Addis Ababa University
School of Earth Sciences

Assessment of the loss of gold to the tailing dam: A case study on Meli Gold Plant, Northwestern Tigray, Ethiopia.

A Thesis Submitted to School of Graduate Studies of Addis Ababa University in Partial Fulfillment of the requirements for Degree of Master of Science in Mining geology

By
Liyou Kifle

Advisor: Solomon Tadesse (Prof.)

September, 2020
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This is to certify that the Thesis Prepared by Liyou Kifle, entitled: Assessment of the loss of gold to the tailing dam: A case study on Meli Gold Plant, Northwestern Tigray, Ethiopia submitted in the partial fulfillment for the requirements for the Degree of Master of Science in Mining geology complies with the regulations of the University and meets the accepted standard with respect to originality and quality.

Approved by Board of Examiners:

Signed by the Examining Committee

Prof. Solomon Tadesse Advisor	________________ Signature	____________________ Date
____________________________ Examiner	________________ Signature	____________________ Date
____________________________ Examiner	________________ Signature	____________________ Date
____________________________ Chair of School or graduate Program coordinator	________________ Signature	____________________ Date

Declaration of Originality

I hereby declare that the thesis entitled "Assessment of the loss of gold to the tailing dam: A case study on Meli Gold Plant, Northwestern Tigray" has been carried out by me under the supervision of Prof. Solomon Tadesse, School of Earth Science, Addis Ababa University, during the year 2019-2020 as part of Masters of Science program in Mining geology. I further declare that this work has not been submitted to any university or institution in the past for the award of any degree or diploma at any other university.

Liyou kifle

Addis Ababa University
Addis Ababa

Date: 07/ 9/ 2020
Signature: ---

Acknowledgement

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Abstract

This study is designed to evaluate and calculate a possible gold loss by metallurgic operation of meli gold plant which is located in northwestern Tigray region, northern Ethiopia. To understand the problem systematic samples has been carried out from tailing dam. Geochemical analysis in support with secondary data review has been done.

From geochemical data analysis of samples, concentration of gold in the tailing dam was found to be significant amount. The gold grade in the tailing dam falls within the range of 1.14 to 0.34g/t with an average grade of 0.61g/t. This indicates loss of gold in the tailing dam. As evidenced from the geochemical results, there exists a clear gold grade decrement as it moves from outlet to inlet.

One of the possible cause for gold loss to the tailing dam is the presence of base metals (Cu, Ni, Zn..). They affect the rate of gold dissolution because they consume both cyanide and the dissolved oxygen and forming stable complexes with cyanide and retarding gold cyanidation. In addition to these the lime proportion and pH regulation problems affect the gold recovery and leads to loss of gold. For mitigating gold loss for ore containing soluble copper, multi stage leaching, Ammonia-cyanide mixtures, treating by non-cyanide leaches such as thiorea leaching can be employed. Moreover, EMD needs to use 60-75% of particle size percentage range. In addition to these proper maintenance of the lime feeder should have to be done.

Acronyms

AAS	Atomic Absorption Spectrometry
ANS	Arabian Nubian Shield
CIL	Carbon-in-leach
CIP	Carbon-in-pulp
CIS	Carbon-in-solution
CRM	Certified reference material
DUB	Duplicate
EMD	Ezana Mining Development PLC
EMRDC	Ethiopian Mineral Resources Development Corporation
FMV	Felsic metavocanic
IMV	Intermediate Metavolcanics
IMVC	Intermediate Metavolcaniclastic
MB	Mozambique Belt
MMV	Mafic Metavolcanics
PPM	Parts per million
RIP	Resin-in-pulp
SIP	Solvent-in-pulp
VA	Volcanic Arc
VMS	Volcanogenic Massive Sulfide

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CHAPTER ONE

INTRODUCTION

1.1 Background and Justification

The accelerating growth of the world's population combined with an improving standard of living throughout the world greatly increases demand for mineral products of all types. The everlasting imbalance between the limited available resources and the unlimited needs of the society calls for due planning in explorations, exploitations and beneficiation of these resources. It is known that mineral resources have a great importance in the economy development of a country. However success in mining business does not end up with the discovery of mineral of economic deposit alone. The development, exploitation and beneficiation shall be handled in a very sophisticated and controllable way so that optimum recovery of the sought resource can be achieved. Nowadays developing countries like Ethiopia are taking initiatives by participating foreign investment, and collaborations to exploit the mineral resources.

Ethiopia is gifted with substantial amounts of metallic and non-metallic mineral resources and has a long history of mining (Assefa, 1985; Gebresilassie 2009; Tadesse et al., 2009). Mining for gold in the southern region of Ethiopia dates back to mid1930's. Since then nearly 80 tons (EMRDC, 1985) of gold has been produced from placers of the Adola area alone and nearly 35 tons of gold from the Legadembi primary gold deposit (Midroc Legadembi) between 1991 and the end of 2007.

Ezana Mining Development Plc (EMD) (2008) is a pioneer exploration company which has identified and reported presence of auriferous poly-metallic VMS deposits near Rahwa/Meli in Adi Ekele belt and Terer in northwestern Tigray (Samuel et al., 2015). These deposits are identified mainly based on the cap rock gossan developed on these sulfide deposits. Among these occurrences, the focus has been the Meli deposit which has about 30m thick gossan and is auriferous (having about 2t gold) (Bheemalingeswara and Atakilty, 2012).

In order to obtain the metals and other minerals needed for industrial processes, fertilizers, homes, cars, and other consumer products, large quantities of ores are mined, crushed, pulverized, and processed. Since mining industries often use fine grind ore to release metals, they produce enormous quantities of powdered rock materials as west to the tailing dam. Nowadays, due to increasing demand, it has become economical to mine lower-grade deposits by utilizing advanced mining equipment. This has greatly increased the habit of reprocessing tailings and other wastes generated by individual mining projects and by the mining industry as a whole.

Ezana Mining Development Plc (EMD) mine gold by using CIL processing method. Though various works have been done by EMD in the gossan of the study area, still there is some gap in tailing disposal point of view and the grade of gold in the tailing dam is not systemically studied. Therefore, this research is premeditated to study the detailed geochemical framework of tailing dam.

1.2 Description of the study area

1.2.1 Location

The study area is located 61km southeast of Shire Indasillasie and about 371kms northwest of Mekelle, capital city of the regional state of Tigray, northern Ethiopia (Fig.1.1). Geographically, it is bounded between 1540000m to 1545000m N and 390000m to 398000m E in UTM.

1.2.2 Accessibility

The study area is accessed via the main asphalt road from, Mekelle to Shire Indasillassie and Endaba Guna. From Endaba Guna to the study area can be reached through gravel road. It can also be accessed via airplane from Addis Ababa to Shire Indasillassie and from Shire Indasillassie to Endaba Guna by asphalt road. (Fig. 1.1)

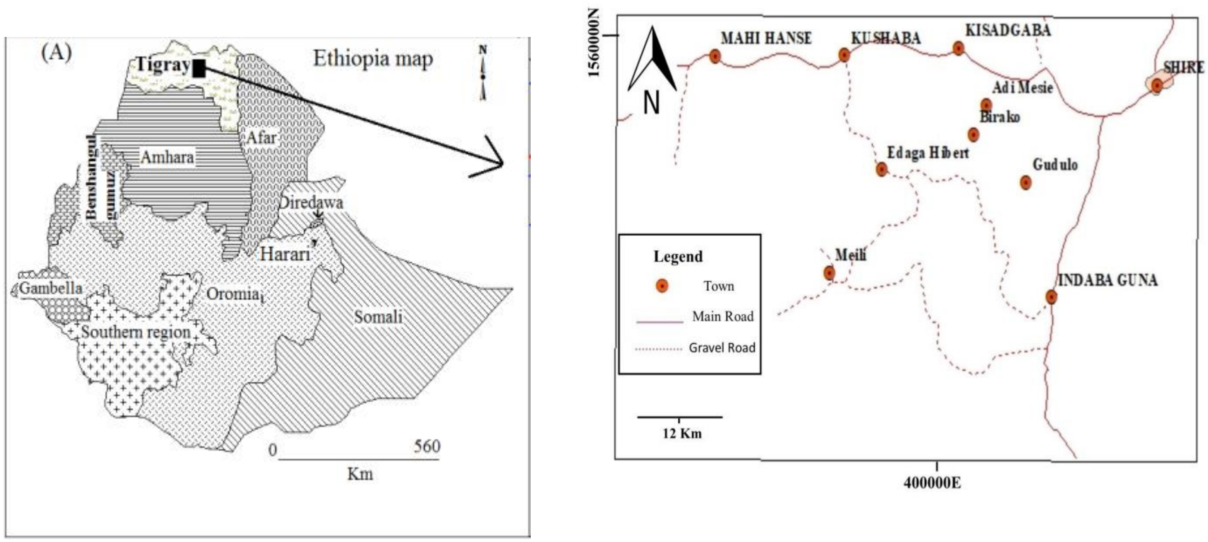

Figure 1. 1 Map showing A) Tigray region, and B) *Location and accessibility of Meli area (Mickiale and Bheemalingeswara, 2017)*

1.2.3 Physiography and Drainage

1.2.3.1 Physiography

The study area is part of the northwestern Ethiopian lowland, characterized by rugged to flat topography. Precambrian metamorphic and intrusive rocks are dominant at rugged mountains of the northern, western and southeastern parts of the area. The southern and eastern parts of the area show gentle, undulating and dissected topography. The area generally lies at an average elevation of 1080m and it reaches up to 1460m above mean sea level in the Shire plateau (Fig.1.2).

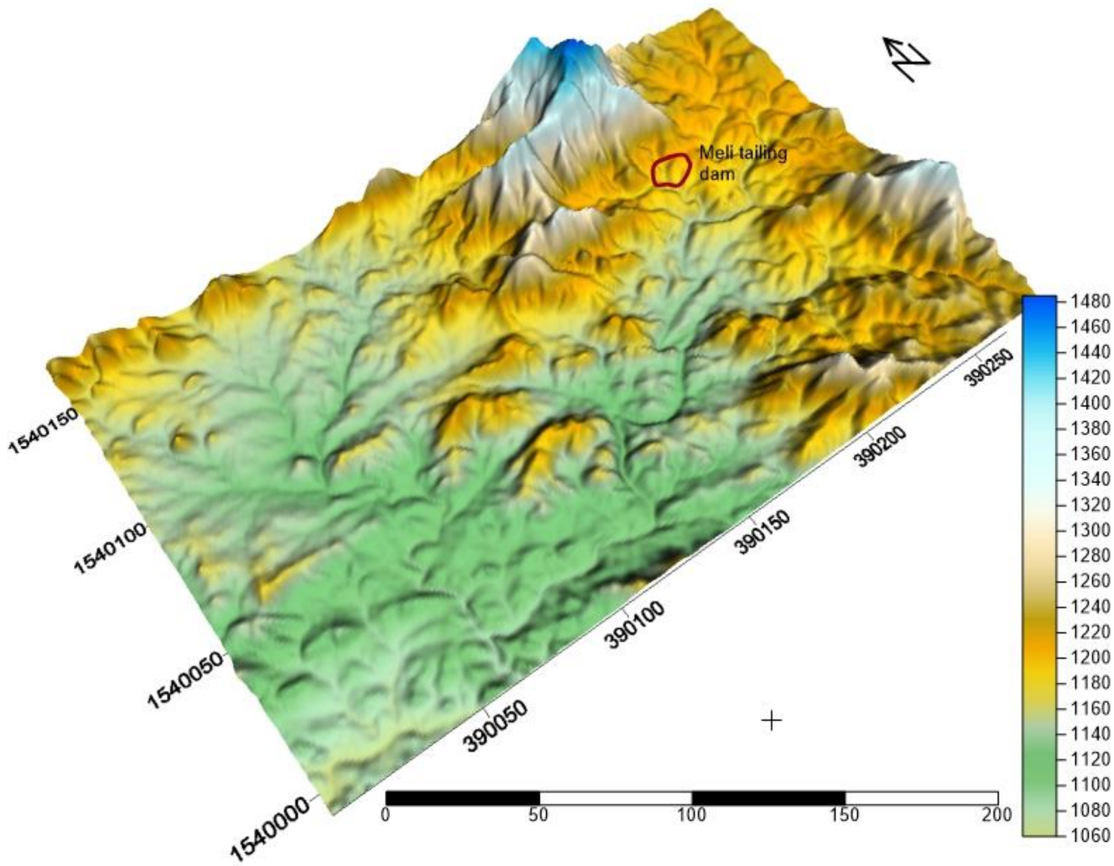

Figure 1. 2 *Digital elevation model of the Meli area (modified after Samuel et.al., 2015)*

1.2.3.2 Drainage

The drainage of the study area is mainly characterized by dendritic pattern. The 1st and 2nd order streams of the area flows to Tinishu Lehote and Tiliku Lehote rivers, in which both in turn merges to form Lehote river. Lehote River drains towards southwest, into the Tekeze River. The streams and rivers of the area are dry and few are intermittent. Broadly speaking radial and concentric drainage patterns are manifested in the dome and subdued granitic outcrop, whereas, dendritic drainage pattern is dominant in the flat lying topographic areas.

1.2.4 Climate and Vegetation

The area is characterized by semi-arid climatic condition, which can be called as hot tropical climate (Atakilt, 2009). From June to mid-September are the main rainy season and an intermittent rain during March. The average annual rain fall is 600 to 800 mm. In the dry season the temperature of the area reaches up to 40ºC at day time and 18ºC at night; and in the wet season 20ºC during day and 8ºC at night (Atakilt, 2009). Generally, the mean maximum, minimum and annual rain fall is 30.2ºC, 15.3ºC and 734.3 mm respectively.

The vegetation of the area is very scarce; most parts of the area are devoid of any vegetation. The prevailing vegetated areas are covered by thorny bushes, which are characteristics of the semi-arid climate. Along the main drainage system in both sides of Lehote River, scattered trees are commonly observed. Long savanna grass and incense trees are grown, in rainy season.

1.2.5 Human settlement and land use

Generally, the study area is inhabited by sparsely populated farmers. Most of the settlements are concentrated around Meli. Barley, maize and wheat are main crops of harvest of inhabitants of the area to get their income from rain-fed agricultural activities. Livestock breeding is also practiced in conjunction with agriculture. In addition, there are artisanal mining activities which used to generate substantial income for the local community.

1.3 Problem Statement

Tailing dam assessment in gold mining companies is crucial work in order to recover significant amount of gold, which has been rejected from each processing units. Through time these disposed gold concentrations accumulated and would potentially give recoverable gold content. In addition of these lower grade gold from these processing units, other loss of gold by poor machinery efficiency and unfavorable beneficiation technique could lead to accumulation of significant gold in the tailing dam.

Ezana Mining Development Plc (EMD) Company has been mining gold for about three years. However, they did not conduct detail tailing dam grade assessment. Conducting assessment on ore loss and gold grade distribution in tailing dam, before the closure, tells how efficient the machineries and the beneficiation techniques are, and leads to conduct necessary remedial measurement. Therefore, this research study tries to assess loss of gold accumulated in the tailing dam and analyze their grade distribution within the tailing dam and suggest areas of recovery plan if the company plans to reprocess the tailing during stage.

1.4 Objective

1.4.1 General Objective

The general objective of this work is to know and evaluate the loss of gold to the tailing dam of Meli Gold Plant.

1.4.2 Specific Objective

- To calculate the possible loss percentage from Meli gold after mill processing circuit.
- To determine the gold grade in the tailing dam.
- To determine the gold distribution in the tailing dam.
- To investigate the cause of loss of gold and provide viable solution

1.5 Methodology

Different methodologies were employed in this study in order to achieve the above mentioned objectives. The methodology includes literature review, secondary data collection, sampling, geochemical analysis.

1.5.1 Pre-field works

- ❖ Literature review: related information to this research, in published and unpublished literatures from relevant sources, institutes, bureaus, Universities and internet was collected and revised.

1.5.2 Field work and sampling

- ❖ 20 representative samples from the tailing dam are systematically collected to identify the gold grade and to outline its distribution throughout the tailing dam (Fig1.3). These samples are selected to perform AAS (Atomic Absorption Spectrometry) laboratory analysis for analyzing mineral geochemistry. And the AAS laboratory results were collected, analyzed and processed to meet the target of the study (Fig1.4).

Figure 1. 3 *Sampling from tailing dam at Meli gold plant*

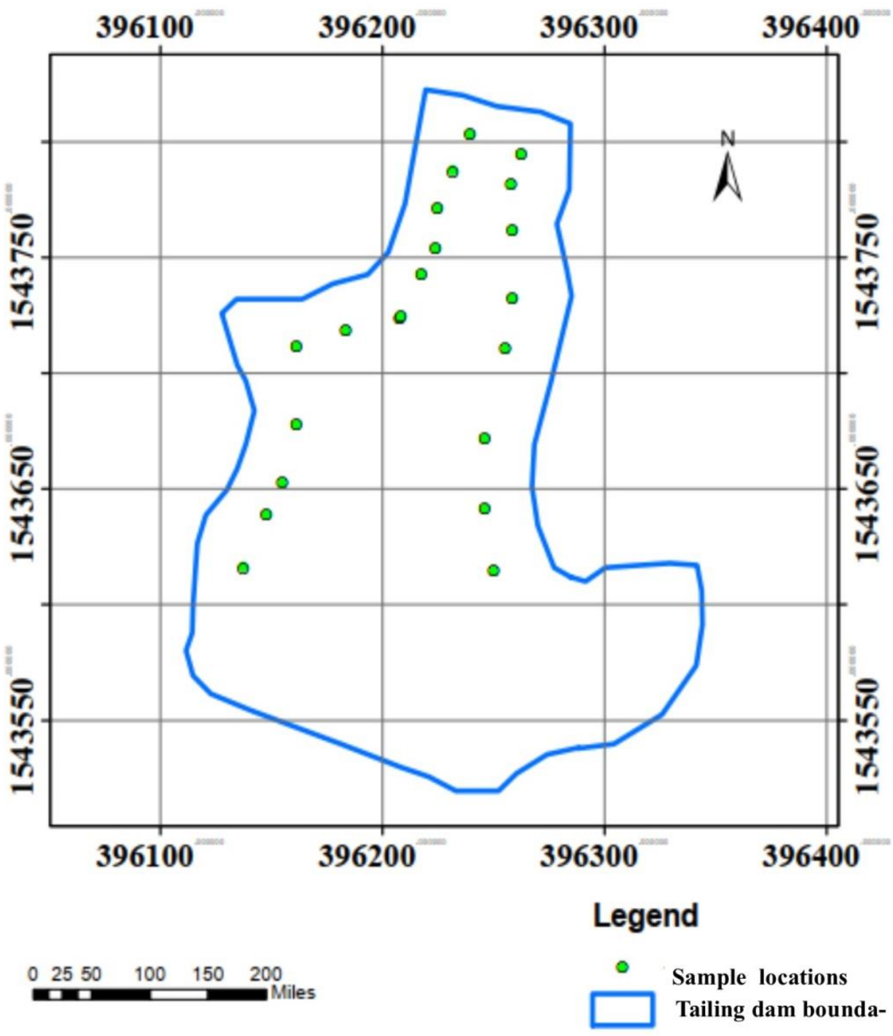

Figure 1. 4 *Sample location map*

- ❖ Software packages were employed during data processing, among these Microsoft Excel 2010, Arc GIS.
- ❖ Representative Samples were powdered at Ezana analytical laboratory and the fine powders were analyzed for major, element analysis using AAS.

1.5.3 Geochemical analysis

This chemical analysis data will be used for the calculation of percentage loss at processing circuit and the amount of loss gold in the tailing dam.

The sample preparation for geochemical analysis is to make the sample in a powder form. To do these three steps will be followed; break the sample in to desirable sizes in confined container not to lose the fines, crush the broken sample in a crusher and finally the crushed sample will be milled in automatic milling machine. If the powder is in preferred size it's good to go to the final stage to pack and label them. The same process will be applied for the other samples. Then finally when all samples are finished the samples will be sent to "Ezana Mining development plc. Ezana Analytical laboratory'' for chemical analysis (AAS).

- Arc GIS software is used to model the gold grade, and distribution of gold in the tailing dam.
    - The sample is crushed until sample size reduced to 2-3mm. The crusher should be clean by using air gun between every sample in order to minimize contamination.
    - Riffle used to take a representative sample as well as homogenized sample so 150-200g homogenized sample were taken. Take a representative sample from the splinted sample and Weigh the sample and pan or container before inserted to the oven. Then dry the sample at the temperature of 105±5ºC for two hours. Then pulverized the dried sample. If the sample fines is >85% it is acceptable and if < 85% it needs further grinding.
    - Quality control sample used to control the accuracy of the laboratory activities by inserting standards in between of the sample. Example:- blank, dup and CRM are the main quality control parameters.

After that weigh 50g of pulverized sample and add 80 ml HCL and 30 ml HNO3. Put the sample on hot plat and digest it for 1 hour at a temperature of 160ºC. Remove the sample from hot plate and cool the digested sample. Then add 2ml of sigma flock solution for settling purpose. Transfer the digested sample to 250ml volumetric flask. Dilute the slurry to 250 mark with deionized water. Then transfers the 250ml diluted solution to 400ml beaker, from the decanted solution measure 40ml to a culture tube and add 15 ml sodium di hydrogen ortho phosphate for iron suppressing and 5ml of DBIK for gold extraction. Shake the solution for 3 minutes. Directly read the gold content by AAS. The Atomic absorption can measure down to parts per billion of a gram (μg dm–3) in a sample. The measurement of metals can be done by using calibration curves generated by a series of patterns, standard additions and the internal standard method.

1.5.4 Data Processing

20 samples were analyzed for gold grade determination and the data obtained was processed through the following methods.

- Samples gold grade versus their distance from outlet to inlet were plotted in diagram in order to understand the trend of gold grades and evaluate their variability in increasing distance

1.6 Significance of the research

The study can greatly contribute in determination of the gold concentration, the possible cause for the loss of gold and the gold distribution in the tailing dam. Hence, this would have a role for the next potential of reprocessing of gold in the area, since the gold price is not constant.

CHAPTER TWO

GEOLOGICAL SETTING

2.1 Introduction

Clear understanding of the very fact that mineralization can be directly related to the plate tectonic processes during a Wilsonian Cycle, is a significant advance in defining the nature and significance of metallogenic zones or provinces. Therefore, proper synthesis of the information related to plate tectonics and related metallogeny strongly enhances studies in the better understanding of the tectonic, stratigraphic and regional metallogeny.

The regional geological setting of the study area is related to the formation of the Arabian Nubian Shield (ANS). Therefore, this chapter deals with discussion of the regional tectono-stratigraphic evolution and associated metallogeny related to ANS and the adjacent areas.

2.2 Geology of the Arabian-Nubian Shield

The ANS is the northern half of the great collision zone called the East African Orogen, which is formed at the end of Neoproterozoic time when east and west Gondwana collided to form the supercontinent Gondwana (Vail, 1985; Stoeser and Camp, 1985). The main units in the Arabian–Nubian Shield (ANS) consists of Neoproterozoic metasedimentary rocks, migmatites, metavolcanic suites, serpentinites, dismembered ophiolite complex, gabbro-diorite-tonalite complexes, unmetamorphosed volcanic and pyroclastic sequences that are extensively intruded by monzonite-granodiorite-granite complexes (Stern, 1994). It extends 2200 kilometers along north-south and 1200 kilometers east-west and is represented by Precambrian crystalline rocks, exposed along the flanks of the Red Sea. Geographically the ANS includes countries Israel, Jordan, Egypt, Saudi-Arabia, Sudan, Eritrea, Ethiopia, Yemen and Somalia from north to south (Fig. 2.1). Abdelsalam and Stern (1996) have divided the deformation belts in the Arabian-Nubian Shield (ANS) into: arc-arc and arc-continent collisions and accretionary structures. Arc - arc deformation belts are characterized by the occurrence of N-S and E-W verging ophiolities and they are oriented east to northeast and north to north northeast in the northern and southern parts of the ANS respectively (800 -700 Ma). Arc-continent deformational belts are related to the collision of East and West Gondwana and define the eastern and western boundaries of the ANS (750-650 Ma). Post accretionary structures are formed from continuous shortening of the ANS

(~650-550 Ma). The Arabian-Nubian shield hosts numerous VMS deposits, orogenic lode gold deposits and placer deposits practiced by artisanal workers (Barrie et al., 2007). The ANS hosts approximately sixty VMS) deposits in several districts found all over the shield most notably in Saudi Arabia, Sudan, Ethiopia and Eritrea (Barrie et al., 2007; Barrie and Hannington, 1999).

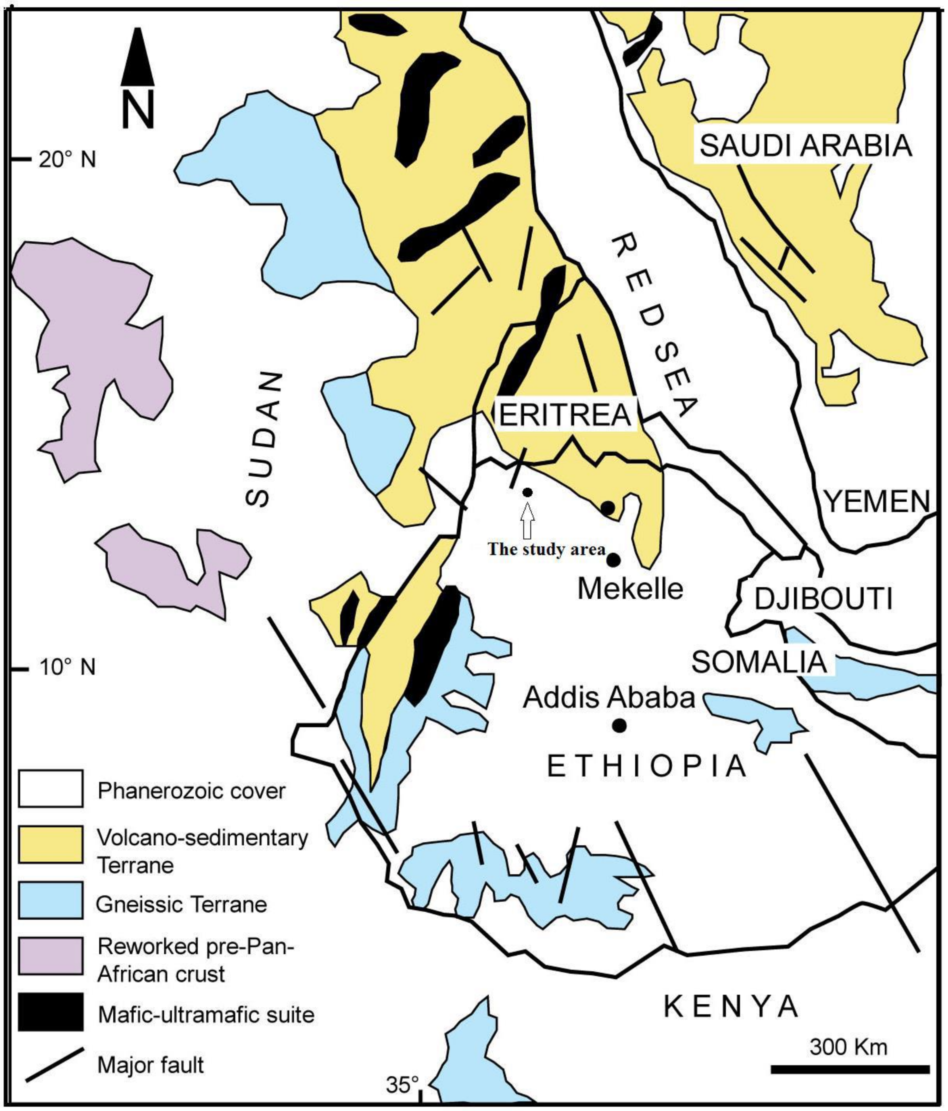

Figure 2. 1 *Distribution of rocks of Arabian-Nubian Sheild, which form the basement rocks Arabian Peninsula, northeast Africa(Egypt and Sudan), and Ethiopia(Samuel,.et al as cited in Gebresilassie, 2009)*

2.3. Geology of the Ethiopian Basement rocks

According to Kazmin et al. (1978) the Precambrian basement rocks of Ethiopia are divided in to Upper, Middle and Lower Complexes on the basis of the grade of metamorphism. The low grade rocks belonging to Upper Complex and high grade to Lower Complex and medium to Middle Complex. They also considered the Upper Complex rocks to be Late Proterozoic in age and Middle and Lower Complexes to be Middle and Early Proterozoic (or Late Archean). But recent studies on the basis of the geochronological and isotopic data (Gerra, 2000, Teklay et al., 1998 and others) have suggested that the Precambrian basement rocks are dominantly Neoproterozoic in age and have experienced different grades of metamorphism. They are exposed in north, south, southwest, western, and eastern parts of the country. In similar way, the Precambrian rocks of Ethiopian have been studied by various scholars; some of them are: [Beyth (1971, 1972); Kazmin (1971, 1975); Kazmin et al., (1978); Garland(1980); Seife Michael (1990); Mengesha et al., (1996); Beraki (1995);Tarekegn (1996);Tarekegn et al., (1997, 1999,2000); Asfawossen (1997); Tadesse (1998); Mulugeta and Barker (1993,1997); Mulugeta (1998) ; Mulugeta et al. (2000a, 2000b, 2006) and Asfawossen et al. (2001, 2003, 2004)]. Accordingly Ethiopian basement is composed of two major blocks: (i) a gneissic and migmatic terrain, which essentially consists of the Lower and Middle Complex (Kazmin 1971, 1975), and correlated with the Mozambique Belt; (ii) a low-grade volcano sedimentary terrain, which comprises all the rocks of the upper complex and is correlated with the ANS. Besides, pre-, syn- and post-tectonic granitoids intruded the Ethiopian basement rocks as shown in fig.2.2 (Asrat et al., 2001).

Recently, the three fold geosynclinal stratigraphy of Kazmin (1971, 1975) has been revised and rejected on the basis of geochronological, thermo chronological, geochemical and lithotectonic data. The Ethiopian basement rocks are regrouped into two major blocks; (1) Volcanic-sedimentary terrain and (2) Gneisic–migmatitic terrain (Asfawossen et al., 2001). Asfawossen et al., (2001) identifies three periods of granitic magmatism from three complexes of Kazmin (1971) both in ANS and MB: 800-885Ma, 700-780Ma and 540-660Ma.

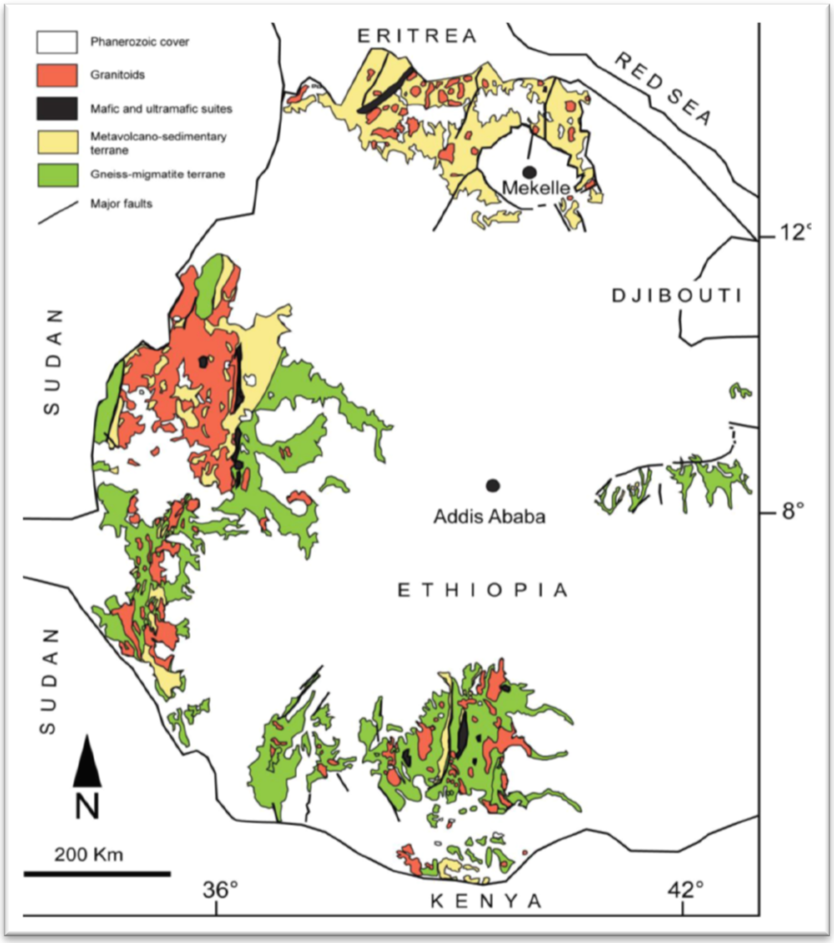

Figure 2. 2 *Distribution of the low grade volcano-sedimentary sequences of the ANS and high-grade gneisses and migmatites of the Mozambique Belt in Ethiopia (Samuel,.et al as cited in Gebresilassie, 2009)*

2.4. Geology of the Tigray Basement rocks

The Precambrian basement rocks of northern Ethiopia are predominantly composed of metavolcano-sedimentary assemblage (Kazmin, 1973; Tadesse, 1996). They form part of the southern part of the Arabian-Nubian Shield (ANS) (Kazmin et al., 1978; Vail, 1983, 1988), which in turn constitute a large segment of juvenile Neoproterozoic crust formed by accretion of oceanic arc terrains (Stoeser and Camp, 1985; Stern, 1994; Genna et al., 2002; Johnson and Woldehaimanot, 2003. In northern Ethiopia, lowgrade, meta-volcanic, meta-volcanoclastic, and meta-sedimentary rocks are intruded by synto late tectonic granitoids where the meta-volcanic and meta-volcanoclastic rocks together forming the largest unit (Kazmin et al., 1978). The basement rocks of central and eastern Tigray has been studied by different researcher [Levitte (1970), Beyth (1971, 1972); Garland (1980); Tadesse (1998); Tarekegn (1996); Tarekegn

Tadesse et al., (1997,1999); Mulugeta (1998) ; Mulugeta et al. (2000a, 2000b, 2006); Asfawossen et al., (1997, 2003, 2004); Kibret et al., (2005) and Avigad et al.(2007)].

Tarekegn et al., (1999) studied the geochemistry of metavolcanic rocks from Axum area; the geochemical result indicated that accreted intra-oceanic arc sequences with varied lithological characteristics. In similar way, the geochemistry of metavolcanic rocks from Mai Kenetal-Negash and Werii area display depleted magma source and volcanic arc setting (Mulugeta et al., 2000; Kibret et al., 2005).

Beyth (1972) divided the basement rocks of Northern Ethiopia into Tsaliet and Tembien Groups based on stratigraphic relationships. Tsaliet Group is mainly composed of metavolcanic/volcanoclastic, metagreywacke, metacarbonates, slate, calcareous siltstone, intermediate to acidic welded tuffs, and lapilli tuff (Beyth 1972; Beyth et al., 2003; Tarekegn et al., 1999; Mulugeta et al., 2006). Its thickness is estimated more than 1500m. Though the age of Tsaliet Group is not well established but based on lithological similarities, these rocks are correlated with the metavolcanic rocks of the Ghedem terrain in Eritrea (Alene et al., 2006), which were dated ~854±3Ma using the Pb/Pb single zircon evaporation method (Teklay, 1997). Tambien Groups are mainly composed of slate, phyllite, graphitic schist and metalimestone. The Tambien Group was deposited in a shallow marine environment during a period of regional arc-magmatic lull (Avigad et al., 2007). This is mainly exposed in a series of synclinal inliers surrounded by the Tsaliet Group rocks. Furthermore, the lithostratigraphic sections of the Tambien Groups from the oldest to the youngest describe as follows; Werii Formation, Assem Formation, Tsedia Formation, Mai kenetal Formation, Amota Formation, Didikama Formation, Matheos Formation, Marian Bohkakho Formation, and Negash Diamictite (Mulugeta et al., 2006; Swanson-Hysell et al., 2015).

Swanson-Hysell et al., (2015) have been reported that the Tambien Group consists of approximately 5-km thick mixed carbonate-siliciclastic succession. These sediments are accumulated above volcanics and volcaniclastics of the Tsaliet Group. In similar way, the Tambien Group consists of a number of inliers each containing 2-3 km thickness of interbedded carbonates and clastic sediments (Mulugeta et al., 2006). The contact between the Tsaliet and Tambien Groups is transitional, and input of volcanic ash into the basin continued during the deposition of the Tambien Group (Mulugeta et al., 2006; Swanson-Hysell et al., 2015).

The Northern domain extending northwards in Eritrea, composed of several meta-volcanosedimentary belts and sub-belts, bounded by mafic–ultramafic rocks, host gold and base metal occurrences. A good example for this is the Adi Zeresenay gold (Solomon et.al, 2003). Beyond that the geology of the Adyabo area which is neighboring to Terakimti is analogous to the host rocks for gold-rich VMS deposits located at Bisha and the Hassai districts, which occur within the Adi Nebried back arc basin (Archibald et.al, 2015).

According to Tarekegn Tadesse (1997) the northern part of the Ethiopian geological domain particularly the Axum area is divided in to six tectono-stratigraphic, structural bounded geological blocks. These are the Mai Kenetal block, the Adwa block, the Chila block, the Adi Nebrid block, the Adi Hageray block, and the Shiraro block.

2.5 GEOLOGY OF THE STUDY AREA

Geology of the study area forms part of the low-grade metavolcano-sedimentary succession of the Neoproterozoic stratigraphy of northern Ethiopia. The Precambrian rocks of the Meli(Rahwa) area comprises of felsic to mafic metavolcanics, metavolcaniclastic, and detrital and chemically precipitated metasedimentary rocks and chaotically intermixed linear belts of mafic-ultramafic rocks. The whole sequence is intruded by the syn and post -tectonic granitoids. The rocks have undergone intense tectonic activity and associated poly-phase deformations. This has resulted in the dismembering of the rocks which .hampered proper establishment of the stratigraphic sequence. Spatial distribution and structural trends the metavolcanics and metasediments suggest that they are the southwestern continuity of the Adi-Nebrid and Chila blocks. These are truncated by mafic-ultramafic and granitoids in the southeastern part of the study area and generally shows tectonic contact with the adjacent rocks.

Mafic-ultramafic rock outcrops are encountered in the southeastern and central part of the Meli area. The southeastern part is represented by ultramafic core surrounded by mafic rocks while in the central part; the ultramafic rocks have engulfed the intermediate metavolcanics. The ultramafic units are represented by talc-chlorite and talc schists with relicts of serpentinite pods. Intense quartz vein development is observed in the northwestern, central and northeastern parts of the area.

2.5.1 Mafic metavolcanic rock

Mafic metavolcanic rock (MMV) is found having pale greenish tint, dark to dark grayish color, medium grained and mafic in composition. It is the dominant rock unit of the study area and it is mainly exposed in northwestern, southwestern, and southern parts of the study area (Fig 2.3). MMV is found in contact largely with felsic and intermediate metavolcanics, phyllite and granite outcrops. The foliation orientation which is developed in this lithology is SW-NE to E-W (250°-275°) and it dips moderate towards southeast and sub-vertical to south.

Figure 2. 3 *(b) Outcrops of the rock unit (Samuel, A. 2012).*

3.5.2.2 Intermediate metavolcanics rock unit (IMV)

IMV is exposed on the southern, eastern and central parts of the study area in contact with felsic metavolcanic unit, phyllite and intermediate metavolcaniclasts (Fig. 2.4). It was observed having light to dark greenish, gray with pale greenish color and fine grained texture. It is composed of mainly biotite, chlorite, quartz, and some actinolite and plagioclase minerals and it is found slight contact with mafic metavolcanics and granite units. IMV is observed to have massive to strongly foliated fabric. At places quartz veins and veinlets are found cutting this unit in different directions. The foliation orientation is 260° to 280°.

Figure 2. 4 *(b) Outcrop of the rock unit (Samuel, A. 2012).*

3.5.2.3 Intermediate metavolcaniclastic unit (IMVC)

Intermediate metavolcaniclastic unit is exposed on the central part of the study area extending towards northeast (Fig.2.5). It shows light to dark greenish gray color. Texturally, it is fine to medium grained, and it is moderately foliated. The rock is commonly comprised of silicified, tuffaceous and epidotized rock fragments within a matrix of chlorite and amphibole. The size of these clasts ranges from few millimeters to about 20 centimeters and some big clasts are observed with quartz grains within them. The clasts have sub angular, elliptical to sub-rounded shapes and they show alignment due to stretching along down dip. This rock unit is foliated with orientation ranging 245°-270° dipping 35°-65°due SE and it occupies lower topography having a contact with the intermediate metavolcanic rock unit.

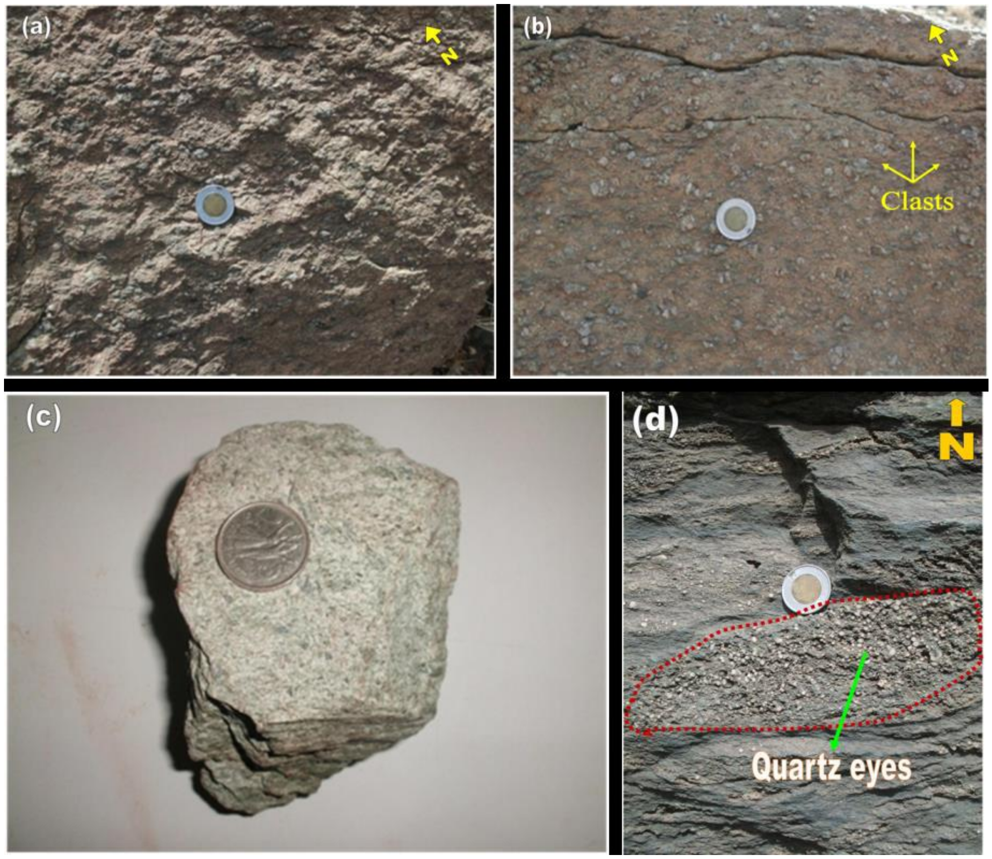

Figure 2. 5 *Intermediate metavolcaniclastic rock unit (a) and (b) Exposures of the unit (c) Sample of the rock from its outcrop, (d) Quartz grains within a big clast in the rock unit. (Samuel, A. 2012)*

3.5.2.4 Felsic metavolcanic rock (FMV)

Felsic metavolcanic rock is mainly exposed in northern part of the study area (Fig.2.6). It has white and pale-gray color and it has fine grained glassy texture. In some parts it shows variegated color of reddish brown, whitish gray and dark brown, and it also shows banded color of gray and light gray. The foliation developed in this unit is oriented SW-NE to E-W dipping sub vertically towards south. There are some disturbances measured dipping 25°-45° due north. At some parts of its outcrop, this rock exhibit variably distributed grains of quartz and it is observed transverse by quartz veins having different orientation and thickness ranging from 1cm to 25cm.

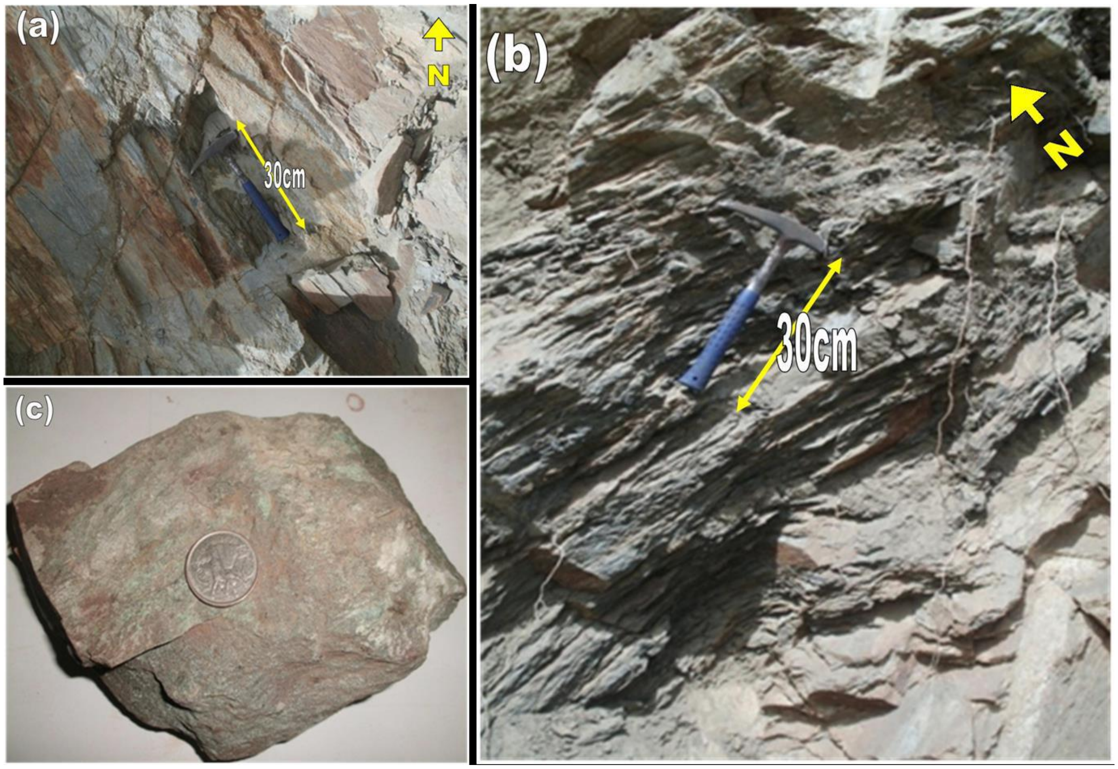

Figure 2. 6 *(a) and (b) Outcrops felsic metavolcanics rock, (c) Sample brought from the exposure (Samuel, A. 2012)*

3.5.2.5 Phyllite

This unit is a dominant outcrop in the central part of the study area extending E-W (Fig.2.7). It has contact with mafic metavolcanic and intermediate metavolcanic rocks. This rock is light gray and pale pinkish in color and fine-grained in texture. It is fissile and graphitic in nature. Generally this unit has E-W foliation orientation. It is cut by aplitic dikes, quartz veins and at places shows bedding structure.

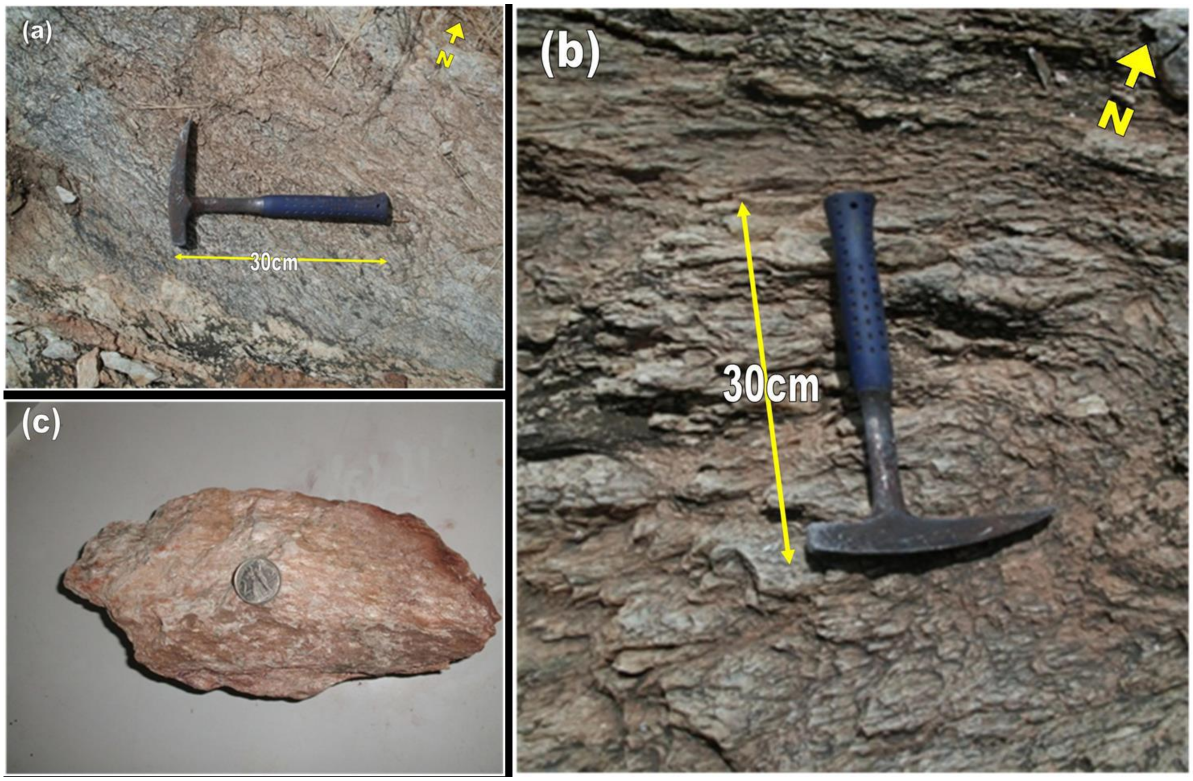

Figure 2. 7 *Metasediment rock (a) and (b) exposures of phyllite rock unit, (c) sample collected from the rock. (Samuel, A. 2012)*

3.5.2.6 Granite

Granite of the study area is syn or post tectonic (EMD, 2009). This granitic intrusion is light gray to pink colored with black spot, medium- to coarse-grained, massive to weakly weathered unit. It is exposed in south western and eastern parts of the study area (Fig.2.8). It is composed of quartz, feldspar, and minor biotite and muscovite. Besides, this unit exhibit xenoliths of dark color within it. The xenoliths are angular to sub rounded and are up to 20cm in size.

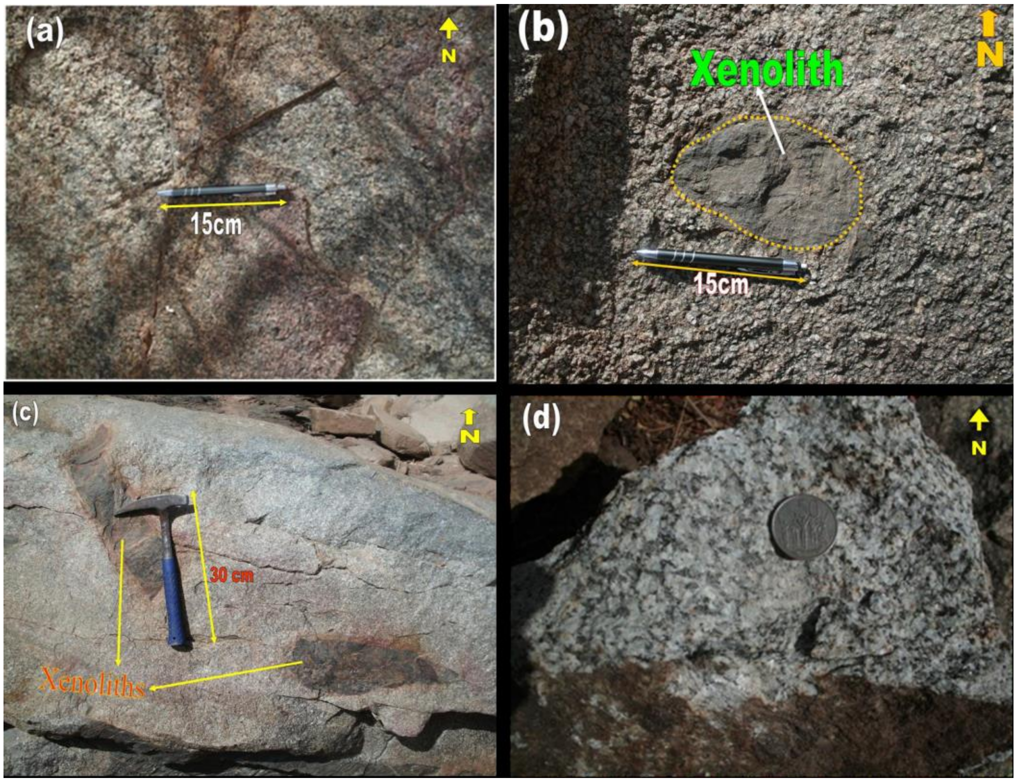

Figure 2. 8 *Granite rock unit (a), (b), and (c) outcrops of the unit, the dashed line in b indicating the Xenoliths in the granite (d) Sample collected from the exposure (Samuel, A. 2012)*

3.5.2.7 Aplitic Dike

In the study area swarms of aplitic dike that ranges from 20cm to 1m in thickness are observed cutting phyllite and intermediate metavolcanic rock units (Fig.2.9). The aplitic dikes are light pinkish and pinkish gray colored, fine- to medium-grained composed of mainly quartz and feldspar. The orientation of the dikes is almost similar with the foliation orientation of the rocks which are oriented SW to NE. Quartz veins are also seen invading the dikes in the some parts.

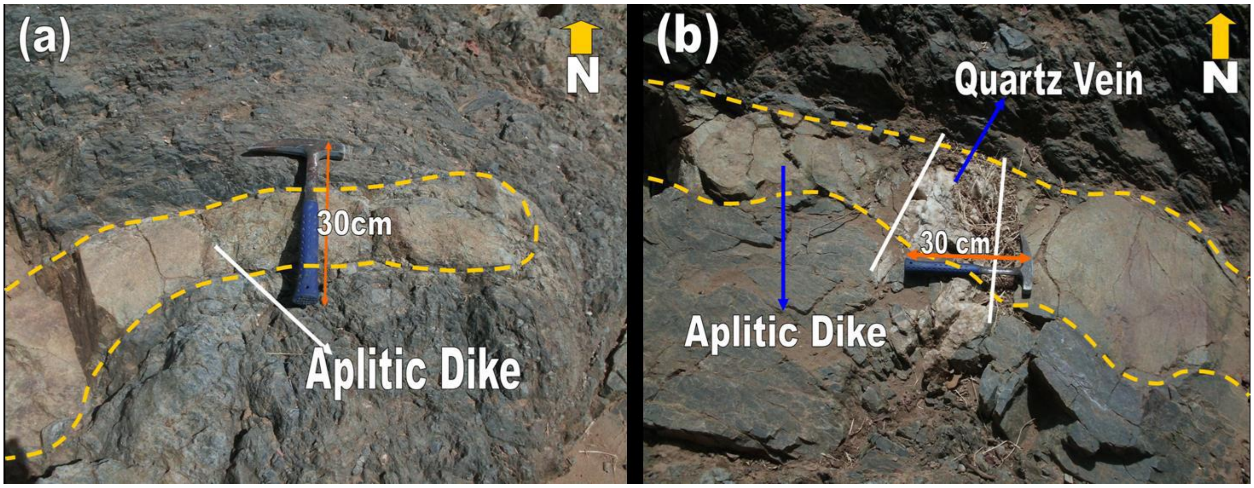

Figure 2. 9 *(a) Aplitic dike cutting intermediate metavolcanics unit (b) Quartz vein cutting an aplitic dike. (Samuel, A. 2012)*

3.5.2.8 Quartz Vein

Quartz veins are observed throughout the study area. These veins are intensely invaded the mafic metavolcanics, intermediate metavolcanics, intermediate metavolcanoclasts, felsic metavolcanics and phyllite rock units (Fig.2.10). Thickness and orientation of the veins varies but generally the thickness ranges from few cm to 50cm. The overall orientation of these veins and veinlets lies in NNW, NW and E-W. Most of the quartz veins are white and glassy but some are smoky. There are also some milky type quartz veins and veinlets having minor openings and some are auriferous. Most of the veins are highly brecciated; fragments of quartz cover a wide area as quartz float.

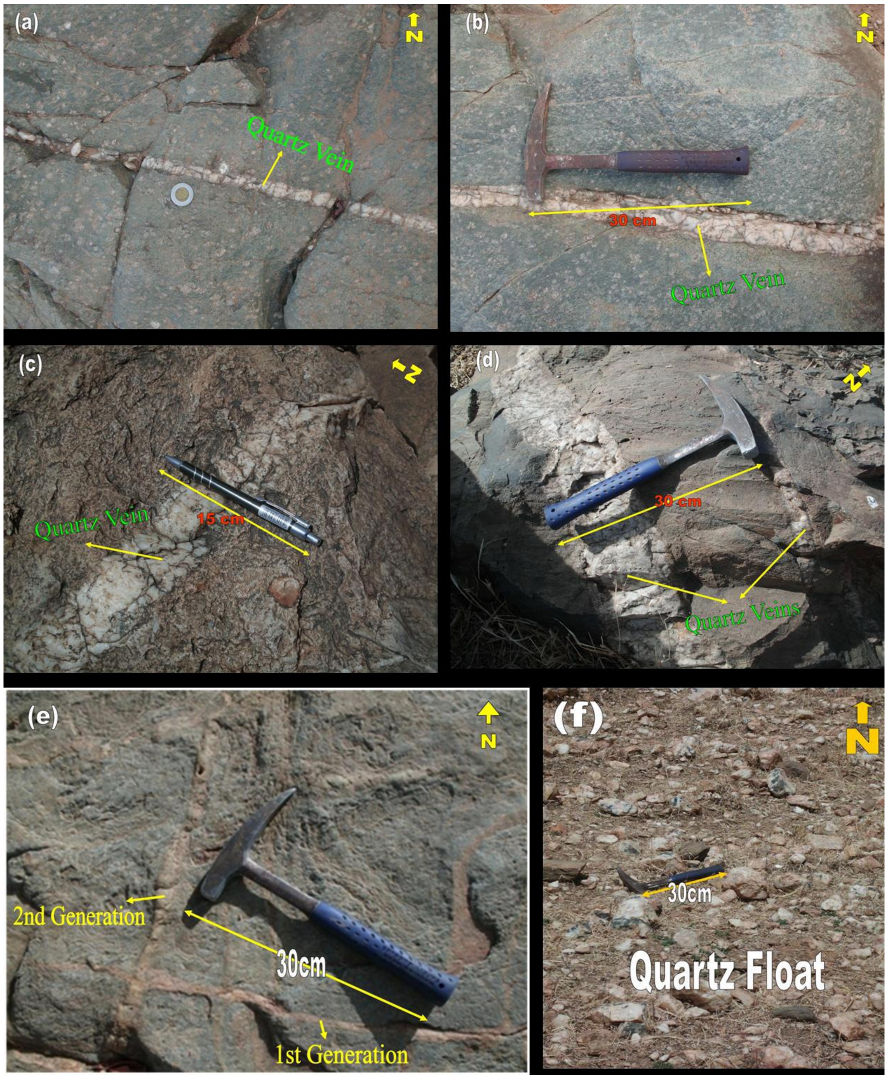

Figure 2. 10 *Quartz veins (a), (b), (c) and (d) quartz veins cutting intermediate metavolcanoclasts, intermediate metavolcanics, phyllite, and felsic metavolcanics respectively (e) 1st and 2nd generation quartz veins in felsic metavolcanics. (Samuel, A. 2012)*

3.5.2.9 Gossan

This auriferous gossan is underlain by an acidified zone and hosted in dacitic tuff (Atakilt, 2009). It is mainly represented by limonite, goethite and hematite, and its outcrops exhibit different colors which probably could be due to factors like mineralogy, grain size, aggregation and moisture content. The exposure of this unit generally forms flat topography with small undulated hills. It displays variegated colors of reddish brown, dark red, yellowish and light reddish (Fig.2.11 and 2.12). The geologic map of the area is presented below (Fig.2.13).

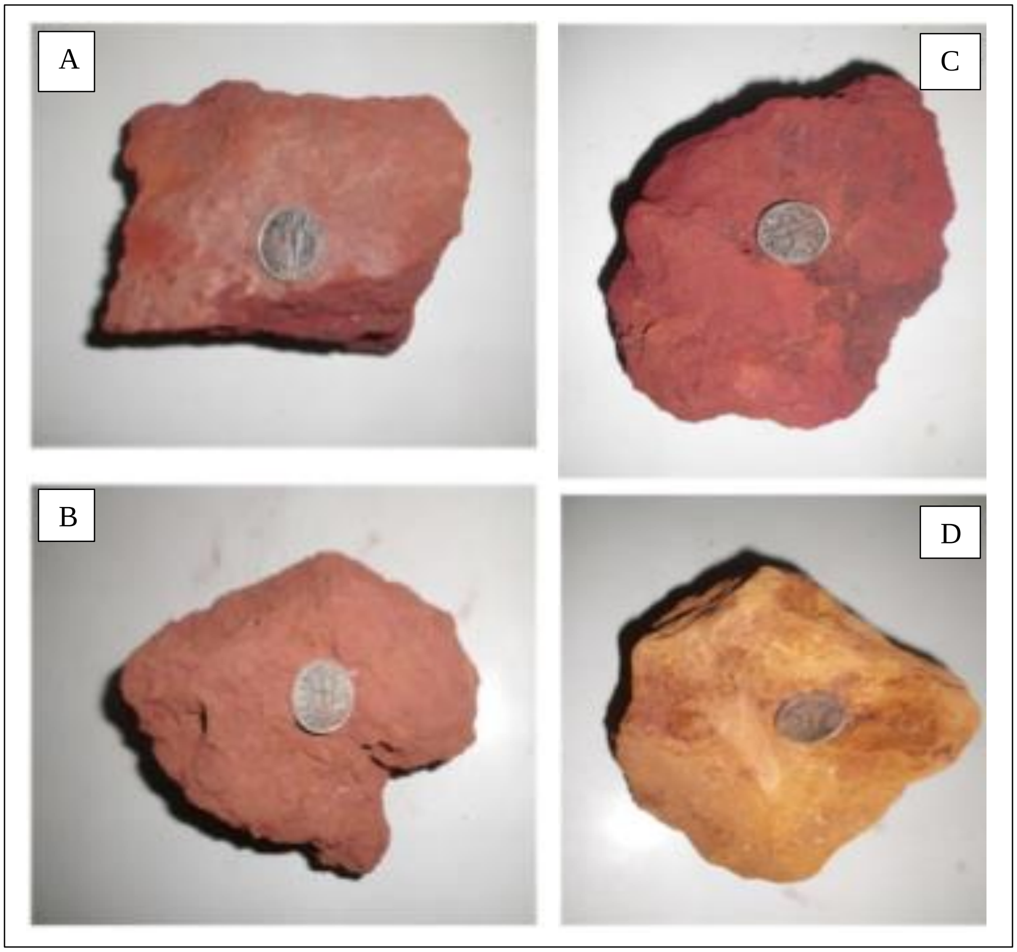

Figure 2. 11 *Gossan Samples collected exhibiting variegated colors (A) light reddish (B) reddish brown (C) dark reddish, and (D) yellowish gossan (Samuel, A. 2012)*

Figure 2. 12 *exposure of gossan at the study area*

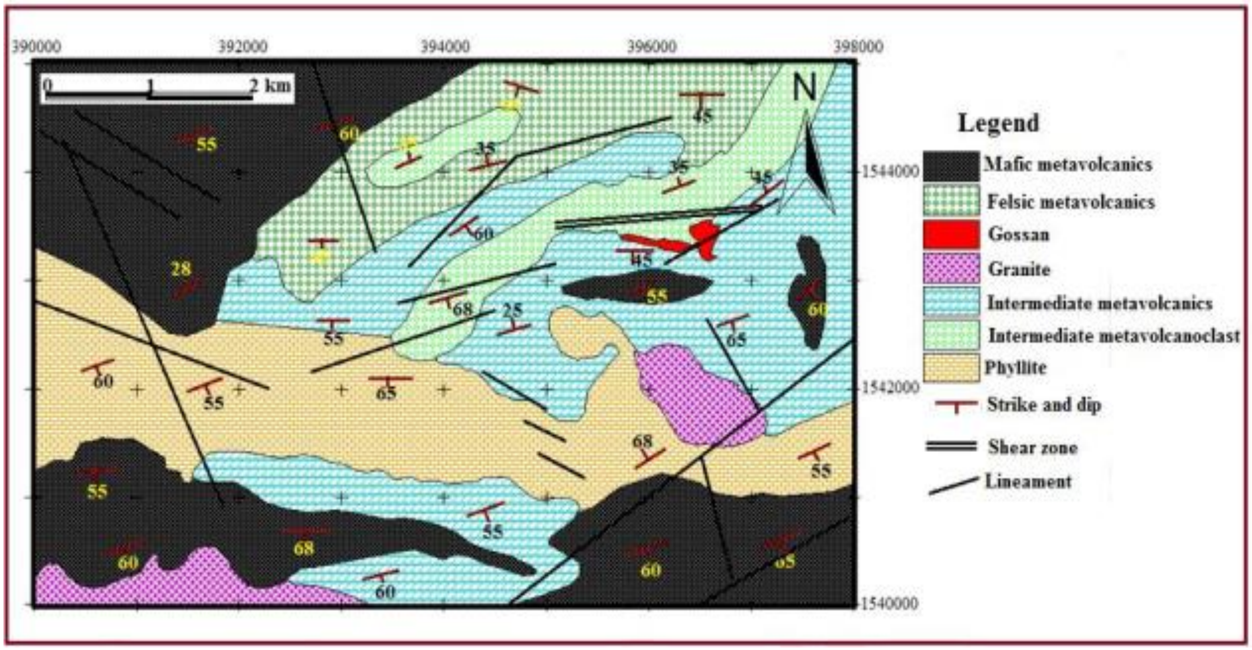

Fig.2.13 Geological Map of Meli area (Samuel, A. 2012).

CHAPTER THREE

ORE EXTRACTION & GOLD RECOVERY: LITERATURE REVIEW

3.1 Definition

A dam is a barrier constructed for the retention of water, water containing any other substance, fluid waste, or tailings.

Tailings are waste material. It is a by-product of the ore extraction process. The ore bearing rock is finely ground to either liberate the desired mineral or allow it to be chemically processed. Most extractive processes in mining are wet processes, and the tailings discharged at the end of the process are therefore in slurry form (Heidi, 2007). Disposal facilities or impoundments are used for the disposal of the tailings slurry. These impoundments are known as tailings dams (Henderson, 1998).

3.2 Fundamental steps of ore extraction

There are fundamental steps in the processes of ore extraction, which are common to many ore and also used to understand tailings and tailings dam (Fig3.2).

3.2.1Crushing and grinding

Crushing is performed in stages with the aim of reducing rock fragments from mine-run size to a size that can be accepted as feed to grinding equipment. Grinding further reduces size of the rock fragments produced by crushing (Vick, 1990) (Fig 3.1).

A short head cone crusher in closed circuit, by means of a screen (preferable double-decked), should deliver the crushed material required for grinding by steel–media mill (about 80 % -1/2).

A closed crushing circuit delivers the proper size of crusher ore to the grinding circuit, optimizes the energy consumption, and may alleviate wet–winter problems (Nordberg, 1989).

The advantages and disadvantages of the various comminution circuits are many, but the final analysis involves practical considerations such as the availability of good quality grinding media, trained people to operate a sophisticated plant , etc, as well as economics (Barratt and Sochlcky 1982).

The mineralogy of the gold ore dictates the required fineness of grinding for adequate gold liberation and the economically optimum extraction recovery (Yannopouls et al., 1990). Each ore has its own gold-liberation characteristic. Therefore, tests and feasibility studies have to define the optimum comminution circuit that can deliver adequate the mill feed throughout the year under the specific climatic conditions of the mill site.

The three basic types of comminution circuits are:-

i- Conventional three-stage closed-circuit crushing and rod-mill / ball-mil grinding.

ii- Primary crushing and autogenously grinding.

iii- Primary crushing with autogenously mill followed by fine grinding in a ball mill.

It is essential that mill feed be kept as close as possible to that called for in the original design specification of the mill and concentrator. Regular reconciliations will be required between the estimated stope grades, the grades indicated from stope/truck sampling and those reported by the mill.

3.2.2 Concentration

The purpose of concentration is to separate those particles with high values (concentrate) from those with lower values (tailings). Methods for concentration vary according to one type, but three general classes are in use: gravity separation, magnetic separation, and froth flotation.

3.2.3 Leaching or heating: Optional processes following or supplanting concentration may include leaching or heating.

3.2.4 Dewatering

The final stage in the process is recovering excess water from the tailings in preparation for pumping the tailings slurry to the disposal impoundment. The term dewatering in a mill process context refers not to complete drying of the tailings, but rather to the process of removing some of the water in the tailings-water slurry following concentration. The most common means of dewatering is by thickeners (Vick, 1990).

Figure 3. 1 *Milling of the ore at Meli Gold plant*

Tailing production

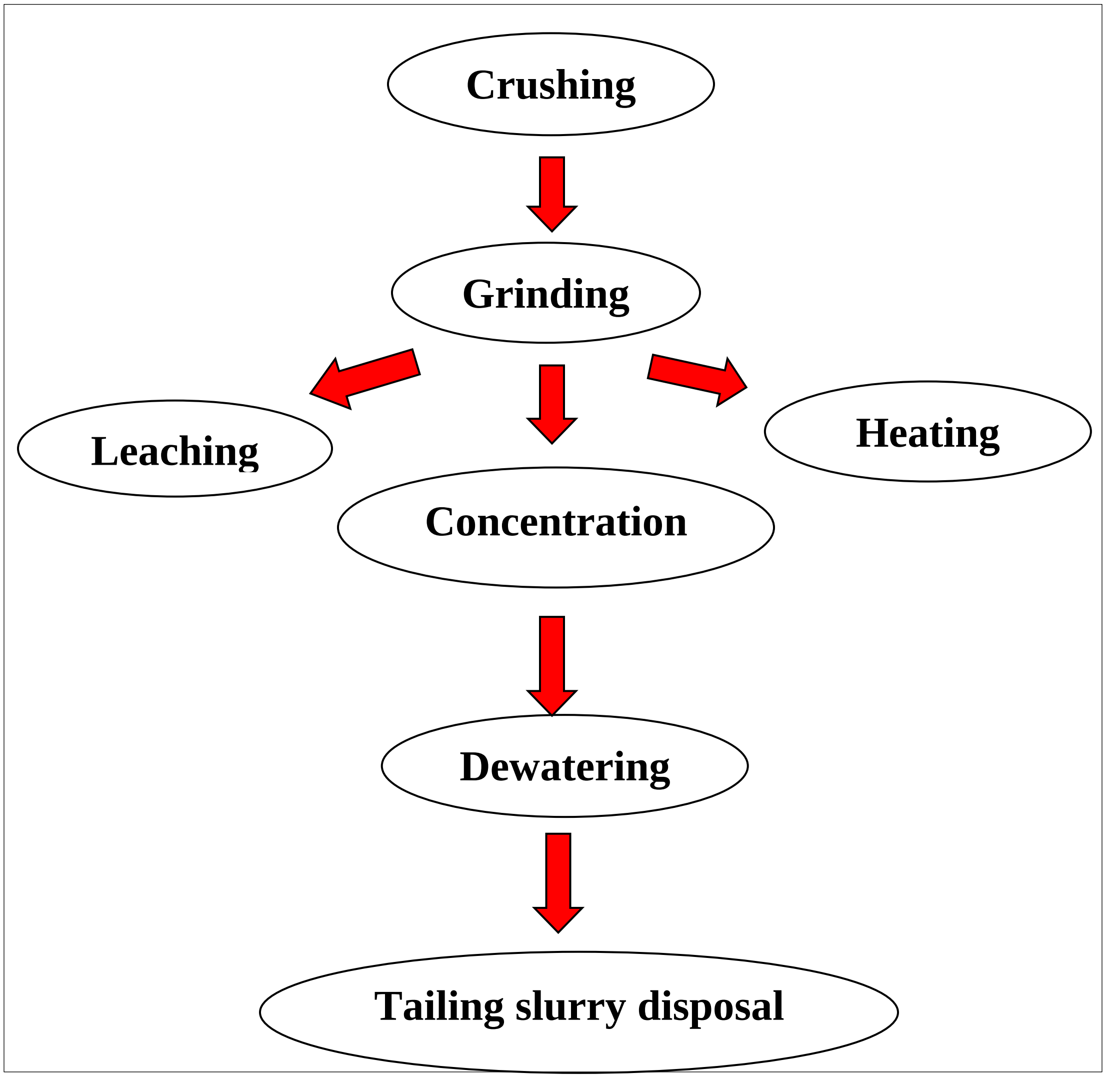

Figure 3. 2 *fundamental steps in the processes of ore extraction*

The particle size of tailings may vary between medium sand to silt or clay size. Tailings slurries are normally transported to the disposal area by a pipeline. The slurries can be distributed by different techniques such as sub aerial discharge (with spigots), sub aqueous discharge (slurry is injected below the water surface) and thickened discharge (slurry with low water content) (Jewell, 1998).

There are basic four types of tailings transport and disposal operations. These are conventional, thickened, paste and dry filter tailings.

The conventional tailings slurry is generally characterized by turbulent flow and segregation of coarse and fine tailings during transport and at the final disposal points. The conventional tailings slurry could be transported long distances away from the mill by gravity flow in open launder chutes or enclosed pipelines to the impoundment (Daniel et al., 2011). Energy dissipation drop boxes or centrifugal pumping were added as needed for discharge from single or multiple spigot points around the tailings impoundment. The settled tailings density for conventional tailings would be relatively low with the more coarse tailings forming a flat alluvial fan beach surface at each disposal point. The finer tailings slimes with low sand content flowed further into the interior toward the water pool. Rotation of the slurry disposal points from active to inactive areas around the impoundment perimeter allowed thin layers of the low-density beach material to drain and dry for densification and increased strength. Clear water from the water pool surface would be decanted by gravity flow in towers and pipelines extending beneath the tailings dam or by lower risk floating barge pumps in modern times for water return to the plant.

Thickened tailings disposal: Visually the thickened tailings slurry would appear to have a "soupy" non-turbulent discharge flow with less segregation of coarse and fine tailings in the tailings beach surface materials (Daniel et al., 2011). The end result of thickened tailings impoundment disposal is a more uniform tailings gradation from beach to water pool at a higher settled density and whole tailings strength with less water demand at the plant compared to conventional tailings.

Paste tailings disposal: The mud like paste tailings are developed with high wall and high angle thickeners at a slower plant production rate. The paste tailings have high hydraulic friction losses for pumping and require a switch from centrifugal to positive displacement pumping for pipeline transport. The paste tailings settled density is significantly higher. Visually the paste tailings discharge slurry would appear to be a mud flow with a low amount of "bleed" water seeping from the beach slope.

Dry filter tailings disposal: Dry filter tailings refer to dewatered tailings that can no longer be pumped and require mechanical transport by trucks or conveyors for disposal in the tailings impoundment. The dry filter tailings solids to water ratio at 75 to 85 percent typical allows the material to be stacked at a relatively high density and strength compared to the pipeline transported tailing slurry (Daniel et al., 2011). A few gold and silver operations have used dry stack tailings since the 1980's and a secondary benefit of the dry filter water recovery at the plant includes capturing the majority of the cyanide water for reuse in operations in addition to lower cyanide destruction costs.

The coarse particles of tailings (i.e. sand) settle close to the point of discharge, and the fine particles (i.e. silt and clay, generally termed as slimes) run down the beach into the pond and settle there (Vick, 1990). The particle size of tailings varies from medium sand to silt or clay size. During spigotting, the coarse particles (sands) lie close to the embankment, while the fine particles (slimes) move downwards to the impoundment.

Tailings are produced in huge quantities annually, and may contain some toxic chemicals that are harmful to the environment. Therefore, it is necessary to store tailings in an environmentally safe and economical way.

3.3 Gold recovery methods

This section will outline the various mineral processing methods commonly used to recover gold, giving an indication of the size range of material processed and typical gold recovery figures.

Table 3. 1 : different gold recovery methods

Method	Effective size range (μm)	Recovery efficiency
Sluice boxes	2500 to 100	As low as 20% for-<100 μm gold upto 96% for < 1000 μm gold
Jigs	2500 to 75	As low as 50%for 100μm gold upto 98% for 1000 μm gold
Shaking table	3000 to 15	As low as 20% for 20-40 μm gold up to 90% for gold >140 μm
Spirals	3000 to 75	65%-80%

Rotating cones and Bowl concentrator	6000 to 30	Up to 99%
Amalgamation	1500 to 70	As low as 65% for <75 micrometer gold-98% for < 500 µm gold
Cyanidation	Finer than 200	At least 80%-99%

3.3.1 Cyanidation

The ore is comminuted (using grinding machinery), and was then concentrated by froth flotation or by centrifugal (gravity) concentration depending on the mineralogy of the ore. The alkaline ore slurry is combined with solution of sodium cyanide or potassium cyanide. The negatively charged cyanide ions (anions) release the positively charged gold ions (cations) from the ore as a metal complex. The gold oxidizes to from the soluble aurocyanide metal complex Na Au $(CN)_2$ (Jiang et al., 2001).

3.3.1.1 The mechanism of cyanidation

The process of gold dissolution in cyanide (and consequently the extraction of gold from it) involves heterogeneous reaction at the solid-liquid interfaces. Hence, the following sequential steps may be assumed as leading to the dissolution of gold (from its ores) by cyanide:

i- Absorption of oxygen in solution.

ii- Transport of dissolved cyanide and oxygen to the solid-liquid interfacial.

iii-Adsorption at the reactions (CN- and O2) onto the solid surface

iv- Electrochemical reaction

v- Desorption of the soluble gold cyanide and other reaction products from the solid surface

vi- Transport of the desorbed product into the bulk of the solution**.** (Yannopulos, 1991**)**

During cyanidation the gold is leached from a thickened pulp with soluble cyanide salts (sodium cyanide, NaCN or calcium cyanide, CaCN) according to the following equation:

$2Au + 4NaCN + \%02 + H20 \leftrightarrow 2NaAu\,(CN)_2 + 2NaOH$

Other essentials are oxygen and calcium hydroxide, $Ca(OH)_2$, to stabilize the cyanide radical, control alkalinity and maintain a minimum pH of 10 to 11. The pulp is usually prevented from sedimenting by mechanical or compressed-air agitation to facilitate leaching. Depending on the grade of the ore up to 98% gold can be extracted in the leaching process. Cyanide leaching results in a solution containing anionic metal cyanide complexes from which the gold complexes must be recovered. Two major routes for recovery may be followed. The first, zinc precipitation, requires the liquid (gold in solution) to be separated from any insoluble material before recovery; the second, carbon-in-pulp (CIP) process, is able to recover the gold directly from the leached pulp. Separation of liquid and solids is usually done through continuous vacuum filtration followed by filtrate clarification, to remove the remaining fine suspended solids.

3.3.1.2 Zinc Precipitation or Cementation

Vacuum de-aeration is applied to remove dissolved oxygen, since this greatly improves the efficiency and economics of the process. Soluble lead salts (lead nitrate) are then added together with zinc shavings or dust, whilst satisfactory cyanide and alkalinity levels (using calcium oxide, CaO) have to be maintained for efficient operation. Mixing can take place prior to precipitation in a small emulsifier tank. Zinc precipitates the gold due to its high electro-negative charge in relation to gold's electropositive charge, according to the following formula:

$2NaAu(CN)2 + Zn$ ::::: $Na_2Zn(CN)4 + 2Au$

The slime of gold and zinc salts produced in this way is recovered by filtration and then treated with dilute sulphuric acid to remove as much zinc and other impurities as possible. The resulting precipitate is then calcined to dry the pulp and to oxidise impurities such as lead and zinc prior to smelting with a flux of borax and silica. Smelting oxidises the base metals in the gold slime and combines them with silica to form a slag, which separates from the gold and silver due to its lower density. Gold smelts at 1063°C, requiring the ovens to work at between 1200 - 1400°C. The bullion is poured into bars, cleaned, numbered and assayed before dispatch to the Rand Refinery. The slag is also dispatched to the Rand Refinery as a by-product.

A minimum "critical" cyanide concentration is required for gold cementation [reported as 0.002 M (0.1 g/l NaCN) by Nicol et al., (1979) and 0.035M (1.7 g/l NaCN) by Barin et al., (1980)]. The gold concentration has a direct influence on the rate of cementation, which is essentially a first order reaction controlled by the mass transfer of $Au(CN)_2^-$ ions. Although a change in the pH of the solution within the range of 9 to 12 has no appreciable effect on the rate of cementation, higher pH may cause the formation of $Zn(OH)_2$ as an intermediate compound. Precipitation of $Zn(OH)_2$ tends to take place at the surface of the zinc particles and can retard or even stop the zinc cementation. Finkelstein (1972) reported that sulfide, sulfate, thiosulfate, and ferrocyanide anions in the concentration range 10- 3M-10 - 2M may reduce the gold recovery by 1-2% from 10 -3M cyanide solutions. The sulfate ion may precipitate as gypsum on zinc particles and reduce their reactivity. Nicol et al., (1979) found that sulfide ions can passivate the zinc surface even at concentrations as low as 1 x 10-4 M.

3.3.1.3 Carbon-In-Pulp Process

Activated carbon is used to recover gold by adsorption directly from the leachate. The carbon must be coarser than the pulp (usually > 1 mm) so that it can be readily screened from the pulp in the end. Adsorption occurs as the pulp flows continuously by gravity through a series of tanks and inter-stage screens. The inter-stage screens ensure that the carbon remains in the tanks but allows the pulp to pass. The pulp loses gold progressively down the train, with barren value in the last tank. The carbon, removed once a day by counter current from the last to the first tank, is then washed by dilute hydrochloric acid (HCI), followed by washing with water to remove tramp material (wood chips etc.) as well as slime. During washing the system is raised to the desired elution temperature of 110-120°C. Eluates such as potassium cyanide. Sodium sulphide or caustic cyanide (NaCN + NaOH) are then used to release the gold and bring it into solution. Gold is recovered from the eluate either by electro winning or zinc Precipitation. A similar technique called the carbon-in-solution (CIS) process may be used to scavenge gold from other solutions that arise from existing mine circuits.

3.3.1.4 Carbon-in-Leach (CIL) Process

The leach and adsorption circuits are occasionally combined. The carbon enhances the leach efficiency by removing surface coatings and has advantages from a capital cost point of view. However, CIL presents considerable operational problems. Other methods employed include: resin-in-pulp (RIP) using ion-exchange resins instead of carbon, solvent-in-pulp (SIP), reverse osmosis and membrane technology. electro winning, pressure elution and electro elution.

3.3.1.5 The effect of foreign ions on cyanidation process

Many investigators agree that the dissolution of gold by cyanide is diffusion controlled, but in the industrial cyanidation of ores, cyanide-and the hydroxides which are oxygen consuming substances may decidedly affect the rate of gold extraction. Pyrrhotite (and, to a lesser extent, pyrite), copper, Zinc, (and all base metals), arsenic, and antimony minerals also consume cyanide, some of the known cyanide reactions are below:

Fe^{++} +6CN- → Fe (CN) 4-6

$2Cu^{++}$ + 7CN- + 2H2O → 2 Cu (CN) 2-3 +CNO- + H_2O

Zn O + 4Na CN + H2O → Na Zn (CN) 4 + 2Na OH

Base Metal ions (Cu^{2+}, Fe^{2+}, Fe^{3+}, Mn^{2+}, Ni^{2+}, and Zn^{2+}) form stable complex with cyanide, thus consuming it, reducing its activity, and retarding gold cyanidation (Fink and Pugnam, 1950) Generally, the presence of Base Metals elements such, Cu, Fe, Zn, Au, As, Sb affects the rate of gold dissolution because they consume both cyanide and the dissolved oxygen and forming stable complexes with cyanide.

3.3.1.6 The effect of cyanide concentration

At atmospheric pressure, the cyanidation is not depending on cyanide concentration in the sulrry. At room temperature, and atmospheric pressure, 8.2mg/l of oxygen are dissolved in water, equivalent to 0.26×10-3 mole/l. Hence, the required Sodium Cyanide (molecular weight 49) concentration should be at least is equal to 4× 0.26 × 10-3× 49 = 0.051 g/l Higher than 0.05 g/l Sodium cyanide concentration cannot affect the rate of gold dissolution, since at atmospheric pressure it is controlled by constant oxygen concentration in solution (Yannopoulos, 1990).

The cause of the wide variation in the solution strengths found by various investigators to give maximum rate of gold dissolution probably lies in the variations include such factors as the ratio of volume of cyanide solution is used and relatively small surface of gold exposed to the cyanide solution and if agitation is sufficiently intense to remove the products of the reaction from the surface of the gold as rapidly as they are formed, then the controlling factor governing the rate dissolution of the gold should be the oxygen concentration of the solution in contact with the gold. If air is used and the tests are run at sea-level the maximum concentration of oxygen in solution will be 8 mg per liter, as follows.

4Au + 8 NaCN + O_2 +2H_2O = 4Na Au $(CN)_2$ + 4Na OH

392 32

There would be no advantage in having more than 392 parts by weigh of NaCN for every 32 parts by weigh of oxygen or 98 parts of NaCN for every part of oxygen in the cyanide solutions.

(Hedly et al., 1945) found that in 1 liter of cyanide solution and aerating vigorously (28 liters per hour) the maximum rate of dissolution and took place between 0.011 and 0.05 per cent Na CN: the rate of dissolutions in the former solution was 95% of maximum.

In practice most cyanide plants treating gold ores use solution containing less than 0.05 per cent NaCN; the general average is probably in the neighborhood to 0.02 per cent NaCN (Norman et al., 1968).

3.3.1.7 The effect of alkali additions

The purpose of adding bases (CaO, NaOH, or Na_2CO_3) to the cyanide process includes the following:-

i- To prevent the loss of cyanide by hydrolysis

ii- To prevent the loss of cyanide by the action of carbon dioxide in the air

iii-To decompose bicarbonates in the mill water before using it in cyanidation

iv- To neutralize acidic compounds such as ferrous, ferric salts and magnesium sulfate in mill water before adding it to the cyanide circuit

v- To neutralize acidic constituent-pyrite, etc. – in ore

In addition, the use of lime promotes the settling of fine ore particles so that clear pregnant solution can be easily separated from cyanide ore pulps.

Although the use of an alkali is essential in cyanidation, many researchers have stated that alkali such as sodium hydroxide, and particularly calcium hydroxide, retard the dissolution of gold in cyanide solutions. Barasky et al. (1934), investigated the effects of calcium hydroxide and sodium hydroxide on rate of gold dissolution in cyanide solutions containing (0.1%) NaCN. They found that when calcium hydroxide was used, the rate of dissolution decreased rapidly as the pH of the solution neared 11, and dissolution was null at pH 12.2. The effect of sodium hydroxide was much less pronounced, the rate of dissolution was start to decrease rapidly above PH 13.4.

With sodium hydroxide that in solution of the same cyanide strength containing Calcium hydroxide at PH 12.2. The effect of the calcium ion on the dissolution of gold was then investigated by using $CaCl_2$ and $CaSO_4$ as addition to cyanide solution. Neither of these salts affected the rate of gold dissolution to any appreciable extend. Oxygen solubility in cyanide solution containing calcium hydroxide and containing about 5 per cent has been reduced. Therefore, it was conclude that the reductions caused by the addition of $Ca(OH)_2$ are due to the formation of calcium peroxide on the metal surface, which prevent the reaction with cyanide.

Therefore, the pH of leaching cyanide solution should be carefully optimized to prevent formation of KCN and achieve a higher gold leaching rate. As a rule, the optimum PH range in practice is 11 to 12.

3.3.1.8 Effect of ore Particle Size on the rate of Dissolution

When free coarse particles of gold are present in the gold Ores, the usual practice to remove it by means of gold traps, Jig, blanket, etc., a head of cyanidation. Otherwise, these coarse particles might not be completely dissolved in the time available for cyanidation. Another practice which reduces the size of the gold particles going to cyanidation by grinding and then classification of gold ore in closed circuit. This practice keeps returning the heavier gold particles to the grinding mill until they are small enough or thinner to overflow the classifier into the cyanidation circuit (Asghar and Reza, 2015). Under what might be considered ideal condition with respect to aeration and agitation (Barsky et al., 1934) found that the maximum rate of dissolution of gold to be 3.25 mg per squ. cm. per hour. By calculation this is equal to a penetration of 1.68 microns on each side of a fat gold particle, or a total reduction in thickness of 3.36 microns per hour. Thus, a piece of gold 44 microns thick (equals in thickness to 32.5 mesh Tyler standard) would take no longer than 13 hours, and a piece 149 microns thickens of gold would take twice as long to be dissolved. With an actual ore and under plant conditions the rate of dissolution is affected by such factors as the association of the gold .i.e. whether or not it is completely liberated, coatings 52 on the gold and the efficiency of the cyanide solution. Not infrequently, however, the gold in an ore is so fine that 80 % to 85 % of it will dissolve in the grinding and classification circuit (Norman et al., 1968).

CHAPTER FOUR

RESULTS

4.1 Geochemical Analysis

20 representative samples from tailing dam are systematically collected. Thus systematically collected samples are sent to geochemical analysis to analyze the concentration of Au and other selected base metals; such as Cu and Ni, using atomic absorption spectrometry (AAS). The samples are arranged in two groups, right side samples and left side samples with respect to the discharging point (outlet). Geochemical analysis result of samples collected from the left side of the tailing dam shows maximum a (0.84 g/t), a minimum (0.34 g/t) and average (0.51 g/t) grade of gold. Samples in the right side of the tailing dam also display a maximum (1.14 g/t), a minimum (0.43 g/t) and average (0.69 g/t) grade gold. Other base metals (i.e. Cu and Ni) geochemical result indicates the samples contain elevated Cu content (~303.5 g/t) and low Ni content (~0.44 g/t). Results of the analysis are presented in the table below (table 4.1).

Table 4. 1 *geochemical analysis data for Au, Cu and Ni from right and left side tailing samples*

Sample ID	Analysis Results (ppm)		
	Au	Cu	Ni
DrS S1	0.83	304	0.17
DrS S2	1.14	316	1.08
DrS S3	0.99	317	1.06
DrS S4	0.73	313	0.47
DrS S5	0.60	304	<0.01
DrS S6	0.49	302	0.73
DrS S7	0.63	301	0.36
DrS S8	0.56	303	0.66
DLS S1	0.67	295	0.71
DLS S2	0.57	299	0.32

DLS S3	0.53	304	0.39
DLS S4	0.60	299	0.76
DLS S5	0.49	308	0.41
DLS S6	0.53	299	<0.01
DLS S7	0.47	300	0.42
DLS S8	0.50	302	0.32
DLS S9	0.34	299	0.17
DLS S10	0.34	296	0.30
DLS S11	0.34	298	0.17
DLS S12	0.43	311	0.26

Summarized 8 month secondary data provided by the company is presented in the table below (table 4.2). The overall ore feed during the 8 month period is 75859.06t, and out of this ore feed the average head grade is 5.21g/t. In other hand, the overall gold grade lost during this time is presented in two categories; insoluble and soluble (undissolved and dissolved). Therefore, the total gold grade loss in insoluble form is 0.46g/t where-as, soluble form is 0.03g/t. Hence, the total gold recovery is 90.57 % (395268.1g). On the other hand, the total gold loss 9.43 % and (36702.7g). Metal loss and recovery percentage loss is calculated from 8 months data and it is illustrated in the table below (Table 4.3).

Table 4. 2 summary *of 8 month gold recovery result*

Month	Ore Feed(t)	Ore Grade			Gold Recovery (%)
		Head G(g/t)	Insoluble (g/t)	Soluble (g/t)	
Jul-19	7443.911	4.12	0.523	0.035	86.45631
Aug-19	5143.107	4.31	0.38	0.02	90.71926
Sep-19	6052.096	5.05	0.424	0.0275	91.05941
Oct-19	20130.88	5.52	0.466	0	91.55797
Nov-19	2329.231	7.551	0.57		92.45133
Dec-19	5074.155	6.65	0.47	0.08	91.72932
Jan-20	14771.97	5.4	0.47	0.04	90.55556
Feb-20	14913.71	4.67	0.411	0.04	90.34261

Table 4. 3 *Calculated loss of gold from 8 month data*

Month	Ore Feed(t)	Metal(g)	Loss metal(g)	Loss Percentage(%)
Jul-19	7443.911	30668.9	4153.7	13.54369
Aug-19	5143.107	22166.8	2057.2	9.280742
Sep-19	6052.096	30563.1	2732.5	8.940594
Oct-19	20130.88	111122.5	9381.0	8.442029
Nov-19	2329.231	17588.0	1327.7	7.548669
Dec-19	5074.155	33743.1	2790.8	8.270677
Jan-20	14771.97	79768.6	7533.7	9.444444
Feb-20	14913.71	69647.0	6726.1	9.657388
Sum	75859.06	395268.1	36702.7	

CHAPTER FIVE

DISCUSSION

5.1 Gold loss in the tailing dam

Primary geochemical results of the tailing dam and secondary 8 month consecutive gold loss analysis indicate the presence of significant gold loss disposing to the tailing dam. The company has a cutoff grade of 0.3g/t gold from the overall processing circuit; however, according to analytical result grade of gold disposed to the tailing dam is greater than this value. Primary data collected from the tailing dam shows an average grade of 0.61g/t with the maximum grade of 1.14g/t and minimum 0.34g/t. In addition, secondary data of the company's 8 month gold grade loss analysis indicates average grade of 0.46g/t insoluble and 0.02g/t soluble gold with maximum grade of gold 0.57g/t and minimum 0.4g/t is disposed to the tailing dam. Hence, this indicates the presence of possible loss in the tailing dam.

The reasons for this gold grade loss could be the cumulative effects of particle size, beneficiation technique; cyanide concentration, carbon size, effects of other ions and effects of alkali additions. (Yannopulos, 1991; Asghar and Reza, 2015; Jiang et al., 2001)

Once the ore material is delivered to the mill, the proportion of the particle size percentage is determined according to the expected gold recovery. Percentage of grounded ore passes through -200 meshes with acceptable (>88%) gold recovery is 60-75% (Yannopulos, 1991). The average gold recovery in EMD is 90.57%, for this gold yielding ore they allow 80% ground ore pass through -200 meshes, this deviate from the standard by 5%. This could affect the effectiveness of the cyanidation processes; hence, it could lead to additional gold loss.

The other contributing factor for possible gold loss is effects of cyanide concentration. The most commonly used cyanide concentrations in solution and solid form are 0.2-0.5g/l NaCN (Norman et.al., 1968) and 0.45-1.81kg/t respectively (Yannopulos, 1991). Yannopulos, 1991 recommended low contents of cyanide for gold leaching. The NaCN concentration used by EMD in solution and solid form are 0.3-0.35g/l and 1kg/t respectively. This result shows gold loss in EMD could not be caused by improper cyanide concentration both in solution and in solid form.

The next possible beneficiation technique that could affect gold recovery percentage is carbon size in cyanidation process. Carbon is added to the ore slurry in the cyanide leaching tank in order to adsorb the gold cyanide complex from the solution (Yannopulos, 1991). The size of the carbon added to the solution has a great role in recovering gold efficiently, the coarser the grain size the more effective the gold recovery will be. EMD uses carbon size of 1.7-3.35mm, while the size ranges of activated carbon used are 1.18-2.36mm, 1.18-3.35mm and 1.70-3.35mm. Therefore, the carbon size also could not be the reason for the gold loss in the tailing dam.

Unlike the previous two beneficiation techniques, effects of other ions in cyanide solution negatively affect the efficiency of the gold recovery percentage (Jiang et al., 2001). Cyanide gold extraction method is one of the favorable techniques to recover gold from the ore mineral. However, it needs effective proportion and require detail investigation on the presence of other active element which form CN complex so that it reduce the effectiveness in gold recovery process (Fink and Putnam, 1950). As discussed in previous sections EMD uses cyanide extraction method, but the ore mineral (gossan) they use contains significant amount of base metal (i.e. Cu in the tailing has the maximum value of 317g/t, minimum 296g/t and average grade of 303.5g/t) with ions similar to Au that form cyanide complex.

In addition to these effects of base metal in cyanidation, addition of alkali to ore slurry could also affect gold recovery percentage and cause gold loss to the tailing (Barasky et al. 1934). According to John, (2017) the optimum alkalinity of the ore slurry tank should be in the range of 8-11 PH value. EMD used to keep this range of PH value using lime feeder, but since the lime feeder is failed they use manual way of feeding lime to the ore slurry tank. Therefore, this manual operation is prone to minor deviation which leads to gold loss to the tailing.

Generally, gold loss in mining process is the cumulative effect of many individual factors. In the case of EMD the contribution of the above mentioned factors are investigated. Therefore, particle size in milling, effect of other ions and addition of alkali in beneficiation technique could cause the gold loss in the tailing dam.

5.2 Tailing dam reserve estimation

The area of the tailing dam is calculated by using Arc GIS and it is $405m^2$. The thickness obtained from elevation difference which is at the bottom of the tailing dam has an elevation of 1177m and at the top 1198m; overall 21m. Using conventional reserve estimation method the tailing dam reserve is estimated as follows:

$$T = A \times H \times \rho$$

$$T = 402m^2 \times 21m \times 2.65g/cm^3$$

$$T = 22{,}371.3\ t$$

Where, T- Tonnage in the tailing dam; A-Area of the tailing dam; H- thickness of the tailing dam; ρ- bulk density of the ore.

This ore disposed in the tailing contains average minable grade of 0.61g/t, which is above the cutoff grade of 0.3g/t. Therefore, the net lost metal content which could obtain in the tailing is;

$$Metal = T \times G$$

$$= 22{,}371.3t \times 0.61g/t$$

$$= 13{,}646.5gm$$

Where, *T- Tonnage in the tailing dam; G- tailing average grade*

The reserve estimation indicates that 13,646.5gm of gold is dumped to the tailing dam.

5.3 Spatial Variation of Gold Discharge

Samples from tailing dam show decrement of gold grade as we go away from point of discharge (outlet) to inlet. As shown on the graph below from DrS S1 to DrS S8 and DLS S1 to DLS S12 the grade of gold decreased progressively (Fig. 5.1 & 5.2) According to Vick, 1990, the coarse particles of tailings settle close to the point of discharge, and the fine particles run down the beach into the pond and settle there. This could indicate that the higher gold grade concentration near to the outlet is due to coarseness of the gold grains. In other case, higher density of gold ($19.3g/cm^3$) may cause concentration of gold near the outlet. It is known that denser materials can settle down first than of light ones. This shows that, since the gold grade is above the cutoff grade, the company should conduct tailing dam reprocessing nearby the outlet.

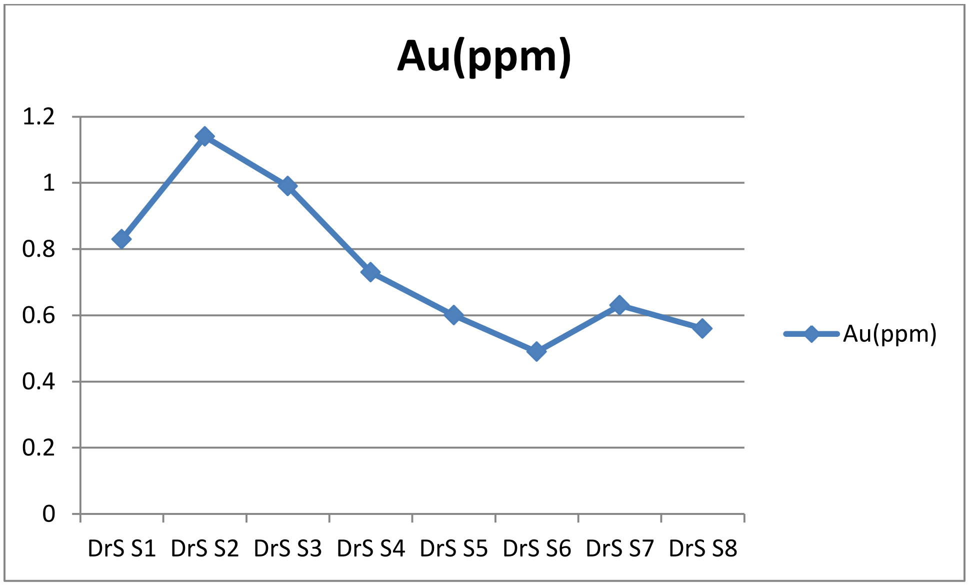

Figure 5. 1 gold grade distribution on the right side of the tailing dam

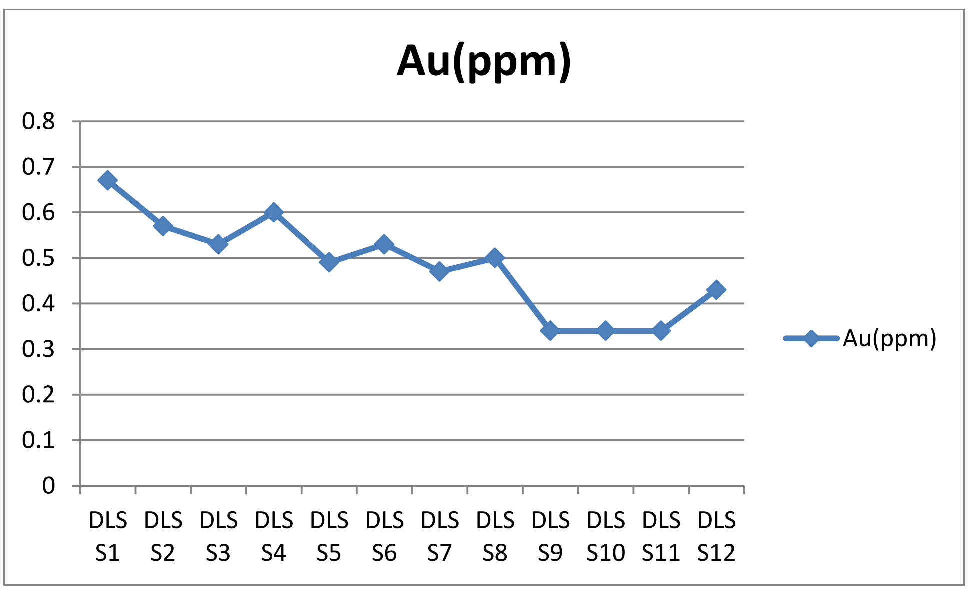

Figure 5. 2 *gold grade distribution on the left side of the tailing dam*

5.4 Mitigation Measurements for the loss of gold

As discussed earlier different factors can lead for the loss of gold. This can affect the overall profit of the mining company. In order to reduce the loss of gold different mitigation measures have to be employed. For ores containing easily soluble copper in cyaniding, multi stage leaching can be used. Ammonia-cyanide mixtures have also been reported for treatment of copper bearing gold ores. Furthermore, it needs to be decopperred prior to cyanidation , or to be treated by non-cyanide leaches such as thiorea leaching which has been reported to be insensitive to copper. Compared to cyanide as a gold lixiviant, thiourea has certain advantages.

- Low sensitivity to base metals (Pb, Cu, Zn, As, Sb)
- Low sensitivity to residual sulfur in calcines
- High gold recovery from pyrite and chalcopyrite concentrates
- Satisfactory recovery of gold from carbonaceous (refractory) ores (Chen et al., 1982).

According to Yannopulos, 1991 the percentage of grounded ore passes through -200 meshes with acceptable (>88%) gold recovery is 60-75%. For this reason the company should conduct with this range in order to get a good recovery of gold.

A good recovery of certain mineral commodity depends on different conditions. One of them is efficiency of the machine. The machineries should be free from problems and damage otherwise the results they give would be uncertain. And also, there should be permanent technician or maintenance for the machineries in order to increase workability life. As in the case of EMD, the lime feeder which was used for regulating the PH has been failed. So, It needs a proper maintenance in order to reduce the gold loss.

CHAPTER SIX

CONCLUSION AND RECOMMENDATION

6.1 Conclusion

- Geochemical results reveal that the presence of a gold loss to the tailing dam. It ranges from 13.54 to 7.54% with average loss percentage of 9.43. From total 8 month ore feed (75859.06t) total metal gained is 395268.1gm and total metal lost is 36702.7gm. The average grade of gold in the tailing dam is 0.61g/t which is above the cutoff grade (0.3g/t). Hence, from reserve estimation the net metal loss is about 13,646.5gm. As a result this much of gold is lost to the tailing dam.
- The gold grade of the tailing dam fall in the range of 1.14 to 0.34g/t with an average grade of 0.61g/t. This indicates loss of gold in the tailing dam.
- Geochemical results shows that a gradual decrement in grade of gold when it moves away from discharging point (outlet). Moreover the geochemical data reveals that the insoluble gold (0.46g/t) is much greater than the soluble (0.03g/t) ones. This indicates un recoverable gold is present in the tailing dam which related to coarseness of the gold.
- This study also revealed that the presence of base metals like Cu in the solution, the lime proportion used and the percentage of ground ore passes through -200 mesh are the main aggregate effects for the loss of gold in Meli gold processing.

- For mitigating gold loss for ore containing soluble copper, multi stage leaching, Ammonia-cyanide mixtures, treating by non-cyanide leaches such as thiorea leaching can be employed. Moreover, EMD needs to use 60-75% of particle size percentage range. In addition to these proper maintenance of the lime feeder should have to be done.

6.2 Recommendation

Based on the obtained results and interpretations the following recommendations can be forwarded:

- Further study using pit and auger sampling can help to understand the amount of gold in the tailing dam.
- Full mineral processing and metallurgic study is recommend for pointing out the problem clearly.
- From the finding of gold grade distributions in the tailing dam, it is highly recommended that the outlet or discharging area is favorable for reprocessing of gold.
- It is recommended that environmental impact studies of cyanidation extraction process should be done; since nobody has done on that before.

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Appendices

Tailing dam locations

No	X coordinate	Y coordinate	Elevation
1	396189.236	1543541.83	1198.569
2	396146.914	1543557.51	1198.609
3	396130.075	1543563.44	1198.775
4	396123.184	1543570.36	1198.944
5	396120.931	1543577.04	1198.817
6	396121.699	1543595.14	1198.052
7	396125.258	1543624.32	1199.087
8	396126.781	1543633.13	1199.354
9	396129.658	1543635.81	1197.619
10	396136.313	1543647.24	1198.196
11	396141.395	1543654.78	1198.307
12	396141.768	1543654.57	1198.047
13	396147.727	1543671.17	1198.348
14	396149.317	1543677.42	1198.337
15	396149.186	1543686.09	1198.82
16	396146.647	1543697.73	1198.297
17	396144.596	1543701.19	1198.236
18	396136.154	1543721.89	1198.667
19	396148.908	1543723.89	1198.75
20	396160.468	1543724.17	1199.276
21	396172.642	1543728.04	1198.961
22	396185.686	1543731.96	1198.08
23	396196.166	1543733.76	1197.968
24	396201.826	1543736.78	1197.609
25	396203.127	1543736.58	1197.206
26	396207.629	1543743.7	1198.022
27	396209.908	1543748.72	1197.918
28	396213.753	1543760.36	1198.381
29	396217.692	1543773.03	1198.57
30	396220.534	1543782.55	1198.911
31	396221.915	1543789.01	1199.024
32	396221.479	1543795.75	1199.017
33	396225.029	1543815.09	1198.647
34	396235.182	1543811.36	1199.43

35	396246.724	1543807.77	1199.204
36	396257.95	1543807.45	1199.179
37	396272.56	1543804.38	1198.726
38	396278.92	1543802.1	1198.95
39	396278.966	1543802.01	1198.947
40	396277.079	1543779.21	1200.15
41	396272.893	1543759.52	1199.298
42	396275.401	1543744.89	1199.256
43	396277.07	1543735.37	1199.707
44	396266.948	1543691.08	1199.35
45	396262.431	1543674.06	1199.477
46	396262.057	1543664.34	1199.021
47	396260.747	1543656.88	1199.067
48	396261.715	1543639.42	1199.366
49	396263.135	1543626.85	1198.179
50	396268.126	1543617.25	1198.912
51	396266.682	1543609.55	1196.501
52	396278.052	1543603.73	1196.645
53	396281.721	1543605.92	1198.419
54	396286.4	1543602.17	1196.675
55	396301.104	1543601.66	1196.786
56	396302.587	1543608.34	1198.853
57	396320.096	1543610.79	1198.282
58	396320.432	1543602.9	1196.945
59	396327.753	1543598.49	1197.018
60	396337.45	1543610.07	1202.229
61	396337.122	1543592.86	1200.33
62	396334.642	1543573.47	1200.057
63	396321.002	1543554.74	1198.867
64	396318.364	1543556.61	1197.172
65	396304.684	1543547.88	1197.006
66	396303.471	1543545.74	1197.642
67	396286.336	1543542.06	1198.321
68	396283.783	1543546.07	1196.77
69	396272.312	1543544.16	1196.726
70	396274.353	1543538.74	1198.465
71	396261.932	1543534.09	1198.155
72	396260.737	1543538.61	1196.658
73	396247.732	1543531.27	1196.543
74	396254.407	1543527.39	1198.505

75	396249.096	1543523.69	1198.051
76	396244.583	1543528.64	1196.516
77	396240.615	1543527.91	1196.43
78	396237.029	1543524.41	1198.621
79	396217.851	1543531.14	1198.567
80	396219.279	1543535.69	1196.333
81	396226.04	1543532.59	1196.395

Tailing dam sample locations

Sample Id	**X coordinate**	**Y coordinate**	**Elevation**
DLS S1	396239	1543603	1195
DLS S2	396232	1543621	1196
DLS S3	396223	1543632	1196
DLS S4	396223	1543651	1195
DLS S5	396217	1543680	1193
DLS S6	396208	1543684	1195
DLS S7	396183	1543689	1196
DLS S8	396161	1543704	1194
DLS S9	396161	1543712	1194
DLS S10	396154	1543726	1193
DLS S11	396147	1543740	1194
DLS S12	396136	1543752	1194
DrS S1	396262	1543745	1196
DrS S2	396257	1543734	1195
DrS S3	396257	1543719	1194
DrS S4	396258	1543695	1194
DrS S5	396255	1543678	1193
DrS S6	396246	1543647	1194
DrS S7	396246	1543622	1193
DrS S8	396250	1543601	1193

8 months ore production results

July 2019					
Feed slurry status					
Date	**Total ore feed(t)**	**Weighted Average of**	**Weighted ave of**	**Weighted ave of soluble(g/t)**	**Gold Recovery(%)**

		Head grade(g/t)	insoluble(g/t)		
1	198.22	5.54	0.67	0.02	87.5%
2	345.00	4.90	0.47	0.00	90.4%
3	500.75	4.01	0.50	0.04	86.6%
4	55.75	4.25	0.57	0.00	86.6%
5	-	0	0	0.00	-
6	209.29	3.65	0.50	0.00	86.3%
7	0	0	0	-	-
8	271.25	3.41	0.53	-	84.6%
9	393.84	3.56	0.54	-	85.0%
10	394.46	3.40	0.59	-	82.6%
11	185.22	5.12	0.68	0.01	86.4%
12	216.31	3.61	0.56	0.00	84.5%
13	213.86	3.55	0.70	0.03	79.5%
14	322.85	3.90	0.62	0.01	84.1%
15	180.97	3.69	0.66	0.05	80.8%
16	211.29	4.35	0.65	0.01	84.9%
17	195.07	4.64	0.66	0.00	85.7%
18	289.78	3.86	0.52	0.01	86.2%
19	222.63	4.09	0.50	0.01	87.5%
20	501.07	3.91	0.47	0.00	88.0%
21	315.85	3.92	0.42	0.00	89.2%
22	281.30	4.44	0.42	0.00	90.6%
23	123.76	4.33	0.48	-	88.9%
24	150.23	4.55	0.44	0.01	90.1%
25	337.82	4.13	0.43	-	89.5%
26	332.29	4.73	0.43	0.00	90.8%
27	241.90	4.57	0.49	0.01	89.1%
28	438.93	4.14	0.37		91.0%
29	61.40	4.12	0.52	0.04	86.5%
30	-	0	0.00	-	-
31	252.84	4.23	0.42	-	90.0%
SUM	**7443.91**				

AUG-2019					
Feed slurry status					
Date	**Total ore feed(t)**	**Weighted Average of Head grade(g/t)**	**Weighted ave of insoluble(g/t)**	**Weighted ave of soluble(g/t)**	**Gold Recovery (%)**
1	208.66	3.93	0.46	0.01	88.1
2	34.11	4.31	0.38	0.02	90.7
3	94.36	4.59	0.47	0.01	89.6
4	-	-	-	-	-
5	-	-	-	-	-
6	-	-	-	-	-

7	267.12	4.31	0.39	0.02	90.4
8	398.77	4.54	0.43	0	90.5
9	350.62	4.42	0.46	0.00	89.6
10	104.46	4.07	0.44	0.00	89.2
11	183.81	4.32	0.36	0.00	91.6
12	128.71	5.11	0.39	0.00	92.4
13	-	-	-	-	-
14	-	-	-	-	-
15	241.02	4.26	0.35		91.8
16	-	0.00	0.00	0.00	
17	181.76	4.28	0.34	0.01	92.0
18	100.72	4.31	0.38	0.02	90.7
19	-	0.00	0.00	-	-
20	210.14	4.04	0.30	0.00	92.4
21	496.99	4.01	0.31	0.00	92.2
22	267.72	4.89	0.30	0.00	93.9
23	298.78	4.38	0.30	0.00	93.2
24	166.80	4.22	0.37	0	91.1
25	101.36	4.31	0.38	0.02	90.7
26	42	4.31	0.38	0.02	90.7
27	330.81	4.61	0.38	0.00	91.7
28	284.98	4.15	0.36	0.00	91.3
29	103.32	4.44	0.40	0.00	91.0
30	289.77	4.32	0.37	0.00	91.3
31	256.31	3.72	0.40	0.00	89.2
SUM	**5143.105**				

Sep 2020					
Feed slurry status					
Date	**Total ore feed(t)**	**Weighted Average of Head grade(g/t)**	**Weighted ave of insoluble(g/t)**	**Weighted ave of soluble(g/t)**	**Gold Recovery (%)**
1	369.35	4.40	0.45	-	89.9
2	193.00	4.45	0.40	-	91.0
3	188.68	4.07	0.32	-	92.3
4	230.29	4.05	0.36	0.01	90.9
5	161.44	6.30	0.40	-	93.7
6	53.59	0.00	0	-	-
7	228.47	4.81	0.28	0.01	94.1
8	209.66	4.63	0.26	-	94.3
9	238.02	7.49	0.49	-	93.5
10	-	0.00	0.00	-	-
11	219.83	5.59	0.56	-	90.0
12	520.48	5.74	0.41	0.01	92.6
13	361.30	5.32	0.47	0.01	91.0

14	265.61	4.65	0.43	-	90.7
15	414.90	5.35	0.39	-	92.7
16	328.85	5.29	0.40	-	92.4
17	187.53	3.47	0.25	0.00	92.8
18	652.95	4.71	0.35	0.01	92.3
19	484.37	4.86	0.34	0.02	92.7
20	240.40	4.06	0.35	0.01	91.1
21	203.99	4.43	0.00	0.02	99.6
22	191.00	0.97	0.11	-	88.5
23	108.40	4.40	0.49	0.02	88.4
24	-	0.00	0.00	0.00	-
SUM	**6052.10**				

October 2019					
Feed slurry status					
Date	**Total ore feed(t)**	**Weighted Average of Head grade(g/t)**	**Weighted ave of insoluble(g/t)**	**Weighted ave of soluble(g/t)**	**Gold Recovery(%)**
1	176.7				
2	71.88	3.92	0.53		86.49
3	465.5315	4.83	0.65		86.64
4	446.2985	3.79	0.50		86.86
5	483.5375	3.32	0.38		88.43
6	472.44	3.29	0.38		88.35
7	389	4.14	0.45		89.14
8	74.35	0.00	0.00		
9	354.95	3.67	0.36		90.15
10	277.584	3.74	0.31		91.58
11	244.2	3.99	0.40		90.08
12	448.53	4.32	0.40		90.08
13	417.085	4.81	0.45		90.61
14	297.5	3.39	0.33		90.24
15	380.815	4.65	0.44		90.54
16	481.95	5.29	0.52		90.24
17	538.85	4.50	0.43		90.53
18	490.28	4.30	0.46		89.38
19	644.94	4.46	0.41		90.84
20	501.77	3.48	0.30		91.30
21	477.13	3.31	0.46		86.14
22	445.72	2.89	0.28		90.16
23	576.21	5.90	0.34		94.27
24	413.64	5.06	0.35		
25	476.01	5.48	0.45		91.84
26	530.27	6.98	0.52		92.53

27	363.57	6.08	0.48		92.12
28	116.29				
29	418.88				
30	648.10	6.32	0.50		92.05
31	603.49	7.64	0.53		93.11
1	422.26	8.03	0.54		93.23
2	403.78	7.96	0.56		93.01
3	257.29	6.18	0.47		92.40
4	643.88	7.14	0.62		91.35
5	371.72	5.74	0.43		92.46
6	230.30	5.20	0.41		92.15
7	-				
8	-				
9	-				
10	-				
11	333.10	2.98	0.18		93.94
12	270.12	5.12	0.36		93.00
13	96.31				91.68
14	376.31	6.31	0.53		91.68
15	278.46	4.39	0.31		93.04
16	518.80	6.96	0.48		93.17
17	266.78	9.29	0.60		93.51
18	518.42	8.84	0.54		93.89
19	134.31	4.78	0.56		88.28
20	-				
21	-				
22	178.12	4.82	0.43		91.14
23	312.32	5.28	0.46		91.28
24	55.17				
25	97.93				
26	232.73	6.28	0.36		94.32
27	262.81	2.46	0.17		93.00
28	352.64	6.14	0.39		93.63
29	379.52	6.48	0.41		93.68
30	187.55	4.53	0.32		92.88
31	222.75	5.72	0.47		91.78
Sum	**20130.88**				

November 2019					
Feed slurry status					
Date	**Total ore feed(t)**	**Weighted Average of Head grade(g/t)**	**Weighted ave of insoluble(g/t)**	**Weighted ave of soluble(g/t)**	**Gold Recovery(%)**

1	422.26	8.03	0.54		93.23
2	403.78	7.96	0.56		93.01
3	257.29	6.18	0.47		92.40
4	643.88	7.14	0.62		91.35
5	371.72	5.74	0.43		92.46
6	230.30	5.20	0.41		92.15
Sum	**2329.23**				

Dec -2019					
Feed slurry status					
Date	**Total ore feed(t)**	**Weighted Average of Head grade(g/t)**	**Weighted ave of insoluble(g/t)**	**Weighted ave of soluble(g/t)**	**Gold Recovery(%)**
1	-				
2	-				
3	-				
4	-				
5	-				
6	-				
7	-				
8	-				
9	-				
10	-				
11	333.10	2.98	0.18	-	93.9
12	270.12	5.12	0.36	-	93.0
13	96.31	0.00	0.00	-	-
14	376.31	6.31	0.53	-	91.7
15	278.46	4.39	0.31	-	93.0
16	518.80	6.96	0.45	0.02	93.2
17	266.78	9.29	0.60	-	93.5
18	518.42	8.84	0.53	0.07	93.1
19	134.31	4.78	0.56	-	88.3
20	-	0.00	0.00	-	-
21	-	0.00	0.00	-	
22	178.12	4.82	0.43	-	91.1
23	312.32	5.28	0.46	-	91.3
24	55.17	0.00	0.00	-	-
25	97.93	0.00	0.00	-	-
26	232.73	6.28	0.36	-	94.3
27	262.81	2.46	0.17	-	93.0
28	352.64	6.14	0.39	-	93.6
29	379.52	6.48	0.41	-	93.7
30	187.55	4.53	0.32	-	92.9
31	222.75	5.72	0.47	-	91.8
SUM	**5074.16**				

Jan-2020					
Feed slurry status					
Date	**Total ore feed(t)**	**Weighted Average of Head grade(g/t)**	**Weighted ave of insoluble(g/t)**	**Weighted ave of soluble(g/t)**	**Gold Recovery(%)**
1	393.56	0.73	0.07	-	91.0
2	344.77	6.16	0.58	-	90.6

3	594.64	5.54	0.45	0.02	91.4
4	517.68	6.22	0.48	0.02	91.9
5	465.90	4.56	0.41	0.01	90.9
6	741.84	6.49	0.47	-	92.8
7	643.98	5.76	0.51	-	91.1
8	621.43	5.21	0.50	0.01	90.2
9	626.26	4.75	0.48		89.9
10	368.25	3.98	0.45	0.01	88.6
11	180.72	0.00	0.00	-	-
12	-	0.00	0.00	-	-
13	121.58	3.86	0.62	-	83.9
14	506.74	3.84	0.49	-	87.2
15	393.72	4.81	0.45	0.09	88.9
16	567.30	0.00	0.00	-	-
17	590.90	5.25	0.42	0.06	90.8
18	582.65	3.65	0.34	0.02	90.2
19	566.51	4.18	0.40	0.02	89.8
20	561.02	5.03	0.66	0.03	86.3
21	542.32	6.08	0.47	0.02	91.9
22	399.44	5.93	0.56	0.01	90.4
23	196.45	4.38	0.40	-	90.9
24	250.29	3.70	0.29	0.00	92.1
25	359.91	4.22	0.32	-	92.4
26	599.63	5.58	0.45	0.01	91.7
27	724.72	5.57	0.49	0.00	91.2
28	433.57	5.52	0.42	0.01	92.4
29	529.72	5.27	0.32	-	93.9
30	643.87	5.25	0.32	-	93.8
31	702.61	4.95	0.29	-	94.2
SUM	**14771.97**				

Feb-2020					
Feed slurry status					
Date	**Total ore feed(t)**	**Weighted Average of Head grade(g/t)**	**Weighted ave of insoluble(g/t)**	**Weighted ave of soluble(g/t)**	**Gold Recovery(%)**
1	611.82	4.84	0.29		94.0
2	463.75	5.40	0.26		95.2
3	692.32	4.39	0.35		92.1
4	567.15	4.10	0.30		92.8
5	612.96	5.22	0.36		93.0
6	697.73	5.23	0.37		93.0
7	645.41	5.03	0.39		92.3
8	585.67	5.24	0.44		91.6
9	709.38	5.60	0.37		93.2

10	540.77	5.46	0.46		91.7
11	598.76	4.81	0.38		92.0
12	589.32	3.62	0.35		90.2
13	510.11	5.36	0.43		92.0
14	532.81	5.19	0.39		92.4
15	451.68	4.66	0.36		91.7
16	503.75	4.51	0.47		89.6
17	461.01	2.75	0.30		88.2
18	425.70	3.94	0.32		91.8
19	642.23	3.76	0.33		91.3
20	518.12	3.70	0.36		90.3
21	337.05	3.37	0.29		91.3
22	462.06	3.12	0.29		90.6
23	381.10	4.12	0.42		89.8
24	552.34	4.25	0.53		87.6
25	427.57	2.77	0.41	0.02	85.2
26	490.39	4.17	0.59		85.8
27	700.23	2.87	0.38		86.6
28	71.45	0.00	0.00		
29	132.19	0.00	0.00		
SUM	**14913.71**	0	0.00		