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สวัณเขต สปป ลาว

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The Pre-
Feasibility Study of Gold Mine Development, A Case Study of Sepon Deposit, Savannakhet Province, Lao PDR

Mr. Seelae Phaisopha

A Thesis Submitted in Partial Fulfillment of the Requirements
for the Degree of Master of Engineering Program in Georesources Engineering
Department of Mining and Petroleum Engineering
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Thesis Title	The Pre- Feasibility Study of Gold Mine Development, A Case Study of Sepon Deposit, Savannakhet Province, Lao PDR
By	Mr. Seelae Phaisopha
Field of Study	Georesources Engineering
Thesis Advisor	Assistant Professor Sunthorn Pumjan, Ph.D.

Accepted by the Faculty of Engineering, Chulalongkorn University in Partial Fulfillment of the Requirements for the Master's Degree

....Dean of the Faculty of Engineering
(Professor Bundhit Eua-arporn, Ph.D.)

THESIS COMMITTEE

....Chairman
(Associate Professor Pinyo Meechumna, Ph.D.)

....Thesis Advisor
(Assistant Professor Sunthorn Pumjan, Ph.D.)

....Examiner
(Thitisak Boonpramote, Ph.D.)

....External Examiner
(Songwut Artittong, Ph.D.)

สีแล่ ไพโสพา : การศึกษาความเป็นไปได้เบื้องต้นการพัฒนาเหมืองทอง กรณีศึกษาแหล่งเซโปนสวัณเขต สปป ลาว (The Pre- Feasibility Study of Gold Mine Development, A Case Study of Sepon Deposit, Savannakhet Province, Lao PDR) อ.ที่ปรึกษาวิทยานิพนธ์หลัก: ผศ. ดร. สุนทร พุ่มจันทร์, 76 หน้า.

งานวิจัยนำเสนอการศึกษาความเป็นไปได้เบื้องต้นการพัฒนาเหมืองทอง ในประเทศ สปป. ลาว โดยมีพื้นที่วิจัยที่แหล่งแร่ทองคำเซโปน จ. สวรรณเขตุ งานวิจัยชิ้นนี้มีวัตถุประสงค์หลักคือ 1) สร้างแบบจำลองธรณีวิทยาเพื่อนำไปสู่การคำนวณปริมาณสำรองทางธรณีวิทยาและเป็นการเตรียมข้อมูลสำหรับการออกแบบและการทำเหมือง 2) สร้างแบบวิเคราะห์ด้านการเงินซึ่งเกี่ยวข้องกับการคำนวณส่วนลดจากกระแสเงินสด (DCF) มูลค่าปัจจุบันสุทธิ (NPV) อัตราผลตอบแทนภายใน (IRR) และค่าใช้จ่ายเฉลี่ยของน้ำหนัก (WACC) 3) เพื่อศึกษาผลกระทบสิ่งแวดล้อมจากกิจกรรมการทำเหมืองและเสนอแนวทางการลดผลกระทบ

แบบจำลองทางธรณีวิทยาสร้างจากข้อมูลวิเคราะห์ตัวอย่างแร่จำนวน 3,218 ตัวอย่าง รวบรวมจากหลุมเจาะจำนวน 54 หลุม พื้นที่ศึกษาครอบคลุม 500 ม. x 700 ม. ในแนวตะวันออกและตะวันตก มีความหนา 250 ม. ผลการประเมินปริมาณแหล่งแร่ทางธรณีวิทยาพบว่ามีทรัพยากรแร่ทั้งหมด 34.87 ล้านตัน ที่ค่าเฉลี่ยความสมบูรณ์แร่ 0.58 กรัม/ตัน ในขั้นตอนการศึกษาบล็อกโมเดลขนาด 12.5 ม. x 12.5 ม. x 5 ม. ถูกสร้างขึ้นจำนวน 17,412 บล็อก บ่อเหมืองถูกออกแบบโดยวิธีการของ เลอ–กรอสแมน ซึ่งสามารถประเมินแหล่งแร่ที่สามารถทำเหมืองได้ที่ 2.44 ล้านตัน ที่ค่าเฉลี่ยความสมบูรณ์แร่ 1.53 กรัม/ตัน ขั้นตอนการทำเหมืองได้ออกแบบให้เหมืองมีปริมาณการผลิตที่ 6 แสนตัน/ปี ตลอด 4 ปีของอายุหมือง ระบบจัดการวัสดุประกอบด้วยรถบรรทุกจำนวน 5 คัน รถตักล้อยางจำนวน 1 คัน และขุดตักจำนวน 1 คัน เพื่อรองรับการขนส่งแร่ที่อัตรา 160 ตัน/ชม. ผลวิเคราะห์ทางการเงิน คำนวณค่ามูลค่าปัจจุบันสุทธิ (NPV) ได้ 1.93 ล้านดอลล่าร์ อัตราผลตอบแทน (IRR) ที่ร้อยละ 22 ซึ่งสูงกว่าค่าเฉลี่ยของน้ำหนัก (WACC) ที่คำนวณได้ที่ร้อยละ 12.6 งานวิจัยได้เสนอแนะด้านการลดผลกระทบสิ่งแวดล้อม โดยปฏิบัติตามมาตรฐานด้านสิ่งแวดล้อมของประเทศลาว ในเรื่องผลกระทบจากระดับเสียง ปริมาณฝุ่นละออง แรงสั่นสะเทือน และนอกจากนี้ยังเสนอแนะข้อปฏิบัติในการระเบิดเพื่อลดผลกระทบสิ่งแวดล้อม ในภาพรวมโครงการพัฒนาเหมืองทองนี้ มีความเป็นไปได้ทั้งในแง่ด้านเทคนิค การเงิน และด้านสิ่งแวดล้อม

ภาควิชา	วิศวกรรมเหมืองแร่และปิโตรเลียม	ลายมือชื่อนิสิต	....
สาขาวิชา	วิศวกรรมทรัพยากรธรณี	ลายมือชื่อ อ.ที่ปรึกษาหลัก	....
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KEYWORDS: OPEN PIT MINE DEVELOPMENT/MINE OPERATIONS/MINE DESIGN/MINE PLANNING/MINE ENVIRONMENT

SEELAE PHAISOPHA: The Pre- Feasibility Study of Gold Mine Development, A Case Study of Sepon Deposit, Savannakhet Province, Lao PDR. ADVISOR: ASST. PROF. SUNTHORN PUMJAN, Ph.D., 76 pp.

This study is about the pre-feasibility study of gold mine development in Lao PDR. The research area is located at Sepon Basin, Vilabouly District, Savannakhet Province. This research focuses mainly on technical development of open pit gold mine with three objectives. First, to construct the geological block model which provides the basis for resource/reserve estimation, and will be used as inputs in the stage of mine optimization, mine design and operation, Second, to develop a financial model using Discounted Cash Flow (DCF), and the criteria of Net present Value (NPV), the Internal Rate of Return (IRR), and the Weight Average Cost of Capital (WACC). Third, to address the environmental and social impacts from mine operation and proposed some mitigation guidelines.

The geological model with 3,218 assays data collected from collected from 54 drillholes is prepared in MineSight input format. The geological volume extends 700 meters in x direction (Easting), 500 meters in y direction (Northing) and 250 meters in z direction (thickness).The results from resources estimation gives the geological resource of 34.87 million tonnes with the respective average ore grade at 0.58 ppm. The geological block model is constructed with the block dimension of 12.5m. x 12.5m. x 5m bringing to the total of 17,421 blocks. The ultimate pit is designed using Lerchs-Grossmann algorithm providing the mine mineable reserve at 2.44 million tonnes at the average grade of 1.53 ppm. This mine is schedule with 4 years of mine life at the average annual production capacity of 600,000 tonnes. For material handling, it requires five trucks, one wheel loader and one excavator to handle the production of 160 t/hr. In terms of financial estimations, the NPV is calculated at 1.93 million USD, and IRR of 22 percent that is higher than WACC of 12.6 percent. For the environmental mitigations, it is suggested to follow the Lao National Environmental Standard to control the noise, dust, vibration and some recommendation for blasting practice. In summary, this gold mine project is proven feasible taking into consideration of technical, financial and environmental issue.

Department:	Mining and Petroleum Engineering	Student's Signature	....
		Advisor's Signature	....
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CHAPTER I
INTRODUCTION

1.1. Background

Laos is located in South East Asian with an abundance of natural resources such as forests, rivers, fertile land and mineral deposits. Nowadays, Laos has become well-known for its mineral resources such as gold, copper, lead, iron, coal and limestone. There are a number of enterprises which invest in mining in Laos, such as Sepon Company, Phubia Mining Company, and Twenty kilometer Gold mining. As a developing country, Laos need an upgrade for infrastructure especially roads, residential housing, hospitals and schools. For this reason, infrastructure development is playing an important role in every aspect of the development in Laos.

Since 1986, Lao PDR has carried out a comprehensive renovation policy, shifting from a centralized to a market-oriented economy. This means implementing market mechanisms, opening up the country, and cooperating with foreign countries. Currently, Laos has progressed in many aspects, especially in mining industry. To get the most out of this industry, it is essential that the country need to upgrade infrastructure concerning mine operations. This is a significant factor that encourage Lao's government to increase the cooperation with many organizations in the mining industry. Due to the fact that Laos has an abundance of precious mineral resource but lack of experience in the mining development. Hence, the Lao government recognizes the opportunity in this section and allows many mining companies to invest in Laos, notable example is Lanxang Mineral Limited which is now extracting copper is Sepon Mineral District (SMD).

Sepon is located in Vilabouly district, Savannakhet province in the South-Central part of Laos. The Sepon project comprises a 1,947 square kilometer located approximately 40 kilometers north of the town of Sepon, as shown in Figure 1.1. Gold and copper deposits at Sepon contain over 4.75 Moz of gold and 1.4 Mt of copper. Seven gold and two copper deposits exist within a distance of 5 km and comprise the Sepon Mineral District. The Sepon gold mine commenced operations in late 2002 and produced 156,000 oz of gold in the first year. Expansion of the gold plant and construction of the adjacent copper plant were completed in late 2004 and early 2005, respectively. Current gold resource stands at 83 Mt @ 1.8 g/t, and 4.75 Moz Au at 0.5 g/t Au cut-off grade. While copper resources comprise 27 Mt @ 4.3 percent, and 1.14 Mt Cu at Khanong, and copper reserve at 9.8 Mt @ 2.3 percent and 230 Kt Cu at Thangkham South at 1.0 percent Cu cut-off grade.

Source: http://www.smedg.org.au/Tiger/Sepon.htm

Figure 1.1 Location of Sepon Project.

1.2. General Information of Sepon District

1.2.1. Location, Geography and Access to Sepon Mineral District

Laos is a small landlocked country, located in the center of Indochina. Lao PDR has a population of 6.9 million (statistics of 2015), with an area of 236,800 km^2. It shares a border with China, Cambodia, Vietnam, Thailand and Myanmar. The capital city of Laos is Vientiane. It is also the main government administrative center. There is Wattay national airport in Vientiane which provides flights to Thailand, Vietnam and China. The Mekong river forms most of the Eastern border of Laos and is a transportation route along the length of the country.

This section discusses Sepon Mineral District geomorphology and it is general information. Sepon Mineral District is set at an elevation of 250 m above sea level. It is based in the Sepon Basin, along the Southern boundary of the NW-trending Truong Son fold belt in the South central part of Laos. It is located at longitude 105°59’E and latitude 16°58’N. The distance from Sepon district to the copper/gold deposit is 40 km North of the town and 130 km East of the provincial center of Savannakhet province. Access to the Sepon project is either by a direct 90 minutes company charter flight from Vientiane scheduling 6 - times per week, or by road starting from the provincial town of Savannakhet taking about two hours. Actually local people commute by bus service at the Sepon project, only few use their own vehicle to the project site.

Vilabouly is located on a mountain belt with 20 - 25 percent of flat terrain area. From Top view, it looks like an alternation basin and mountain range. The Xanghae mountain in the West, Kasard ridge mountain in the Center and Koi mountain in the South East, Katon mountain in the South and Pouluang mountain in Boualapha district in the South. Vilabouly district was separated from Sepon district in 1990 due to the fact that Sepon district covers big area and difficult for administration. Therefore, it is separated from Sepon in order to be more easily managed. The name Sepon district was used before the separation. The gold and copper project is located only 5 km away from Vilabouly district.

The main tourist attractions in the Sepon district are Pathad Ngang Lao stupa, Ongsaen cave in Nateu village, deity sink in Nateu village, the natural limestone rocky mountains in Nayom, and the nature of Xanghae mountain, and beautiful big rocks and Nangeuk rice paddy fields in Vilabouly district.

There are 13 regions in Vilabouly district and 8 - 10 villages (statistics of 1999). It is made up of 39,000 population with 70 percent of Puthai tribe, and 30 percent of Laotheung.

The local tradition and culture is characterized by speaking and culture and traditional like Phuthai character. They are mainly Buddhist.

The roads are not permanent as they affected by the weather. Travel in the rainy season is very difficult. There are number of main roads which are road # 28A, road #10, and road #9. Road #28 A (Ho Chi Minh Road) leads to Bualapha district, Xaibouathong district and Thakek district. Road #10 separates from road # 28A to Nasalor village - Paknao village - Kokkataiy to Seno. Road #9 leads to Vilabouly district and to Boualapha district. In the past during the war, road #9 was called Ho Chi minh Trail. Nowaday, road #9 to Vilabouly village is very convenient for travelling. The main occupation here is agriculture, such as rice farming, mountain rice fields, animal husbandry likes buffalos, ox, goat, chicken and other animals.

1.2.2. Climate

The climate in Villabouly district is mainly comprised of two seasons; the rainy season and the winter season. The rainy season starts from June to September and winter starts from October to January.

According to the statistics, the maximum rain intensity is during July to September with an average rainfall of 350 mm - 500 mm. The heaviest rain was recorded in July, 1994 at 820 mm; and again in 2002 of 990.4 mm. which was the highest in the ten years recorded.

1.2.3. The Main River in Vilabouly District

Sekok and Sengy rivers are the most important rivers of Vilabouly district. Sekok river originates from the mountain top and flows to Sebanghieng river in Sepon district. Sengy river is a branch of the Sekok river, it is headwater locates in the East of Pusok mountain, and flows to Sekok river at Meuangkao village near Phapayor.

These rivers can be accessed by boat during the rainy season. The Sekok river can be travelled by boat from Sepon up along the river until Thongluang village and also Houayhong village. For the Sengy river, it is possible to travel by boat as far as Meuangarngkham village and Sobsalong village. The Sengy viver and Sekok river flows to the South.

All of these river tributaries are very important for local people living in the region. Local people pan for gold by using gold pans and water streams to wash sediment away and leave only gold at the bottom of the pan, as it shows in Figure 1.2. These rivers are Sekok river, Sengy river, Sebai river, Sebanghieng river and small branch like Semala river, Houaynampa river, Houaynammy river, Houaykhieng river, Houaybang river, Houaynamkhang river and other small tributaries.

Source: http://www.rfa.org/english/news/laos/gold-01282014150414.html

Figure 1.2 Panning for gold activity at Sepon District.

1.2.4. The Important Plateaus in Vilabouly Province

There are many plateaus in Vilaboulay district such as Nayom Bannamsang, Nakeekang, Nongkapang, Nameuangsan, Naxienglae, Nasou, Nakeekak, Napilang, Nammalou, Naharng and Dongyang plateau. Some plateau positions along the Senoy and Sebai river. And, many small plateaus located along the Sekok and Sengy river area are Nahoi, Vangthakhouay, Thongluang, Naluang-Nakaxin, Naloo, Nongkadaeng, Jorlor, Nammahy, Vangmahang, and Banmeuangarngkham plateau.

1.3. Geology of Gold and Copper Mineralization at Sepon Mineral District

1.3.1. Geology and the Structural Geology

Copper-gold mineralization at Sepon region is based on the evaluation of Indochina plate through geologic time resulting in a favorable condition for copper - gold deposit. The structural control mineralization process in here is the normal fault with an advance inclination to vertical dip, which made a favorable condition for hydrothermal fluid to move along the fault plane and their surrounding rocks within the region. In addition, this region is crossed by strike slip fault, which is also provide a favorable condition to copper - gold mineralization and their associated minerals such as malachite, azurite, chalcocite and the others.

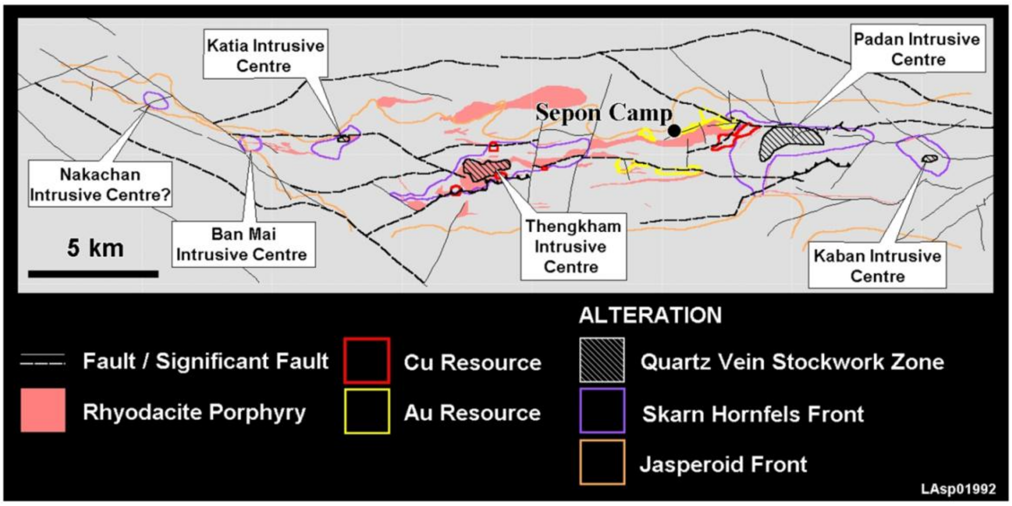
In-situ supergene enrichment of this sulphide resulted in the development of a high grade chalcocite enrichment blanket immediately beneath, which lies a very high grade copper oxide zone comprising malachite ± azurite ± cuprite ± minor native copper (Loader, 1999).

1.3.2. The Characteristics of the Surrounding Rocks

The surrounding host rocks are disseminated with copper-gold minerals. In Sepon region, host rocks compound with calcareous sedimentary rocks such as calcareous shale, dolomitize limestone, and calcareous siltstone.

1.3.3. The Characteristics of Volcanic Rocks

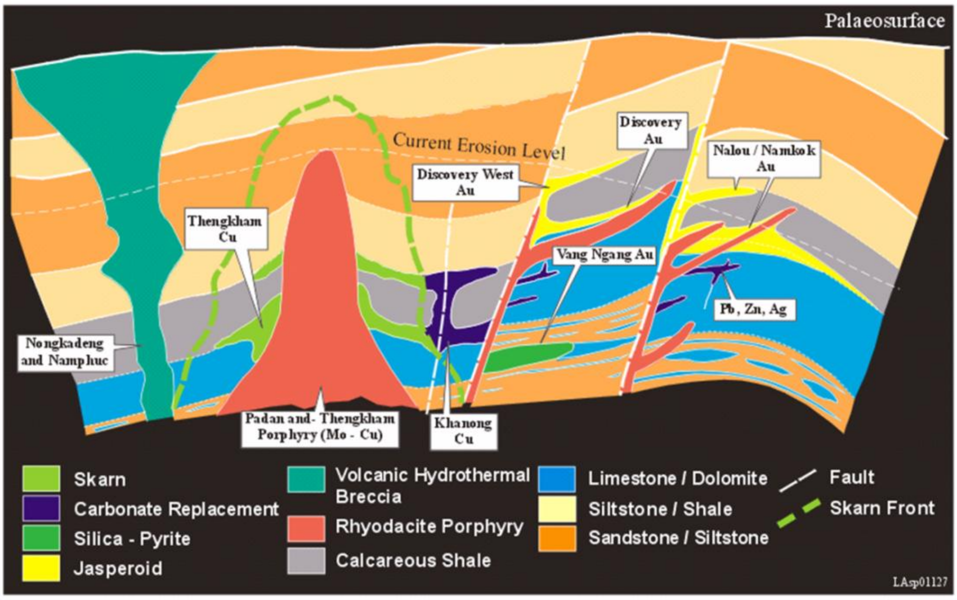
Sepon Mineral District is controlled by igneous intrusive center at Thangkham Mountain and Phadang Mountain as shown in Figure 1.3. The characteristic of this igneous intrusion is CA series (Calcite Alkaline series), S - type, and the spot structure of Rhyodacite Porphyry. This geology factors in addition to the hydrothermal alternation significantly control the process of Cu - Au mineralization. The alteration processes and its mineralization are as followed;



Source: https://www.google.co.th/imgres?imgurl

Figure 1.3 Padang & Thengkham Intrusion Centers.

- Cu - Au - Mo in porphyry intrusion center are altered by alkali feldspar in plagioclase with disseminate sulfide.
- Cu - Au skarn is altered by garnet and pyroxene in prograde skarn stage and chlorite epidot with disseminate sulfide in retrograde skarn stage.
- Au sediment hosted carlin type is altered by silica which is called jasperoid with disseminate pyrite.



Source: http://www.smedg.org.au/Tiger/Sepon.htm

Figure 1.4 Schematic model of mineralization in the Sepon Mineral District (after Sillitoe 1990).

This research work utilizes the geological information, gathered from Sepon Mineral District. Such information includes basin geology, structural geology, drillhole information, theirs lithology and assay data, to construct the geological block model. The geological block model provides a basis for resource/reserve evaluation, and finally the mine design and operation stage.

1.4. Statement of the Problem

- Owning to a strong potential of Cu - Au mineralization at Sepon Mineral District, it is important to carry out the pre-feasibility study of the gold mine development project in order to evaluate the resource/reserve potential, pit optimization and mining development, and its overall economic assessment.

- According to the increasing domestic as well as global demand of gold commodity, the gold mine development in Laos provides an economic incentive in boosting the country revenue. Nowaday, mining industry is playing more and more important role in Laos growing economy. Therefore, the feasibility of the gold mine development is deemed important in the moment, as it can assess the full potential of gold deposit of interested in light of technical, economical, and social and environmental considerations.

1.5. Objectives

The objectives of this research work are as followed;

1. To construct the geological block model which provides the basis for resource/reserve estimation, which will be used as inputs in the stage of mine optimization, mine design and operation.
2. To develop a financial model using Discounted Cash Flow (DCF), and the criteria of Net present Value (NPV), the Internal Rate of Return (IRR), and the Weight Average Cost of Capital (WACC).
3. To address the environmental and social impacts from mine operation and proposed some mitigation guidelines.

1.6. Usefulness

In this study, it will benefit the open pit mine development in the following;

1. It provides the framework for gold mine pre-feasibility study in Laos by taking a full consideration of local factors (infrastructure, cost factor, environmental issue, etc.)
2. It illustrates the technical requirements and designed parameters for gold mine development, such as resource/reserve estimation, mine optimization, mine planning and design, mine operation including drilling, blasting, loading and hauling, and mine environment.

CHAPTER II
LITERATURE REVIEW

2.1. Literature Review

This chapter presents the literature survey related to this research study.

- Rex Bryan et al. (2013) carried out pen pit operation feasibility study of Pumpkin Hollow Copper Project at Nevada Copper Corp. The project is located approximately eight miles South East of Yerington in Lyon country. They are two projects: underground and open pit operation. A optimum pit design used the moving cones as guidelines to exploit the minable reserve. The mine is designed with the bench height of 25 feet, haul road of 150 feet width, and the ultimate pit utilizes switchback haul road to maintain the road and ramp. Open pit mining operation has an estimated mine life of 22 years started from 2012, at 0.2 percent COG copper. The operation is 70,000 tpd with milling rate of 63,000 tpd. The Pumpkin Hollow FS plans two open pit areas designated as North and South pit. The open pit mine design were developed using GEMS Whittle pit optimization software. The results from the economic analysis are; mine life of 22 years, pre-tax NPVof $961 million, pre-tax IRR of 20.2 percent, post-tax NPV of $ 726 million, post-tax IRR of 17.9 percent, payback (post-tax) of 52 months, and federal income tax paid of $162 million.

- Kem (2013) studied the pre - feasibility study of limestone quarry for cement industry in Cambodia. The research area is located at Prey Ta Pret village, Banteay Meas district, Kampot province, Cambodia. The research focusses mainly on three objectives. First, to construct geology block model in order to estimate the potential resource. Second, to formulate quarry development model including ore model, mineable reserve estimation, pit design and mine scheduling. Third, to evaluate the financial model, such as NPV, IRR, DCF, and WACC of the project. The geological block model generates 560,000 blocks with block discretization of 15m x 15m x 10m in (x,y,z) dimension. The geological resource of 204 million tonnes is estimated by using the inverse distance square method. The pit optimization using the Lerchs-Grossmann method yields the selected ultimate pit with 30 million tonnes mineral reserve. The mineable reserve with capacity of one million tonnes per year of limestone production covering 25 years mine life. For the material handling, it requires 5 trucks, 1 loader and 2 excavators to handle 550 ton per hour of limestone production. In terms of financial analysis, 19.3 million USD of NPV is estimated with the IRR of 45 percent, that is higher than the WACC of 15 percent. For environmental consideration, the maximum noise and ground vibration level recommended for blasting are 115 dB linear and 5 mm/s (peak particle velocity), and the air pollution control for silica should not greater than 0.2 mg/m^3. The final mine closure is planned to convert the final pit into a water reservoir which benefit to local communities.

- The Gemell mining engineers reported the Technical Review of Chatree Gold Project, Akara (2013). The Akara mining operates gold mine at Pichit province, Thailand. The cutoff grade is designed at 0.32 g/t (Au) for gold price of US$ 1,600 per ounce. In the beginning, the pit boundaries were determined by applying Whittle 4X optimization software to the resource model. The resource estimation is done by Multiple Indicator Kriging (MIK), a non-linear geostatistical technique. For the pit optimization, the block model was constructed with blocks dimensions of 29 m (east) x 25 m (north) x 6 m (elevation). The bench height is 9 meter with 3 meter flitches, the hole diameter is 102 mm to 115 mm, and mine life of minimum 10 years. According to the financial year 2013, 5,699,013 tonnes of most hard and soft ore were processed. The current long term mine plan indicates that about 26 percent of future waste production will be NAF. Dust and noise preventions are both achieving the ISO 9001:2008 (quality), ISO 14001:2004 (environmental), and OHSAS 18001:2007 (occupational health and safety) standards.

- Carl Murray (2008) reported the pre-feasibility of mining development of Weld Rang iron ore project. It is located in Western Australia some 600 km N-NE of Perth. This mine is designed as open pit mine operation with the project ore reserve of 225 Mt, and the Lerchs - Grossman algorithms was used to find the ultimate pit. This project adopted three software 1). Whittle™ for open pit mine optimization, 2). MineSight® for mine design and mine scheduling package, 3). AutoCAT® for drafting software. Mine life is calculated at 15 year. The mineral resource models with the block sizes of 40 x 10 x 12 m. and was re-block to the block size of 40 x 50 x 12 m. The production of this project was focused on three specification of mineralization. 1) Fe with average grade greater than 58.0 percent, 2) Al_2O_3 with average grade below 2.6 percent, and 3) SiO_2 with average grade below 5.5 percent. For equipment selection, they are 140 tonnes truck Caterpillar model 785D, excavator model Hitachi Ex2500-6 with 11 m^3 (27 t LCM) bucket size, and a combination of 8 excavators along with 64 haul trucks.

- Tt TETRA TECH (2010) studied the pre-feasibility study of gold mine project at Mount Todd Northern Territory, Northwest Australia. For mineral resource estimation, geostatistics block kriging techniques was used. This method is carried out statistical and geostatistical analysis of the gold assay data with the block size of 12 x 12 x 6 m; and the result of resource estimation is 94,008,000 tonnes with average cutoff grade of 0.4 g/t (Au). The total reserve is estimate at 24,458,000 tonnes with cutoff grade of 1.09 g/t (Au) and the stripping ratio of 2.37. For equipment selection, CAT 789C and CAT 785C trucks with payload of 140 tonnes and 180 tonnes are used, respectively. Hitachi EX3600 hydraulic shovel and CAT 992G wheel loader with bucket size of 21 m^3 and 12 m^3 are also selected. For economic analysis, this project gives pre-tax IRR of 14.9 percent, pre-tax NPV of $ 210.175 million, Cash flow analyses at US$ 950/ton (Au) and 5 percent discount rate. The mine life of this project is estimated at 15 years with annual product 6.77 Mtpy. To achieve the ultimate pit design, the Whittle software is adopted.

CHAPTER III
METHODOLOGY

The methodology adopted in this study presented in Figure 3.1. The process starts with the collection of geological information which is necessary for ore block model construction. The geological information, drillhole lithology are input into Minesight Software to define the composite data, ore domain and the IDWS resource estimation. Then, the following processes of pit optimization, mine planning and operation can be evaluated prior to the economic and environmental considerations.

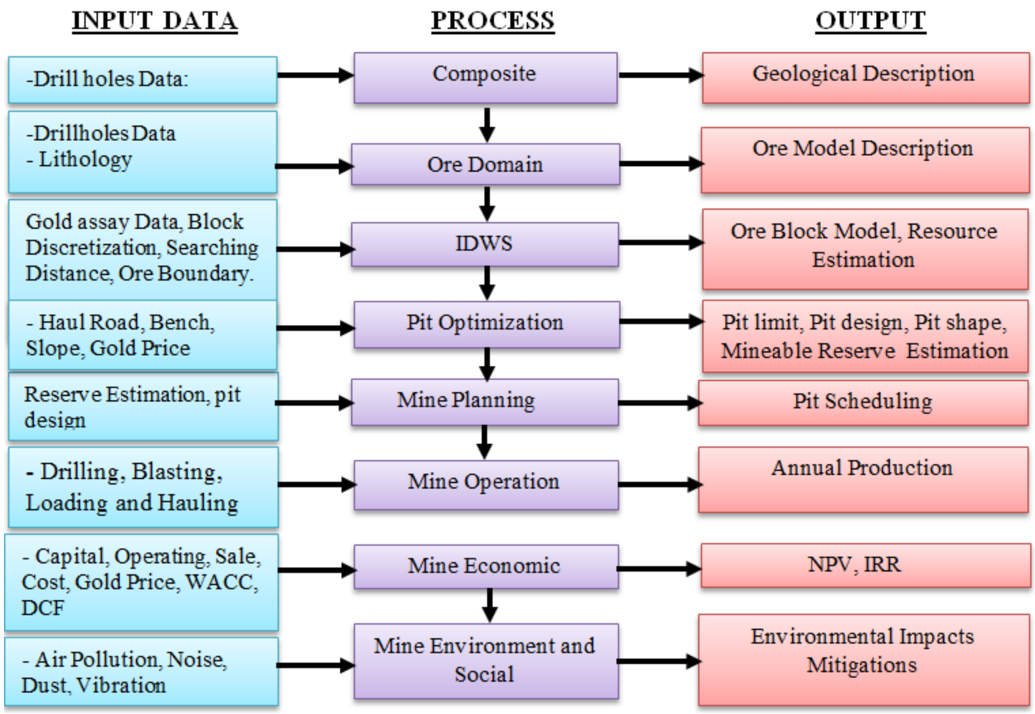


Figure 3.1 The flowsheet of the study.

3.1. Geological Model

The geological model reviews the 3D structure of the studied deposit. It is executed by inputting the geological information into the Minesight Software such as geological data, drillhole description data and other related information. The 3 D - description geological model is constructed from the incomplete information and is largely relied on the correlation technique along the drillhole data.

3.2. Resource and Reserve Estimation

The resource and reserve definitions are as the following;

Resource: mineral resource can be defined as the whole resource within the confine of geological boundary, whereas ore reserves are the parts of mineral resource that can be economically mined. Resource estimation is calculated based on an increase in geological knowledge, the exploration information. When the economics information is input to resource, then the resource estimation is converted to reserve estimation.

Reserve: It is defined as the part of resource that meet minimum physical and chemical criteria related to the specified mining and production practices, including those of grade, quality, thickness and depth. It can be reasonably assumed to be economically and legally extracted or produced at the time of determination.

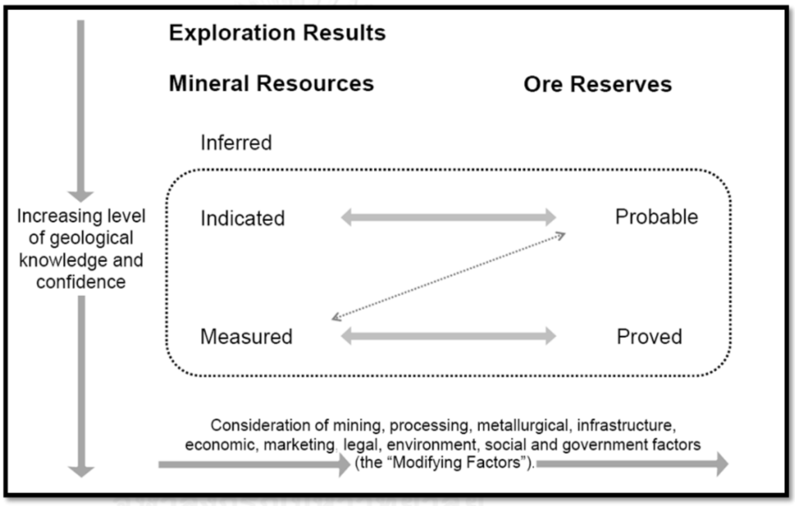


Figure 3.2 Relationship between exploration, resource and reserve (SME, 1991).

3.3. 3D Block Model

Nowadays, most of exploration and mining operations revolve around block model. In this model, all relevant deposit characteristics are organized in a transparent and manageable way. 3D block model is the simplified mathematical description of the deposit. The deposit is subdivided into small block that each block represents a planning unit which contains information about raw material properties such as chemical grade, rock type, geology, structural conditions, and etc.

As a rule of thumb, the minimum block size should not be less than 1/4 of the average drillhole interval, say 50 ft. block for a 200 ft. drilling grid and 200 ft. block size. for an 800 ft. drilling grid (David, 1977).

The height of the block is often related to the bench height which will be designed in mining operation. The location of blocks depends on many factors, for example overburden ore contact, the interface between types of mineralization (oxides-sulfides, transition and primary), high grade and low grade zones, and etc.

The interpolation techniques are used to assign ore grades to these blocks. The tonnage of each block can be easily found from the block volume (the same for all blocks) and the tonnage factor (which may vary).

The individual block grade values are usually assigned by interpolating drillholes assay data to the center of the estimated blocks by means of complex interpolation algorithms.

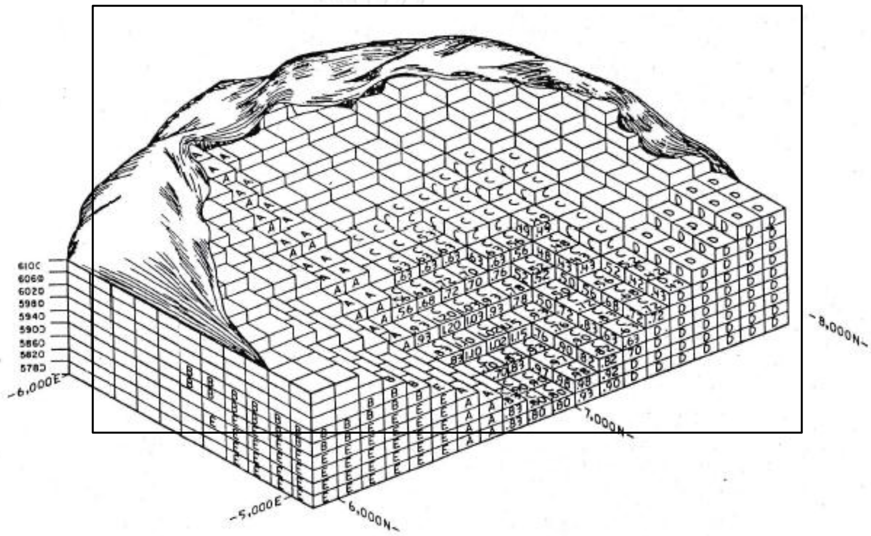


Figure 3.3 3D block model (Crawford & Davey,1979).

3.4. Inverse Distance Weighting Square method (IDWS)

The IDWS is the deterministic interpolation method for ore resources estimation. This method is based on Inverse Distance Weighting Technique. The influence of surrounding grades varies inversely with the distance separating the grade and the estimated block center. It is obvious that the grade of the block should be more similar to nearer points than those far away. To emphasize the dependency between the weighting and the distance, this is done by changing the power function of d_i in Equation 3.1.

$$g = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i^m}}{\sum_{i=1}^{n} \frac{1}{d_i^m}} \qquad (3.1)$$

where,

g = the calculated weight

d = the distance between samples and block center

m = power function

As shown in relationship 3.1, the power function (m) is selected based on the characteristics of the mineral deposit. In practice, the less variation ore grade deposit will adopt the power function of 1. On the contrary, the power function of 2 is selected for the high variation ore grade deposit.

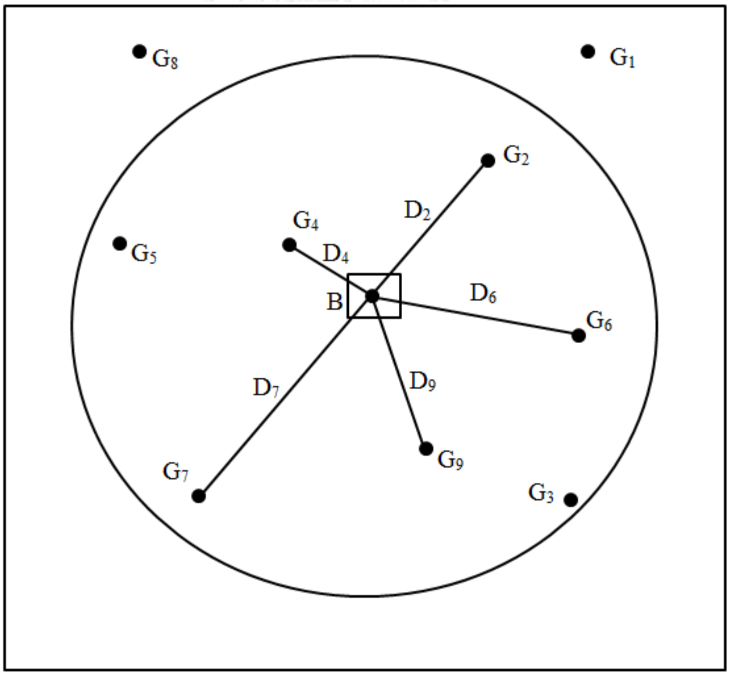


Figure 3.4 IDWS block calculation.

As shown in Figure 3.4, the distance of sample points to the center of the estimated block will be used to calculate their weights. The linear combination of sample grade and weight gives the estimated block grade. The sum of all weights must be equal to one in order to ensure its unbiaseness.

3.5. Pit Optimization

Pit optimization uses Lerchs - Grossmann (LG) 3-D Algorithm to evaluate the final pit dimensions based upon reserves expressed in the form of grade block model. The overall objective of LG pit optimization is to find the selected blocks based on the constraints, and maximizing the profit, metal content, and marginal value. This is a 3-dimentional problem to gain a true optimum condition of ore blocks. For an orthogonal set of ore blocks, there exist two basic geometric configurations for accessing the selected blocks (Hustrulid and Kuchta, 1995). They are:

- the 1-5 pattern, where 5 blocks must be removed to gain access to one block on level below them, and
- the 1-9 pattern, where 9 blocks must be removed to gain access to one block on the level below them too.

The geometric configurations and equivalent graphic representations for these two geometries are shown in Figure 3.5. The nodes represent the physical blocks. The arrows (directed arcs) point toward those blocks immediately above which must first be removed before underlying block can be mine. Each block has a weight associated with valuable grade that can be the net economic value. The weight may be either positive or negative based on their economic merit.

Lerchs & Grossmann (1965) published the basic algorithm which when applied to the 3-D block model would yield the optimum final pit outline. Figure 3.5 illustrates the basic 3-D block model.

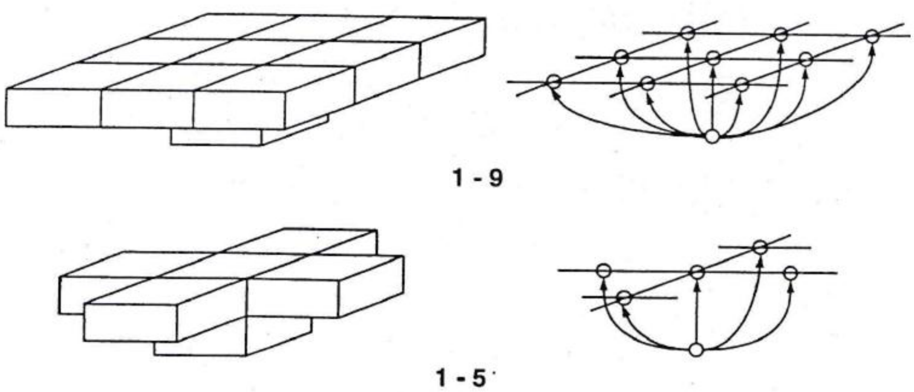


Figure 3.5 Representation of the 1-5 and 1-9 block constraints (Laurent et al., 1977).

3.6. Basic Bench Geometry

Many ore deposits are excavated by open pit technique which vary considerably in size, shape, orientation and the depth below the surface. Ore body is excavated from the top horizontal to bottom horizontal in a uniform layer called ‘bench’. Mining starts from the upper most bench until it is completed, and then moves to the lower benches. Roads and ramps are created to connect all bench levels for transportation trucks, cars, and dozers.

Each bench has an upper and a lower surface separated by ‘H’, the bench height. The exposed sub-vertical surface is called the bench surface which is described by toe and crest, face angle (the average angle of bench face is made against the horizontal distance). Creating a bench needs to consider the orientation and blasting practice. In general, a bench slope angle in hard rock varies from 55°- 80°. The typical initial design value might be 65°. This should be carefully designed since the bench face angle can have a significant effect on the overall slope angle.

The excavated bench produces the bench floor. The width of the bench is the distance between the crest and the toe measured along the upper surface. The bench width is a horizontal projection of the bench face as can be seen in Figure 3.6.

There are many types of bench, the bench which is being mined is called the process of pit. The bench width is exposed from working bench is called cut. The width of working bench B_w is defined from crest of bench floor until new toe position after the cut has been extracted as shown in Figure 3.6.

The purposes of bench are

- to collect the broken material sliding down from the upper bench, and
- to stop downward movement of boulders.

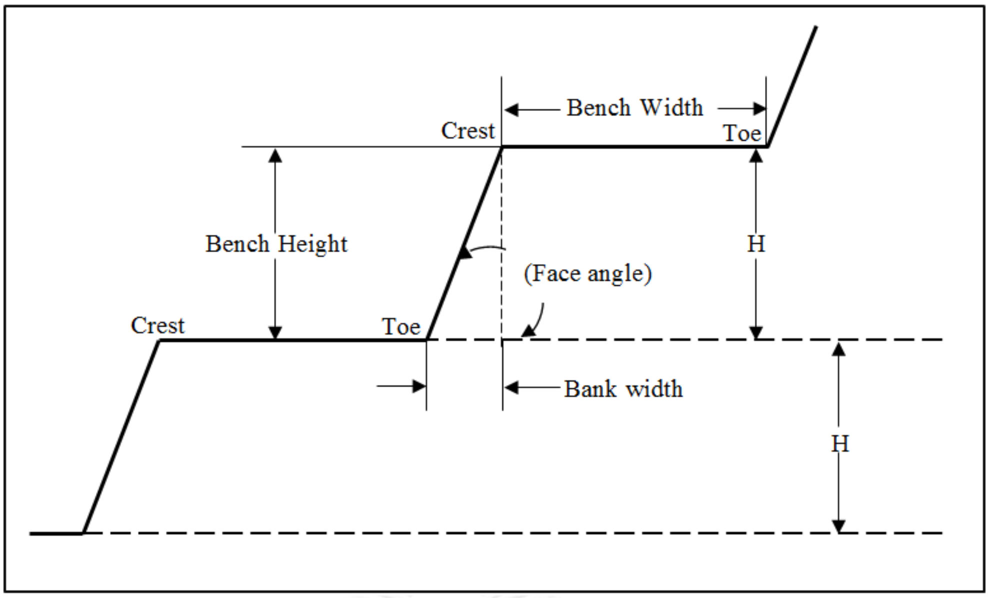


Figure 3.6 Bench geometry (Open pit mine planning and design, Vol1, 1995).

3.7. Mine Operation

There are four principals in mine operation, drilling, blasting, loading and hauling systems. These are important mine operations especially in hard rock mining.

3.7.1. Drilling and Blasting

This is very important mining operation. In open pit mining, drilling and blasting are the first activities for purposes of breaking down hard rock material. To achieve this aim, it is necessary to drill holes and filled with explosives so as to break the rock into small pieces before loading onto transport equipment, and sending to other processes. There are many type of drilling equipment as illustrated in Figure 3.7, such as rotary systems, percussion systems, and a combination of rotary and percussion system.

Figure 3.7 Drilling machine.

Drilling and blasting are the initial activities in mining operations, which will affect the loading and hauling processes in the mine. For efficient drilling and blasting, it is obviously important to consider the structure of the rock and the bench design. This, in turn, relates to the hole pattern, the hole diameter, explosive types and the firing system. It is necessary to record all the related processes in order to adjust and evaluate from the previous results for the next round. The optimum blast control categories are shown in Figure 3.8.

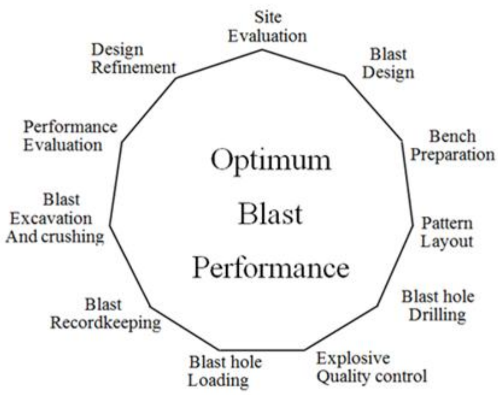
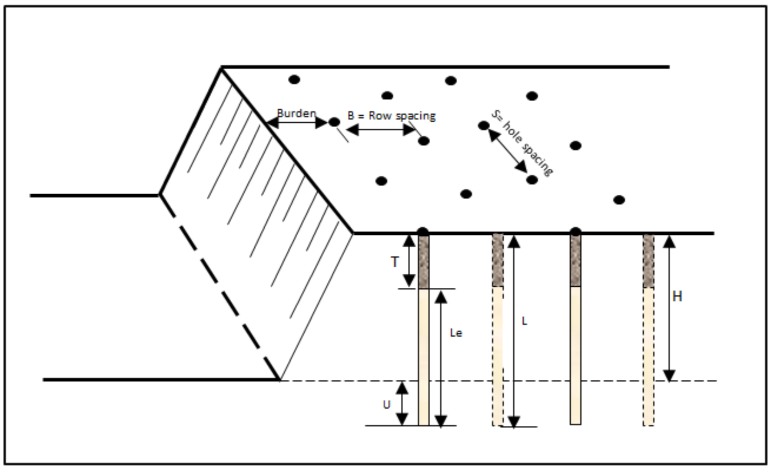


Figure 3.8 The procedure of optimum blast performance.

Blasting patterns important design criteria are burden, spacing and the hole depths as shown in Figure 3.9. These depend on the characteristic of the rock and the value of the material that needs to be blasted for instance, waste or ore.



B - Burden
S - Spacing
U - Sub drilling
T - Stemming distance
H - Bench height
Le - Charge distance
L - Hole depth

Figure 3.9 Show drill holes pattern.

Table 3.1 illustrates the relationship between explosive density to blasthole diameters which are used in open pit mine design. The bench height is related to hole diameter and other measurements. These parameters can be pre-determined to suit blasting design. In quarry and open pit mine operations, efforts are made to fix the bench height in relation to blasthole diameters(Jukka, 1987-88), as shown in Figure 3.10.

Table 3.1 The explosive density and blasthole diameter (Blaster's handbook, 1989)

Blast hole size	inch	1.5	2	2.5	3	3.5	4	5	6
	mm	38	51	64	76	89	102	127	152
AN-FO (kg/m)		0.9	1.6	2.6	3.6	5.0	6.5	10.1	14.5
Emulite 150 (kg/m)		-	2.3	3.7	5.0	7.1	9.3	-	-
Bulk emulate (kg/m)		-	2.4	3.9	5.3	7.5	9.9	15.3	21.9
Dynamex M (kg/m)		-	2.6	4.0	5.6	7.8	10.2	-	-

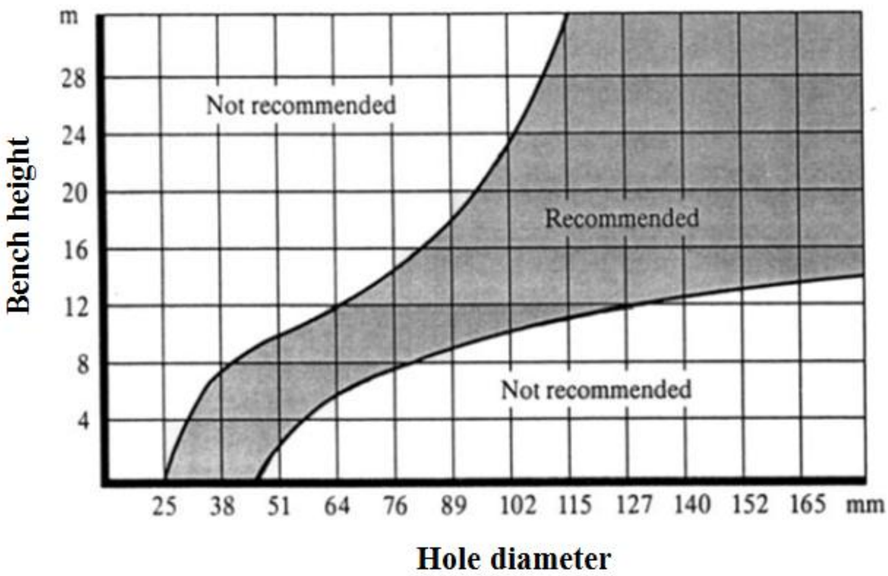


Figure 3.10 Determination of drillhole diameter for various bench heights (TAMROCK, 1987-88).

3.7.2. Loading and Hauling

In surface mining operations, loading is a significant activity for transporting waste or ore onto trucks from the mine to the processing plant. This operation requires the loading equipment which is appropriate to the bench geometry and trucks capacity. There are many types of loaders such as hydraulic excavator, backhoe excavator, dragline and front-end loader (wheel loader). The equipment selection in surface mining depends on the type of mineral to be extracted and specifications of the mine geometry, such as bench height and other factors. Figure 3.11 illustrates the loading and hauling in open pit mine operation.

Figure 3.11 Loading and hauling system.

The selection of mine machinery for loading and hauling system is very important for efficient production. The selection criteria mainly make based on the conditions of the mine such as the slope angle, the distance, the climate and also the communities surrounding the mine.

For truck selection, the number of trucks selected should be based on overall economic system, ensuring lowest operating and capital costs that fit with target capacity. This section estimates the cycle time for loading which is a function of bucket size, fill factor, excavator, cycle time, loose material density, and truck capacity. Other fixed times depend on loading methods and dump configuration such as spot and maneuverability in loading areas (typically 6 - 8 min), and dumping (typically 1-1.2 min). The unit production calculation considers truck payload, truck cycle times, hours per shift, and operating efficiency.

Therefore, it is essential to evaluate the number of wheel loaders and trucks in order to handle ore transportation to the processing plant. The wheel loader production can be determined from this relationship below.

$$O = \frac{BC*BF*D*MA*JF*3600sec}{(1+SF)*CT} \quad (3.2)$$

Where,

O = Production, tons/hour, (t/hr)

BC = Bucket size, cubic yard (cy) (usually heaped at 2:1)

BF = Bucket fill factor, %

D = In place density, tons/cy

MA = Mechanical availability, %

JF = Job factor, %

SF = Material swell, %

CT = Average cycle time, seconds

The number of truck can be estimated by this equation,

$$N_{truck} = \frac{truck\ cycle\ time}{loading\ time} \quad (3.3)$$

Where,

N_{truck} = the number of truck.

3.7.2.1. Cycle Time of Truck

The truck cycle time comprises loading time, hauling time (full travel), dumping time, returning time (empty travel), queuing or waiting time (and other delays) and spotting time. Generally, a cycle time usually begins at a loading site where the truck receives its load from excavating equipment. The truck travel full of ore to dumpsite by designated route along a haul road. The dumpsite may be a stockpile or waste dumpsite. After dumping the material to the stockpile, the truck turns around and travels empty back to the loader again. Spotting is the act of maneuvering the truck under the loader for serving, this can take several minutes. In large mine, the truck cycle time may be 20 - 30 minutes in total and can vary significantly over time if the stockpile move and as the mine deepens (Burt and Caccetta, 2013). Figure 3.12 illustrates the cycle time of the truck from the loader to dumpsite.

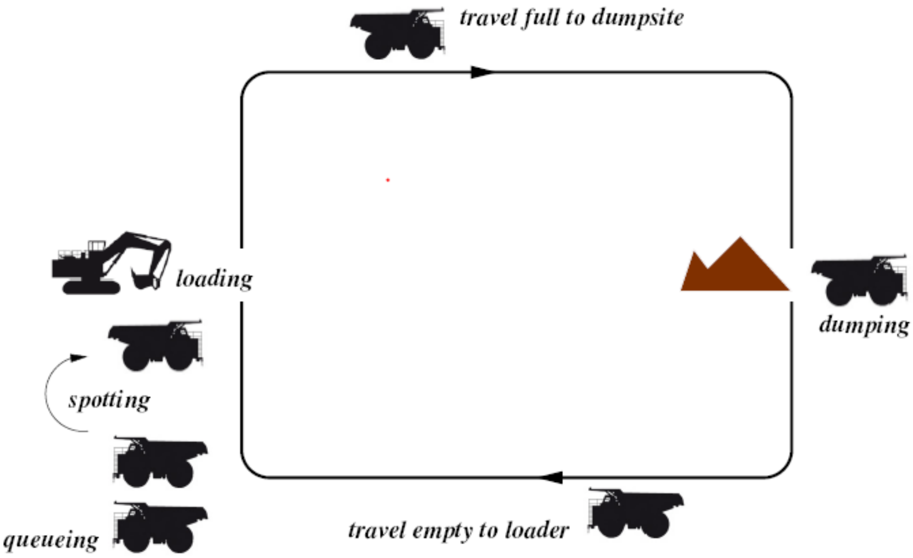


Figure 3.12 Cycle time of truck (Kennedy, 1990).

The average cycle time is another parameter to determine the production rate in each cycle time of equipment. Table 3.2 illustrates the cycle time that increase with the size of equipment (Caterpillar, 1979-2004).

Table 3.2 Average cycle time (Caterpillar Performance Handbook, 1979 - 2004)

Loader size (cubic yard)	Cycle time (minute)
1.7 - 4.5	0.45 - 0.50
5.0 - 7.5	0.50 - 0.55
7.5 - 11	0.55 - 0.60
15 - 21	0.60 - 0.70

3.7.3. Stripping Ratios

One method describing the geometrical efficiency of mining operations is through the use of the term “stripping ratio”. It refers to the amount of waste which will remove to release a given ore quality. The ratio is most commonly expressed as;

$$SR = \frac{Waste\ (ton)}{ore\ (ton)} \qquad (3.4)$$

Where,

SR = striping ratio

For most mining projects, a large amount of waste in the early mining years of the project is not of interest.

3.7.4. Cutoff Grade (COG)

Cut-off grade is defined as the grade that is normally used to discriminate between ore and waste within the given deposit. For open pit mining, the definition of cut-off grade was based on the quality of the ore. Cut-off grade is the criterion that specifies the amount of ore and waste. The material with grade equal to or higher than the cut-off grade is classified as ore. The material with grade less than cut-off grade is considered was waste.

In generally, the use of COG is considered by two characters such as

1). Technology COG: it bases on production technology such as mining and processing technology.

2). Breakeven/Economic COG: it employs economic aspects as profit or loses to determine the cutoff grade. In mining operation, it can classify as Mining COG and Milling COG that depends on time period and many factors such as price of ore or revenue, capital or expenditure, and the productivity

The formula below identifies cutoff grade associated with their cost structure.

$$\text{COG} = \frac{c+m}{(P-S)*Y} \tag{3.5}$$

Where,

P - Price ($/oz)

S - Sales cost ($/oz)

c - Processing cost ($/t)

y - Recovery (%)

m - Mining cost ($/t)

g_B - Block grade (g/t)

BV - Block value (USD)

The ore block value (BV) is also defined as;

$$BV = (P-S)*g_B*y - c - m \qquad (3.6)$$

Where,

P - Price (\$/oz)

S - Sales cost (\$/oz)

c - Processing cost (\$/t)

Y - Recovery (%)

m - Mining cost (\$/t)

g_B - Block grade (g/t)

BV - Block value (USD)

For the waste block BV = - m

Noted; If Grade > COG, the block is considered as ore. If Grade < COG, the block is considered as waste.

3.8. Mine Economic

Discounted Cash Flow (DCF) Model is generated in this study which providing Net Present Value (NPV), Internal Rate of Return (IRR), and the Weight Average Cost of Capital (WACC). These economic models will be discussed intern.

Discounted Cash Flow Model

The cash flow is referred to as net inflow or outflow of money that occurs during a specific time period. The elementary cash flow calculation is:

- Gross profit = Gross revenue - Operating expenses
- Net profit = Gross profit (taxable income) - Tax
- Cash flow = Net profit - Capital cost

The discounted cash flow method is widely accepted and used in the industry for all types of capital investment evaluations. The discounted cash flow method recognizes the time value of money. This is critical when assessing the profitability of long term investments. Future and past values of money can be converted into their present value equivalent by using the time value of money concepts. The model provides many economic evaluations as the following:

3.8.1. Net Present Value

The Net Present Value (NPV) also refers to the present value of cash surplus or present worth, which is obtained by subtracting the present value of periodic cash outflows from the present value of periodic cash inflows. The present value is calculated using the Weighted Average Cost of Capital (WACC) of the investor, also referred to as the discount rate or minimum acceptable rate of return.

When NPV of the investment at a certain discount rate is positive, it pays for the cost of financing the investment, or the cost of the alternative use of funds. If the investment generates revenue is equal to the positive present value, it also implies the rate of return on the investment is at least equal to the discount rate. The net present value method of evaluating the desirability of the investments is mathematically represented by the following equation:

$$NPV = \frac{S1}{(1+id)} + \frac{S2}{(1+id)^2} + \frac{S3}{(1+id)^3} + \cdots + \frac{Sn}{(1+id)^n} - Io \qquad (3.7)$$

$$NPV = \sum_{t=1}^{n} \frac{St}{(1+id)^t} - Io$$

$$NPV = \sum_{t=1}^{n} \frac{NCFt}{(1+id)^t}$$

Where,

St - The excepted net cash flow (gross revenue-LOE-tax) at the end of year t

Io - The initial investment outlay at time zero

id - The discount rate

n - The project's economic life in year "n"

NCFt - Net cash flow at the end of the year t

If the NPV is positive, the proposal is accepted.

If the NPV is negative, the proposal is rejected.

If the NPV is zero, the analyst will be indifferent because the proposal is generating the same return as the alternative use of funds will generate.

3.8.2. Internal Rate of Return

Internal Rate of Return (IRR) is another important economic parameter reported measure of profitability. IRR is reported as a percentage rather than a dollar figure such NPV. IRR is the discount rate at which the net present value is exactly equal to zero, or the present value of cash inflows is equal to the present value of cash outflows. Another definition of IRR is the interest rate received for the investment consisting of payments (negative values) and income (positive values) that occur at regular periods. The equation to calculate IRR is shown as below:

$$\sum_{t=1}^{n} \frac{NCFt}{(1+IRR)^t} = 0 \qquad (3.8)$$

Where,

NCFt = Net cash flow at year t

The role for making the investment decision when using IRR are:

- Accept the investment, if the calculated IRR is greater than the return on the alternative use of funds or cost of capital.
- Reject the investment, if the calculated IRR is less than the return on the alternative use of funds or cost of capital.

Discount rate that using in this present value calculation derived from the Weighted Average Cost of Capital (WACC). The WACC calculations are all capital sources, including common stock, preferred stock, bonds, and any other long - term debt. WACC is calculated by multiplying the cost of each capital component by its proportional weight.

The WACC is quantified from the following relationship.

$$\text{WACC} = \frac{D}{V} *(R_d)*(1-T_c) + \frac{E}{V} *(R_e) \qquad (3.9)$$

Where,

R_e - Cost of equity

R_d- Cost of debt

E - Market value of the firm's equity

D - Market value of the firm's debt

V = E + D: firm value

T_c = Tax rate.

WACC is used in discounting cash flows to calculate NPV, equity interest and other evaluations for investment analysis. WACC represents the average risk faced by the organization. It would require an upward adjustment, if it is used to calculate NPV of project which poses more risk than the company's average projects, and a downward adjustment in case of less risky projects.

3.9. Mine Environment and Society

Today, there is a strong measure of government control and inspection of mines under registration specific to the mining industry that is intended to safeguard the health and safety of the miners (Anon, 1977).

In open pit mining operation, environmental problems are dust, vibration and air blast pollution, noise pollution, visual pollution and air pollution. These problems occur during the mine operations such as drilling, blasting, loading and transportation. Noise from blasting is the main issue which can disturb the comfort of people who live close to the mine. It makes residents frightened and disturbs their relaxation time. Vibration from blasting can damage houses and buildings. Besides this, dust and air blast pollution generate from blasting and travelling of trucks in mine, the problem can decrease by using water spray along the transported road.

For blasting design, to minimize the impact from ground vibration to houses, buildings and historic sites in general, the scale distance should not be less than 70 feet/√pound, and should be not less than 120 feet/√pound for the sensitive historic site. Table 3.3 displays the relationship between the amount of explosive allowed to the required safety distance.

Table 3.3 The relationship between the amounts of explosives allowed to the required safety distance

SD (feet)	W70 (pound)	W120 (pound)
300	90	31
500	249	85
700	489	166
1,000	998	340
1,200	1,437	489
1,500	2,245	764
2,000	3,992	1,358
2,500	6,237	2,122
3,000	8,982	3,056

Note:
SD = Safe distance (feet)
W70 = the amount of explosive allowed to ignite at once 70feet/√pound
W120 = the amount of explosive allowed to ignite at once 120feet/√pound

3.9.1. The Vibration Safety Standard of Australia

The sensitivity of ground vibration to human feeling from blasting is tabulated in Table 3.4

Table 3.4 The relationship between peak velocity and human sensitivity.

The maximum of particle velocity (mm/s)	Feeling vibration
0.1	Not recognized
0.15	Almost begin to feel
0.35	Begin to feel sometime
1.0	Can be felt every time
2.2	Clearly felt
6.0	Violence of vibration
14.0	Felt much more violence of vibration

The Australian Vibration safety standard from blasting, AS 2187 - 1983, determines the peak velocity limits as below:

- For historic sites have the peak velocity should less than 2.0 mm/s.
- For houses or residential areas, the peak velocity should less than 10 mm/s.
- For the commercial building and other construction with concrete material, the peak velocity should less than 25 mm/s.

In addition, the Australian Environment Council has set up the standard of vibration from blasting in order to prevent annoyance in proximity to the local community:

- The peak velocity of not more than 5 mm/s and not exceed 5% of the amount of blasting.
- The maximum peak velocity must less than 10 mm/s.

3.10. Royalty

3.10.1. Mining Law Issue

The current mining law of Lao PDR was promulgated by the national assembly in April 1997. It consists of 8 chapters and 63 articles. The purpose of the mining law is to provide a system of management for conservation, exploration, mining and processing of minerals for both local consumption and for export.

3.10.2. The History of Mining Law

The first Mining Law was enacted on May 31st, 1997 and then later in 2006 Mining Law was revised and changed from Mining Law to Mineral Law, enacted on December 8th, 2008. In April 2011, the Draft of Implementing Decree on Mineral Law was approved by Government meeting. After that in August 2011, the Government declared to set up the new Ministry of National Resources and Environment (MONRE) and transferred the Department of Geology and Mineral and their responsibilities, Ministry of Energy and Mine to the MONRE (DOM, 2008).

3.10.3. Taxation Issue

According to Laos taxation system for investment, the foreign investment have to pay annual profit tax at the rate of 10%, 15%, and 20% depending on the promotion area (other investments are 35%). For mining industry, every project is subjected to taxation profit of not less than 25 - 35% according to The Ministry of Energy and Mines. Foreign investment may be 100% owned or a joint venture with a minimum of 30% investment between foreign and domestic company. But the government has the right to hold 10% shares borrowed from the Investment, reimbursed thru suspension of future capital and income tax. For personal income taxation, the employer needs to withhold taxes from foreign employee at a rate of 10% and Lao employees at 5 - 25%.

3.10.4. Land Rent

Exploration companies will be charged an annual rent for the land covered by their exploration license. This will commonly be increased as the company moves from exploration to the production stage. Table 3.5 illustrates the rental fee of land from the initial prospecting to mining stage.

Table 3.5 Rental fees (Ministry of Energy and Mine)

No	Type of mineral	The rate of concession ($ /hectare/year)			
		Prospecting	Exploration	Assessment of economic - technique	Mining
1	Precious metal: (Gold, Platinum, Silver)	1	2	3	100
2	Base metal: • (lead, zinc) • copper)	 1 1	 2 2	 3 3	 60 80
3	Diamond, Ruby, Sapphire, Emerald, jade	2	2	3	700
4	Tin and Tungsten	1	2	3	100

3.10.5. Royalty

The mineral products and their storage provide the basis of the total revenue calculation. The rate of royalty ranges from 1 - 7% depending on types of minerals and products. Cooperate income tax is 25 - 35% depending on the size of a project and mineral production. Table 3.6 illustrates the Lao royalty system.

Table 3.6 Illustrates the Lao royalty system (Department of Geology and Mine)

Mineral Production	Loyalty Rate (%)
Iron	2
Copper	3
Lead	3
Zinc	3
Tin	3
Gold	5
Silver	4
Platinum	5
Sapphire	5
Ruby	5
Emerald	5
Potash	2
Gypsum	2

For example, LXML’s royalty at Sepon were increased, by agreement in late 2004, from 2.5% to increasing rate of 0.5% per year up to a maximum of 4.5% in 2007, and Phu Bia Mining’s current royalty rate is 2.5%.

3.10.6. Procedure to Obtain Mining License

According to the MPI investment guidelines (2008), in order to receive a mining license, investors have to follow roughly seven steps, 1). Prepare documents required by MPI, 2). Submit project documents to OSU MPI, 3) IPD presents project documents to CPMI meeting, 4). Consideration in CMPI meeting and report to Prime

Minister Office, 5). Prime Minister's meeting, 6). Project negotiation (Mineral Exploration and Production Agreement MEPA, and 7). Sign MOU with government (Khyophivong, 2007). The flowchart illustrating the process for receiving mining license is shown in Figure 3.13.

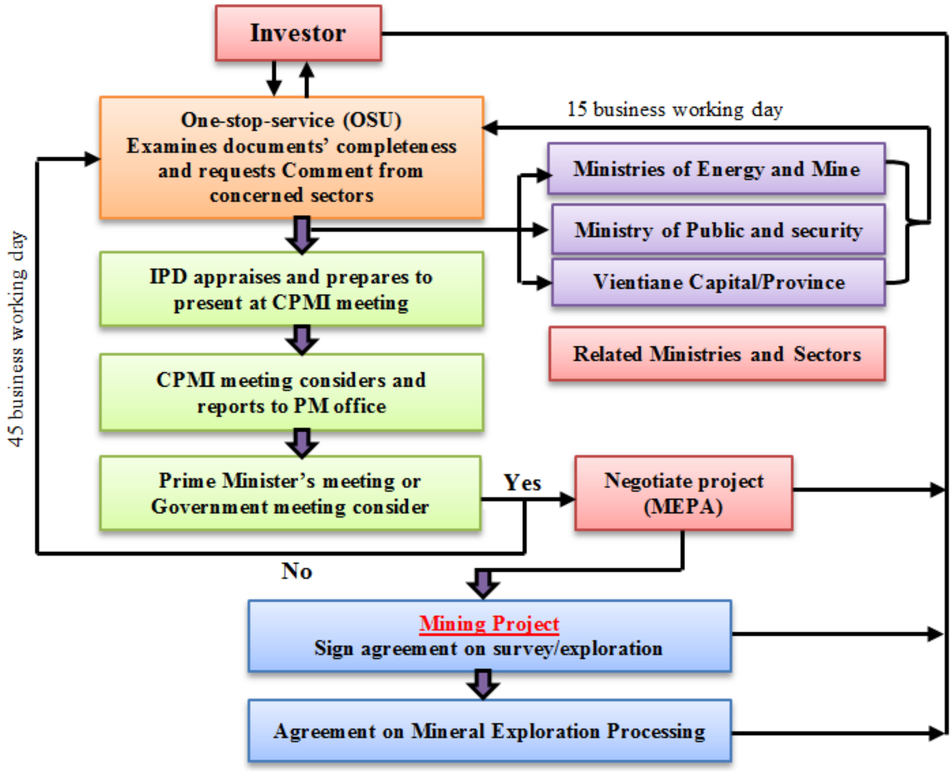


Figure 3.13 Flowchart for mining license application (MPI, 2008).

According to Lao PDR Development Report 2010 prepared by (Morten Larsen), revenues from Lao mining operations have surged in recent years. Relevant questions have been raised concerning the fairness of profit and benefit sharing between project owners and Government of Lao PDR. It has assessed the actual profit-sharing in Lao mining projects of around 50:50 percent (Larsen, 2008). Table 3.7 illustrates the calculation sheet parameters for evaluating "government share" or "Effective tax rate

Table 3.7 The “government share” or “Effective tax rate” calculation sheet (Parsons, 2008)

No	Description	Projects Cash Flow (US$)	Government Cash Flow (US$)
A	Sale	1000.0	
B	Operating Costs	- 650.0	
C	Operating Profit (A-B)	350.0	
D	Royalty (4% of A)	- 40.0	40.0
E	Taxable Profit (C-D)	310.0	
F	Profit Tax (35% of E)	- 108.5	108.5
G	Available for Shareholders (E-F)	201.5	
H	Dividend to government (10% of G)	- 20.2	20.2
I	Dividend to Investor (G-H)	181.4	
J	Dividend withholding Tax (10% of I)	- 18.1	18.1
K	Net Distribution to Investor (I-J)	163.2	
L	Total Government Cash flow	-	186.8
M	**Effective tax rate (L/C)*100**	**46.6 percent**	**53.4 percent**

Source: Parsons, 2008

Surtaxes

Surtaxes are simply determined based on the total revenue of national government’s royalty. The surtax usually is an additional tax imposed on royalty by some local government. For instance, a mine generates 1,000,000 USD in sales revenues and the national government assesses a royalty for 2%. The royalty payable to government is then 20,000 USD. A provincial or local government entity may be authorized to assess a surtax on the 10% royalty tax basis: 20,000 x 0.10 = 2,000 (in effect, 0.02 x 0.10 x 1,000,000). The mining company would pay 20,000 USD nationally and 2,000 USD locally. Table 3.8 compares rate of taxation and royalty rate in developed and developing countries (Ralbovsky, 2012).

Table 3.8 CIT and royalty rate (www.pwc.com/gx/mining)

No	country	Corporate income tax (CIT)	Royalty (gold)	Royalty (copper)	Level at which applied
1	Argentina	35%	3%	3%	provincial
2	Australia	30%	3.5%	2.5%	State, federal
3	Brazil	34%	1%	2%	federal
4	Canada	15%	5%	5%	provincial
5	Chile	18.5%	0 - 14%	0 - 14%	federal
6	China	25%	0.5 - 4%	0.5 - 4%	federal
7	DRC (Congo)	30 - 40%	2.5%	2%	federal
8	Germany	15.8%	N/A	N/A	Federal, local
9	Ghana	25 - 35%	5%	5%	federal
10	India	32.442 - 40.024%	2%	4.2%	Federal, state
11	Indonesia	25%	3.75%	4%	federal
12	Kazakhstan	20%	5%	5.7%	federal
13	Mexico	30%	N/A	N/A	federal
14	Peru	30%	1 - 12%	1 - 12%	Central Gov
15	Philippines	30%	2%	2%	federal
16	Russian Federation	20%	6%	8%	Federal, state
17	South Africa	28%	5%	5 - 7%	federal
18	Tanzania	30%	4%	4%	federal
19	Ukraine	21%	UAH 15.98/ton extracted	according to rate of a main material	federal
20	United Kingdom	26%	N/A	N/A	federal
21	United States	35%	2 - 5%	2 - 5%	Federal, state

CHAPTER IV
THE RESULTS OF THE STUDY

In parallel with the adopted methodology, this chapter discusses the results of model development comprising drillholes data input, statistical analysis of gold variable, geological model construction, resource estimation, pit optimization, and mineable reserve estimation. The results of mine planning and scheduling, and financial analysis are also discussed in detail.

4.1. 3D Block Model Development

The 3D block model is constructed on MineSight software (MS). The MS work flow starts with drillholes data input, statistical analysis of variables, and geological model construction. The results of these processes will be discussed in the next section.

4.1.1. Drill Holes Data Preparation

The area of this project is approximately of 500 x 700 sq. meters, extending from the grid coordinates of 24200 to 24900 in the Easting direction and grid coordinates of 74000 to 74500 in the Northing direction, and the vertical distance of 250 m, as shown in Table 4.1. This project study comprise of 54 drill holes and 3,218 gold assay data collected from drillholes DH01 to DH54. The drillholes locations are plotted in Figures 4.2 - 4.3. The drillholes data contain gold assay data with 1 meter interval and their assigned 3-D (x,y,z) coordinates.

Table 4.1 Project setting (boundaries and block discretization)

	Minimum (m)	Maximum (m)	Block size (m)	Scale distance (m)	Number of Block
Easting	24200	24900	12.5	700	56
Northing	74000	74500	12.5	500	40
Elevation	0	250	5	250	50

Figure 4.1 illustrates the input file format for inputting into MS software. The first column represents the name of drillhole (DH01 and DH02) and the first row of data also represents the name of drillhole and its coordinate (x= 24538.71, y = 74329.06 and z = 217.6), azimuth, dip, and depth respectively. The assay data are in the fifth, sixth, and seventh column represented the grade of gold (Au), copper (Cu), and silver (Ag), respectively.

Drillhole ID	Easting	Northing	Elavation	Azimuth	Dip	Depth
	24538.71	74329.06	217.6	170	-45	119.8
	from	to	interval	au	cu	ag
DH01						
DH01	0	1.5	1.5	0.01	0.01	0.1
DH01	1.5	3	1.5	0.01	0.02	0.1
DH01	3	5	2	0.01	0.02	0.2
DH01	5	7	2	-0.01	0.03	0.2
DH01	7	9	2	-0.01	0.02	0.2
DH01	9	10.2	1.2	0.07	0.01	0.4
DH01	114	116	2	0.02	0	-0.1
DH01	116	118	2	0.01	0	-0.1
DH01	118	119.5	1.5	0.01	0	-0.1
DH01	119.5	119.8	0.3	-0.01	0	-0.1
DH02	24663.13	74352.63	217.01	360	-90	13
DH02						
DH02	1	2	1	-0.01	-99	-99
DH02	2	3	1	-0.01	-99	-99
DH02	3	4	1	-0.01	-99	-99
DH02	4	5	1	-0.01	-99	-99
DH02	5	6	1	-0.01	-99	-99
DH02	6	7	1	0.04	-99	-99
DH02	7	8	1	0.53	0.01	6
DH02	8	10	2	-99	-99	-99
DH02	10	11	1	2.17	0.01	6
DH02	11	12	1	1.32	0.01	7
DH02	12	13	1	0.55	0.01	7

Figure 4.1 Drillhole data input file format.

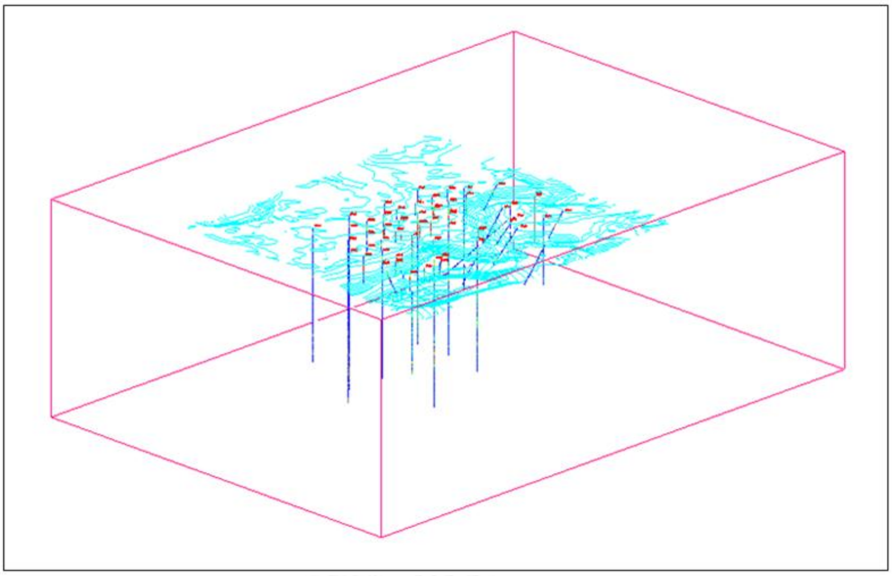


Figure 4.2 The project setting in 3 D model and drillholes.

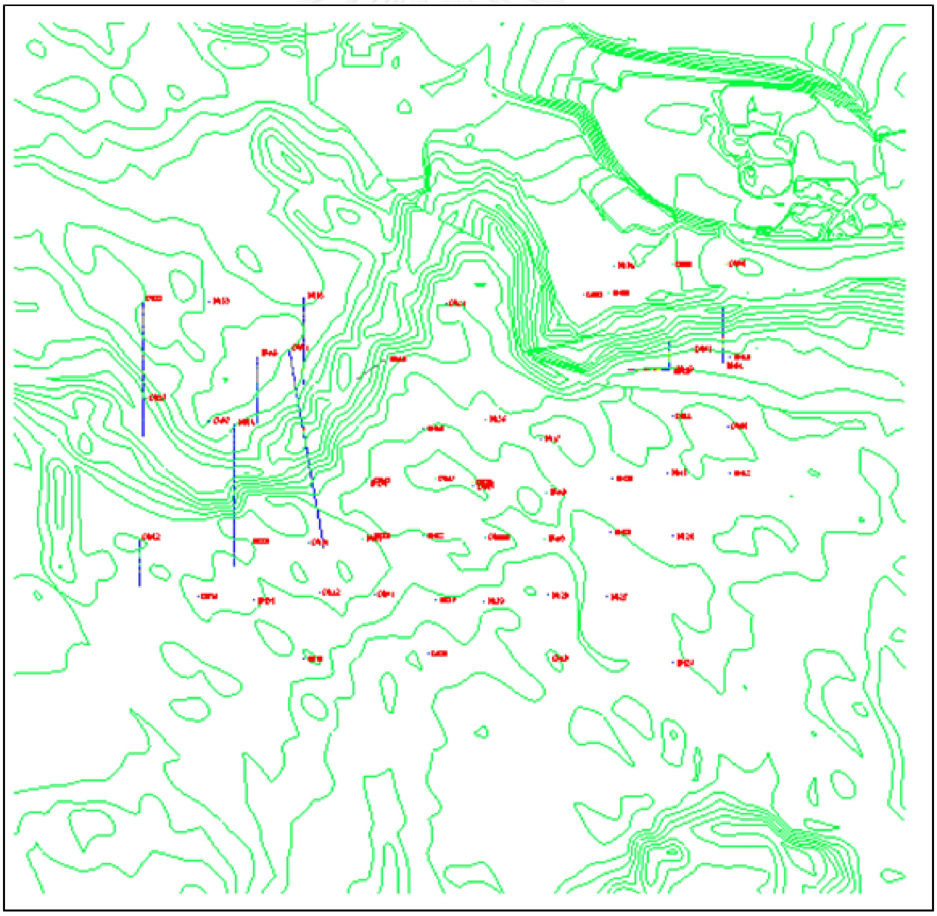


Figure 4.3 Topo surface and drillholes.

The average 1 meter interval assay gold data is grouped into 5 meter composited gold assay as shown in Figure 4.4.

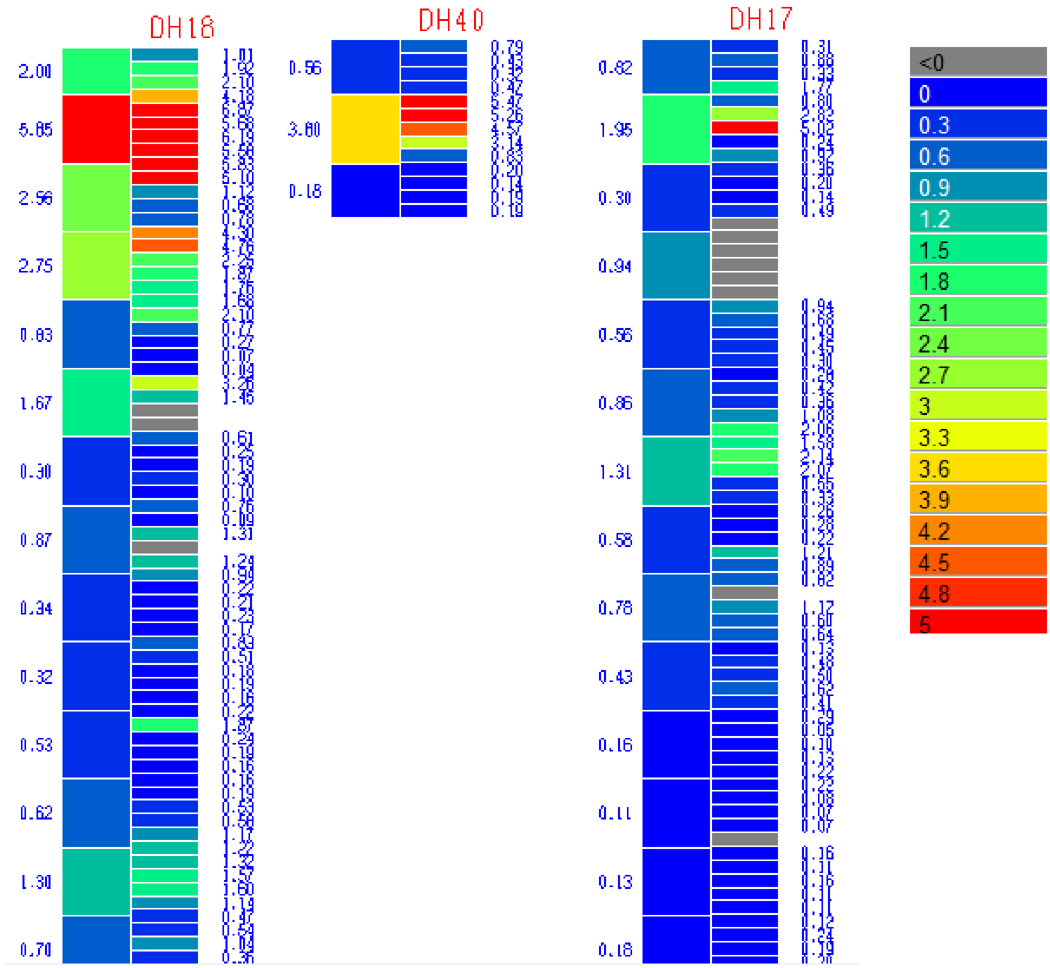


Figure 4.4 Composite data of drillholes.

As shown in Figure 4.4, the number on the left hand shows the average grade of composite of gold assay in each 5 meter interval, and the number on the right hand is represented the average gold grade of the original interval of 1 meter. The grade variation is represented by color code bar starting from low grade (blue shade) to high grade ore (red shade).

The rock codes identification is carried out to differentiate different rock types, and the rock codes input into the MineSight Software are shown in Table 4.2.

Table 4.2 The classification of the rock code

Lithology Codes			
Description		Codes	Index Number
1	Calcareous Shale	001	8
2	Clay	002	430
3	Dolomite	003	463
4	Jasperoid	004	477
5	Limestone	005	470
6	Ryodacite Porphyry	006	-

The drillholes characteristics are summarized in Table 4.3. It accounts for 44 vertical and 10 inclined drillholes. The minimum and maximum depth are 12, and 200 meters, respectively. The total number of gold assay data is 3,218 assays.

Table 4.3 The drillholes statistics and assays

No	Description	Result
1	Vertical hole	44 holes
2	Inclined hole	10 holes
3	Maximum depth	200 m
4	Minimum depth	12 m
5	Total number of data	3,218 assays

4.1.2. Statistical Analysis of Gold Assay Data

The histogram of Au, illustrated in Figure 4.5, and the statistical analysis the gold data yields the basic statistics parameters for minimum, maximum, mean, variance and skewness of 0.01, 18.8, 0.56, 1.82, 6.03 respectively, as shown in Table 4.4 The gold grade exhibits a moderate average grade and strong skewness with cluster of low grade data at the tail area and a record of a few high grade values. The gold grade statistics together with the studied volume and its discretization will be used in the resource estimation step.

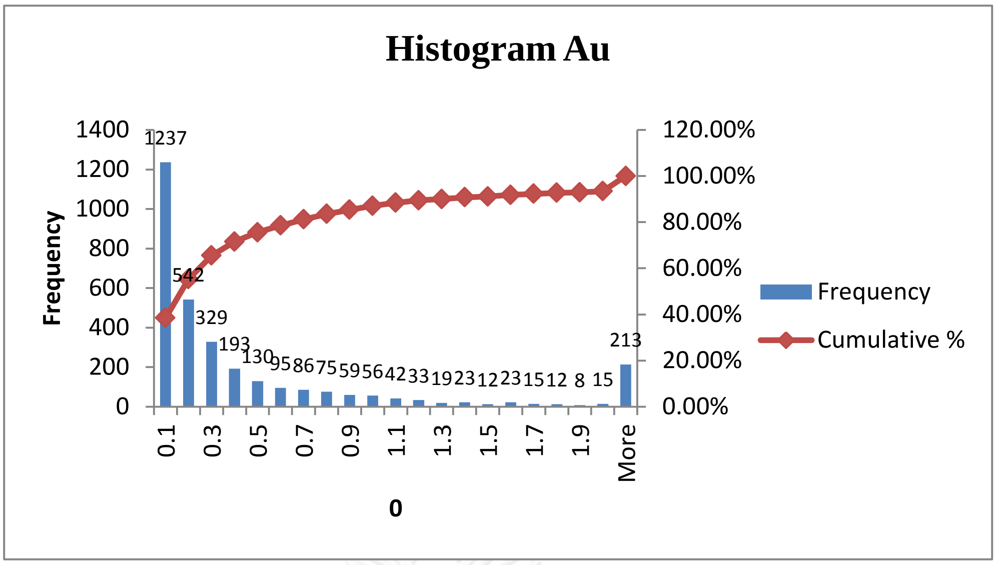


Figure 4.5 Histogram and Cumulative Distribution Function (CDF) of gold assays.

Table 4.4 The primary statistics of gold assays data.

Statistics of Au	
Parameter	**Result**
Total number of assays	3,218 assays
Mean	0.58 ppm.
Minimum	0.01 ppm.
Maximum	18.8 ppm.
Variance	1.82
Mode	0.02
Median	0.16
Skewness	6.03

4.2. Geological Resource Estimation and Grade Tonnage Curve (GTC)

The geological resource estimation is calculated using the Inverse Distance Weighting Squared (IDWS) technique. The IDWS estimates the gold grades into the center of the block giving the separating distance between the center of the block to the data locations as the criteria to the assigned weight. In this study, the block volume is discretized as 12.5 x 12.5 x 5 meter in x, y, z direction, respectively. The isotropic searching volume of 50 meters radius is adopted with the power factor of 2. It is important to note that the adopted searching distance of 50 meters is based on previous case study of other gold deposit and their variogram model. The geological resource of this gold deposit stands at 34.8 million tonnes of gold at an average grade of 0.56 ppm. from the total number of 17,412 estimated blocks.

The estimated block grades and tonnages are used for the Grade Tonnage Curve (GTC) construction. The cutoff grade is varied from 0.2 to 2 ppm. at the increment step of 0.2 ppm. The corresponding tonnages, and average grades are shown in Table 4.5. It is anticipated that the calculated tonnage are reduced, and the average grade are increased, while increasing the applied cutoff grades as show in Figure 4.6. The 3-D block model from IDWS was used as a conditioned input for GTC calculation shown in Table 4.5. The GTC allows the sensitivity analysis of cutoff grade variation in responding to the gold price, mining and processing technology.

Table 4.5 The cutoff grades, tonnages, and average grade estimation from IDWS geological block model

Cutoff grade (ppm)	Tonnages	Average grade (ppm)	Number of block
0.2	16,427,470	0.52	8,237
0.4	8,126,727	0.77	4,115
0.6	4,074,708	1.04	2,103
0.8	2,058,225	1.40	1,088
1	1,165,044	1.81	631
2	357,622	2.95	197

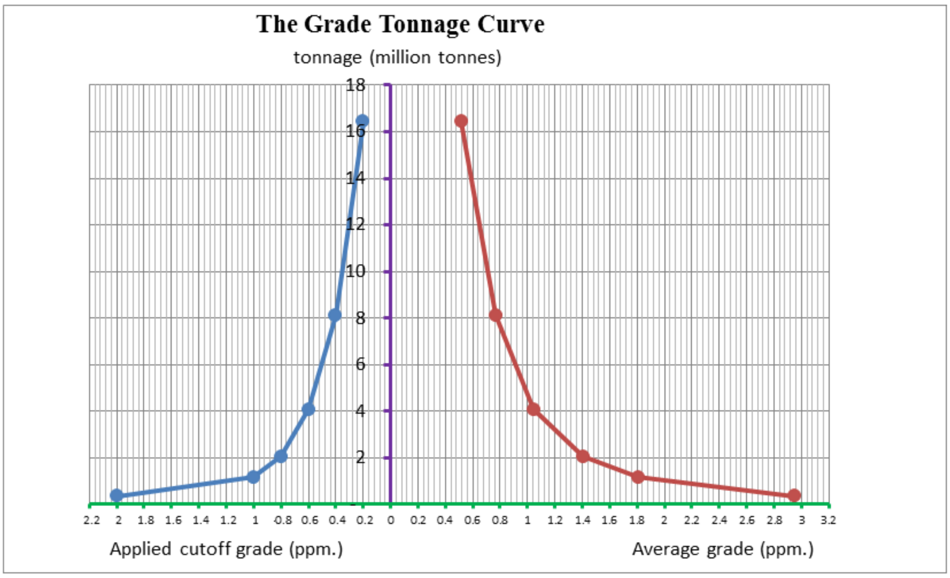


Figure 4.6 The Grade Tonnage Curve (GTC).

Table 4.6 The Inverse Distance Weight parameters, and tonnage, from the generated blocks

No	Parameter	Searching distance (m)	Value (tonnes)	Number of bock
1	IDW	25	16,026,420.31	8,031
2	IDW	50	34,876,826.56	17,412
3	IDW	75	51,357,596.88	25,896
4	IDW	100	66,552,809.38	33,772

4.3. Pit Optimization

The pit optimization uses Lerchs-Grossmann algorithm providing the conditioned data of 3-D geological block model from IDWS as shown in Figure 4.7, and the related technical and economic parameters. The economic parameters are given in Tables 4.7 - 4.9.

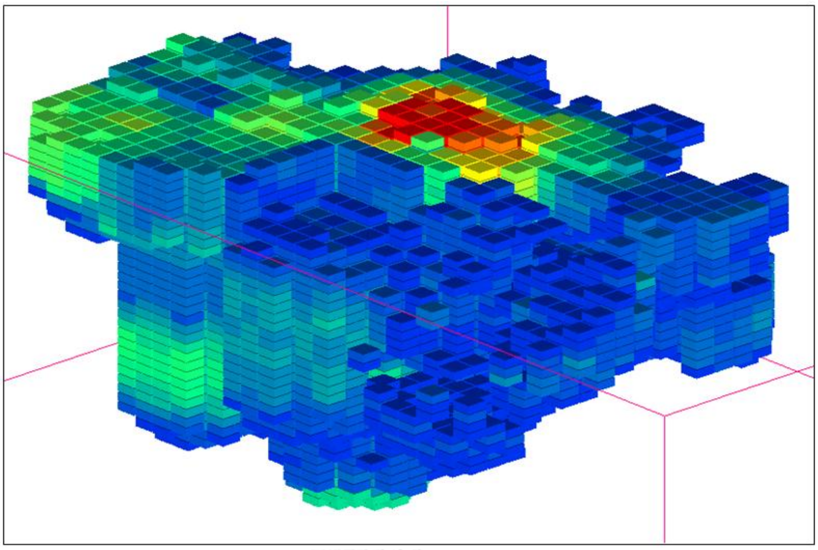


Figure 4.7 3-D Block model from IDWS.

Numerous assumptions have been made in this step. The assumption of gold price is 1,200 \$/oz which is equal to 42.40 \$/g. Processing cost is assumed at 20\$/t, mining cost of 1.5 \$/t, and percentage of recovery of 85 percent. From GTC calculation at cutoff grade of 0.61 g/t yields the average grade of 0.58 g/t. It is observed that the relationship between the gold price and cutoff grade is significantly related. As the price of gold increases, the cutoff grade will be lower. That means mining operation can select a low grade ore to feed to the processing plant. On the other hand, when the gold price decreases, the cutoff grade has to be higher, the mining operations have to choose only high grade ore to feed to the processing plant that will result in shortening mine life. Table 4.7 shows the cutoff grade estimation based on gold price and operating cost. Table 4.8 illustrates the sensitivity of gold prizes to the cutoff grades.

From Table 4.9, the economic factors are 1,200 \$/oz. of gold price, 0.61 ppm.of cutoff grade, 20\$/ton of processing cost, 1.5 \$/ton of mining cost, and 85 percent recovery. For the mine design input parameters, they are 10 meters bench height, block discretization of 12.5 x 12.5 x 5 in x,y,z direction respectively, 40 degree overall slope angle, and 3 meters berm width. The haul road is designed for 2 lanes with 8 meters width for each lane, bringing the total of 16 meter width. A ramp gradient of 10 percent, 5 meter bench width, 55 meters of the maximum pit floor depth, and standard density of 2.6 t/m^3 are used as inputs for the mine optimization process. The input parameters for pit optimization are summarized in Table 4.9. Note that the geological resource estimation and mine optimization are carried out using Minesight Software.

Table 4.7 COG Parameter Calculation

Parameter	Relationship	Value
C (Processing Cost)	-	20 \$/t
P (Gold Price)	-	1,200 \$/oz = 42.40 \$/g
Y (Plant Recovery)	-	85% , (0.85)
M (Mining Cost)	-	1.5 \$/t
S (Service Cost)	-	1.2 \$/t
Cutoff Grade (COG)	COG = $\frac{c+m}{(P-S)*Y}$ $= \frac{20+1.5}{(42.40-1.2)*0.85}$	0.61 g/t

Table 4.8 The relationship between cutoff grade and gold price.

Gold Price \$/g	Gold Price \$/oz	Cutoff Grade g/t	mark
49.47	1,400	0.52	Better case
42.40	1,200	0.61	Base case
35.34	1,000	0.74	Worse case

Table 4.9 The input parameters for open pit optimization using LG

Assumption	**Value**
Bench height	10 m
Processing cost	20 \$/t
Mining cost	1.5 \$/t
Plant recovery	85 percent
Cutoff grade	0.61 g/t
Block size	12.5 x 12.5 x 5 (x,y,z)
Overall slope	40°

Haul road (two lanes)	16 m (8 m per lane)
Berm width	3 m
Inter-ramp slope	10 percent
Density of material	2.6 t/m^3
Gold price	1,200 $/oz
Bench slope	70°

The pit optimization step yield the ultimate pit design as shown in Figure 4.8 by which the mineable reserve and the bench production are determined. For this gold deposit, the mineable reserve of 2.44 million tonnes of ore is generated with the corresponding average grade of 1.53 ppm. The bench production outputs include total ore and waste, strip ratio, and the bench average grade are reported for each bench layer. As shown in Table 4.10, the total ore and waste of 1.24 million and 1.19 million tonnes are generated respectively, resulting the overall stripping ratio of 0.96, and the overall grade of 1.53. The maximum depth of pit floor is recorded at 55 meters with 40° degree overall pit slope angle. The spiral main haulage road with a 10 percent ramp gradient are shown in Figures 4.8 - 4.9.

Table 4.10 The bench production

Bench	**Total ore (tonnes)**	**Total Waste (tonnes)**	**Total waste & ore (tonnes)**	**SR (t/t)**	**Average grade (ppm.)**
220	333	3,604	3,937	10.82	3.68
215	114,077	152,266	266,343	1.33	1.97
210	165,182	216,459	381,641	1.31	1.85
205	211,962	219,726	431,688	1.04	1.60
200	200,002	174,999	375,001	0.87	1.54
195	179,688	133,124	312,812	0.74	1.46
190	143,750	102,030	245,780	0.71	1.38
185	114,062	83,438	197,500	0.73	1.24
180	68,750	52,655	121,405	0.77	1.15
175	32,812	43,438	76,250	1.32	1.06
170	17,188	14,062	31,250	0.82	0.91
	1,247,806	**1,195,801**	**2,443,607**	**0.96**	**1.53**

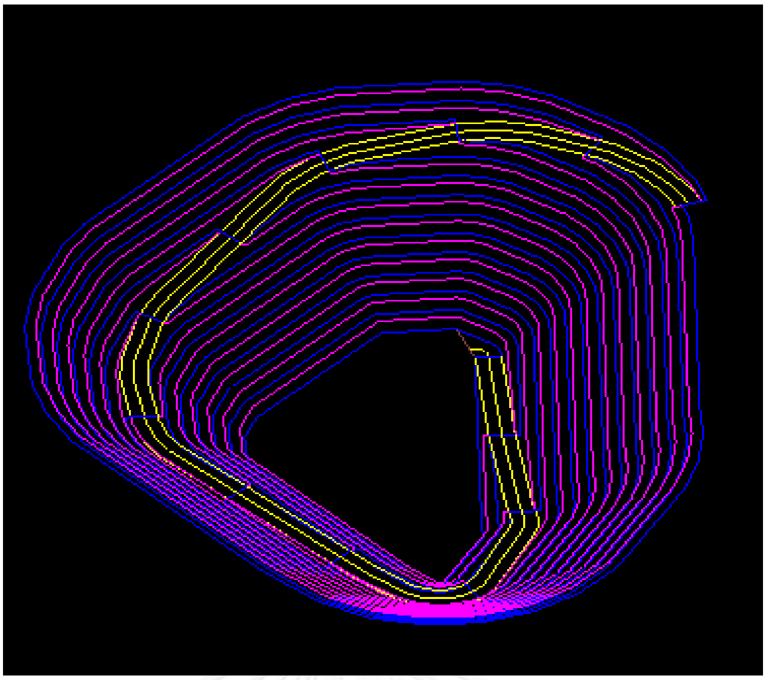


Figure 4.8 The ultimate pit design (contour level)

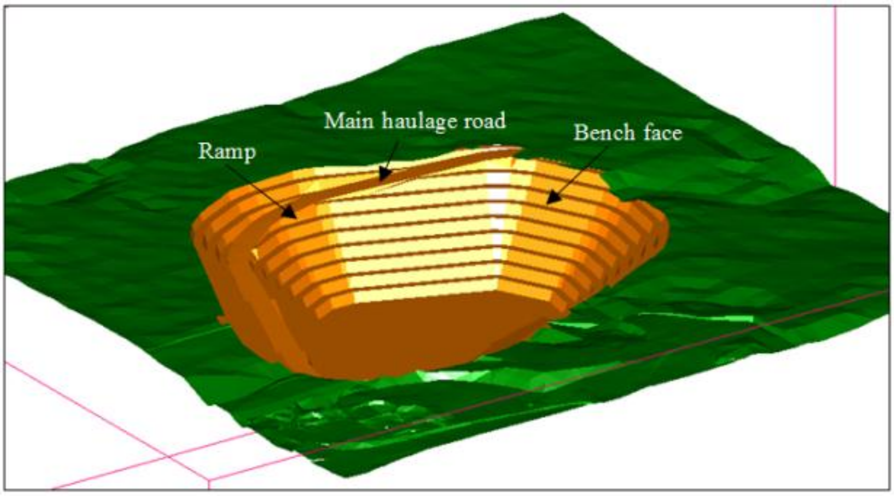


Figure 4.9 The ultimate pit design (solid model).

4.3. Mine Planning and Scheduling

Given the mineable reserve at 2.44 million tonnes, it is considered a very small deposit. Owning to the nature of gold mineralization at Sepon Mineralization District, it generates a series of gold deposit with a limited extension of gold occurrence. Then, in order to generate a meaning full amount of ore tonnage, the multiple pit development should be investigated to sufficiently provide ore tonnage to the processing plant and prolong the mine life to a standard development period as 20

-25 years. For the sake of this study, this mine design is planned within four years production schedule. The average annual production of 600,000 tonnes is scheduled as shown in Table 4.11. The pit profiles associated with their production years are shown in Figure 4.10. As mentioned above, this short life mine must be accompanied by a series of pit developed consecutively to provide a stable supply of ore tonnage and meet the plant designed capacity.

Table 4.11 The mine scheduling from year one to year four

Bench	**No**	**Waste (Tonnes)**	**Ore (Tonnes)**	**Total (Tonnes)**	**Average grade (ppm.)**
220 - 210	Year 1	279,5920	372,329	651,921	1.90
205 - 195	Year 2	411,964	394,725	806,689	1.60
190 -185	Year 3	323,4380	235,154	558,592	1.54
180 -170	Year 4	232,812	193,593	426,405	1.16
Total				**2,443,607**	

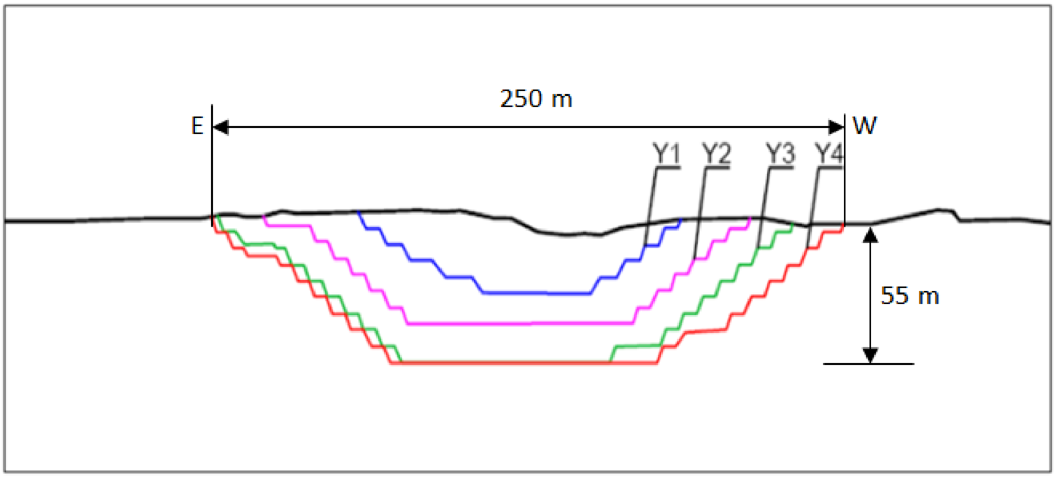


Figure 4.10 The pit profiles for production years 1- 4.

4.4. Bench Slope Parameters

Typically, the bench slope is defined based on the geology of surrounding material such as rock characteristic, rock strength, structure of rocks, bed layer. The bench slope design parameters are summarized in Table 4.12 and Figure 4.11

Table 4.12 Bench slope parameters

Parameter	Relationship	Value
H (bench height)	-	10 m
D (total pit depth)	-	55 m
bw (berm width	-	3 m
B	B = $\left(\frac{D}{H} + 1\right)$* bw = ($\frac{55}{10}$ + 1) x 3	19.5 m
M	$\frac{B}{A} = \frac{19.5}{55}$	0.35 m
cos θ = 0.35 m	θ = $\cos^{-1}(0.35)$ = 69.5°	≈ 70°
M	$\frac{A}{B} = \frac{55}{19.5}$	2.8 m
tan θ = 2.8 m	θ = $\tan^{-1}(2.8)$ = 70.3°	≈ 70°

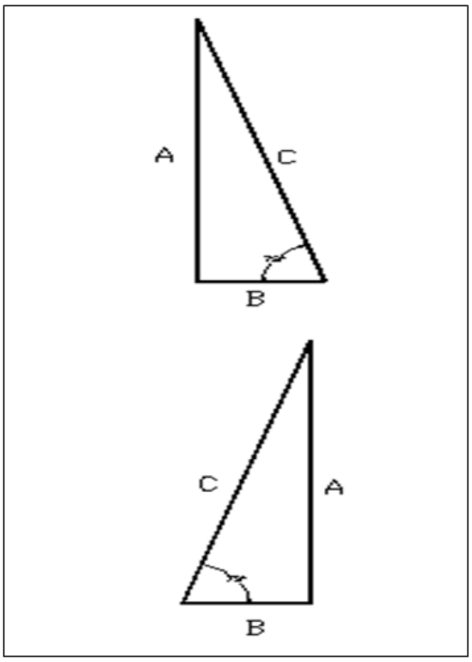


Figure 4.11 The slope face angle.

From Table 4.12, it calculates the bench slope angle of about 69° - 70°. The total depth of the pit is 55 meters, and the berm width of 3 meters.

4.5. Mine Operation

Mine operations comprise of four main tasks, that are drilling, blasting, loading, and hauling. These processes are necessary for open pit production as shown in Figure 4.12.

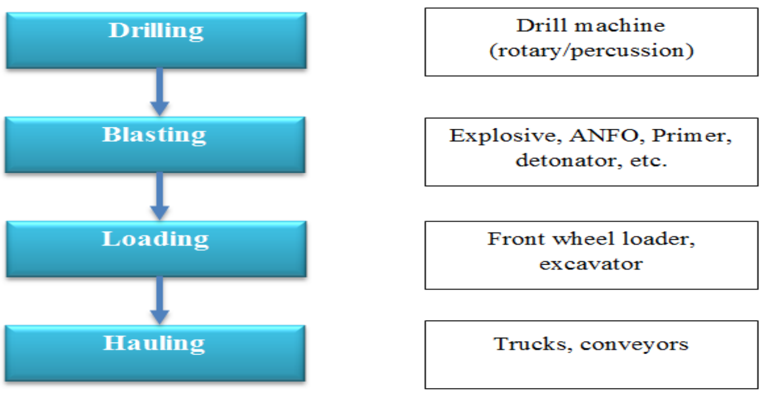


Figure 4.12 Mining operation schedule.

4.5.1. Drilling and blasting

Both drilling and blasting are significant process in open pit mining in order to have the suitable rock size fragmentation. Also, to ensure minimum impacts from fly rock, vibration, and dust generated from blasting sequence. In this study the blasting pattern design is shown in Table 4.13. Two types of explosive are used such as ANFO and Emulsion.

Table 4.13 Blasting pattern parameters

Parameter	Relationship	Value
Drilling pattern	Staggered	-
Diameter of drill hole	ø	102 mm
Burden	B = (25…40) x D	3 m
Spacing	S = (1.25) x B	3.5 m
Effective Sub-drill	U = (0.3…0.4) x B	1 m
Stemming length	T = (0.8) x B)	2.5 m
Bench height	H = (4 x B)	5 m
ANFO density	SGe	0.85 g/cm^3
Emulsion density	SGe	1.3 g/cm^3

Rock density	SG_R	2.6 t/m^3
Volume of rock	$V = B \times S \times H \times N_{hole}$	945 m^3
Charge distance	$L_e = H + U - T$	4.5 m
Total explosive	$Q = C \times N_{hole}$	522 kg
ANFO	95% of total explosive	495 kg
Booster or Primer	5% of total explosive	26 kg
Charge per meter	$D_e = \frac{SGe\ x\ (D)^2 x\ \pi}{4000}$	6.5 kg/m
Total charge per hole	$C = L_e \times D_e$	29 Kg/hole
Hole length	$L = H + U + 1/\sin\alpha$	6 m
ANFO per hole	ANFO/N_{hole}	27.5 Kg/hole
Booster per hole	Emulsion/N_{hole}	1.5 Kg/hole
Powder factor	$PF = \frac{C}{B\ x\ S\ x\ H\ x\ SGr}$	0.25 kg/t
Specific drill	Hole length/volume	0.11 m/m^3
Quality of material blasted	Q	90%

Table 4.13 illustrates the blasting pattern design, it generates about 2,545 tonnes of broken ore for one round of blasting. The assumption has been also made that only 40 percent of total ore and waste material will be done by blasting (from oral discussion with the mining engineer at Sepon Mine). The design requires 18 holes per round and schedule to blast twice a week. The blasthole design is used staggered pattern which contains 3 rows and 6 columns. In this study, two sizes of blasthole diameter will be chosen which are 3.5 inch (89 mm.) and 4 inch (102 mm.) with a bench height of 5 meters. In this section, the 89 mm hole diameter is used for pre-splitting without sub drill and fill with only emulsion and attached to the detonator cord. For pre-splitting, it requires to blast in advance before blast holes production. In normal practice, the pre-splitting blast first, and then follow by blastholes production by employing cap delay. The volume for one blast is around 945 m^3, and 522 kg of total explosive is needed. In this case, ANFO (Ammonium Nitrate 94% mix with Fuel

Oil 6%) of 495 kg are used as explosive, and primer of 26 kg, achieving the powder factor of 0.25 kg/t.

The results of blasting before pre-splitting and after pre-slitting are shown in Figures 4.13 - 4.14. As the result, without using presplitting, it generates uneven and unstable slope face as shown in Figure 4.13. On the other hand, with pre-splitting it generates a smooth and stable slope face without hanging rock. Therefore, it can be concluded that the pre-splitting method is suitable with this hard rock condition.

Figure 4.13 Slope face using pre-splitting method.

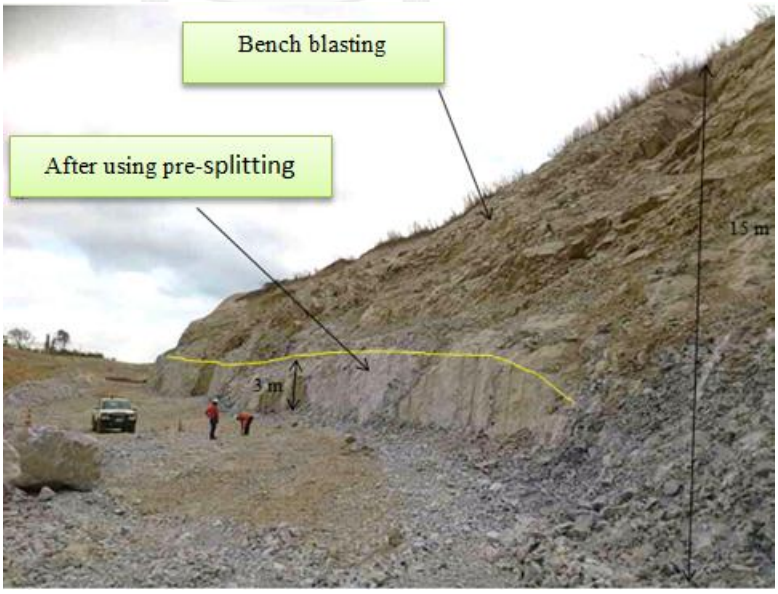


Figure 4.14 Slope face with pre-splitting method.

According to the given blasting design, topography of the area and the practical of block size of 12.5 m. x 12.5 m. x 5m. in (x,y,z) dimension, respectively, bench height is planned with 10 m. blasted on two 5 m. benches. The optimum blasting pattern design in shown Figure 4.15.

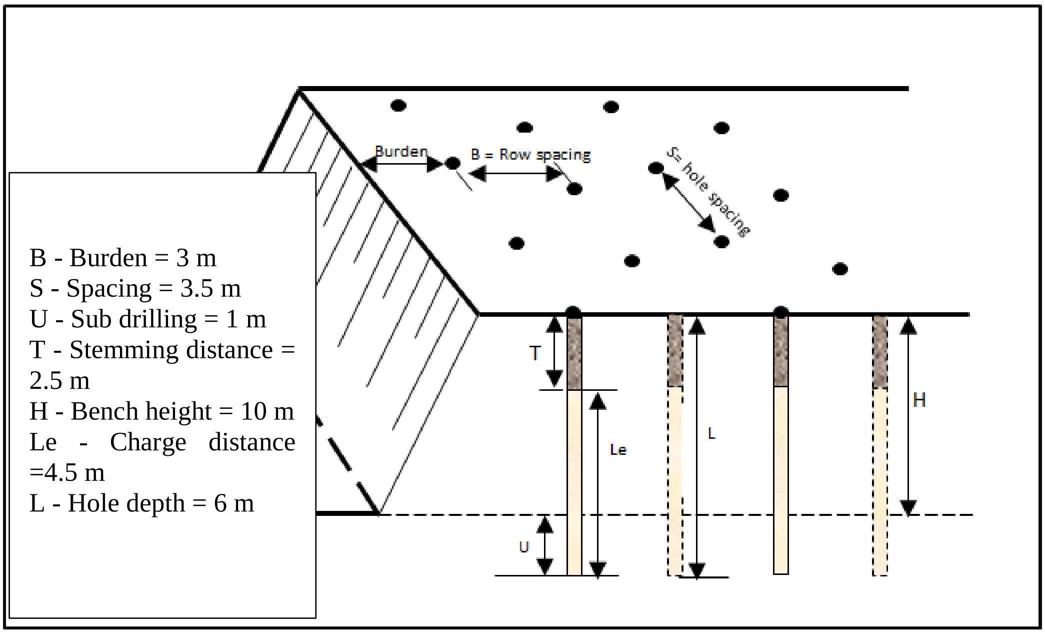


Figure 4.15 Blasting pattern design.

4.5.2. Loading and Hauling

For open pit mine operations, transportation equipment (such as excavators, wheel loaders, and truck systems) are required. The challenge is, how to optimize the number of equipment required with respect to the production capacity. In this study, it is assumed that the average productivity including ore and waste is 600,000 tonnes per year. It also assumes 12 months of working operation, and 26 working days per month, Therefore, the total capacity is 1,923 tonnes per day and there is one shift per day with 12 hours operation, and 160 tonnes per hour production.

In this case, the wheel loader (Model Caterpillar 924Gz, capacity 1.8 m^3), excavator (Model Caterpillar 325C, capacity 1.6 m^3), and dump trucks (Model caterpillar C770, capacity 16.4 m^3) are used. (Caterpillar Performance Handbook, October 2004, Edition 35). The production calculations are illustrated in the following sections.

4.5.3. Caterpillar Wheel Loader Production Estimation

In this study, the caterpillar wheel loader model CAT 924 Gz with a capacity of 1.8 m^3 is used in this calculation. From the equation (3.2) at section 3, the estimation of the production per hour of wheel loader is illustrated in Table 4.14, altogether with input parameters.

Table 4.14 Parameters for wheel loader estimation

Parameter	Relationship	Value
BC (bucket size)	2.6 t/m^3 x 1.8	4.6 t
BF (bucket fill factor)	-	(95 %) = 0.95
D (density in place)	-	2.6 t/m^3
MA (machinery available)	-	(85%) = 0.85
JF (job factor)	83.3% (assume 50min/60min)	0.833
SF (swell factor)	-	(30%) = 0.3
CT (average cycle time)	0.55 min	33 seconds
O (operation)	$O = \frac{BC*BF*D*MA*JF*3600sec}{(1+SF)*CT}$	675 t/hr

From the calculation in Table 4.14, the production of the wheel loader is approximately 675 t/hr. It requires only one wheel loader to handle 160 t/hr of ore and waste material.

4.5.4. Excavator Production Estimation

For excavator production calculation, Caterpillar Model 325C with bucket capacity of 1.6 m^3 is selected in this study. The calculation is similar to wheel loader as illustrated in Table 4.15. It estimates the production capacity of 587 t/hr with 33 seconds cycle time, as shown in Table 4.15.

Table 4.15 Excavator Production Estimation

Parameter	Relationship	Value
BC (bucket size)	2.6 t/m^3 x 1.6	4 t
BF (bucket fill factor)	-	(95 %) = 0.95
D (density in place)	-	2.6 t/m^3
MA (machinery available)	-	(85%) = 0.85
JF (job factor)	83.3% (assume 50min/60min)	0.833
SF (swell factor)	-	(30%) = 0.3
CT (average cycle time)	0.55 min	33 seconds
O (operation)	$O = \frac{BC*BF*D*MA*JF*3600sec}{(1+SF)*CT}$	587 t/hr

4.5.5. Trucks Production Estimation

Typically in open pit operations, trucks are significant equipment for transporting ore or waste from the extraction site to the processing plant, stock pile, or dump site. It is very important to have the exact number of truck to meet capacity required. In this study, the number and type of trucks are chosen based on the capacity needed and also related to the cycle time of each truck.

The distance of hauling and returning are divided into three segments base on the road conditions (percent grade, working condition) as shown in Table 4.16. The truck average speed and time are calculated for hauling and returning segments. The hauling time is 7 minutes, while the total returning time is 5 minutes. The loading time of excavator is 3.3 minutes. Therefore, taking every truck cycle time into consideration, the total time of truck is 18.3 minutes, as shown in Table 4.17.

Table 4.16 Hauling and returning time of truck

Hauling				
Segments	Length (m)	Grade (%)	Avg. Speed (Km/hr)	Time (min)
1	500	0	30	2
2	1,100	10	30	4
3	500	0	30	2
Total	2,100			7
Returning				
1	500	0	30	1.5
2	1,100	-10	30	2
3	500	0	30	1.5
Total	2,100			5

Table 4.17 The cycle time of truck

Total loading time	3.3 min
Haul	7 min
Dump	2 min
Return	5 min
Spot and waiting time	1 min
Total time	18.3 min

Therefore, the calculation of the number of trucks is provided below.

The capacity of excavator bucket is:

Tons/cycle = 4 t/bucket * 0.95 = 3.8 t/cycle

Of cycles/truck = $\frac{20\ t/truck}{1 cycle/3.8\ t}$ = 5.26 $\approx$ 6 cycle

Total loading time = 6 cycles * 0.55 min/cycle = 3.3 min

$N_{truck} = \frac{Total\ time}{Load\ time} = \frac{18.3}{3.3} = 5.5 \approx$ 6 trucks

4.5.6. Unit Production of Truck

The unit production calculation is based on the optimization number of trucks needed to operate in open pit mining operation. Unit production is calculated by the truck payload, truck cycle time, hours per shift, and operating efficiency. The calculation is shown in Table 4.18.

Table 4.18 The unit operation estimation of truck

Parameters	Equation/assumptions	Value
Working time	50 min/hr (assumption)	0.833 min
Operation efficiency	85% (assumption)	0.85
productivity	20 t*1cycle/18.3*50min/hr*0.85/unit	46.4 t/hr
Total production	4 trucks * 46.4 t/hr	185.7 t/hr
N (number of truck)	Total time/ total loading time	6 trucks
To obtain 160 t/hr	160 t/hr/productivity	4 trucks

The unit operation calculation shows that 6 trucks are required. However, based on the actual required production, it is needed only four trucks to handle 160 t/hr, for material handling. In addition, 10 percent of truck is needed for a spared time, that brings to total of 5 trucks.

In conclusion, for the loading and transportation requirements at 160t/hr) material handling, the total cycle time of the truck operation is approximately 18.3 minutes per cycle, and it requires 1 wheel loader , 1 excavator and 5 trucks.

4.6. Financial Analysis

The financial analysis for long term investment project involves Discounted Cash Flow (DCF) analysis that provides the criteria for Net present Value (NPV) at discounted rate, and Internal Rate of Return (IRR).

For all project investments, significant parameters to determine the project feasibility are IRR and NPV. The DCF model is constructed to estimate the cash inflow and cash outflow on any given year in the future. IRR is used to compare with the discount rate of the project. NPV is used to validate the feasibility of the project investment.

4.6.1. Weight Average Cost of Capital (WACC)

Typically, most of the projects especially mining project, there are mainly two sources of capital budget. The first budget is taken by borrowing from the bank and the second one is from the project owner or from the shareholders. This study is assumed that 40 percent of the fund will get from the bank with interest rate of 6.5, percent and the rest of 60 percent is the fund from shareholders with some expect rate of return. The cost of capital is a combination of debt cost and equity cost. The formula (3.9) is used to compute the Weight Average Cost of Capital (WACC). Table 4.19 provides the parameter inputs for WACC calculation. The calculated WACC is 12.6 percent, which provides the interest rate of the project.

Table 4.19 Input parameters for Weight Average Cost of Capital (WACC)

Parameter	Formula	Results
Capital Cost	100%	8,371,787 $
E (equity) Cost	60% * 8,371,787 $	5,023,072 $
D (debt) Cost	40% * 8,371,787 $	3,348,715 $
V (total debt and equity cost)	D + E	8,371,787 $
R_d (cost of debt)	6.5 percent	0.065
T_c (tax rate)	35 percent	0.35
R_f (free risk)	4.5 percent	0.045
R_m (medium risk)	17 percent	0.17
ß (coefficient)	10% high risk (ß =1+10%)	1.10
R_e (cost of equity)	R_f + ß (R_m - R_f)	0.18
WACC	$\frac{D}{V}*(R_d)*(1-T_c) + \frac{E}{V}*(R_e)$	**12.6** percent

4.6.2. Discounted Cash Flow (DCF) Model

The parameters needed for discounted cash flow model are capital cost, operating cost, sale cost, depreciation cost, corporate income tax, royalty. The input parameters for DCF calculation and its calculation sheet is shown in Tables 4.20 - 4.21.

Table 4.20 The input parameters for DCF model

Input parameters	Value
Capital Cost (assumption)	8,371,787 $
Mining Cost	1.5 $/t
Processing Cost	20 $/t
Drilling and Blasting Cost	0.5 $/t
Transport Cost	1.9 $/t
Gold Price	1,200 $/ounce
Corporate Income Tax (35 percent)	35%
Royalty Rate (4.5 percent)	4.5%
Mine life 4 years (assumption)	600,000 t/year
Milling Capacity	250,000 t/year

Table 4.21 The discounted cash flow calculation

No	Year	0	1	2	3	4	5
A	Gross Revenues	0	14,526,715	12,233,023	11,774,285	13,456,326	8,791,108
B	Royalty (4.5%) of (4.5%*A)	0	653,702.18	550,486	529,843	605,535	395,600
C	Net Revenues (A - B)	0	13,873,013	11,682,537	11,244,442	12,850,791	8,395,508
D	Operating Costs	0	8,145,131	8,703,843	7,808,213	7,331,018	5,548,472
E	Operating Income (C - D)	0	5,727,882	2,978,694	3,436,229	5,519,773	2,847,036
F	Depreciation	0	1,564,357	1,564,357	1,564,357	1,564,357	1,564,357
G	Taxable Income (E - B - F)	0	3,509,822	863,850	1,342,029	3,349,881	887,079
H	Tax Rate (35%)	0	0.35	0.35	0.35	0.35	0.35
I	Tax Due (G*H)	0	1,228,437.87	302,348	469,710	1,172,458	310,478
J	Net Income After Tax (G - I)	0	2,281,385	561,503	872,319	2,177,423	576,601
K	Net Cash Flow (J + F)	- 8,371,787	3,845,742	2,125,860	2,436,676	3,741,780	2,140,959
L	Cum. Cash flow	- 8,371,787	- 4,526,045	- 2,400,185	36,491	3,778,271	5,919,230
M	CF PV @ 12.6% (Profit)	- 8,371,787	3,415,401	1,676,710	1,706,800	2,327,688	1,182,814
N	Cum. CF PV @ 12.6% (Profit)	- 8,371,787	- 4,956,386	- 3,279,676	-1,572,876	754,812	1,937,625

Interest Rate :	12.6%	
Net Present Value (NPV) :	1,937,625	USD
Internal Rate of Return (IRR) :	22%	
Payback Period (PP) :	3.0	years
Discounted Payback Period (DPP) :	3.7	years

The results from the DCF model, given interest rate of 12.6%, yields the NPV of 1.93 million USD, IRR of 22%, payback period of 3 years, and discounted payback period of 3.7 years as shown in Table 4.21.

As it shows in Figure 4.16, the total capital is 8.3 million USD investment in year zero (0), and the gross revenue from year one is high due to high gold grade as 1.90 ppm. in the first production year. The production gold grade decreases to 1.60 ppm. in year two, 1.54 ppm. in year three, 1.76 ppm. in year four and 1.16 ppm. in year five. The project records a profit in year 4 to year 5. The PP is paid in year 3 and the DPP is in year 3.7.

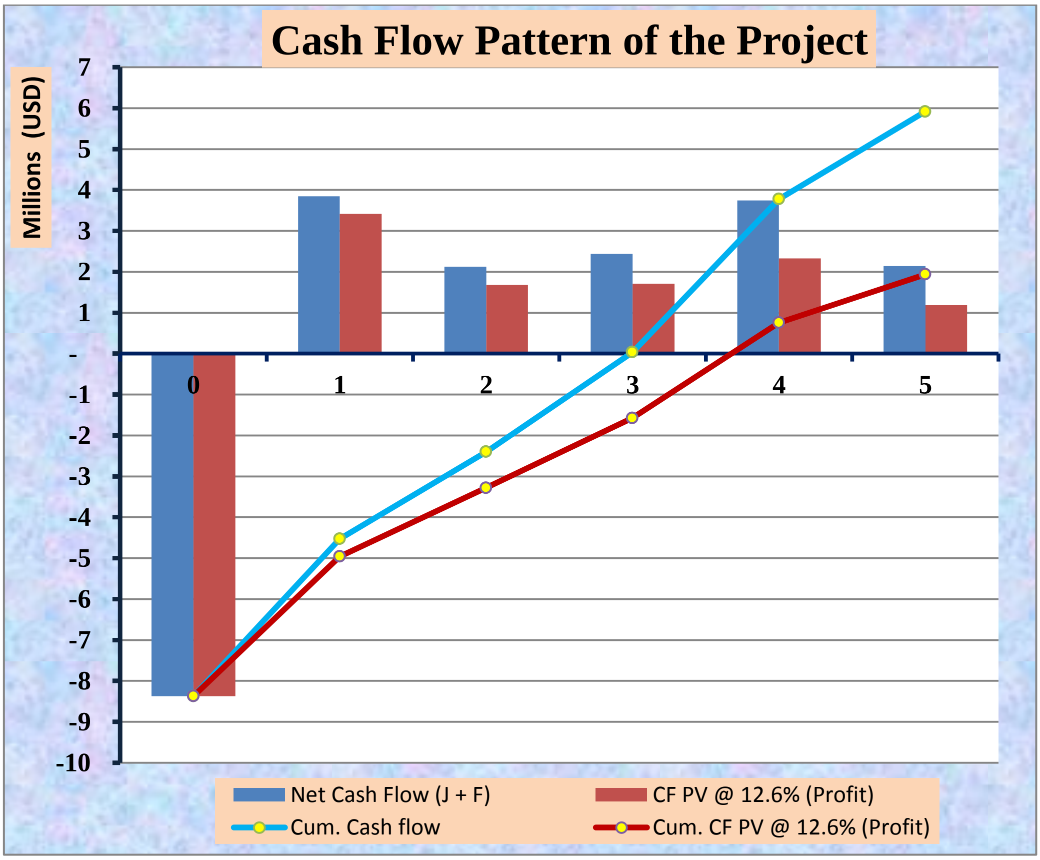


Figure 4.16 Cash Flow Pattern of Project.

Three sensitivity analysis are carried out which are 1) the sensitivity analysis of gold price to NPV and IRR, 2) the sensitivity analysis of tax rate to NPV and IRR, and 3) the sensitivity analysis of royalty rate to NPV and IRR.

The sensitivity analysis of gold price to NPV and IRR determination is summarized in Table 4.22. It can be seen that the NPV and IRR are significantly sensitive to gold price. The gold prices are varied from 1,100 to 1,400 $/oz. When the gold price increase to 1,400 $/oz, NPV is 6.2 million USD, and IRR is 42 percent. But, if the gold price drop to 1,100 $/oz, NPV is negative, and IRR is less than interest rate. This means that the project is not feasible when gold price drop to 1,100 $/oz. The plot of gold price vs NPV at 35% tax rate, 4.5% royalty rate, and the average cutoff grade of 1.53 ppm. is shown in Figure 4.17.

Table 4.22 Sensitivity analysis of gold prize to NPV and IRR

Sensitivity Analysis 1		
Gold Price $/oz	**NPV@12.6%**	**IRR**
1400	6,277,735	42%
1300	4,107,680	32%
1200	1,937,625	22%
1100	- 232,429	11%

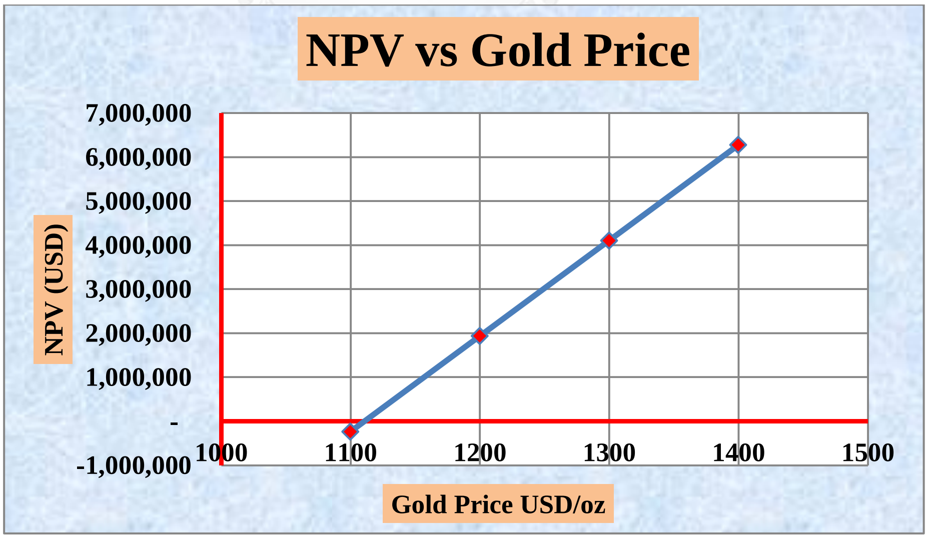


Figure 4.17 Gold price vs NPV (with tax rate 35%, royalty 4.5% and cutoff grade at 1.53 g/t).

For the sensitivity analysis of NPV and IRR to tax rate variation, when the tax rate increase to the maximum of 65 percent as shown in Figure 4.18. NPV becomes negative, and IRR is less than the interest rate, then the project is not feasible. It is recommended that the tax rate of 35 percent is suitable for investors and government. On the other hand, the tax rate is less than 35 percent, NPV and IRR will increase and government will lose tax revenue, as shown in Table 4.23.

Table 4.23 Sensitivity analysis of tax rate to NPV and IRR

Sensitivity Analysis 2		
Tax rate	**NPV@12.6%**	**IRR**
65%	- 256,103	11%
60%	109,519	13%
55%	475,140	15%
50%	840,761	17%
45%	1,206,383	19%
40%	1,572,004	20%
35%	1,937,625	22%
30%	2,303,247	24%
25%	2,668,868	26%

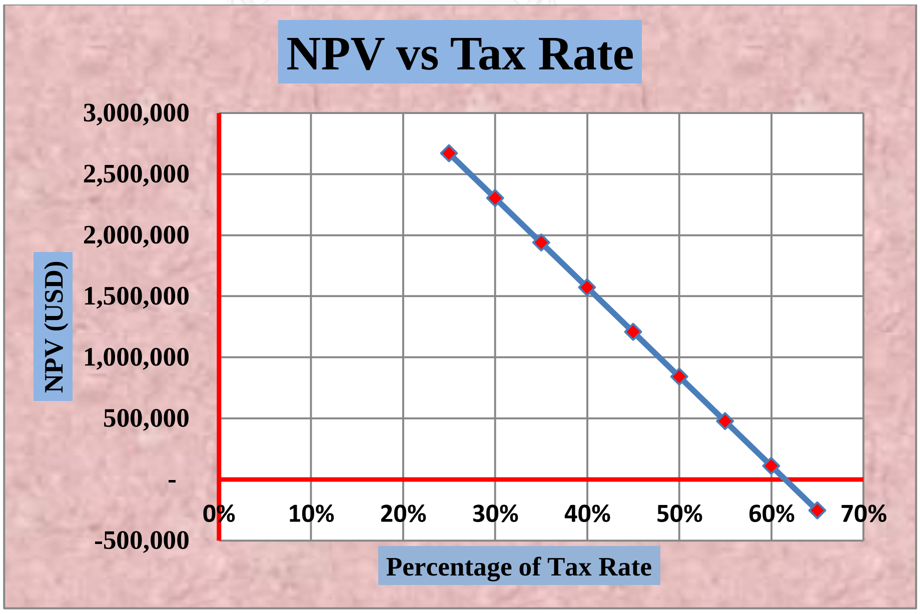


Figure 4.18 Tax rate vs NPV (gold price 1,200 $/oz, royalty 4.5% and cutoff grade at 1.53 g/t).

For the sensitivity of NPV & IRR to royalty rate variation, when royalty rate increase to the maximum of 8 percent, IRR and NPV become negative, as shown in Figure 4.19. On the other hand, when royalty rate decreases to 3 percent, NPV is 2.7 million USD, and IRR is 26 percent as shown in Table 4.24.

Table 4.24 Sensitivity analysis of royalty to NPV and IRR

Sensitivity Analysis 3		
Royalty	**NPV@ 12.6%**	**IRR**
8%	- 65,502	12%
7%	506,820	15%
6%	1,079,142	18%
5%	1,651,464	21%
4.5%	1,937,625	22%
4%	2,223,786	24%
3%	2,796,109	26%

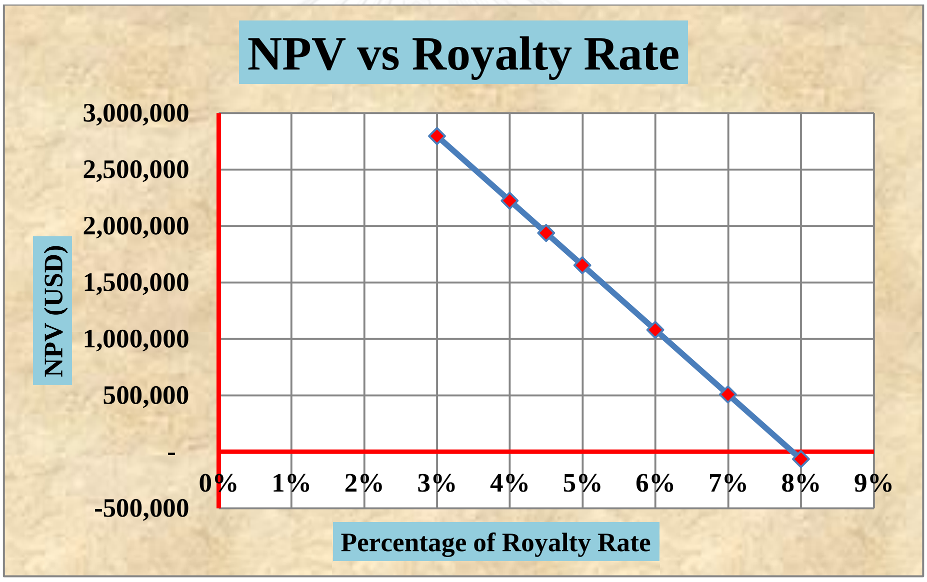


Figure 4.19 Royalty rate vs NPV (gold price 1,200 \$/oz, tax rate 35% and cutoff grade at 1.53 g/t).

4.6.3. The Internal Rate of Return (IRR)

In this study, the calculated IRR is 22 percent which is much higher than WACC (12.6 percent) as shown in Figure 4.20. The higher IRR results from the fact that the sell cost of gold production (1,200 USD per ounce) is much higher than operating cost, processing cost and mining cost of 20 USD/ton, and 1.5 USD/ton, respectively. Therefore, this mining investment project is considered financially feasible.

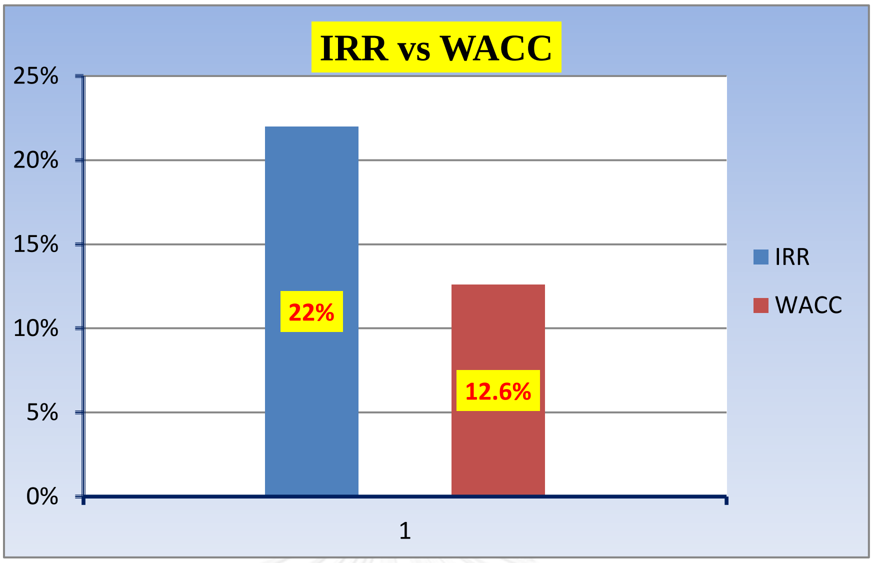


Figure 4.20 The percentage of WACC and IRR comparison.

4.6.4. Net Present Value (NPV)

In this project the NPV is positive accumulated to 1.93 million USD. Therefore, this open pit gold mine considers financially feasible. The calculation of NPV is illustrated in Table 4.21.

The overall financial economic model analysis can be summarized that the WACC is calculated at 12.6 percent with two sources of capital which are 40 percent from the bank loan and 60 percent from the shareholder. The Internal Rate of Return (IRR) is 22 percent which is much higher than WACC rate of 12.6 percent, and NPV of 1.9 million USD.

4.6.5. Operating Cost Comparison

The operating cost comparison includes mainly mine operation and processing cost. These costs comprise of ore mine cost, waste mine cost, blasting cost, transport cost and milling cost. As shown in Figures 4.21 - 4.22, the milling cost is much higher than other costs. While mining cost, blasting cost and transport cost are more ore similar.

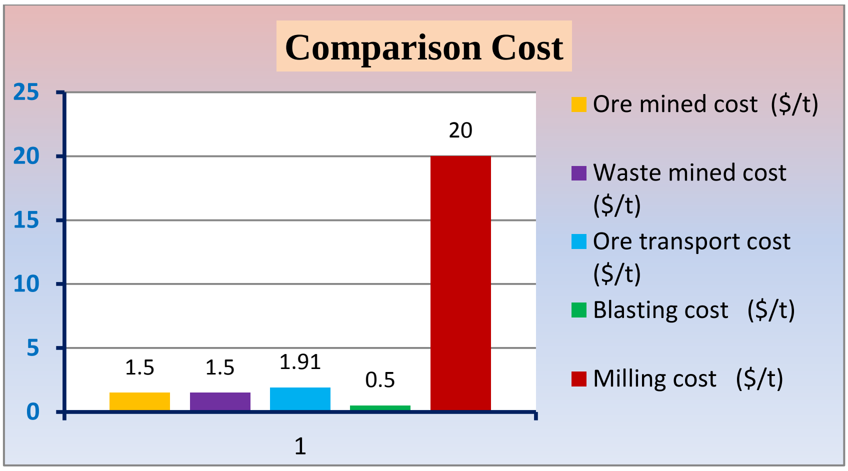


Figure 4.21 Operating cost comparison.

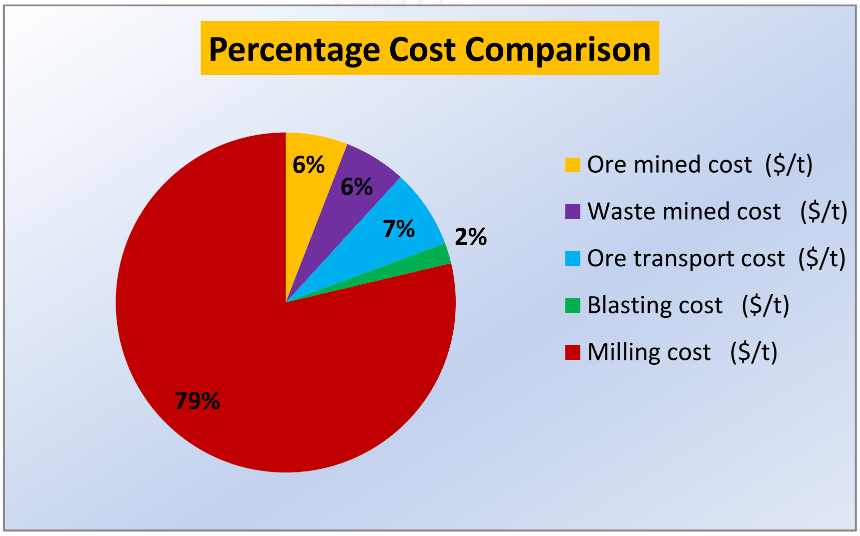


Figure 4.22 Operating cost comparison in percentage.

4.7. Environmental and Social Consideration

4.7.1. Noise Pollution Control Standard

In order to mitigate the impact of the noise in this project, the mining practice has to strictly follow the noise standard from Lao National Environmental Standards for mining operation and processing plant.

As quoted in Table 4.25 , the maximum sound level (L_{max}) should not exceed 11 dB (A), and the L_{eq} 24 hr. must not exceeding 70 dB (A).

Table 4.25 Noise pollution control standard

Standards	Method of Measurement
Maximum Sound Level (L_{max}) should not exceed 115 dB(A)	Equivalent Sound Level (L_{eq}) from Fluctuating Noise
L_{eq} 24 hour should not exceeding 70 dB(A)	Equivalent Sound Level (L_{eq}) from Steady Noise

4.7.2. Vibration Control

This study recommends to adopt the safety standard from vibration of blasting from Australian Standard AS 2817-1993. The recommended guidelines are;

- Historic sites should not have the peak velocity more than 2.0 mm/s.
- Houses or residential areas, should not have the peak velocity more than 10 mm/s.
- The commercial building and other construction site with concrete material should not have the peak velocity more than 25 mm/s.
- The peak velocity should not be more than 5 mm/s and not exceed 5% of amount of blasting.
- The overall maximum peak velocity should not be more than 10 mm/s.

4.7.3. Air Pollution Control

The air pollution standard follows the dust emission standards of Lao National Environmental Standard which state that;

- The particulate matter is less than 10 microns, and not more than 0.12 mg/m^3 average time within 24 hour.
- Dust from mining can be minimized by using a water spray during the mine operation as well as when transporting ore and waste.
- Dust from blasting can be reduced by using the cap delay, and it is recommended to use NONEL (non- electric cap) due to the high safety performance.

4.7.4. Emission Cyanide Standard from the Processing Plant

In an attempt to mitigate the cyanide contamination, it is advised to use a good quality pipelines to transfer cyanide from the processing plant to the tailing dam. The top surface of the tailing dam must be covered by a reliable plastic material, and also at the bottom of the basin in order to prevent the leakage of cyanide into ground water. In addition, it is very important to have a good design and standard construction control of tailing dam in order to prevent failure. In this study, the Lao National Environmental Standard for wastewater discharging is used as a general guideline, as shown in Table 4.26.

Table 4.26 Standard of wastewater discharge of Lao National Environmental Standard

No	Parameter	Symbol	Unit	Maximum concentration
1	Cyanide	CN^-	mg/l	0.1
2	Sulphide	S	mg/l	1.0
3	Potential of Hydrogen	pH	-	6 - 9.5
4	Ammonia Nitrogen	NH_3-N	mg/l	4
5	Silver	Ag	mg/l	0.1
6	Zinc	Zn	mg/l	1.0
7	Iron	Fe	mg/l	2
8	Copper	Cu	mg/l	0.5
9	Lead	Pb	mg/l	0.2
10	Mercury	Hg	mg/l	0.005
11	Oil and Grease	-	mg/l	5

4.7.5. Communities Aspects

In relation to communities and mining relationship, it is recommend to establish the regulation plans in conjunction with the local resident. The mining sector should directly contribute to economic and social development in the mining areas. Initially, mining project should develop road networks and electricity connections. In addition, mining project should provide funds for developing rural areas, such as buildings, schools, hospitals, temples etc. Secondly, mining project has the spillover effects on Small and Medium Enterprises (SMEs) creating enterprises which will facilitate the transfer of technology and improve knowledge and skills to domestic SMEs. It can also generate new business for agriculture, retail trade and livestock farming which will benefit the whole community.

CHAPTER V
CONCLUSIONS AND RECOMMENDATIONS

5.1. Conclusions

From the results of this study, there are many important points that can be concluded according to the results from each steps of work. These results are primary used for the pre-feasibility of open pit gold mine development in the SMD area in Laos. These points are:

1. From the 3D block model, 54 drillholes data with 3,218 assays is prepared in MineSight input format. The drillhole assay interval is 1 meter and gold assay is composited into 5 meters base on mining bench and some factors such as the interface between types of mineralization (oxides, sulfides, transition and primary), high grade and low grade zones. The geological volume extending 700 meters in x direction (Easting), 500 meters in y direction (Northing) and 250 meters in z direction (Thickness).The statistical results of gold assays show a mean grade of 0.58 ppm. and variance of 1.82.
2. The geological block model is constructed with the block discretization of 12.5m x 12.5m x 5 m in (x,y,z) dimensions, resulting a 17,412 block. The Inverse Distance Weight Square method for resource estimation with searching distance of 50 meters is planned for gold resource estimation. The result gives the total geological resource of 34.8 million tonnes. The Grade Tonnage Curve (GTC) is also constructed allowing the sensitivity analysis amongst applied cutoff grade, tonnage, and the average ore grade. In current technology and economic condition, the applied cutoff grade of 0.58 ppm. is selected yielding the corresponding ore tonnage of 2.44 million tonnes, and average ore grade of 1.53 ppm.
3. Pit optimization employs Lerchs - Grossman method in order to achieve the optimum pit design. It yields the mineable reserve of 2.44 million with four years mine life and the average production rate of 600,000 tonnes per year.
4. In this study, mine planning and mine scheduling are planned for production rate of 600,000 tonnes annually with the overall slope angle of 40 degree. The bench slope is 70, degree and the ultimate pit depth is at 55 meters.

5. For blasting design, 18 holes are needed in total within 3 rows and 6 columns blast pattern. In this case, two sizes of blasthole diameter as 102 mm. and 89 mm. are used with the bench height of 10 meter. The production volume for one round of blasting is about 945 m^3, that requires total explosive of 522 kg with two types of explosives as 495 kg of ANFO and 26 kg of Emulsion, and powder factor is 0.25 kg/t with specific drill of 0.11 m/m^3.
6. For material handling, the productivity rate of 160 t/hr requires one wheel loader with bucket size of 1.8 m^3, one excavator with bucket size of 1.6 m^3, and 5 trucks with each capacity of 16.4 m^3.
7. For financial model, the WACC is calculated at 12.6 percent with two sources of capital which are 40 percent borrowing from the bank, and 60 percent from the shareholder. IRR is estimate at 22 percent that is much higher than WACC rate. This project calculates NPV at 1.93 million USD at gold price of 1,200 USD/oz. This mining investment project is proven feasible in terms of technical and financial viability.
8. The environment mitigation is followed standard from Lao National Environmental Standard. The maximum of noise level recommended for blasting is 115 dB linear. For ground vibration, the peak velocity should not be more than 5 mm/s and not exceed 5% of amount of blasting (Australian standard AS 2187 - 1983). It is also recommended to use water truck to spray water during the transport of material.

5.2. Recommendations

It is emphasized that owing to the nature of this gold mineralization district, gold deposit occurs is a multiple clustered area. Therefore, a series of developed open pits along with the gold assay clustered area is envisaged for the full scale development. This study is just one attempt to evaluate and plan for mining development at the one selected gold clustered area. The same procedure can be applied for the other nearby area. In overall, the selection and number of developed pit must be taken into consideration the spatial distribution of gold deposit, the production tonnage and grade, to ensure a stable supply of ore tonnage and the continuity of one pit to the next pit development.

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APPENDIX

VITA

Seelae Phaisopha

Adress: Phetchinda Masion, Urapong, soi 23, Bangkok, Thailand

Tell : 096 7398796

Email : pspseelae@gmail.com

PERSONAL DATA

Nationality : Lao

Marital Status : Single

Sex : Male

Date of birth : 02 July 1985

Place of birth : Salavan Province, Lao PDR

EDUCATION/QUALIFICATION

2013 - present : study Master degree at Chulalongkorn University

2008 - 2013 : student at Polytechnic College, Vietiane Capital, Lao PDR

LANGAUGES

Lao : Mother tongue

English : Good

REFERENCE

Thitisak Boonpramote, Ph.D: Head, Department of Mining and Petroleum Engineering

Asst. Prof. Dr. Sunthorn Pumjan: Lecturer at Department of Mining and Petroleum Engineering

Tell: +66870721626

Email: sunpumjan@gmail.com