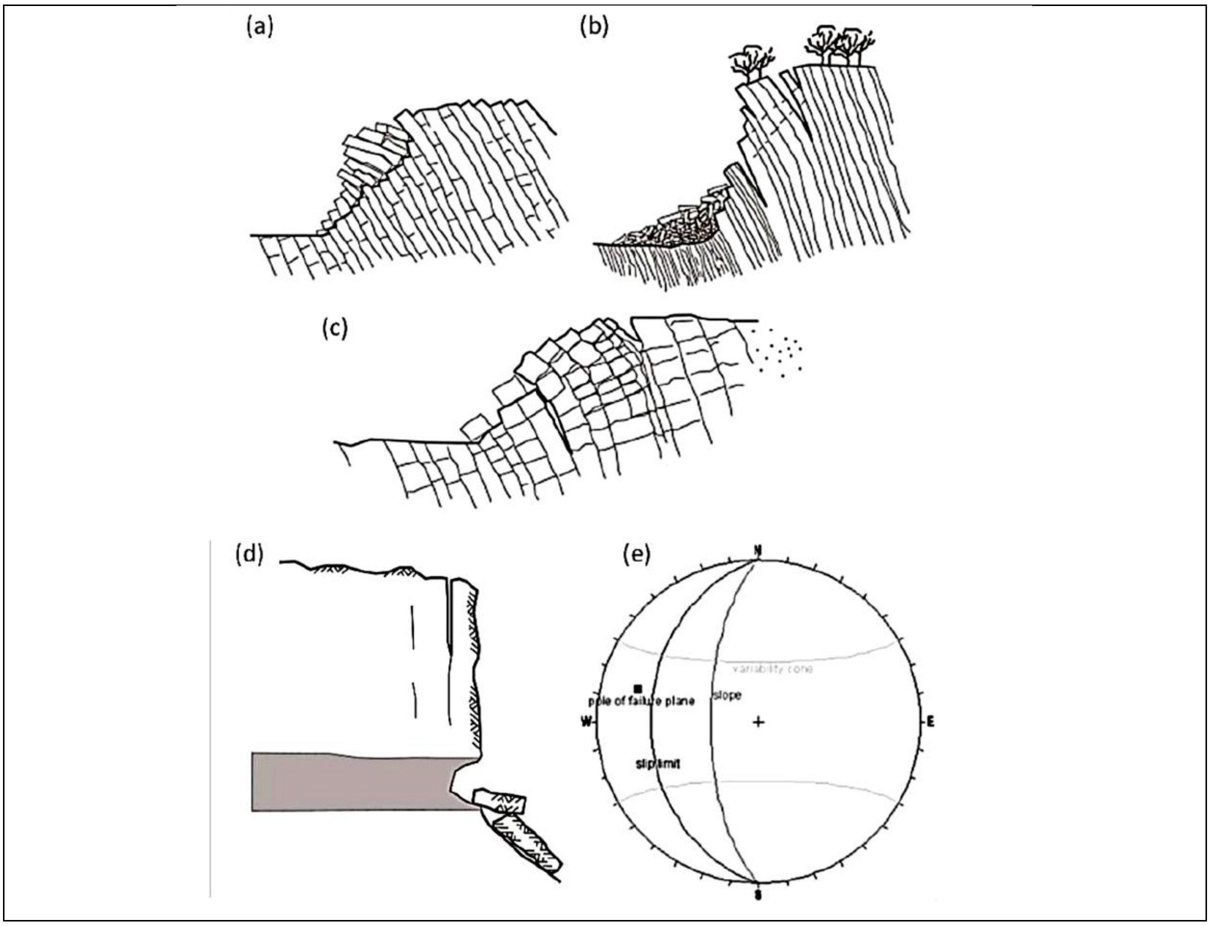


Fig. 5.6 Common types of toppling failures (a) Block toppling (b) Flexural toppling (c) Block-Flexural toppling (d) Secondary toppling (e) Stereonet representation of the kinematical condition required for toppling failure (Wyllie & Mah, 2004)

According to Goodman and Bray (1976) toppling failure occur in case such conditions are fulfilled;

$(90-\Psi f)+\varphi d\leq\Psi d$

Where,

Ψf= Dip of slope face

φd= Internal friction angle of plane/joint (discontinuity)

Ψd=Dip of plane/joint (discontinuity)

If discontinuity dips into the slope face and strikes within 30° of the face, toppling failure is possible to occur.

Rock mass (Circular) Failure Mechanisms

Circular failure generally occurs in highly weathered or closely jointed rock masses. The failure surface is mostly in the form of circular shape by developing the line of least resistance path through the slope. This type of failure is not controlled by structural geology for stability and takes place when the individual particles in a rock mass are very small compared with the size of the slope (Wyllie and Mah, 2004). In Figure 5.6, two- and three-dimensional illustrations of circular failure are presented. Limit equilibrium method is a commonly used analysis method for circular shear failure by applying the method of slice in which a circular failure surface is assumed. Additionally, finite element, finite difference and distinct element method are frequently preferred numerical modeling tools for analysis of rock mass failure.

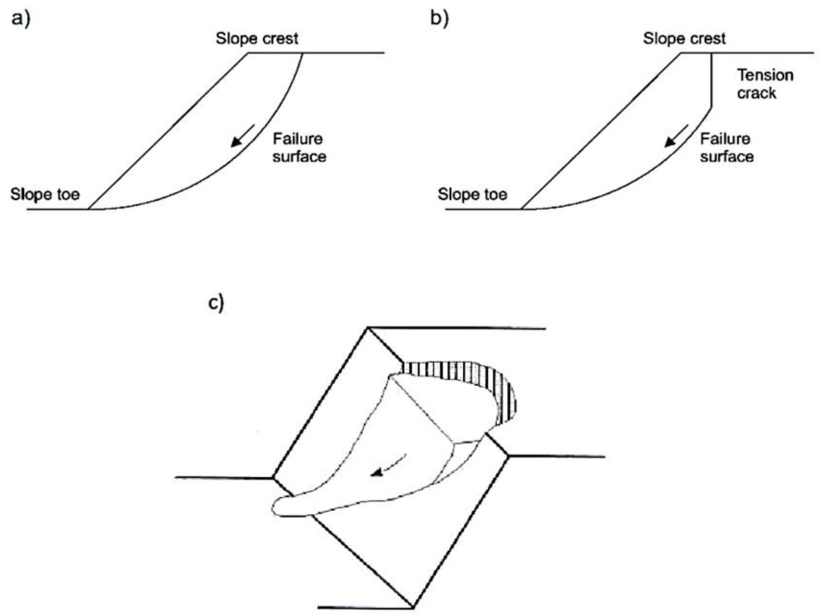


Fig. 5.7 Typical circular failure (a) without tension crack (b) with tension crack (c) three dimensional geometry of circular shear failure (Hoek and Bray, 1981)

5.6. Rock Slope Design Methods

The purpose of design methods for rock slopes is basically to determine and predict whether or not failure occurs. In fact, they are intended to determine when the acting stress on a slope exceed the strength of the rock mass. Various methods were proposed for the design analyses, including kinematic analysis and empirical design methodology, limit equilibrium methods and numerical modeling analysis.

5.6.1. Kinematical Analysis

Kinematic analysis is a useful method to investigate the possible structurally controlled slope failure modes examining the sliding direction by stereographic projection. Kinematic is described as the motion of bodies without reference to the forces that cause them to move (Goodman, 1989). Maximum safe slope angle can be estimated based on the basic failure modes such as planar failure, wedge failure and toppling failure. The analysis is conducted by using the orientation of discontinuities and the slope generally in terms of dip and dip direction. However, strength conditions, bench geometry, external forces, seismic or groundwater conditions are not considered in this technique. Stereonet plots are used to determine the failure type and the direction of the slide which gives data about the stability conditions. A preliminary slope stability evaluation of a mine by conducting kinematic analysis associated with susceptible failure mode is presented in Figure 5.7. Although kinematic analysis is a relatively simple to use, it just gives an initial indication of failure potential. Therefore, kinematic analysis is only utilized for preliminary design purposes by eliminating stable slopes for further detailed analyses.

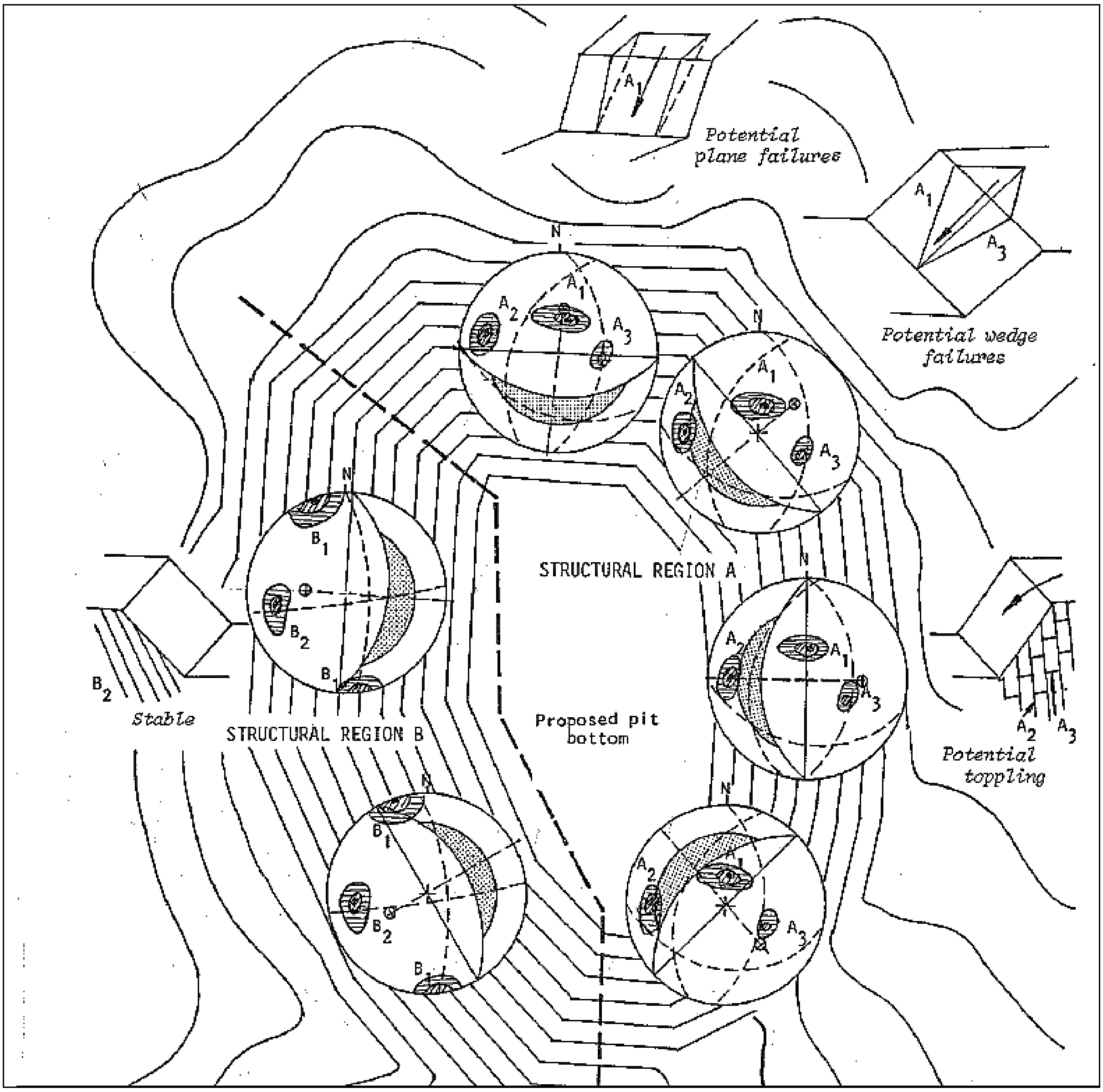


Fig. 5.8 Example of preliminary evaluation of slope stability of an open pit mine by kinematic analysis (Hoek and Bray, 1981)

CHAPTER SIX

6. GEOTECHNICAL DATA COLLECTION AND ANALYSIS

6.1. Geotechnical characterization

6.1.1. Overburden

Overburden in the pit area is typically composed of orangey-brown silty soil on the top ranging in thickness from 0.5 to about 2 meters. It is stiff to dense and has angular to sub angular particles. The bedrock near the surface or at the overburden/bedrock interface is typically rippable due to weathering of the rock mass. The depth of overburden and rippable bedrock combined is 8 meters in average.

The overburden is shallow to negligible in depth around the pit area, existing as thin veneer of topsoil to a one to three meters thick layer of weathered rock materials, overlying the rippable bedrock.

Orangey-brownish topsoil at the northern part of the proposed open pit. This soil overlies the thicker bedrock made up of massive weakly foliated metagabbro, intensively foliated carbonate amphibole schist, carbonate chlorite schist, slightly weathered metagranodiorite and rarely talc schist.

Fig. 6.1 Topsoil in the northern part of the pit

6.1.2. Bedrock Units

The bedrock in the pit area is light grey to grey green to dark grey in color with occasional carbonate, chlorite and quartz contents. The bedrock is strongly foliated with foliation plane dipping towards the West at approximately 68^0 to 74^0. Numerous shearing structures exist mainly in the central part of the pit area, dipping towards the West, similar to the orientation of the foliation planes within the rock mass. The main shear zone bisects the proposed pit, running subverticaly along northeast to southwest trend.

Multiple geological units were defined within the open pit including massive weakly foliated metagabbro, intensively foliated carbonate amphibole schist, carbonate chlorite schist, slightly weathered metagranodiorite. The individual lithologies are complex due to multiple deformation and alteration phases, and correlating lithologies across drill holes is difficult task and may produce unreliable geological model. Re-grouping the lithological units into packages of similar characteristics allowed for easier understanding of the geology.

a) Intermediate-mafic-ultramafic rocks

Metagabbro/Amphibolite, carbonate chlorite schist and chlorite amphibole schist and rarely metadiorite are the common lithologies exposed in the pit area. They have good quality with average RMR value of 60 and average RQD of 85. The average laboratory rock strength test results range from 17 to 232MPa with mean of 112MPa for all failures.

Fig. 6.2 Massive weakly foliated metagabbro/Amphibolite (left) and highly foliated carbonate chlorite schist (right).

Felsic rocks

Metagranodiorite and aplitic dikes are the common felsic rocks exposed in the pit area. Metagranodiorite has very good quality. It has average RMR value of 72 and average RQD value of 89. The aplitic dikes are very small stocks intruding the mafic-ultramafic sequences and they have little geotechnical significance.

6.2. Structural discontinuity mapping

Foliation is the main structural control in the study area. It has a general trend of NNE-SSW and dipping towards west direction. The main lithologies from which persistent foliation structures are observed are carbonate chlorite schist, chlorite amphibole schist and metagranodiorites. The general strike and dip directions of foliation and fracture plane data collected from the field (fig.6.3)

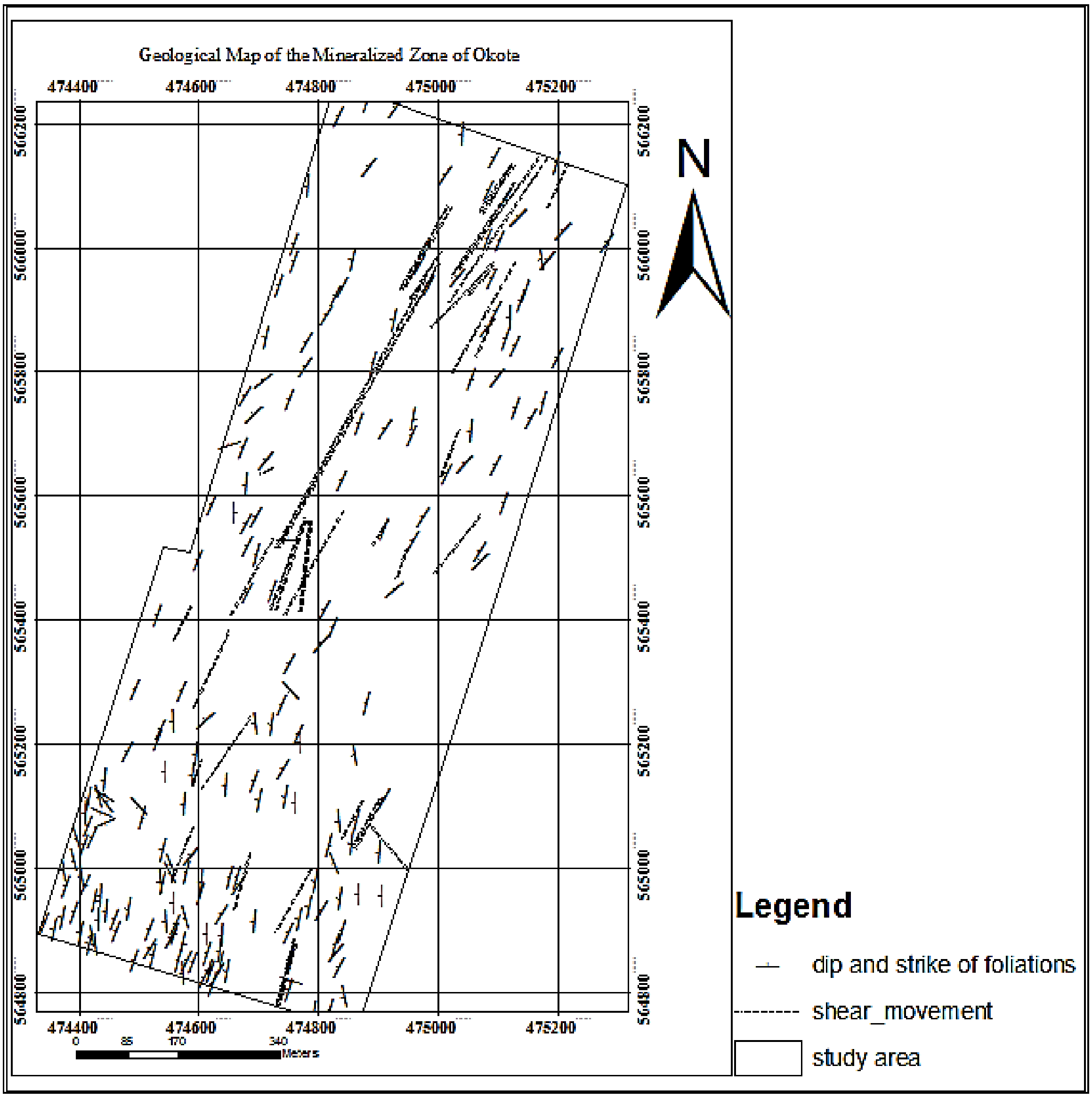


Fig. 6.3 Structural discontinuity map

6.3. Geotechnical logging

While geotechnical logging the following activities were carefully carried out:

- ✓ Recording of the basic drill hole identification data on the log,
- ✓ Geotechnical logging of the solid core, including the rock type, the degree of weathering and/or alteration, the strength of the intact rock, total core recovery (TCR), solid core recovery (SCR) and the rock quality designation (RQD)

- ✓ Geotechnical logging of the fractures that intersect the solid core, including details of fracture, foliation and shear orientation relative to the core axis (figure 5.4) and fracture spacing and infillings.

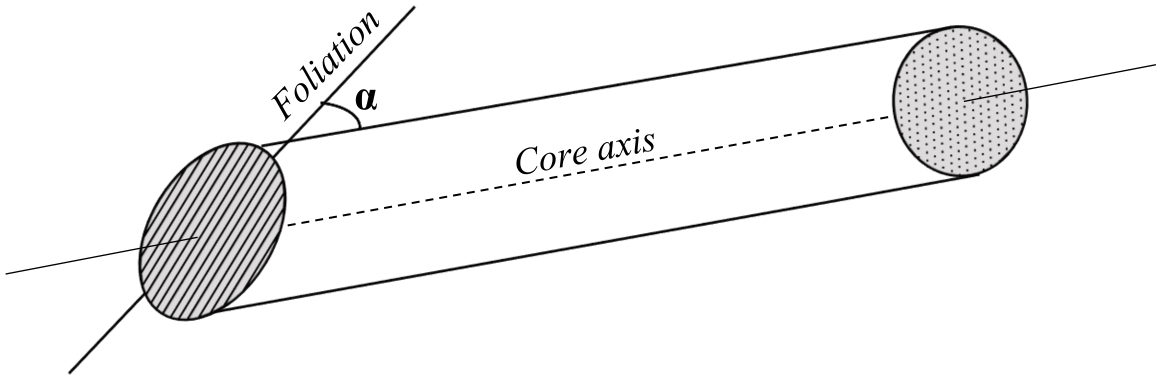


Fig. 6.4 Diagram showing foliation orientation and dip (α angle) measuring method from drill core (Read and Stacy, 2009).

Mechanical defects caused by drilling, handling and recovery processes are also carefully separated from the natural discontinuities found in rock mass to avoid mishap of data generation

To evaluate the rock mass property, representative drill holes (DH 1300N/1, DH 1650N/2, DH 1700N/3, DH 800N/1, DH 1650N/2, DH 950 N/1, DH 1250 N/1, and DH 1100 N/1) were selected for characterization of ore body and waste materials found in and around North Okote area. The later four drill holes were logged by NMiC. The main purpose of geotechnically logging the solid core is to divide the core into geotechnically similar intervals, then ascribe geotechnical parameters to each domain.

Fig. 6.5 Drill Cores aligned in core box (photo from NMiC archive).

6.3.1. RQD and fracture frequency per meter

The most common types of the geotechnical data and the rock strength index, the Rock Quality Designation (RQD) and fracture frequencies are routinely compiled from selected drill holes. The rock quality designation (RQD) is a commonly used index for the description of rock mass fractured state. The RQD was initially introduced for civil engineering applications, and it has been quickly adopted in mining, engineering geology as well as geotechnical engineering. The success of the RQD is in great part, due to its simplicity. The rock quality designation (RQD) is important and used as input for in determination of the rock mass strength.

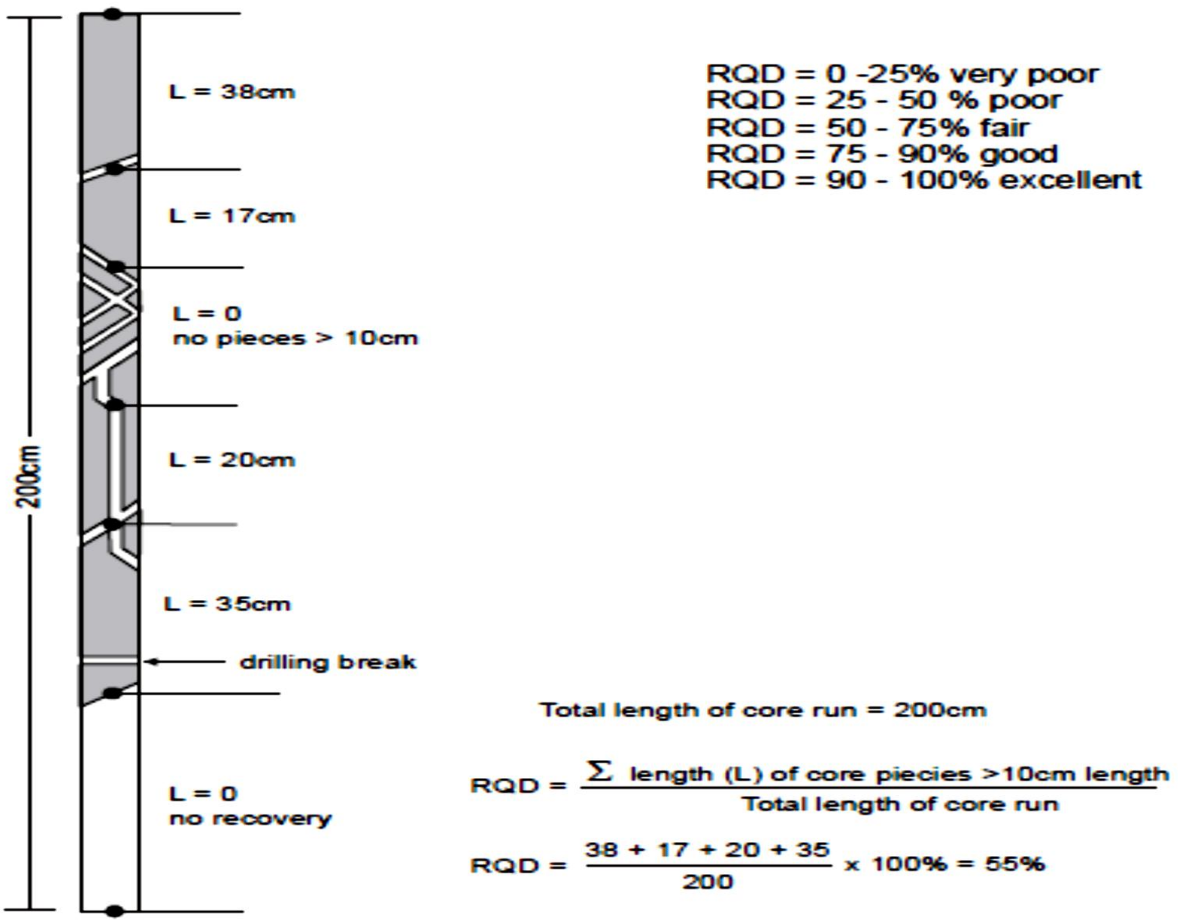


Fig. 6.6 Procedure for measurement and calculation of RQD (Deere, 1989).

Table 6.1 RQD and fracture frequency (FF) data captures from DH 1650 N/2

GEOTECHNICAL CORE LOGGING [RQD and FF (fracture frequency)]										
BH ID:DH 1650N/2			**Area: NORTH OKOTE**					**Logged By:Shibiru M.**		
			page:1 of 1						**Date:April 05,2018G.C**	
Interval		**Core recovery**		**sticks>10cm**		**Fracture frequency (FF)**				**Remark (e.g. fault/dyke)**
From	**To**	**M**	**%**	**M**	**RQD**	**Open**	**SH**	**WH**	**SS**	
0	1.3	1.27	98	0.71	55.91	5				
1.3	6.7	5.4	100	3.31	61.30	12				
6.7	8.25	1.55	100	1.12	72.26	8				
8.25	9.5	1.17	94	0.45	38.46	7				
9.5	11.25	1.75	100	1.31	74.86	10				
11.25	12.65	1.4	100	0.88	62.86	7				
12.65	14.25	1.6	100	0.85	53.13	11				
14.25	15.6	1.35	100	1.13	83.70	9				
15.6	17.05	1.45	100	1.05	72.41	12	3			
17.05	20.05	2.95	98	1.6	54.24	12		2		
20.05	23.25	3.12	98	1.87	59.94	10				
23.25	26.15	2.73	94	1.7	62.27	11				
26.15	29.15	2.73	91	2.45	89.74	17	1			
29.15	31.55	2.4	100	2.25	93.75	16	2	1		
31.55	31.8	0.25	100	0.2	80.00	3				
31.8	34.8	3.01	100	2.97	98.67	14				
34.8	37.8	2.98	99	2.27	76.17	11	1			
37.8	41	3.15	98	2.83	89.84	13	2			
41	44	3	100	2.91	97.00	8				
44	47	2.95	98	2.74	92.88	13	3			
47	50	3	100	2.64	88.00	15		3		
50	53	3	100	2.9	96.67	11				
53	56	3.01	100	2.92	97.01	13	1			
56	59	2.95	98	2.86	96.95	7				
59	62	3	100	2.19	73.00	11	2			
62	65	2.91	97	2.27	78.01	11		3		
65	68	2.94	98	2.66	90.48	9				
68	71	3	100	2.6	86.67	8				
71	74	3	100	2.52	84.00	9				
74	76.6	2.36	91	1.45	61.44	3				
76.6	79.65	3.05	100	2.85	93.44	6				
Total		Sum↓	Ave↓	Sum↓	Ave↓	FF/m↓	Sum↓	Sum↓	Sum↓	
		78.43	98	62.5	77.90	3.978	15	9	0	

SH-strongly healed *WH-weakly healed* *SS-slickensided*

Table 6.2 RQD and Lithology from DH 1650 N/2

Drill Hole ID: DH 1650N/2	National Mining Corporation (N.Mi.C)
X: 474979.2188m	Okote Gold Project (North)
Y: 566190.3125m	Geotechnical Core Logging
Z: 1508m	
DH azimuth:	Logged by: Shibiru Merga
DH inclination:	Date: April 5, 2018

Interval (m)		SCR	CR	TCR	>10cm	RQD	Lithology				
From	To	(m)	(%)	(m)	(m)	(%)	From	To	L. Index	Lithology Description	Remark
0.00	1.30	1.27	98	1.30	0.71	54.62	0.00	3.00	MG/A	Metagabbro/Amphibolite	
1.30	6.70	5.4	100	5.40	3.31	61.30	3.00	6.45	CAS	Carbonate Amphibole Schist	
6.70	8.25	1.55	100	1.55	1.12	72.26	6.45	9.25	MGD	Metagranodiorite	
8.25	9.50	1.17	94	1.25	0.45	36.00	9.25	11.20	CAS	Carbonate Amphibole Schist	
9.50	11.25	1.75	100	1.75	1.31	74.86	11.20	12.05	MG/A	Metagabbro/Amphibolite	
11.25	12.65	1.4	100	1.40	0.88	62.86	12.05	12.60	CAS	Carbonate Amphibole Schist	
12.65	14.25	1.6	100	1.60	0.85	53.13	12.60	13.00	MGD	Metagranodiorite	
14.25	15.60	1.35	100	1.35	1.13	83.70	13.00	16.60	CAS	Carbonate Amphibole Schist	
15.60	17.05	1.45	100	1.45	1.05	72.41	16.60	17.20	MGD	Metagranodiorite	
17.05	20.05	2.95	98	3.00	1.60	53.33	17.20	17.85	CAS	Carbonate Amphibole Schist	
20.05	23.25	3.12	98	3.20	1.87	58.44	17.85	18.55	MGD	Metagranodiorite	
23.25	26.15	2.73	94	2.90	1.70	58.62	18.55	21.15	CCS/CS	Carbonate Chlorite Schist/Carbonate Schist	
26.15	29.15	2.99	100	3.00	2.45	81.67	21.15	22.05	MGD	Metagranodiorite	
29.15	31.55	2.4	100	2.40	2.25	93.75	22.05	24.70	CAS	Carbonate Amphibole Schist	
31.55	31.80	0.25	100	0.25	0.20	80.00	24.70	26.85	MGD	Metagranodiorite	
31.80	34.80	3.01	100	3.00	2.97	99.00	26.85	29.25	CAS	Carbonate Amphibole Schist	
34.80	37.80	2.98	99	3.00	2.27	75.67	29.25	32.20	CCS/CS	Carbonate Chlorite Schist/Carbonate Schist	

37.80	41.00	3.19	100	3.20	2.83	88.44	32.20	39.25	CAS	Carbonate Amphibole Schist	
41.00	44.00	3	100	3.00	2.91	97.00	39.25	39.75	MGD	Metagranodiorite	
44.00	47.00	3	100	3.00	2.74	91.33	39.75	46.45	CAS	Carbonate Amphibole Schist	
47.00	50.00	3	100	3.00	2.64	88.00	46.45	58.75	MG/A	Metagabbro/Amphibolite	
50.00	53.00	3	100	3.00	2.90	96.67	58.75	66.65	CAS	Carbonate Amphibole Schist	
53.00	56.00	3.01	100	3.00	2.92	97.33	66.65	67.55	MGD	Metagranodiorite	
56.00	59.00	3	100	3.00	2.86	95.33	67.55	68.75	CAS	Carbonate Amphibole Schist	
59.00	62.00	3	100	3.00	2.19	73.00	68.75	76.60	MG/A	Metagabbro/Amphibolite	
62.00	65.00	2.99	100	3.00	2.27	75.67	76.60	79.65	MG/A	Metagabbro/Amphibolite	
65.00	68.00	2.99	100	3.00	2.66	88.67					
68.00	71.00	3	100	3.00	2.60	86.67					
71.00	74.00	3	100	3.00	2.52	84.00					
74.00	76.60	2.6	100	2.60	1.45	55.77					
76.60	79.65	3.05	100	3.05	2.85	93.44					
			99			**76.87**					

SCR-solid core recovery TRC-total core recovery RC-core recovery

Geotechnical data capturing is aimed at supporting the principal rock mass classification and strength assessment methods used in open pit slope designing. It is good geotechnical practice to record at least the intact rock strength and RQD data in the exploration drill hole core logging procedures from day one (before splitting for other purposes), so that secondary data early collected from core samples are used. The exploration holes are often drilled before the geotechnical holes and provide less reliable and consistent geotechnical data.

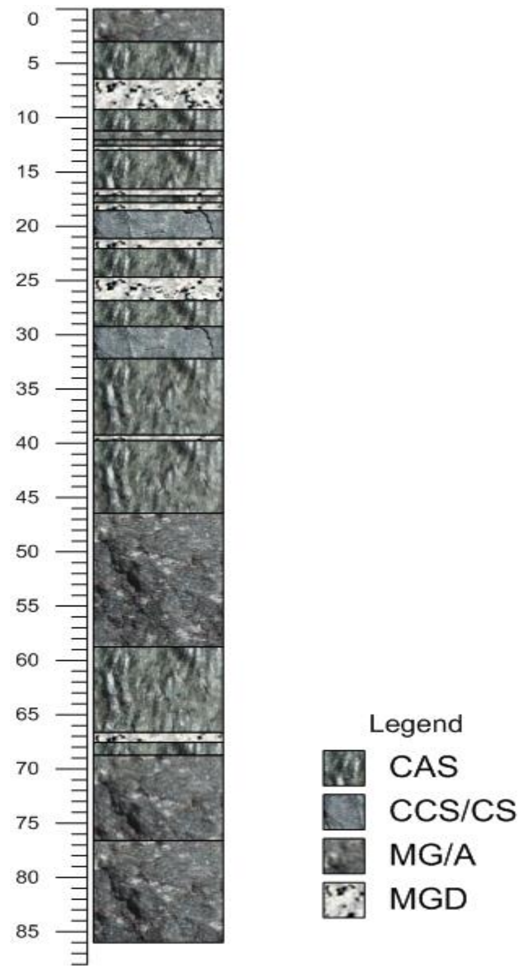


Fig. 6.7 Figure Lithological log of BH 1650N2

6.4. Rock strength test

Intact rock strength (IRS) is a major rock property. Intact rock strength determines the strength of the intact rock block material and as such governs partially the strength of a rock mass. Standard determination of the IRS is by means of an unconfined compressive strength (UCS) test. In, for example, most rock mass classification systems and analytical and numerical calculations intact rock strength is a parameter and is it necessary to obtain the characteristic or mean value of the intact rock strength. ASTM 4543 procedure is followed during rock strength testing at laboratory of Geological Survey of Ethiopia.

Table 6.3 Rock strength and unit weight test result

Sample ID	Sample interval (m)	Rock unit	Pli (MPa)	UCS (MPa)	Ei (GPa)	Ti (MPa)	mi	ϒ (g/cc)
1650_N/2	31.70-32.20	CCS/CS	0.69	17	3267.4	8.03	10	2.82
800_N1_1	121.85-122.35	MG/A	7.73	186	58604	17.23	27	2.93
800_N1_2	42.70-43.20	MG/A	0.89	21	4391.4	8.51	24	2.79
1700_N3	80.25-80.75	MGD	9.70	233	76870	18.58	29	2.69
1300_N1	73.65-74.15	CAS	2.37	57	14181	11.43	11	2.94
Averages			4.28	102.72	31463	12.76	20	2.83

Key: CCS/CS= Carbonate Chlorite Schist
MG/A= Metagabbro/Amphibolite
MDG = Metagranodiorite
CAS = Chlorite Amphibole Schist
Pli = Intact Point Load Index
UCS = Intact Uniaxial Compressive Strength
Ti = Intact Tensile Strength
mi = Intact Material Constant
ϒ = Unit weight of rock

6.5. Hydrogeological condition of the area

To furnish data for the Environmental and Social Impact Assessment (ESIA) of Okote Gold Exploration Project, NMiC employed a consultant to undertake geophysical and hydrogeological investigations in and around the Okote prospect. Accordingly, geophysical survey mainly Vertical Electrical Soundings (VES) and hydro-geological investigations were conducted during October in 2015 by the company. The study considers the Precambrian basement to be regional aquicludes with localized aquifers along their fractured and weathered zones. It is also started that in the vicinity of Okote the thickness of the weathered zone is very small, the fractures appears to be filled with quartz veins reducing the secondary porosity .According to the study team the mining area is located in semi-arid region where rain fall is low which makes the recharge to groundwater from rain fall and runoff insufficient .Alluvial deposits on the course of Okote and on a relatively gentle and flat areas are the only water bearing unit for water holes.

The only source of water for local people is some a water hole excavated on the alluvial sediments along the stream course of small Okote, located at UTM 475148E and 565198N.

Generally, the intercrystalline voids of unfractured metamorphic rock that make up the porosity are minute and many are not interconnected. Because of the small pore sizes and low degree of pore interconnectivity, the primary permeabilities of these rocks are extremely small. Measurements on intact specimens of metamorphic rocks indicate that the primary permeability values in the range of 0.00019 millidarcy (10-11 – 10-13 m/s). Permeabilities of this magnitude indicate that these rocks are impermeable within the context of most groundwater problems.

These rocks are also classified as an aquifuge, in the aquifer classification of Genale – Dawa River Basin Integrated Master Plan.

Except for the water hole on the course of Okote Tiqa (Small Okote), there is no groundwater development activity within the Okote area. The thickness of the regolith is so small to serve as groundwater storage media. Moreover, the gradient between the high elevated area around the Main Camp (1516 meters amsl) and the lowest area within the project area, the valley of Okote (1251 meters amsl) is steep. This steep gradient favor runoff than infiltrations.

Currently, drillings at the Okote site is in progress. the percent of core recovery being almost 100% in all boreholes RC drilling method removes the cuttings by air and the size of cutting chips are very fine. Removal of cuttings by air in a well or wells that encounter groundwater during drilling would have been difficult and the success of drilling would also be very slow. Drilling history of these exploratory wells shows that the area is totally devoid of groundwater.

A deep well to a depth of 75 meters at the bank of Geleba stream, about 10 kilometers west of Okote, drilled and constructed for Hallo village water supply and is a major groundwater development activity in the surrounding area. The course of Geleba stream between Hallo and Geleba village is on a gentle and flat topography, where flood plain of thick sand deposits are common. Hallo village water supply well taps this sand aquifer recharged by the stream. The well is also serving Okote main camp water supply demand. With detail study to delineate the lateral and vertical extent of this inter-granular sand aquifer and recharge rate estimation, it is far more likely to be a reliable source for the future water supply of Okote Project Development.

CHAPTER SEVEN

7. DISCUSSION

7.1. Rock Mass Classification

Rock Mass Classification is the process of placing a rock mass into groups or classes on defined relationships (Bieniawski, 1989), and assigning a unique description (or number) to it on the basis of similar properties/characteristics such that the behavior of the rock mass can be predicted. Quantitative classification of rock masses has become almost routine, since it provides a rapid means of quantifying the quality of a mass, comparing qualities, and assessing support requirements. Classification applied on a routine basis can have tremendous value in open pit mines. During the feasibility and preliminary design stages of a project, when very little detailed information is available on the rock mass and its stress and hydrologic characteristics, the use of a rock mass classification scheme can be of considerable benefit. At its simplest, this may involve using the classification scheme as a check-list to ensure that all relevant information has been considered. At the other end of the spectrum, one or more rock mass classification schemes can be used to build up a picture of the composition and characteristics of a rock mass to provide initial estimates of support requirements, and to provide estimates of the strength and deformation properties of the rock mass. It is important to understand the limitations of rock mass classification schemes (Palmstrom and Broch, 2006) and that their use does not (and cannot) replace some of the more elaborate design procedures. However, the use of these design procedures requires access to relatively detailed information on in situ stresses, rock mass properties and planned excavation sequence, none of which may be available at an early stage in the project. As this information becomes available, the use of the rock mass classification schemes should be updated and used in conjunction with site specific analyses.

The purpose of rock mass classification is to database the engineering properties of the rock mass for use in the stability analyses that will be used to prepare the slope designs at each stage of project development. This includes the properties of the intact pieces of rock that constitute the anisotropic rock mass, the structures that cut through the rock mass and separate the individual pieces of intact rock from each other, and the rock mass itself. It also suggests the selection slope height and slope angles.

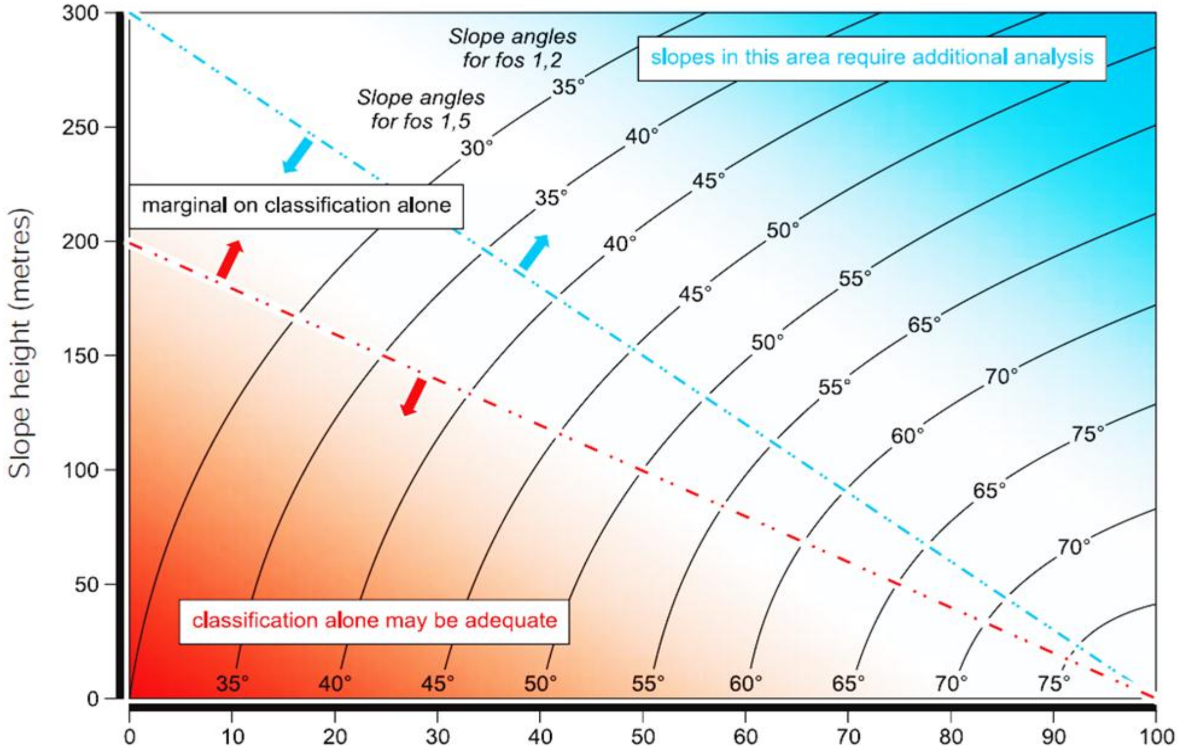


Fig. 7.1 Preliminary slope angle chart showing slope height vs MRMR (Haines and Terbrugge, 1991)

7.1.1. Bieniawski's RMR Classification for Okote

This engineering classification system, which was developed by Bieniawski in 1973, utilises the following six rock mass parameters:

1. Uniaxial compressive strength of intact rock material.
2. Rock quality designation (RQD).
3. Spacing of discontinuities.
4. Condition of discontinuities, given as
   - 4.1.Length, persistence
   - 4.2.Separation
   - 4.3.Smoothness
   - 4.4.Infilling
   - 4.5.Alteration / weathering
5. Groundwater conditions
6. Orientation of discontinuities

All of these are measurable in the field and can also be obtained from borehole data. The rating of each of these parameters are summarised to give a value of RMR. All parameters are measurable in the field and some of them may also be obtained from borehole data. The hanging wall of the Okote gold deposit was classified using Bieniaswski (1989) system of Rock Mass Rating (RMR) and Laubscher (1990) Mining Rock Mass Rating (MRMR) classification system. The RMR was determined using parameters such as intact rock strength (IRS), rock quality designation (RQD), fracture frequency (FF), joint condition (JC) and joint spacing (JS). Data are collected both from the field and drill cores. RMR_{89} classification for ore zone and waste materials in the deposit is presented in the following tables

Table 7.1 Geomechanical classification of Okote area (waste and ore zone)

ROCK MASS CLASSIFICATION

NORTH OKOTE TARGET AREA (WASTE)

DATE	18,04,2018	HOLE ID	BH1150N1, BH1100N1

1) Joints spacing — **RMR index**

Sav	≥2m	0.6m-2m	20cm-60cm	6cm-20cm	<6cm	RMR index
Index	20	15	10	8	5	**10**

2) RQD (Rock Quality Designation)

RQD	90-100%	75-90%	50-75%	25-50%	<25	RMR index
Index	20	17	13	8	3	**13**

3) Compressive strength of intact rock

σ_c (Mpa)	≥250	100-250	50-100	25-50	5_25	1_5	<1	RMR index
Index	15	12	7	4	2	1	0	**12**

4) Joints condition

						RMR index
LENGTH	<1m	1-3m	3-10m	10-20m	≥20m	
INDEX	6	4	2	1	0	**3**
OPENING	0	<0.1mm	0.1-1mm	1-5mm	≥5mm	
INDEX	6	5	4	1	0	**4.5**
ROUGHNESS	very rough	rough	slightly rough	smooth	slikensided	
INDEX	6	5	3	1	0	**5**
FILLING	none	hard&<5mm	hard&≥5mm	soft&<5mm	soft&≥5mm	
INDEX	6	4	3	2	0	**2**
WEATHERING	unweathered	slightly weathered	Moderately weathered	highly weathered	decomposed	
INDEX	6	5	3	1	0	**5.5**

5) Hydraulic condition (ground water)

Condition	dry	damp	wet	dripping	flowing	RMR index
Index	15	10	7	4	0	**13**

6) Orientation of discontinuity - Adjustment

ADVANCE TOWARDS IMMERSION			index	ADVANCE AGAINST IMMERSION			index	RMR index
inclination	**A**	45^0-90^0	0	inclination	**C**	45^0-90^0	-5	
	B	20^0-45^0	-2		**D**	20^0-45^0	-10	
PARALLEL TO IMMERSION			index	**ALL DIRECTION**			index	
inclination	**E**	45^0-90^0	-12	inclination	**G**	0^0-20^0	-5	**-2**
	F	20^0-45^0	-5					

ROCK MASS RATING (RMR)	**≥80**	**60-80**	**40-60**	**20-40**	**<20**	**66**
ROCK MASS CLASS (RMC)	**I**	**II**	**III**	**IV**	**V**	**II**

ROCK MASS CLASSIFICATION

NORTH OKOTE TARGET AREA (ORE)

DATE	18,04,2018	HOLE ID	BH1150N1, BH1100N1

1) Joints spacing — **RMR index**

Sav	≥2m	0.6m-2m	20cm-60cm	6cm-20cm	<6cm	RMR index
Index	20	15	10	8	5	**10**

2) RQD (Rock Quality Designation)

RQD	90-100%	75-90%	50-75%	25-50%	<25	RMR index
Index	20	17	13	8	3	**13**

3) Compressive strength of intact rock

σ_c (Mpa)	≥250	100-250	50-100	25-50	5_25	1_5	<1	RMR index
Index	15	12	7	4	2	1	0	**12**

4) Joints condition

						RMR index
LENGTH	<1m	1-3m	3-10m	10-20m	≥20m	**3**
INDEX	6	4	2	1	0	
OPENING	0	<0.1mm	0.1-1mm	1-5mm	≥5mm	**4.5**
INDEX	6	5	4	1	0	
ROUGHNESS	very rough	rough	slightly rough	smooth	slikensided	**5**
INDEX	6	5	3	1	0	
FILLING	none	hard&<5mm	hard&≥5mm	soft&<5mm	soft&≥5mm	**3**
INDEX	6	4	3	2	0	
WEATHERING	unweathered	slightly weathered	Moderately weathered	highly weathered	decomposed	**5.5**
INDEX	6	5	3	1	0	

5) Hydraulic condition (ground water)

Condition	dry	damp	wet	dripping	flowing	RMR index
Index	15	10	7	4	0	**13**

6) Orientation of discontinuity - Adjustment

ADVANCE TOWARDS IMMERSION			index	ADVANCE AGAINST IMMERSION			index	RMR index
inclination	**A**	45^0-90^0	0	inclination	**C**	45^0-90^0	-5	
	B	20^0-45^0	-2		**D**	20^0-45^0	-10	
PARALLEL TO IMMERSION			index	**ALL DIRECTION**			index	
inclination	**E**	45^0-90^0	-12	inclination	**G**	0^0-20^0	-5	**-2**
	F	20^0-45^0	-5					

ROCK MASS RATING (RMR)	**≥80**	**60-80**	**40-60**	**20-40**	**<20**	**67**
ROCK MASS CLASS (RMC)	**I**	**II**	**III**	**IV**	**V**	**II**

7.1.2. Geological Strength Index (GSI)

Hoek in 1994 introduced the Geological Strength Index (GSI) as a way to facilitate the determination of rock mass properties of both hard and weak rock masses for use in rock engineering. GSI resulted from combining observations of the rock mass conditions with the relationships developed from the experience gained using the RMR system. The relationship between rock mass structure (conditions) and rock discontinuity surface conditions is used to estimate an average GSI value represented in the form of diagonal contours. It is recommended to use a range of values of GSI in preference to a single value. This simple, fast and reliable system represents non-linear relationship for weak rock mass, can be tuned to computer simulation of rock structures and can provide means to quantify the rock strength.

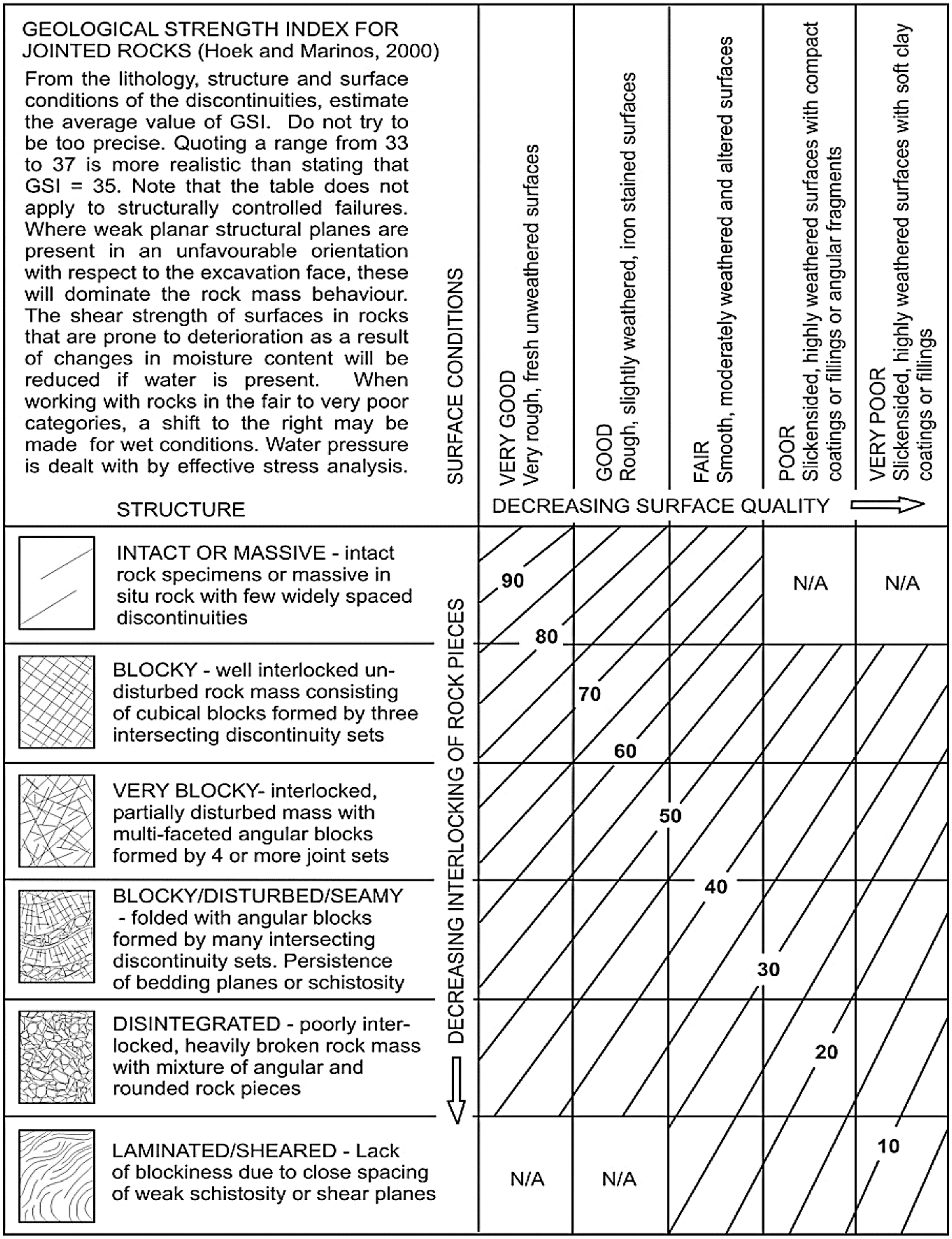


Fig. 7.2 GSI Hoek-Brown rock mass classification system, 1993

Hoek and Brown correlated the RMR to the GSI as follows.

For $RMR_{76} > 18$, $GSI = RMR_{76}$

$RMR_{89} > 23$, $GSI = RMR_{89} - 5$

Since all results of RMR_{89} classification for Okote (both orebody and waste) are above 23, we can choose the second formula to calculate GSI. The RMR_{89} values for the ore zone and waste is 67 and 66 respectively. Accordingly, the GSI value for ore zone is calculated to be 62 and the waste is 61.

7.1.3. MRMR Classification

Laubscher's Mining Rock Mass Rating system (MRMR) was introduced by Laubscher as an extension of Bieniawski's RMR system for mining applications. The modified RMR system (MRMR) adjusts the basic RMR value by considering the in-situ and induced stresses, stress changes and the effects of blasting and weathering, and accordingly support recommendations are proposed for the new value. Adjustment ratings for weathering, blasting, structural orientation and stress effects were applied to the RMR obtained to determine the MRMR of the rock mass. An overall adjustment rating of 0.857 was used and MRMR was calculated to be 57.4 for ore zone and 56.6 for the waste. The indicative overall slope angle (I.O.S.A) and bench slope angle were estimated by interpolation from the adjusted ratings in accordance with Laubscher's system.

7.1.4. Determination of Shear Strength Parameters from RMR

A peak strength criterion is a relation between stress components which will permit the peak strengths developed under various stress combinations to be predicted. Similarly, a residual strength criterion may be used to predict residual strengths under varying stress conditions.

Cohesion $C = 0.05\, RMR$

Angle of internal friction $\phi = 0.5RMR + 5$

Table 7.2 Cohesion and angle of internal friction

	Cohesion C	Angle of internal friction ϕ
Ore zone	3.5	38.5
Waste	3.3	38

7.2. Rock Mass Design Parameters for Stability Analyses

Rock mass and material properties were determined by means of both geotechnical field survey and laboratory tests prior to stability analyses. Generalized Hoek-Brown failure criterion and GSI data were used to determine the design input parameters which represent the rock mass behavior for stability analyses. For the intact rock specimen that constitutes the rock mass, the criterion is expressed by the equation as below:

$$\sigma_1 = \sigma_3 + \sigma_{ci}\left(m_b \frac{\sigma_3}{\sigma_{ci}} + s\right)^a \qquad (7.1)$$

where,

σ_1 *and* σ_3 are the major and minor effective principal stresses at failure,

σci is the uniaxial compressive strength (UCS) of the intact rock material,

mb is the value of Hoek-Brown constant for the rock mass,

s *and* a are the rock mass constants, where s=1 for intact rock.

The value of mb, s *and* a are calculated by,

$$m_b = m_i \exp\left(\frac{GSI - 100}{28 - 14D}\right) \qquad (7.2)$$

$$s = \exp\left(\frac{GSI - 100}{9 - 3D}\right) \qquad (7.3)$$

$$a = \frac{1}{2} + \frac{1}{6}\left(e^{-GSI/15} - e^{-20/3}\right) \qquad (7.4)$$

where, D is the disturbance factor depends upon the degree of disturbance of the rock mass has been subjected to by blasting and stress relaxation.

Moreover, the deformation of the rock mass Em is calculated by using the equation as follows,

$$E_m = \left(1 - \frac{D}{2}\right)\sqrt{\frac{\sigma_{ci}}{100}} \times 10^{\left(\frac{GSI-10}{40}\right)} \qquad (7.5)$$

The Generalized Hoek-Brown criterion assumes isotropic rock and rock mass behavior and it is applicable to intact rock or heavily jointed rock masses which can be considered as homogeneous and isotropic (Hoek *et al*., 1995). The transition from an isotropic intact rock specimen through a highly anisotropic rock mass in which failure is controlled by one or two discontinuities, and to an isotropic heavily jointed rock mass is summarized schematically as shown in Figure 3.18 (Hustrulid *et al*., 2001).

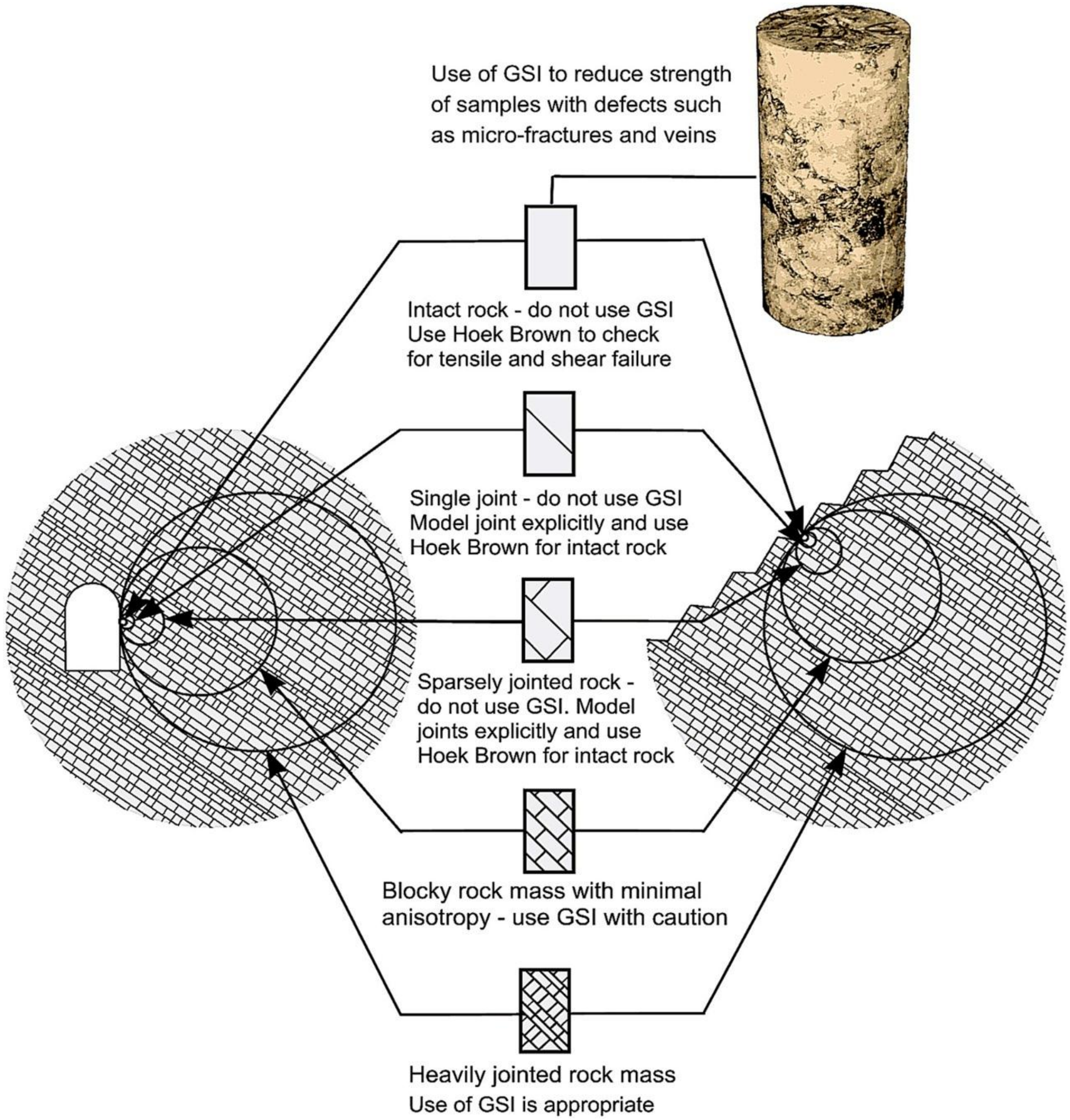


Figure 7.3 Schematic representation of rock mass and material condition (Hoek et al., 1995)

For stability analyses, most of the geotechnical calculations are conducted in terms of the Mohr-Coulomb failure criterion, Therefore, it is required to determine the equivalent Mohr-Coulomb parameters, cohesion (c), and internal friction angle (φ). In the process, Hoek-Brown failure envelope is translated to a linear Mohr-Coulomb failure envelope to estimate the inputs used into stability analysis. Considering the studies of Hoek and Brown (1997) and Hoek *et al*., (2002), equivalent Mohr-Coulomb parameters were determined by using the equations as followed.

$$\varphi' = sin^{-1}\left[\frac{6am_b(s+m_b\sigma'_{3n})^{a-1}}{2(1+a)(2+a)+6am_b(s+m_b\sigma'_{3n})^{a-1}}\right] \tag{7.6}$$

$$c' = \frac{\sigma_{ci}[(1+2a)s+(1-a)m_b\sigma'_{3n}](s+m_b\sigma'_{3n})^{a-1}}{(1+a)(2+a)\sqrt{1+(6am_b(s+m_b\sigma'_{3n})^{a-1})/((1+a)(2+a)}} \tag{7.7}$$

where φ' is the internal friction angle, c' is the cohesion and $\sigma'3n=\sigma'3max/\sigma ci$ is the upper limit of confining stress.

The equivalent rock mass parameters can also be estimated by using RocLab v1.0 software in which all these methods are implemented.

Open pit mine geometries vary depending on the depth, morphology, extent (volume) and attitude of the deposit to be mined. In addition to orebody geometry, geotechnical properties of ore and waste rock materials can also significantly determine pit geometry. Rock mass properties considerably govern the large-scale geometry, which is the overall pit slope angle and depth as well as that of individual benches or bench stacks. Major structures such as faults, thrust or shears, or pervasive fabric such as bedding or foliation may also govern the bench face, stack or overall angles of a pit slope.

7.3. Geometry of Open Pit for Okote Gold Deposit

There is little structural discontinuity data gathered from North Okote area. As a result, consistent determination of bench geometry has not been carried out. That means, either oriented core logging or cable supported TeleViewer technology is recommended to map the subsurface structural discontinuities. Foliation is considered as the most structural plane on which failure may take place. This parameter is inserted into datamine studio3 to derive 3D view of the open pit.

Table 7.3 Input parameters for pit design based on geotechnical study.

Parameters	Values	Maximum allowance
Bench face angle (^{0})	68	±3
Bench height (m)	14	±2
Bench width (m)	7	±2
Road gradient (%)	10	0
Road width (m)	18	±2
Ditch at toe (m)	2	2
Bank width (m)	5	±1
Inter-ramp angle (^{0})	60	±2

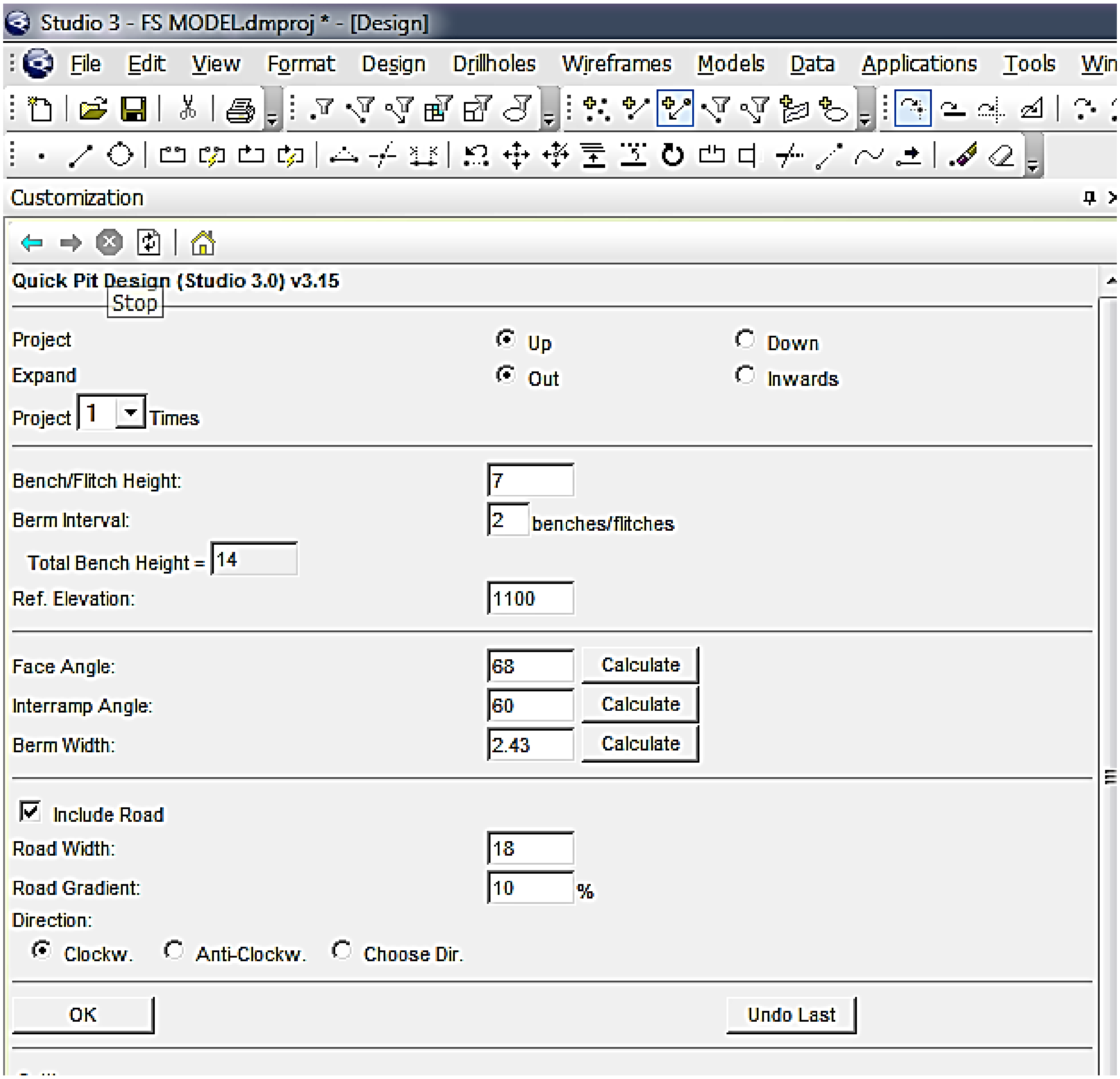


Fig. 7.4 Snapshot photo from Datamine Studio3 (pit designing process)

Selection of the bench geometries is based on the

- ✓ Rock mass properties in the pit area
- ✓ Average dip angle of the foliation planes (~67^{0}) which is considered as failure plane
- ✓ Economic viability of the slope
- ✓ Mining equipments used

Note: $\pm$ is allowed for the engineers because the range is within the acceptable criteria at development stage.

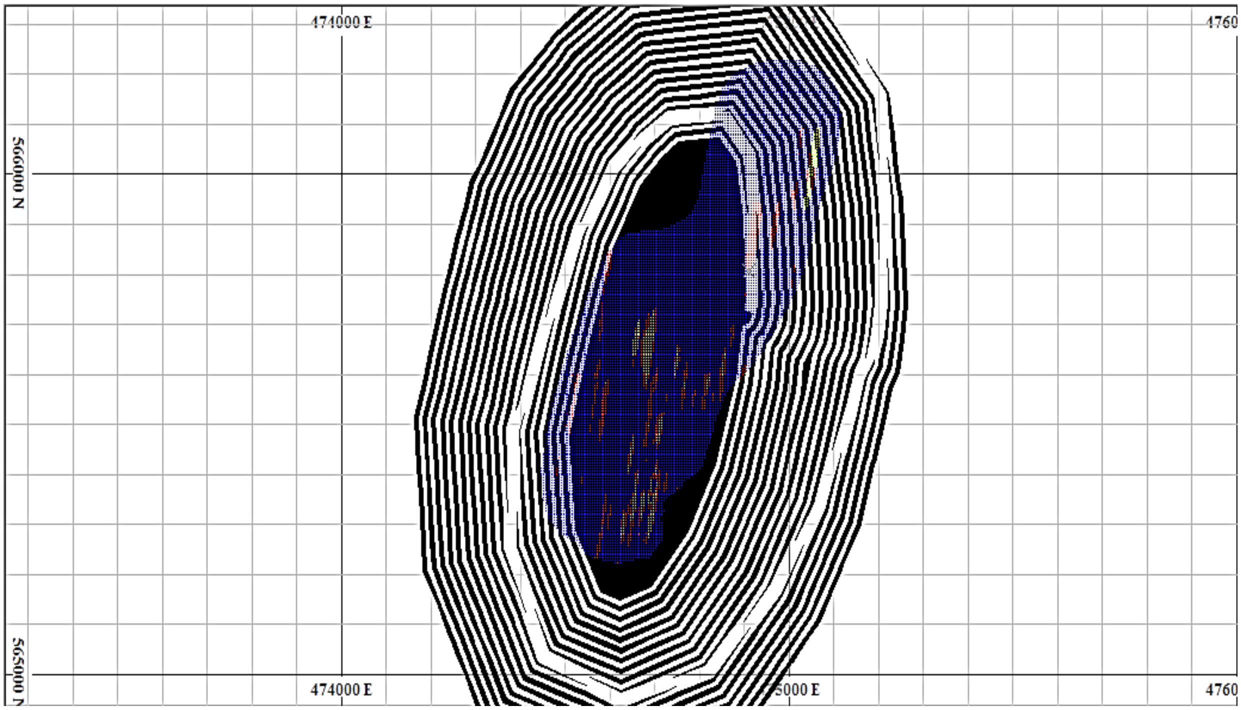


Fig. 7.5 Planar view of the proposed pit design overlapped with the orebody.

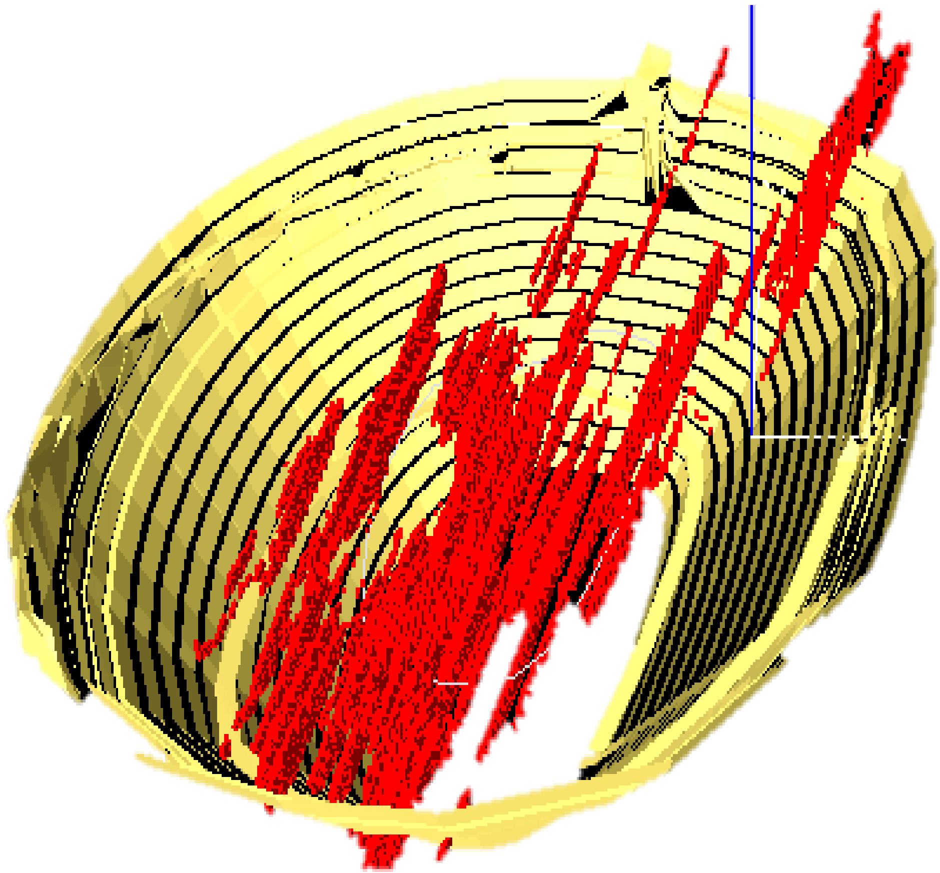
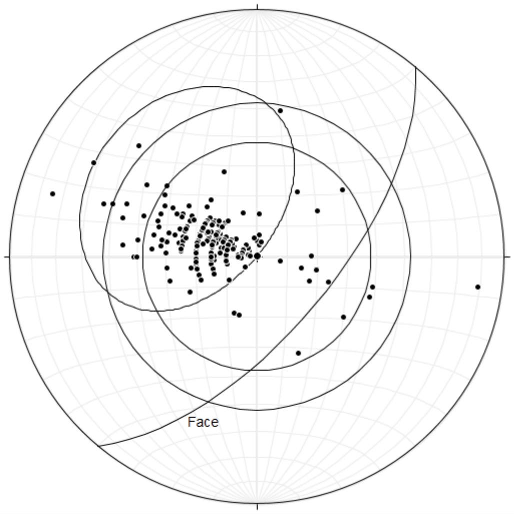


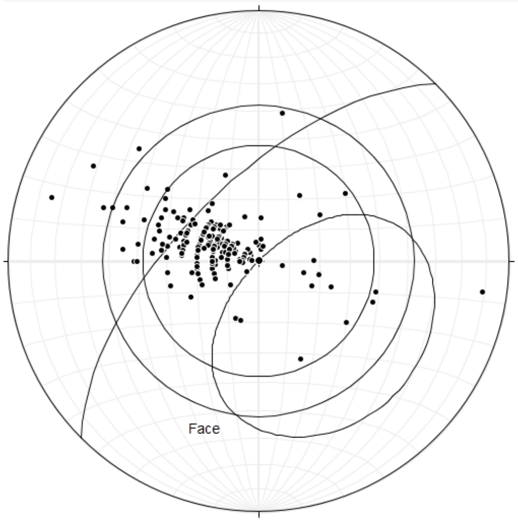
Fig. 7.6 3D visualization of proposed open pit mine of Okote with block models.

7.4. Kinematic Analysis

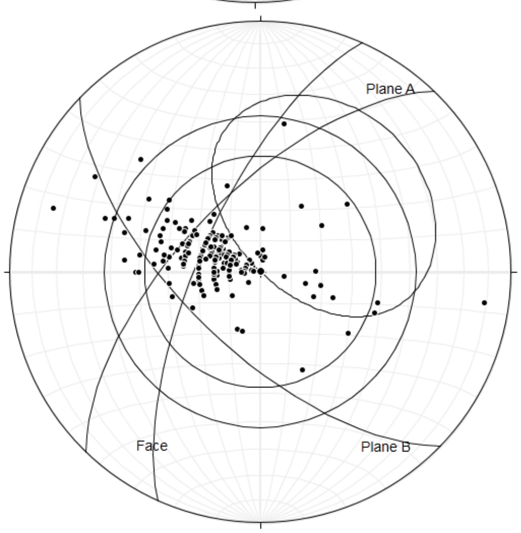
Rock slope stability analysis is routinely performed for assessing safe and functional design of excavated slopes (such as open pit mines, road cuts, etc.) and/or equilibrium conditions of natural slopes. The analysis is usually conducted to determine rock slope stability conditions, investigate potential failure mechanisms, determine slope sensitivity to different triggering mechanisms and design optimal slopes with regards to safety, reliability and economics. Geological investigation of rock slope is an important exercise for stability of slopes and safety of personnel. Kinematic analysis using the stereographic projection was carried out to identify possible mode of failures. Two design domains are selected for kinematic analysis and discontinuity data are plotted on the stereonet version 10.0.



a) Potential plane failure depicted by stereographic projection on the eastern slope face (dipping towards west)



b) Potential toppling failure depicted in stereographic projection in northwestern part of the slope face of the open pit (dipping towards southeast)



c) Potential wedge failures identified by stereographic projection in the southwestern slope face (dipping towards northeast)

Fig. 7.7 Potential failure mechanisms identified by stereographic plots

7.5. Design acceptance criteria

In open pit mining slope failure is not easily defined. Whereas in some engineering systems failure occurs immediately and is not reversible (e.g. the buckling of a structural column or the failure of a dam), in an open pit mine slope failure may take place gradually so that determining the stage at which the pit wall ceases to perform adequately may be highly subjective (Peter and Stacey, 2006)

A slope is considered to be stable when the forces resisting the potentially shearing, sliding or toppling mass of material on the slope are greater than the forces driving the mass. The resisting forces are provided by the strength of the rock material and/or geological structures, dependent on the mode of potential failure. Whereas the driving forces are primarily dependent on the unit weight of the rock, groundwater pressures in the rock mass, and any other forces exerted by in situ stress field or external loads such as loaded trucks on ramps, mine infrastructure near pit crest and seismicity etc. The ratio of the resisting forces to the driving forces is termed the Factor of Safety (FOS) and has been the basis of stability acceptance criterion for many engineering applications. When the FOS = 1 the slope is considered to be in a state of "limiting equilibrium" and if the FOS > 1 the slope is considered to be theoretically stable.

There are no strict criteria that specify the acceptable FOS, but for static loading conditions the values of 1.2 to 2.0 are commonly used depending on the type of slope and its importance. The FOS however is based on single values selected to represent the rock mass parameters used in the stability calculations. The reliability of the computed FOS depends on the selection of the single values from populations with significant distributions.

All the factor of safety calculated for bench, inter-ramp, and overall slope angles for the proposed open pit range between 1.12 to 1.30 for dynamic condition and 1.27 to 1.43 for static condition.

Table 7.4 Typical FOS and POF acceptance criteria values (after Priest and Brown, 1983)

Consequence of failure	Examples	Acceptable values		
		Mean FOS	Minimum P[FOS<1.0]	Maximum P[FOS<1.5}
Not serious	Individual benches; small (<50 m), temporary slopes, not adjacent to haulage roads	1.3	10%	20%
Moderately serious	Any slope of a permanent or semi-permanent nature	1.6	1%	10%
Very serious	Medium sized (50-100 m) and high slopes (<150 m) carrying major haulage roads or underlying permanent mine installation	2.0	0.3%	5%

Table 7.5 Typical FOS and POF acceptance criteria values (Read and Stacy, 2009)

Slope scale	Consequences of failure	Acceptance criteria[a]		
		FoS (min) (static)	FoS (min) (dynamic)	PoF (max) P[FoS ≤ 1]
Bench	Low–high[b]	1.1	NA	25–50%
Inter-ramp	Low	1.15–1.2	1.0	25%
	Moderate	1.2	1.0	20%
	High	1.2–1.3	1.1	10%
Overall	Low	1.2–1.3	1.0	15–20%
	Moderate	1.3	1.05	10%
	High	1.3–1.5	1.1	5%

a: Needs to meet all acceptance criteria
b: Semi-quantitatively evaluated

CHAPTER EIGHT

8. CONCLUSION AND RECOMMENDATION

8.1. Conclusion

Geology of Okote area is part of the metasedimentary and metavolcanics of the Southern Megado terrain. These areas are mainly covered by carbonate amphibole schist, massive weakly foliated metagabbro, carbonate chlorite schist or chlorite schist, talc tremolite actinolite schist, meta-granodiorite, meta-diorite, and aplitic dikes. Most of them are found as lenses distributed unevenly in carbonate amphibole schist.

All rock exposures generally strike along NNE-SSW direction and steeply dipping towards West at an angle ranging between 65 to 70^0.

There are two distinct types of mineralization in Okote area: the shear hosted Carbonate-Chlorite related mineralization (TYPE-I) and the metagranodiorite hosted mineralization (TYPE-II). Both are economically viable at the current stage of exploration and therefore given special considerations in terms of geological and geotechnical characteristics of the orebody for the future mine planning and design.

The delineated deposit for the currently proposed open pit contains 77 million tons of ore at reserve level with average grade of 1.18 g/ton and cutoff grade 0.49 Au g/ton. The contained value (total amount of Au in the deposit in Kg) of the deposit is 23,062 Kg.

The deposit is elongated, NNE-SSW trending with maximum length of 980Kms within the pit area.

Geotechnical study reveals that the possible mining method is hard rock mining which requires heavy blasting due to the strength of both ore body and waste materials. The waste material has RMR_{89} value of 66 (rock mass class II) and that of ore body is 67 (rock mass class II).

All the factor of safety calculated for bench, inter-ramp, and overall slope angles for the proposed open pit range between 1.12 to 1.30 for dynamic condition and 1.27 to 1.43 for static condition.

8.2. Recommendation

3D structural modelling using TeleViewer technology is recommended to map sub-surface structural discontinuities to get detail information prior to design the practical (final) mine design.

National Mining Corporation should conduct detail economic analysis for each bench or at several defined levels by calculating the Net Present Value (NPV). The final economic decision should be carried out based on the viability of the NPV calculated at each level.

South Okote deposit has limited mineralization compared to the North Okote but, clear delineation between both is required.

The deposit has underground mining potential below the final pit floor (900-1100m), so that transition to underground mining is recommended.

RMR only cannot provide detail information to conduct underground mining design. The Barton's Q rock mass classification system is required.

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ANNEXES

1. Discontinuity data of Okote

S.N.	Dip	Dip Direction	Longitude	Latitude	Elevation
1	72	300	474964	565973	1411
2	43	286	475194	566137	1474
3	74	290	475132	565911	1414
4	81	310	475031	565951	1420
5	50	110	475060	565788	1382
6	82	101	474963	565720	1359
7	72	90	474662	565572	1351
8	62	300	474675	565555	1343
9	80	300	474690	565567	1340
10	25	300	474905	565113	1352
11	70	282	474762	564919	1427
12	87	305	474666	564974	1427
13	38	290	474615	564833	1447
14	70	310	474585	565020	1407
15	76	265	474531	564978	1409
16	71	115	474415	564971	1386
17	70	102	474431	564943	1392
18	50	105	474506	565077	1390
19	55	125	475113	566073	1464
20	62	307	474927	566227	1495
21	75	296	474830	566210	1493
22	46	303	474811	565891	1394
23	70	305	475010	566118	1472
24	52	308	474883	566128	1473
25	55	299	475088	566144	1481
26	67	315	475134	566048	1463
27	86	314	475209	566027	1456
28	70	306	475281	566009	1447
29	72	310	475182	565980	1438
30	77	299	475091	565895	1409
31	73	309	475079	565870	1405
32	72	308	475098	565790	1381
33	72	305	474777	565809	1374
34	80	303	474675	565761	1383
35	73	292	474748	565755	1373

36	86	317	474691	565731	1376
37	75	343	474650	565683	1371
38	74	310	474710	565656	1356
39	65	298	474677	565518	1331
40	73	313	475044	565652	1347
41	75	303	474971	565567	1325
42	70	303	475067	565512	1311
43	75	317	475069	565495	1311
44	66	321	474925	565447	1301
45	75	312	474805	565362	1319
46	85	318	474609	565239	1337
47	65	302	474739	565160	1364
48	50	290	474687	565139	1367
49	55	282	474695	565110	1382
50	55	274	474637	565132	1375
51	80	298	474476	565187	1364
52	78	303	474516	564872	1416
53	75	300	474548	564889	1421
54	76	302	474630	564832	1447
55	87	282	474632	564849	1446
56	75	303	474635	564927	1432
57	75	310	474828	564862	1420
58	75	322	474702	565783	1391
59	76	288	474878	566238	1503
60	80	290	474754	566004	1433
61	73	290	474755	565977	1426
62	78	283	474849	565982	1416
63	80	298	474836	565935	1405
64	80	297	474825	565920	1398
65	75	282	474921	565885	1384
66	70	285	474888	565813	1363
67	85	299	474971	565995	1425
68	62	298	474980	565937	1407
69	80	289	474731	565940	1426
70	75	301	474773	565849	1387
71	76	296	475080	566089	1466
72	85	291	475077	566013	1449
73	80	293	475099	566010	1449
74	80	289	475141	565931	1424

75	80	295	475087	565947	1428
76	65	290	475107	565850	1395
77	72	295	475125	565841	1395
78	73	298	475196	565819	1391
79	66	285	475173	565749	1372
80	70	300	475149	565725	1358
81	70	290	475124	565738	1365
82	65	290	474863	565714	1348
83	70	293	474670	565677	1364
84	77	295	474834	565624	1328
85	84	275	474675	565617	1352
86	82	294	474620	565583	1361
87	60	285	474693	565507	1327
88	80	293	474596	565494	1342
89	60	298	474677	565443	1319
90	63	290	474717	565445	1312
91	66	300	474805	565416	1300
92	65	290	474527	565406	1332
93	58	300	474747	565331	1320
94	55	294	474734	565261	1335
95	65	294	474568	565283	1334
96	80	294	474490	565286	1339
97	81	296	475010	565634	1342
98	80	290	475106	565587	1333
99	80	290	474906	565544	1314
100	85	301	474953	565528	1313
101	65	293	474822	565387	1310
102	65	285	474878	565267	1338
103	78	294	474763	565215	1349
104	70	297	474533	565223	1342
105	70	292	474529	565205	1358
106	70	292	474413	565067	1380
107	70	295	474418	565047	1381
108	70	252	474390	565047	1379
109	55	290	474404	565034	1379
110	70	290	474372	564968	1382
111	55	290	474387	564928	1389
112	85	282	474353	564902	1393
113	80	290	474487	564849	1412

114	80	284	474517	564861	1416
115	65	291	474513	564900	1413
116	70	296	474558	564907	1420
117	79	285	474563	564853	1431
118	78	296	474575	564849	1431
119	80	292	474588	564845	1437
120	80	298	474604	564822	1444
121	87	282	474612	564836	1444
122	73	288	474609	564862	1442
123	82	289	474619	564813	1451
124	64	280	474642	564836	1449
125	70	275	474687	564913	1441
126	56	287	474660	564985	1422
127	75	285	474648	564985	1423
128	35	289	474600	564959	1427
129	58	276	474599	564976	1428
130	75	285	474577	564993	1413
131	62	285	474861	565078	1360
132	80	277	474848	565038	1372
133	70	275	474895	565025	1363
134	50	278	474834	564948	1400
135	65	282	474779	564939	1417
136	70	302	474801	564932	1413
137	75	287	474836	564898	1414
138	49	270	474863	564954	1383
139	50	270	474903	564953	1377
140	85	288	474825	564876	1418
141	75	298	474828	564837	1429
142	65	282	474749	564852	1445
143	85	295	474746	564895	1437
144	75	289	474745	564805	1453
145	66	285	474815	564771	1446
146	43	292	474339	564910	1389
147	70	280	474791	564979	1407
148	53	304	474912	565709	1348
149	78	294	474952	565696	1356
150	70	298	475091	565648	1348
151	75	273	475035	566181	1483
152	88	275	474776	566099	1468

153	72	283	474703	565856	1409
154	75	261	475166	565981	1437
155	76	271	475116	565886	1405
156	85	227	474750	565283	1332
157	65	278	474719	565230	1340
158	60	263	474690	565232	1339
159	78	275	475048	565703	1359
160	75	270	474765	565200	1355
161	68	260	474856	565180	1352
162	70	268	474550	565235	1344
163	75	275	474589	565183	1361
164	65	274	474581	565149	1370
165	65	274	474567	565103	1383
166	80	270	474540	565154	1372
167	85	227	474496	565096	1385
168	65	276	474438	565140	1371
169	60	254	474430	565113	1375
170	60	280	474413	565112	1374
171	60	272	474406	565089	1376
172	80	255	474384	565010	1379
173	80	262	474372	564998	1379
174	75	261	474399	564987	1383
175	85	282	474368	564922	1390
176	70	282	474399	564896	1394
177	80	261	474415	564892	1393
178	70	283	474410	564881	1394
179	70	263	474425	564878	1391
180	85	271	474428	564915	1393
181	80	275	474440	564930	1394
182	70	273	474433	564954	1389
183	80	292	474459	564897	1400
184	80	292	474457	564917	1400
185	60	279	474479	564933	1402
186	85	266	474542	564913	1415
187	75	254	474574	564920	1424
188	75	270	474555	564942	1416
189	75	267	474568	564828	1434
190	75	270	474611	564892	1441
191	65	276	474630	564865	1444

192	80	272	474635	564891	1443
193	65	242	474547	564991	1410
194	75	248	474545	565002	1407
195	80	283	474531	564978	1409
196	75	284	474535	565030	1401
197	70	280	474740	565115	1379
198	84	270	474759	565101	1378
199	70	265	474828	565073	1374
200	70	247	474820	565004	1395
201	70	258	474837	564791	1437
202	60	339	474832	564751	1452
203	58	280	474787	564769	1454
204	85	0	474748	565531	1323
205	65	32	474444	565120	1373
206	65	53	474444	565108	1376
207	70	202	474433	565086	1379
208	55	157	474443	565067	1382
209	65	110	474564	564895	1426
210	70	103	474574	564873	1430
211	75	105	474818	565034	1385
212	70	197	474753	564814	1453
213	72	297	474846	566071	1449
214	45	275	474829	565885	1389
215	15	287	475274	565994	1442
216	35	313	474644	565601	1361
217	78	230	474640	565586	1356
218	54	52	474724	565644	1353
219	11	98	474649	565652	1364
220	40	9	475237	566036	1458
221	86	14	475109	566004	1443
222	76	4	475128	565990	1437
223	83	6	475131	565932	1422
224	85	16	475136	565907	1414
225	84	351	475107	565903	1408

2. Rock strength and unit weight test result

Sample ID	Sample interval (m)	Rock unit	Pli (MPa)	UCS (MPa)	Ei (GPa)	Ti (MPa)	mi	ϒ (g/cc)
1650_N/2	31.70-31.80	CCS/CS	0.698	16.752	3267.4	8.032	10	2.82
800_N1_1	121.85-121.95	MG/A?	7.738	185.712	58604	17.238	27	2.93
800_N1_2	42.70-42.80	MG/A	0.893	21.432	4391.4	8.517	24	2.79
1700_N3	80.25-80.35	MGD	9.701	232.824	76870	18.587	29	2.69
1300_N1	73.65-73.75	CAS	2.372	56.928	14181	11.432	11	2.94
Averages			4.2804	102.7296	31463	12.76	20	2.834

Key: CCS/CS= Carbonate Chlorite Schist
MG/A= Metagabbro/Amphibolite
MDG = Metagranodiorite
CAS = Chlorite Amphibole Schist
Pli = Intact Point Load Index
UCS = Intact Uniaxial Compressive Strength
Ti = Intact Tensile Strength
mi = Intact Material Constant
ϒ = Unit weight of rock

3. Okote Gold Mine Project Evaluation Parameters

OKOTE GOLD MINE PROJECT EVALUATION

Technical Parameters		**Financial Parameters**	
Average Grade (g Au/ton)	1.18	Current Gold Price ($/g)	42.45
Cutoff Grade (g Au/ton)	0.49	Mine Operating Cost ($/t)	10.40
Reserve Level at Cutoff (mln tons)	77.000	Mill Operating Cost ($/t)	7.00
Contained Value (kg Au)	23,062	Total Operating Cost ($/t)	17.40
Stripping Ratio	7.43	Mine Capital Cost ($ 000)	20,000
Ore Production Rate (t/d)	4,690	Mill Capital Cost ($ 000)	54,318
Mill Recovery	80%	Total Capital Cost ($ 000)	74,318
Operating days/year	355	Working Capital ($ 000)	12,000
Mine Life (year)	13	Capitalized Exploration Cost ($ 000)	250
		Depletion Allowance (%)	15%
		Royality (% Net Smelter Return)	8%
		Income Tax Rate (%)	25%
		Salvage Value (% of Capital Costs)	10%
		Real Risk-adjusted Discount Rate (%)	10%
		Inflation (%)	3%

4. Geomechanical Classification of Rock Masses (Waste)

GEOTECHNICAL CORE LOGGING [RQD and FF (fracture frequency)]										
BH ID:DH 1650N/2			**Area: NORTH OKOTE**					**Logged By:Shibiru Merga**		
			page:1 of 1						**Date:April 05,2018G.C**	
Interval		**Core recovery**		**sticks>10cm**		**Fracture frequency (FF)**				**Remark**
From	**To**	**m**	**%**	**m**	**RQD**	**Open**	**SH**	**WH**	**SS**	**(e.g. fault/dyke)**
0	1.3	1.27	98	0.71	55.91	5				
1.3	6.7	5.4	100	3.31	61.30	12				
6.7	8.25	1.55	100	1.12	72.26	8				
8.25	9.5	1.17	94	0.45	38.46	7				
9.5	11.25	1.75	100	1.31	74.86	10				
11.25	12.65	1.4	100	0.88	62.86	7				
12.65	14.25	1.6	100	0.85	53.13	11				
14.25	15.6	1.35	100	1.13	83.70	9				
15.6	17.05	1.45	100	1.05	72.41	12	3			
17.05	20.05	2.95	98	1.6	54.24	12		2		
20.05	23.25	3.12	98	1.87	59.94	10				
23.25	26.15	2.73	94	1.7	62.27	11				
26.15	29.15	2.73	91	2.45	89.74	17	1			
29.15	31.55	2.4	100	2.25	93.75	16	2	1		
31.55	31.8	0.25	100	0.2	80.00	3				
31.8	34.8	3.01	100	2.97	98.67	14				
34.8	37.8	2.98	99	2.27	76.17	11	1			
37.8	41	3.15	98	2.83	89.84	13	2			
41	44	3	100	2.91	97.00	8				
44	47	2.95	98	2.74	92.88	13	3			
47	50	3	100	2.64	88.00	15		3		
50	53	3	100	2.9	96.67	11				
53	56	3.01	100	2.92	97.01	13	1			
56	59	2.95	98	2.86	96.95	7				
59	62	3	100	2.19	73.00	11	2			
62	65	2.91	97	2.27	78.01	11		3		
65	68	2.94	98	2.66	90.48	9				
68	71	3	100	2.6	86.67	8				
71	74	3	100	2.52	84.00	9				
74	76.6	2.36	91	1.45	61.44	3				
76.6	79.65	3.05	100	2.85	93.44	6				
		Sum↓	**Ave↓**	**Sum↓**	**Ave↓**	**FF/m↓**	**Sum↓**	**Sum↓**	**Sum↓**	
Total		**78.43**	**98**	**62.5**	**77.90**	**3.978**	**15**	**9**	**0**	

SH-strongly healed ***WH-weakly healed*** ***SS-slickensided***

5. Table RQD and Lithology from DH 1650 N/2

Drill Hole ID: DH 1650N/2
X: 474979.2188m
Y: 566190.3125m
Z: 1508m
DH azimuth:
DH inclination:

National Mining Corporation (N.Mi.C)
Okote Gold Project (North)
Geotechnical Core Logging

Logged by: Shibiru Merga
Date: April 5, 2018

Interval (m)		SCR	CR	TCR	>10cm	RQD	Lithology			Lithology Description	Remark
From	To	(m)	(%)	(m)	(m)	(%)	From	To	L. Index		
0.00	1.30	1.27	98	1.30	0.71	54.62	0.00	3.00	MG/A	Metagabbro/Amphibolite	
1.30	6.70	5.4	100	5.40	3.31	61.30	3.00	6.45	CAS	Carbonate Amphibole Schist	
6.70	8.25	1.55	100	1.55	1.12	72.26	6.45	9.25	MGD	Metagranodiorite	
8.25	9.50	1.17	94	1.25	0.45	36.00	9.25	11.20	CAS	Carbonate Amphibole Schist	
9.50	11.25	1.75	100	1.75	1.31	74.86	11.20	12.05	MG/A	Metagabbro/Amphibolite	
11.25	12.65	1.4	100	1.40	0.88	62.86	12.05	12.60	CAS	Carbonate Amphibole Schist	
12.65	14.25	1.6	100	1.60	0.85	53.13	12.60	13.00	MGD	Metagranodiorite	
14.25	15.60	1.35	100	1.35	1.13	83.70	13.00	16.60	CAS	Carbonate Amphibole Schist	
15.60	17.05	1.45	100	1.45	1.05	72.41	16.60	17.20	MGD	Metagranodiorite	
17.05	20.05	2.95	98	3.00	1.60	53.33	17.20	17.85	CAS	Carbonate Amphibole Schist	
20.05	23.25	3.12	98	3.20	1.87	58.44	17.85	18.55	MGD	Metagranodiorite	
23.25	26.15	2.73	94	2.90	1.70	58.62	18.55	21.15	CCS/CS	Carbonate Chlorite Schist/Carbonate Schist	
26.15	29.15	2.99	100	3.00	2.45	81.67	21.15	22.05	MGD	Metagranodiorite	
29.15	31.55	2.4	100	2.40	2.25	93.75	22.05	24.70	CAS	Carbonate Amphibole Schist	
31.55	31.80	0.25	100	0.25	0.20	80.00	24.70	26.85	MGD	Metagranodiorite	
31.80	34.80	3.01	100	3.00	2.97	99.00	26.85	29.25	CAS	Carbonate Amphibole Schist	
34.80	37.80	2.98	99	3.00	2.27	75.67	29.25	32.20	CCS/CS	Carbonate Chlorite Schist/Carbonate Schist	

37.80	41.00	3.19	100	3.20	2.83	88.44	32.20	39.25	CAS	Carbonate Amphibole Schist	
41.00	44.00	3	100	3.00	2.91	97.00	39.25	39.75	MGD	Metagranodiorite	
44.00	47.00	3	100	3.00	2.74	91.33	39.75	46.45	CAS	Carbonate Amphibole Schist	
47.00	50.00	3	100	3.00	2.64	88.00	46.45	58.75	MG/A	Metagabbro/Amphibolite	
50.00	53.00	3	100	3.00	2.90	96.67	58.75	66.65	CAS	Carbonate Amphibole Schist	
53.00	56.00	3.01	100	3.00	2.92	97.33	66.65	67.55	MGD	Metagranodiorite	
56.00	59.00	3	100	3.00	2.86	95.33	67.55	68.75	CAS	Carbonate Amphibole Schist	
59.00	62.00	3	100	3.00	2.19	73.00	68.75	76.60	MG/A	Metagabbro/Amphibolite	
62.00	65.00	2.99	100	3.00	2.27	75.67	76.60	79.65	MG/A	Metagabbro/Amphibolite	
65.00	68.00	2.99	100	3.00	2.66	88.67					
68.00	71.00	3	100	3.00	2.60	86.67					
71.00	74.00	3	100	3.00	2.52	84.00					
74.00	76.60	2.6	100	2.60	1.45	55.77					
76.60	79.65	3.05	100	3.05	2.85	93.44					
			99			**76.87**					

9. Geomechanical Classification of Rock Masses (Orebody)

ROCK MASS CLASSIFICATION

NORTH OKOTE TARGET AREA (waste)

DATE	**18,04,2018**	**HOLE ID**	BH1150N1,BH1100N1

1) Joints spacing — **RMR index**

Sav	≥2m	0.6m-2m	20cm-60cm	6cm-20cm	<6cm	RMR index
index	20	15	10	8	5	**10**

2) RQD (Rock Quality Designation)

RQD	90-100%	75-90%	50-75%	25-50%	<25	RMR index
index	20	17	13	8	3	**13**

3) Compressive strength of intact rock

σ_c (Mpa)	≥250	100-250	50-100	25-50	5_25	1_5	<1	RMR index
index	15	12	7	4	2	1	0	**12**

4) Joints condition

						RMR index
LENGTH	<1m	1-3m	3-10m	10-20m	≥20m	
INDEX	6	4	2	1	0	**3**
OPENING	0	<0.1mm	0.1-1mm	1-5mm	≥5mm	
INDEX	6	5	4	1	0	**4.5**
ROUGHNESS	very rough	rough	slightly rough	smooth	slikensided	
INDEX	6	5	3	1	0	**5**
FILLING	none	hard&<5mm	hard&≥5mm	soft&<5mm	soft&≥5mm	
INDEX	6	4	3	2	0	**2**
WEATHERING	unweathered	slightly weathered	Moderately weathered	highly weathered	decomposed	
INDEX	6	5	3	1	0	**5.5**

5) Hydraulic condition (ground water)

condition	dry	damp	wet	dripping	flowing	RMR index
index	15	10	7	4	0	**13**

6) Orientation of discontinuity - Adjustment

ADVANCE TOWARDS IMMERSION			index	ADVANCE AGAINST IMMERSION			index
inclination	**A**	45^0-90^0	0	inclination	**C**	45^0-90^0	-5
	B	20^0-45^0	-2		**D**	20^0-45^0	-10

PARALLEL TO IMMERSION			index	ALL DIRECTION			index	RMR index
inclination	**E**	45^0-90^0	-12	inclination	**G**	0^0-20^0	-5	**-2**
	F	20^0-45^0	-5					

ROCK MASS RATING (RMR)	**≥80**	**60-80**	**40-60**	**20-40**	**<20**	**66**
ROCK MASS CLASS (RMC)	**I**	**II**	**III**	**IV**	**V**	**II**

ROCK MASS CLASSIFICATION (RMR_{89})

NORTH OKOTE TARGET AREA (Orebody)

DATE	18/04/2018	HOLE ID	DH 1700 N1, DH 1300 N2, DH 212-10E

1) Joints spacing — **RMR index**

Sav	≥2m	0.6m-2m	20cm-60cm	6cm-20cm	<6cm	RMR index
INDEX	20	15	10	8	5	**10**

2) RQD (Rock Quality Designation)

RQD	90-100%	75-90%	50-75%	25-50%	<25	RMR index
INDEX	20	17	13	8	3	**13**

3) Compressive strength of intact rock

σ_C (MPA)	≥250	100-250	50-100	25-50	5_25	1_5	<1	RMR index
INDEX	15	12	7	4	2	1	0	**12**

4) Joints condition

						RMR index
LENGTH	<1m	1-3m	3-10m	10-20m	≥20m	**3**
INDEX	6	4	2	1	0	
OPENING	0	<0.1mm	0.1-1mm	1-5mm	≥5mm	**4.5**
INDEX	6	5	4	1	0	
ROUGHNESS	very rough	rough	slightly rough	smooth	slikensided	**5**
INDEX	6	5	3	1	0	
FILLING	none	hard&<5mm	hard&≥5mm	soft&<5mm	soft&≥5mm	**2**
INDEX	6	4	3	2	0	
WEATHERING	unweathered	slightly weathered	Moderately weathered	highly weathered	decomposed	**5.5**
INDEX	6	5	3	1	0	

5) Hydraulic condition (ground water)

CONDITION	dry	damp	wet	dripping	flowing	RMR index
INDEX	15	10	7	4	0	**13**

6) Orientation of discontinuity - Adjustment

ADVANCE TOWARDS IMMERSION			Index	ADVANCE AGAINST IMMERSION			index	
Inclination	**A**	45^0-90^0	0	Inclination	**C**	45^0-90^0	-5	
	B	20^0-45^0	-2		**D**	20^0-45^0	-10	
PARALLEL TO IMMERSION			Index	ALL DIRECTION			Index	
Inclination	**E**	45^0-90^0	-12	Inclination	**G**	0^0-20^0	-5	**-2**
	F	20^0-45^0	-5					
ROCK MASS RATING (RMR)			**≥80**	**60-80**	**40-60**	**20-40**	**<20**	**66**
ROCK MASS CLASS (RMC)			**I**	**II**	**III**	**IV**	**V**	**II**

NATIONAL MINING CORPORATION										
GEOTECHNICAL CORE LOGGING (Rock Quality Designation and Fracture Counts)										
Bore hole ID:BH 950N1		Prospect: OKOTE NORTH					Logged By:			
		page:1 of 4						Date: SEP 22, 2011G.C		
Interval		Core recovery		sticks>10cm		Fracture Count				Comments (e.g.fault/dyke)
From	To	m	%	m	RQD	Open	SH[1]	WH[2]	Slikensided	
35.86	37.7	1.84	100							
37.7	40.05	2.35	100	1.92	81.70213	16				
40.05	40.7	0.65	100	0.61	93.84615	6				
40.7	43.42	2.72	100	1.41	51.83824	1				
43.42	44.92	1.50	100	1.12	74.66667	20				
44.92	46.64	1.72	100	1.39	80.81395	10				
46.64	49.64	3.00	100	2.76	92	20				
49.64	50.86	1.22	100	1.12	91.80328	7				
50.86	51.91	1.05	100	0.8	76.19048	6				
51.91	54.42	2.51	100	2.17	86.45418	14				
54.42	55.63	1.21	100	0.89	73.55372	8				
55.63	57.03	1.40	100	1.31	93.57143	19				
57.03	58.61	1.58	100	0.54	34.17722	4				
58.61	61.08	2.47	100	2.08	84.21053	17				
61.08	63.31	2.23	100	2.17	97.30942	16				
63.31	66.31	3.00	100	2.92	97.33333	16				
66.31	67.55	1.24	100	0.62	50	26				
67.55	70.05	2.50	100	1.22	48.8	11				
70.05	71.03	0.98	100	0.44	44.89796	5				
71.03	71.78	0.75	100	0	0	0				
71.78	74.81	3.03	100	1.85	61.05611	18				

74.81	77.54	2.73	100	2.19	80.21978	15				**D=50^0**
77.54	80.14	2.60	100	1.27	48.84615	23				
80.14	82.87	2.73	100	1.47	53.84615	20				
82.87	85.86	2.99	100	2.4	80.26756	18				
85.86	88.86	3.00	100	1.59	53	24				
88.86	91.83	2.97	100	2.26	76.09428	16				
91.83	94.66	2.83	100	2.15	75.97173	15				
94.66	97.66	3.00	100	2.11	70.33333	20				
97.66	100.61	2.95	100	2.3	77.9661	16				
100.61	103.33	2.72	100	2.06	75.73529	17				
103.33	105.8	2.47	100	1.82	73.68421	14				
105.8	107.3	1.50	100	1.5	100	11				
107.3	111.3	4.00	100	2.6	65	7				
111.3	111.42	0.12	100	0	0	3				
111.42	111.53	0.11	100	0	0	2				
111.53	114.15	2.62	100	2.08	79.38931	11				
114.15	115.57	1.42	100	0.9	63.38028	12				
115.57	116.62	1.05	100	0.53	50.47619	11				
116.62	117.64	1.02	100	0.68	66.66667	8				
117.64	118.47	0.83	100	0.83	100	6				
118.47	121.09	2.62	100	2.62	100	16				
121.09	123.82	2.73	100	1.74	63.73626	20				
123.82	125.79	1.97	100	1.41	71.5736	11				
125.79	126.46	0.67	100	0.52	77.61194	5				
126.46	126.73	0.27	100	0.25	92.59259	3				
126.73	129.3	2.57		sampled						
129.3	130.37	1.07		sampled						
130.37	133.31	2.94		sampled						
133.31	136.33	3.02	100	3.03	100.3311	25				
136.33	139.33	3.00	100	2.05	68.33333	19				

139.33	142.33	3.00	100	2.35	78.33333	19				
142.33	145.28	2.95	100	2.56	86.77966	10				
145.28	148.31	3.03	100	1.46	48.18482	24				**D=60^0**
148.31	151.11	2.80	100	2.1	75	18				
151.11	154.09	2.98	100	2.25	75.50336	15				
154.09	156.77	2.68	100	2.12	79.10448	19				
156.77	158.5	1.73	100	1.54	89.01734	10				
158.5	161.17	2.67	100	1.96	73.40824	16				
161.17	162.47	1.30	100	0.71	54.61538	4				
162.47	165.62	3.15	100	3.15	100	20				
165.62	168.63	3.01	100	2.58	85.71429	17				
168.63	169.48	0.85	100	0.34	40	7				
169.48	172.48	3.00	100	1.34	44.66667	27				
172.48	174.55	2.07	100	0.93	44.92754	18				
174.55	175.23	0.68	100	0	0	0				
175.23	175.53	0.30	100	0	0	0				
175.53	175.93	0.40	100	0	0	0				
175.93	176.01	0.08	100	0	0	2				
176.01	176.51	0.50	100	0.19	38	2				
176.51	176.73	0.22	100	0.19	86.36364	2				
176.73	177.68	0.95	100	0.51	53.68421	10				
177.68	179.9	2.22	100	0.94	42.34234	21				
179.9	180.1	0.20	100	0	0	0				
180.1	180.58	0.48	100	0	0	0				
180.58	181.64	1.06	100	0	0	0				
181.64	182.14	0.50	100	0	0	0				
182.14	182.99	0.85	100	0	0	3				
182.99	184.17	1.18	100	0.39	33.05085	9				
184.17	184.79	0.62	100	0.23	37.09677	4				
184.79	187.15	2.36	100	0.98	41.52542	6				

187.15	188.75	1.60	100	1.58	98.75	12				
188.75	190.65	1.90	100	1.7	89.47368	16				
190.65	191.27	0.62	100	0.55	88.70968	8				
191.27	192.89	1.62	100	0.55	33.95062	14				
192.89	194.37	1.48	100	0.3	20.27027	14				
194.37	196.46	2.09	100	0.54	25.83732	12				
196.46	198.86	2.40	100	1.81	75.41667	15				
			100		**59.75009**	982	**4.938147**			
AVERAGES			CORE RECOVERY		RQD			COUNT PER METER		

SH[1]-strongly healed WH[2]-weakly healed

N.Mi.C (Gold Project)