THESIS

MINE PLANNING OF FRONT 8 MERANTI PIT AT PT ANG AND FANG BROTHER, MOROWALI REGENCY, CENTRAL SULAWESI PROVINCE

Compiled and submitted by

TONY CHEN

D111181508

MINING ENGINEERING STUDY PROGRAM
FACULTY OF ENGINEERING
UNIVERSITAS HASANUDDIN
MAKASSAR
2022

LEGALIZATION

MINE PLANNING OF FRONT 8 MERANTI PIT AT PT ANG AND FANG BROTHER, MOROWALI REGENCY, CENTRAL SULAWESI PROVINCE

submitted by

TONY CHEN
D111181508

Has been defended in front of the Examination Committee which established for the Completion of Mining Engineering Undergraduate Program of Faculty of Engineering Universitas Hasanuddin
on September 2nd, 2022
and declared eligible

Approved by,

Supervisor,	Co-Supervisor,
Dr. Aryanti Virtanti Anas, S.T., M.T.	Dr. Eng. Rini Novrianti Sutarjo Tui, S.T., M.BA., M.T.
NIP.197010052008012026	NIP.198311143014042001

Head of Study Program,

Asran Ilyas, S.T., M.T., Ph.D.
NIP.197303142000121001

LEMBAR PENGESAHAN

MINE PLANNING OF FRONT 8 MERANTI PIT AT PT ANG AND FANG BROTHER, MOROWALI REGENCY, CENTRAL SULAWESI PROVINCE

Disusun dan diajukan oleh

TONY CHEN
D111181508

Telah dipertahankan di hadapan Panitia Ujian yang dibentuk dalam rangka Penyelesaian Studi Program Sarjana Program Studi Teknik Pertambangan Fakultas Teknik Universitas Hasanuddin pada tanggal 2 September 2022
dan dinyatakan telah memenuhi syarat kelulusan.

Menyetujui,

Pembimbing Utama,	Pembimbing Pendamping,
Dr. Aryanti Virtanti Anas, S.T., M.T.	Dr. Eng. Rini Novrianti Sutarjo Tui, S.T., M.BA., M.T.
NIP.197010052008012026	NIP.198311143014042001

Ketua Program Studi,

KEMENTERIAN PENDIDIKAN, KEBUDAYAAN, RISET, ... UNIVERSITAS HASANUDDIN FAKULTAS TEKNIK

Asran Ilyas, S.T., M.T., Ph.D.
NIP.197303142000121001

STATEMENT OF AUTHENTICITY

The undersigned below:

Name	: Tony Chen
Student ID	: D111181508
Study Program	: Mining Engineering
Degree	: Undergraduate (S1)

Declare that my bachelor thesis titled

Mine Planning of Front 8 Meranti Pit at PT Ang and Fang Brother,

Morowali Regency, Central Sulawesi Province

is my own writing and not a claim of others. The thesis which I wrote is really my own work. If in the future, it is proven or can be proved that a part or a whole of this thesis is the work of others, I am willing to accept any penalty for such act.

Makassar, September 23th 2022

Tony Chen

ABSTRACT

PT Ang and Fang Brother is a company engaged in laterite nickel mining commodities in Morowali Regency, Central Sulawesi Province. The company has several pits, but the pit that is currently operating is the Meranti Pit with Front 8 as the unmined pit. Mining is an industry that requires good planning activities so that it can be a guide for the implementation of mining operations, easier to achieve goals, and maximize profits. This study aims to conduct mine planning on Front 8 which was carried out using Micromine v.2021.5 and Microsoft Excel. The estimation result of ore resources are 285,983 tons. The optimal pit limit is at pit shell 8 with an NPV of $3,276,964.81. The results of the evaluation of ore reserves in the pit design are 223,125 tons with an overburden of 489,219 BCM. The ore based on the result of the sequence design from sequence 1 to 4 are 69,609; 56,485; 66,562; and 30,469 tons, while overburden are 82,031; 167,813; 153,079; and 86,296 BCM. The selection of loaders from sequence 1 to 4 are 2, 3, 3, and 3 units, while haulers are 15, 16, 18, and 15 units. The results of the cash flow analysis show the NPV of USD 2,791,081.76, IRR of 86.96%, and PBP of 1.17 month. The results of the sensitivity analysis show that changes in nickel prices affect NPV more than changes in operating costs. The mine planning result indicates that this project is feasible to be mined.

Keywords: equipment, laterite nickel, mine planning, net present value, pit design.

ABSTRAK

PT Ang and Fang Brother merupakan sebuah perusahaan tambang nikel laterit yang terletak di Kabupaten Morowali, Provinsi Sulawesi Tengah. Perusahaan ini memiliki beberapa pit *penambangan, namun untuk saat ini* pit *yang sedang beroperasi adalah* Pit *Meranti dengan* Front *8 sebagai salah satu lokasi yang belum ditambang. Pertambangan merupakan industri yang memerlukan kegiatan perencanaan yang baik dalam setiap tahapan operasinya sehingga dapat menjadi pedoman pelaksanaan operasi penambangan, lebih mudah mencapai tujuan, dan memaksimalkan keuntungan. Penelitian ini bertujuan untuk membuat perencanaan tambang pada* Front *8 menggunakan perangkat lunak* Micromine v.2021.5 *dan* Microsoft Excel*. Hasil estimasi sumberdaya bijih nikel laterit adalah 285.983 ton.* Pit limit *optimal berada pada* pit shell *8 dengan NPV sebesar $3.276.964,81. Hasil evaluasi cadangan pijih pada desain* pit *adalah 223.125 tons dengan lapisan tanah penutup yaitu 489.219 BCM. Cadangan bijih berdasarkan desain sekuen secara berurut dari sekuen 1 hingga 4 adalah 69.609, 56.485, 66.562, and 30.469 tons, sedangkan lapisan tanah penutup sebesar 82.031, 167.813, 153.079, dan 86.296 BCM. Pemilihan alat gali-muat dari sekuen 1 hingga 4 adalah 2, 3, 3, dan 3 unit, sedangkan alat angkut adalah 15, 16, 18, dan 15 unit. Hasil analisis aliran kas menunjukkan nilai NPV sebesar USD 2.791.081,76, IRR sebesar 86,96%, dan PBP sebesar 1,17. Hasil analisis sensitivitas menunjukkan bahwa perubahan harga nikel lebih mempengaruhi NPV dibanding perubahan biaya operasional.*

Kata kunci: desain pit*,* net present value*, nikel laterit, peralatan, perencanaan tambang.*

PREFACE

Namo Tassa Bhagavato Arahato Sammāsambuddhassa. The thesis with the title Mine Planning of Front 8 Meranti Pit at PT Ang and Fang Brother, Morowali Regency, Central Sulawesi Province, which was made with perseverance, sincerity, and an unyielding spirit was finally completed. This thesis is the result of research that has been carried out at PT Ang and Fang Brother, Lalampu Village, Morowali Regency, Central Sulawesi Province and the Mine Planning and Valuation Laboratory, Mining Engineering Department, Faculty of Engineering, Hasanuddin University during the world current COVID-19 pandemic.

The author realizes that this thesis could not be separated from the help of various related parties. For all the help, guidance, support, and suggestions in this research activity, the authors would like to thank PT Ang and Fang Brother for the research location. The author would also like to thank Mr. Mathius Mangguwali, S.T. as Project Manager, and Mr. Enrico Pribadi, S.T. as a Junior Mine Plan Engineer who has dedicated their time to guide the author while working. The author also would like to thank Mrs. Nurul Amalia, S.Psi as HRD, Mr. Leri, Ms. Inri, and Mr. Jordi who always pay attention and provide encouragement and motivation to the writer while at the company. Thank you also to the canteen ma'am's who always pay attention to the author, and provide delicious, and nutritious food.

This thesis would not be successful without the motivation, advice, and assistance of those who always guide the author at the Mining Engineering Study Program, Faculty of Engineering, Hasanuddin University. Therefore, the writer would like to express many thanks to Mrs. Dr. Aryanti Virtanti Anas, S.T., M.T. as the Supervisor, Mrs. Dr. Eng. Rini Novrianti Sutardjo Tui, S.T., M.BA., M.T. as the Co-Supervisor, Mr. Dr. Ir. Irzal Nur, M.T. and Mr. Dr. Sufriadin, S.T., M.T. as the examiner at the thesis seminar. Thanks also to Miss. Rizki Amalia, S.T., M.T. as a supervisor for the Mine Planning and Valuation Laboratory as well as to all the lecturers who have educated the author from the first semester until now. The author also thanks the academic staff of the mining engineering department who were very helpful in the management towards the end of the author's study.

The author also expresses his gratitude to the teachers and friends who are always there to encourage the writer when the writer is bored. Of course, the author also expresses his deepest gratitude to friends from TUNNEL 2018, the big family of the Assistant for Basic Physics Laboratory FT-UH, members of the Mine Planning and Valuation Laboratory, as well as friends from various study programs and batches at the Faculty of Engineering, Hasanuddin University who became witness the author's journey in lectures.

It is not easy for the writer to finish all the contents of this thesis. Therefore, the author would like to thank Mr. Chin Hoa Win and Mrs. Sherly Tjahyadi for their great love and struggle to provide college opportunities for the author, as well as brother, sister, and extended family, who always provide prayers, encouragement, and good moral and material support, as well as a source of strength when the author is bored.

The author realizes that there are still many shortcomings in this thesis. Therefore, the author apologizes for any shortcomings encountered in the preparation process. May the good that has been done beget blessings and be a protective force. May Tiratana always protect us.

Makassar, July 2022

Author

TABLE OF CONTENTS

LIST OF FIGURES

Figure Page

Figure Page

Figure Page

LIST OF TABLES

Table	Page

Table Page

APPENDICES

CHAPTER I

INTRODUCTION

1.1 Background

PT Ang and Fang Brother is a company engaged in laterite nickel mining commodities located in Lalampu Village, Bahodopi District, Morowali Regency, Central Sulawesi Province. PT Ang and Fang Brother has two types of IUPs, there are IUP 576 hectares and IUP 199 hectares. Currently, mining operations only take place in the IUP area of 576 hectares with a cut-off grade of 1.3%. The company has several pits, including the Mawar Pit, the Aulia Pit, and the Meranti Pit. The pit that is currently operating is the Meranti Pit. There are eight fronts at the Meranti Pit, six of which are in the mining process and the other two fronts that have not been mined, there are Front 1 and Front 8. Currently, Front 8 is still in the mine planning stage.

Front 8 is one of the locations in the mine planning stage located in the Southeastern part of the Meranti Pit. This front is planned for laterite nickel ore mining with an area of 5.68 hectares. There are 21 drill points with a distance between drill points of 50 meters on this Front to ensure the level of distribution of laterite nickel resources. The drilling results are implemented in a block model for further mine planning stages.

Mine planning involves compiling and incorporating suitable geotechnical, geochemical, geological, mining, engineering, and economic data in order to establish an approach to exploit a specific mineral deposit within the legal and regulatory requirements. Planning and design in mining operations aim to enable the extraction and processing of a mineral deposit at the desired market specification, at a minimum unit cost, and under the existing economic conditions (Genc and Mitra, 2018). Things that need to be considered in making mine planning are the determination of pit boundaries, sequence design, production scheduling, equipment selection, and calculation of operating and capital costs (Hustrulid et al, 1995).

Determining the ultimate pit limit (UPL) is one of the most challenging topics in surface mining that should be investigated in the preliminary stages of mining operations. It should be determined when some parameters, such as the profitability of the mining project, are proven after the exploration stage (Sadeghi et al, 2020). The purpose of pit boundary delimitation is to determine the final opening limit of the ore body and the associated grade and tonnage, which will maximize some economic value or technical criteria while meeting practical operational requirements (Djilani, 1997). The objective of all pit optimization algorithms is to determine the pit design which maximizes the value of the mine (Whittle and Vassiliev, 1997).

Mine design is needed to estimate or predict an area of potential ore resources to be developed into a mining pit location. The design is the determination of detailed and definite requirements, specifications and criteria, and techniques to achieve the goals and objectives of activities as well as the technical sequence of their implementation (Hustrulid et al, 1995).

The next planning step is to identify a mining sequence for these ore blocks, so the order of mining creates a crushed product that remains acceptably close to the target grade in each of the important analytes (Everett, 2011). An open pit mine production sequencing problem can be defined as determining the sequence of material extraction. Production sequencing is termed as block sequencing if the objective is to determine the sequence of block extraction. Expansion of an open pit mine in a mine’s life is done in a series of mining phases referred to as pushbacks or mining cuts. From the planning point of view, pushbacks are designed to maximize payback from a mine (Nogholi, 2015).

The next stage of production planning is production scheduling. Production planning is part of a hierarchical planning, capacity/resource allocation, scheduling, and the control framework. The production plan considers resource capacities, periods, supply, and demand over a reasonably long planning horizon at a high level. The scheduling function typically dispatches the jobs according to their perceived or assigned priorities to align the processing sequence of the jobs with the production plan while using the latest information on job and machine availabilities (Cai et al, 2011).

The final stage of the mine planning process is the assessment of reserves by determining economics (Huang et al, 2022). The mining industry is a very risky industry when compared with the other industries. Therefore, economic analysis and engineering economy are extremely important for the evaluation of a new mine or an operating mine. To reduce projects, investment risk, and make the correct decisions for enterprises, it is necessary to there are some investment decision-making methods, such as Net Present Value (NPV), Internal Rate of Return (IRR), Payback Period (PBP) (Güyagüler and Erdem, 2011).

Based on this description, this research was conducted to carry out mine planning at the Meranti 8 Pit Front, starting from determining optimal mining limits, designing pits, designing mining sequences, scheduling production, to mining investment analysis. This plan is important as a reference for the company in carrying out its actual mining operations.

1.2 Research Problems

Front 8 is an area of the Meranti Pit planned to be mined at the beginning of February 2022. So, Front 8 requires a mine planning study in the form of resource estimation, determination of optimize pit boundary, design of mining sequences, selection of heavy equipment, cash flow, and sensitivity analysis. Therefore, the problems from this research are:

1. How much are the estimated nickel laterite resources in Front 8?
2. Where is the most profitable pit boundary?
3. How is the pit design of the Front 8 Meranti Pit and how much are the evaluated laterite nickel reserves based on the pit design created?
4. How many mining sequences can be formed and how many reserves are contained for each sequence?
5. How many digging-loading and hauling equipment are needed for each sequence?
6. What are the results of the cash flow analysis from Front 8?
7. How is the sensitivity analysis of the obtained NPV based on changes in nickel prices and operating cost?

1.3 Research Objectives

Based on the description of the research questions, the objectives of this research are:

1. To estimate nickel laterite resources in Front 8 based on the block model and topography data.
2. To conduct pit limit optimization studies based on the technical and economic parameters.
3. To create a pit design and evaluate nickel reserves based on the results of optimization studies with several geotechnical parameters.
4. To create mining sequences based on pit geometry parameters and production target per month.
5. To calculate the number of digging-loading and hauling equipment needed for mining operations in each sequence based on data on the average cycle time of

heavy equipment, delay time per day data, equipment availability data, and heavy equipment bucket capacity data.

6. To perform cash flow analysis with NPV, IRR, and PBP parameters based on the mine planning results.
7. To conduct NPV sensitivity analysis based on changes in nickel prices and operating cost.

1.4 Research Benefits

This study discusses topics that are very important to be determined before the actual mining plan begins and is used as a basis for reference or guidance from the actual implementation of activities in the field. This research is useful for the development and application of insights, the knowledge possessed by researchers, and the application of technological advances in mine planning.

1.5 Research Location

The Final Project research activity was carried out at PT Ang And Fang Brother Site Lalampu. The company is located in Lalampu Village, Bahodopi District, Morowali Regency, Central Sulawesi Province. The Final Project research location map is shown in Appendix A.

Lalampu Village astronomically is located at coordinates 122°05'0" to 122°05'10" South Latitude and 2°47'25" to 2°47'40" East Longitude. Lalampu Village geographically is bordered by several areas, namely:

1. The north area is bordered by the Banda Sea,
2. The east area is bordered by Bahodopi Village,
3. The south area is bordered by a BDM (Bintang Delapan Mineral) company IUP.
4. The west area is bordered by Siung Batu Village.

PT Ang and Fang Brother is approximately 385 km from Makassar City, South Sulawesi Province. Access to the research location can be reached by air transportation and land transportation. The journey is taken by land using a bus from Makassar City for approximately 24 hours to Bungku City, then from Bungku City, the journey continues by a car for approximately one hour to the location.

1.6 Stages of Research Activities

There are several stages carried out in achieving the objectives of this research, there are:

1. Preparation

   Preparation is the initial stage carried out before conducting research. At this stage, a research plan is prepared, submitted, and consulted with the Head of the Laboratory. At this stage, administrative preparations are also carried out in the form of research cover letters at both the department and faculty level, searching for research locations, and completing the necessary equipment during the research.

2. Literature study

   Literature study is an activity related to the method of collecting library data, reading and taking notes, as well as processing research materials that have been prepared by the relevant divisions to complete the data needed during the research. The stages of this literature study start from the preparation stage to the data processing and analysis stage.

3. Data collection

   At this stage, the data needed basing on the problem formulation and the results of the literature study were collected. The data needed for this research are primary. Primary data is data taken directly at the research site.

4. Data processing and analysis

   The stages of data processing and analysis are carried out to obtain research results. The results of data processing and analysis are presented in the form of tables and figures. In this study, data processing and analysis were performed using Micromine v.2021.5 software.

5. Presentation

   At this stage, research reports, especially the results of data analysis and processing, are presented in front of supervisors, examiners, and participants.

6. Report collection

   At this stage, the report of the research has been done is collected. This stage is the final stage of research activities. Reporting is done when data processing and analysis have been completed as a whole. The report that have been made are then compiled systematically following the guidelines for writing research report that has been set by the Department of Mining Engineering, Faculty of Engineering, Hasanuddin University.

CHAPTER II

MINE PLANNING OF THE LATERITE NICKEL DEPOSIT

2.1 Laterite Nickel

Laterite is a residual product of chemical weathering of rocks on the earth's surface, where various original or primary minerals are unstable in the presence of water, dissolve or decompose and form new minerals that are more stable to the environment. Laterite is important as a host for economic ore deposits, because the chemical interactions from the lateralization process in certain cases can be very efficient in concentrating some elements. Well-known examples of laterite ore deposits are alumina bauxite and fortified iron ore deposits, but lesser-known examples include lateritic gold deposits (Elias, 2002).

Nickel laterite is a product of the lateralization of Mg-rich or ultramafic rocks containing 0.2-0.4% primary Ni (Golightly, 1981; Butt and Cluzel, 2013). These rocks are generally dunites, harzburgites, and peridotites which form in ophiolite complexes. Lateralization processes produce concentrations by a factor of three to 30 times the nickel and cobalt content of the host rock. The process and character of the resulting laterite depend on regional and local scales by the dynamic interaction of factors such as climate, topography change, tectonics, type, and primary rock structure (Elias, 2002).

Nickel laterite plays an important role in the global nickel industry and currently accounts for about 40% of the total nickel production of approximately one million tonnes. Approximately 70% of all continental or terrestrial nickel resources are contained in laterites (Elias, 2002; Khirbash, 2014; Ito et al, 2021). Laterites that develop in ultramafic/mafic rocks are known to contain important deposits of nickel

(Ni), cobalt (Co), iron (Fe), and aluminum (Al) (Khirbash, 2014). The global distribution of laterite nickel is shown in Figure 2.1 (Elias, 2002).

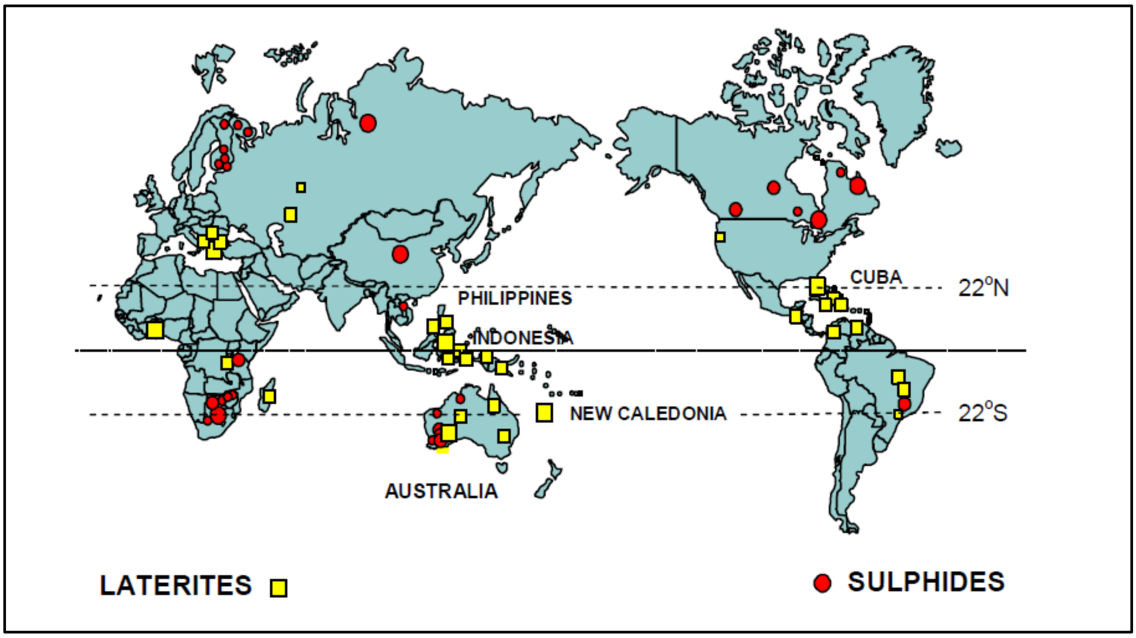
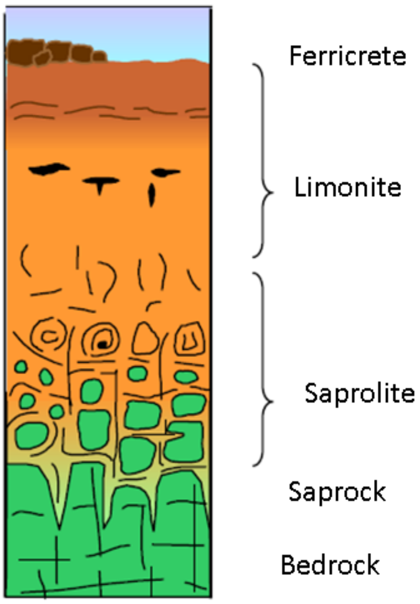


Figure 2.1 Global distribution of laterite nickel (Elias, 2002)

Although it has been mined for about 140 years, by 2000 nickel laterite had accounted for less than 40% of global Ni production, the remainder coming from sulfide ores. The development of nickel laterite mining at that time was relatively slow (Butt and Cluzel, 2013). The main obstacles in the production of nickel laterite are the high capital costs for the processing facilities, the high energy requirements in the pyrometallurgical process line, and the technical challenges to make hydrometallurgical processing more efficient (Elias, 2002; Butt and Cluzel, 2013).

This day, nickel laterite has been discovered and mined in many areas with production having increased due to greater demand, new processing technologies, and reduced availability of sulfide ores. Total production of Ni from laterite ores has increased to 46% of global supply in 2008, exceeding 50% in 2010 and reaching 60% in 2014. Nickel laterite also contributes 20-30% of total Co supply (Butt and Cluzel, 2013).

The lateralization process is chemical weathering that occurs in a humid climate for a long period time under conditions of relative tectonic stability to allow the formation of thick regolith with distinctive characteristics. In summary, the lateralization process involves the decomposition of primary minerals and the release of some of their chemical components into groundwater, leaching of mobile components, concentration of immobile or insoluble components residues, and formation of new stable minerals in the soil. The effects of mineral transformation and differential mobility of the elements involved produce a layered mantle over the host rock which is referred to as the laterite profile. An illustration of the laterite nickel profile is shown in Figure 2.2 (Elias, 2002).



Fe	MgO	Ni	Co
>50%	<0.5%	<0.6%	<0.1%
40-50%	0.5-5%	0.8-1.5%	0.1-0.2%
10-25%	15-35%	1.5-3%	0.02-0.1%
5%	35-45%	0.3%	0.01%

Figure 2.2 Illustration of laterite nickel deposit profile (Elias, 2002)

The profile of laterite nickel deposits is generally divided into three, namely (Bargawa et al, 2021):

1. Limonite zone

   In the limonite zone, there are residual concentrations of non-mobile chemical elements such as Fe, Al, and Mn, while mobile elements such as Ca, Na, K, Mg, and

Si are chemically leaching. The upper part of this zone is yellow limonite which is rich in gutite, while the lower part is red limonite which is rich in hematite. Gutite can be remobilized under acidic conditions to form an iron cap (Fe^{2+} to Fe^{3+}). Insoluble minerals such as spinel, magnetite, maghemite, and talc remain in this zone, not showing the structure and texture of the original rock. The base of the limonite zone is enriched by manganese, cobalt, and asbolite or manganese lumps. The density of limonite is greater than the saprolite zone due to the Fe mineral. The basis for the classification of the limonite zone is based on chemical elements. The classification is Fe (>25%), MgO (<5%), and Ni (<1.5%).

2. Saprolite zone

The saprolite zone is yellowish-brown, greenish-brown, and greenish-yellow. Saprolite is a strongly altered zone where active chemical weathering occurs. The chemical weathering occurs along fractures, cleavages, and fractures in rocks as well as micro-fractures in crystals. The original rock texture and structure are still visible, and most of the original rock minerals are still the same. This zone consists of the original rock fragments, silica, and garnierite which are formed by chemical re-deposition.

In the unserpentinized peridotite, the saprolite formation process is limited to the rock surface because water is difficult to penetrate the rock. Henceforth, the nickel is barely found in the rock chunks. In the serpentinized peridotite, the rock becomes softer and allows water to enter so that the saprolite formation process can occur throughout the rock mass. Rock chunks in the saprolite zone contain significant amounts of Ni. The grain size gradation is increasing to a coarser size along with the depth. The basis for the classification of saprolite zones is based on chemical elements, namely Fe (less than 25%), MgO (more than 5%), and Ni (more than 1.5%).

3. Bedrock zone

The bedrock zone has grayish-black, greenish-black colors or depending on the rock composition. The composition consists of the original rock such as dunite, peridotite, or other ultrabasic rocks. This zone does not contain any economic minerals anymore. The fracture zone at the top is filled with silicate minerals such as garnierite, serpentine, chrysoprase, or other silicate minerals. This fracture is thought to be the cause of the root zone, namely the high-grade Ni zone, but its position is hidden. Generally, fresh, and massive bedrock can be found at the bottom with zone fracturing.

The laterite nickel ore deposit profile explained above is shown in Figure 2.3 (Bargawa et al, 2021).

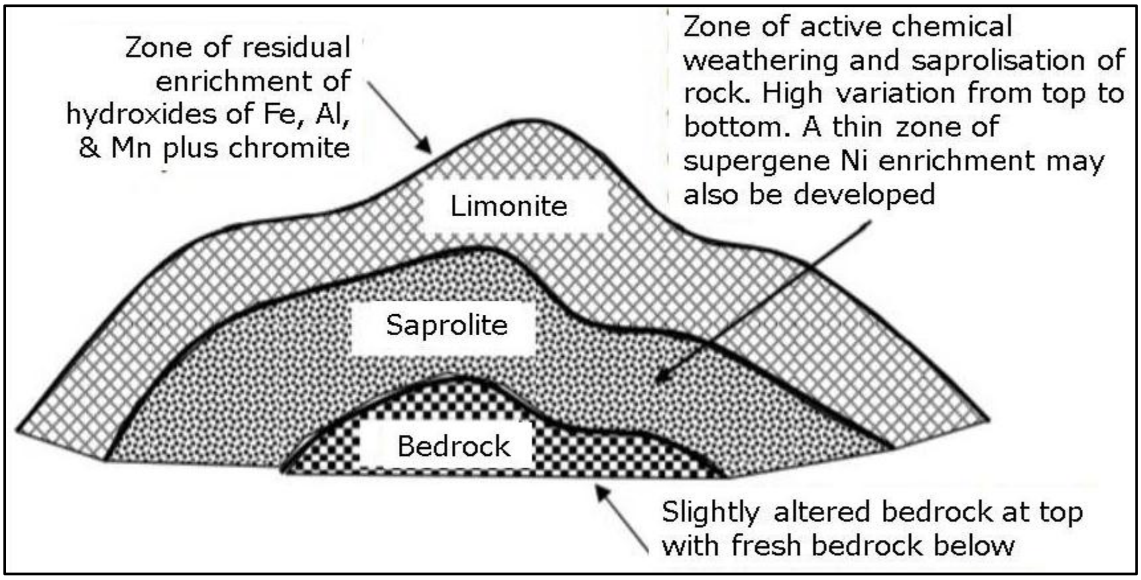
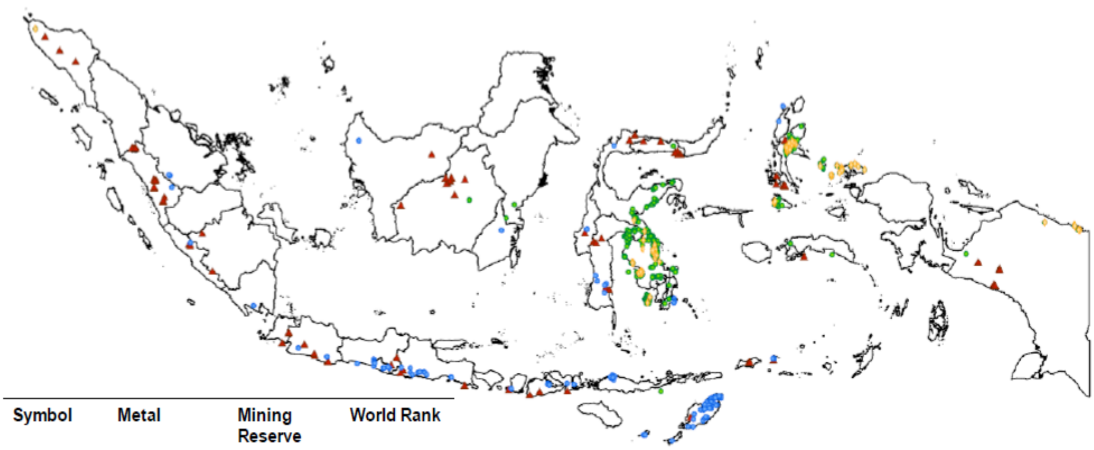


Figure 2.3 Profile of laterite nickel deposits (Bargawa et al, 2021)

Nickel is commonly present in two principal ore types, there are sulfide and laterite ores. Sulfide ores are typically derived from volcanic or hydrothermal processes and usually include copper (Cu) and/or cobalt, and sometimes other precious metals such as gold or platinum and palladium (generally grouped as platinum group metals or PGMs). Laterite ores are formed near the surface following extensive weathering and occur abundantly in tropical climates around the equator or arid regions of central Western Australia or Southern Africa (Mudd, 2009).

Indonesia is the world's largest supplier of nickel ore; the nickel laterite resources are mainly distributed in the Moluccas and Sulawesi regions (Zhang et al, 2020). The laterite nickel resources distribution in Indonesia is shown in the Figure 2.4 below (Prasetyo, 2016).



Symbol	Metal	Mining Reserve (million ton)	World Rank
●	Nickel	21	1st
▲	Copper	51	5th
◊	Cobalt	Unknown	Unknown
Unknown	Aluminum	1200	6th
●	Manganese	43	6th

Figure 2.4 Distribution of laterite nickel in Indonesia (Pandyaswargo et al, 2021)

Many large-scale Ni laterite deposits have been found on Sulawesi Island, Indonesia, and are well developed in peridotite from the obducting ophiolite suite subjected to hot and humid climatic conditions. This deposit allows Indonesia to become the world's leading producer of Ni. The Soroako and Pomalaa mining areas located in the central and southeastern parts of Sulawesi Island, respectively, are the two largest Ni laterite mining sites on Sulawesi Island and mostly produce Ni-rich saprolite ore (Ito et al, 2021).

The biggest nickel consumer globally is Asia, followed by Europe. The steel industry’s global recover, showed China as the greatest nickel user. But in 2008 because of the global economic recession, there has been a global dropping down of nickel consumption and production. Nickel usage is constantly increasing over the years and will lead to the exploration and development of new nickel deposits area and the utilization of new lower-cost metallurgical processes until now (Apostolikas et al, 2009).

2.2 Mine Planning

The strategic mine plan sets out the overall objectives of the mining project. Mine planning is a multidisciplinary action and the aim is to develop an annual extraction plan to meet some predetermined objectives. Mine plans are classified into long-term, medium-term, short-term, and operational plans. Typically, these plans are arranged in such a way that the mining operation generates the highest cash flow or net present value (Osanloo and Rahmanpour, 2017).

Open pit mining is an excavation carried out at ground level to extract ore to the surface during the mine's operational time. To expose and mine the deposit, large amounts of waste rock must be excavated and removed. To achieve the lowest possible cost with the maximum expected profit, open pit mine planning is closely related to the economy, which in turn is influenced by several geological conditions and mining engineering (Jiskani, 2017).

Mine planning consists of several types, there are (Osanloo and Rahmanpour, 2017):

1. Long-term planning (yearly), is modeling of plans for a period of more than one year on an ongoing basis.
2. Mid-term planning (quarterly), is modeling work plan for three months period.
3. Short-term planning (daily/weekly), is modeling of plans for a daily or weekly period.

Things that need to be considered in mine planning are (Hustrulid et al, 1995):

1. Determination of pit limit

   The purpose of determining the pit limit is to determine the final limit of the mining process, where a Mine Plan must be able to plan how much excavated material will be mined, but in determining the pit limit it still does not take into account time and costs.

2. Sequence design

   In mining geometry design, sequence design is an important stage because at this stage it makes the determination of the pit limit into smaller parts, making it easier to work with, and in designing the three-dimensional shape of the mine it becomes easier too.

3. Production scheduling

   The next stage after sequence design is production scheduling. At this stage, it can be determined the amount of overburden with the amount of excavated material to be mined and the amount of heavy equipment needed in a certain period based on the time sequence and production targets set by the company.

4. Equipment selection

   After knowing the production to be achieved, the next stage is the selection of the equipment that will be used in the mining activity. In addition to the selection of production equipment, they are also selected for the mine development process.

5. Calculation of operating and capital costs

   The next stage in mine planning is the calculation of operating and capital costs. The calculation of operating and capital costs is based on the production targets to be achieved and the selection of tools to be used. In addition, at this stage, it can also be determined the amount of working time and work shifts needed to achieve production targets that have been planned by the company.

2.3 Block Model Method

Three-dimensional (3D) geological modeling is an important part of geological research and geological data visualization because it can clearly reveal information such as the shape and structure of underground geological bodies, providing an important role for geological exploration, risk assessment, and so on in intelligent geology. Thus, the accuracy and reliability of 3D geological modeling have a direct impact on geological activities in practice (Liu et al, 2021). An illustration of resource modeling based on topography is shown in Figure 2.5 (Hustrulid et al, 2013).

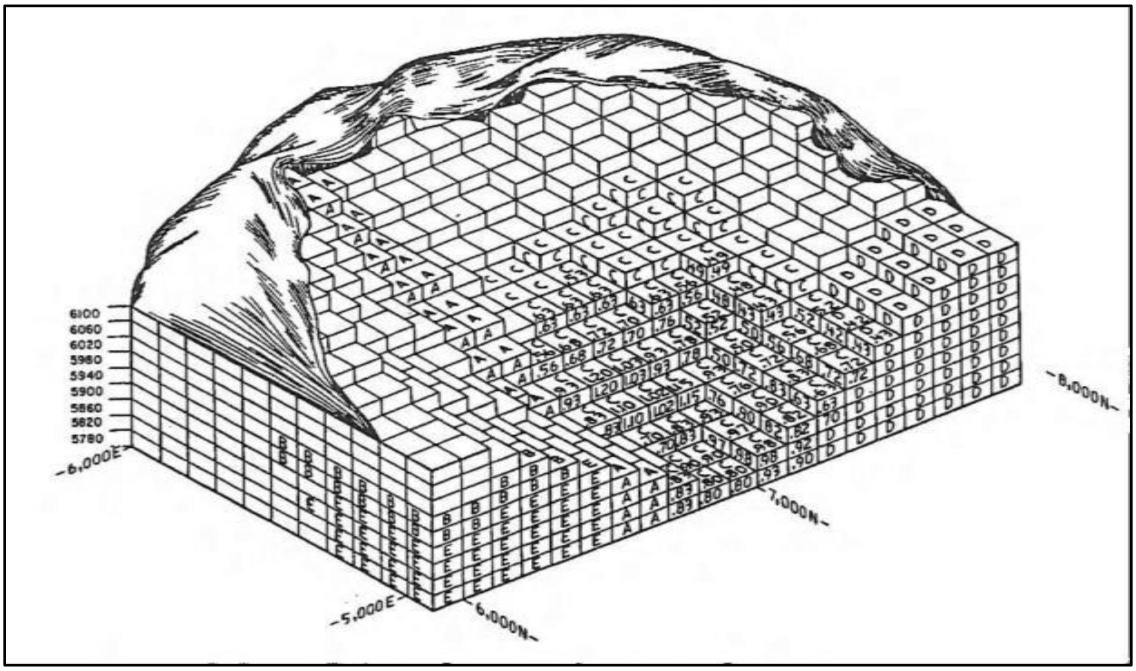


Figure 2.5 Illustration of resource modeling on topography (Hustrulid et al, 2013)

The block model is a simple representation of the ore body and its surroundings which can be thought of as a computer-generated pile of bricks representing the small volumes of rock in the deposit (ore and waste). Each brick, or cell, contains approximate data, such as element level, density, and values of other geological or engineering entities (Poniewierski, 2019). Block models are convenient tools for mine evaluation, resource estimation and mine planning, including pit or stope optimization, and mine scheduling. The vast majority of mineral resource estimates are obtained using block models (Rossi and Deustch, 2014).

Mine planning activities must pay attention to the efficient use of mineral resources. Therefore, reliable mineral deposit models, based on geological models, faults of the deposit, and exploratory borehole data, are essential. These models should be based on conventional and geostatistical grade and tonnage assessment techniques, incorporating errors and uncertainties associated with exploratory data or other uncertainties regarding the spatial tonnage or variability of the grade distribution. This model can be used for optimizing the pit boundary as well as for optimizing long and short-term mine development (Roumpos and Papacosta, 2013).

The block size should be decided based on the drill hole data spacing and other engineering considerations. Larger blocks are easier to estimate than smaller blocks in the sense that the predicted grades are more likely to be close to the actual grade of the block. On the other hand, too large a block size is not useful for pit optimization and mine planning (Rossi and Deustch, 2014).

Typical mine planning packages work on smaller blocks that discretize to the time interval on which the mine plan is based. For example, for long-term planning, incremental monthly planning units are frequently used. The block size should represent a suitable incremental tonnage. Mining engineers will sometimes sub-divide large panels into smaller units if necessary; the easiest choice is to assign the same panel grade to all smaller units (Rossi and Deustch, 2014).

The block size should be less than the data spacing. Journel and Huijbregts (1978, Sect. 5) propose a block size from 1/3 to 1/2 of the drill hole data spacing as an approximate guideline. The logic behind this recommendation is that smaller block sizes will produce artificial smoothing of the model. Adjacent small blocks will receive about the same grade if the same drill hole data are used to estimate them. Too large a block size for the drill hole data grid will not fully utilize the resolution available from the drill hole data. In the context of geostatistical simulation, the grid node spacing and final block size do not depend on data spacing as is shown in Figure 2.6 (Rossi and Deustch, 2014).

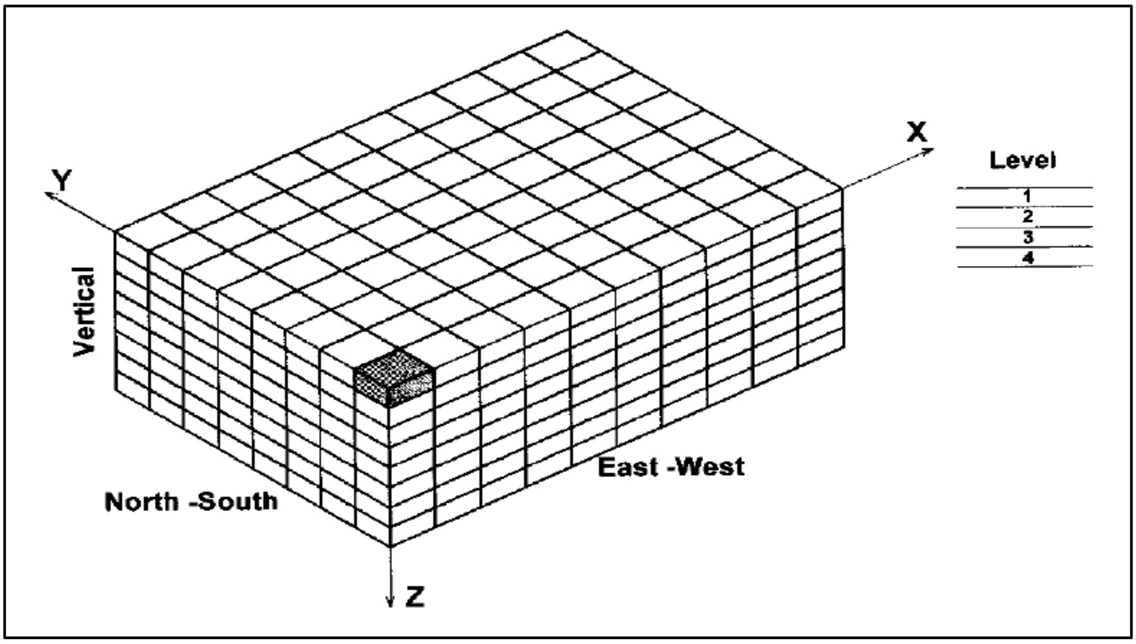


Figure 2.6 Visualization of 3D block models (Khalokakaie et al, 2001)

Block size is a function of the mineralization geometry in the study area and the mining system to be used in three-dimensions (3D) block model sketch. The variables needed for modeling are the topography of the study area, geological information, mineral grade, rock type, density, percentage of blocks as part of ore, and tonnage of each block (Khalokakaie et al, 2001).

2.4 Resource Estimation

The mineral resource is a concentration or availability of minerals that have economic value on or above the earth's crust, with certain forms, qualities, and quantities that have reasonable prospects which can eventually be economically extracted. The location, quantity, grade, geological characteristics, and continuity of mineral resources must be known, estimated or interpreted based on evidence and specific geological knowledge (Calas, 2017).

Mineral resources are further grouped based on the level of geological confidence, into inferred, indicated, and measurable categories (Badan Standarisasi Nasional, 2011; Komite Cadangan Mineral Indonesia, 2017), while mineral reserves are part of mineral resources that can be mined economically. Extraction is justified by considering mining, metallurgical, economic, marketing, legal, environmental, social, and government factors. The classification of mineral resources are (Calas, 2017; Komite Cadangan Mineral Indonesia, 2017):

1. Inferred mineral resources

   An inferred mineral resource is a resource for which tonnage, grade, and mineral content can be estimated with low confidence. This is based on information obtained through adequate engineering of the mineralization site but the quality and level of confidence are limited or unclear. The distance between the observation points is a maximum of 200 meters.

2. Indicated mineral resources

   An indicated mineral resource is a resource for which tonnage, grade, and mineral content can be estimated with a reasonable degree of confidence. This is based on exploration results, sampling and testing obtained through appropriate techniques from mineralized sites. The maximum distance between points is 100 meters.

3. Measured mineral resources

   A measured mineral resource is a resource for which the tonnage, grade, and mineral content can be estimated with a high degree of confidence. This is based on reliable and detailed exploration results, and information on sampling and testing obtained with appropriate techniques from mineralized sites. The distance is a maximum of 50 meters to ensure geological and grade continuity.

The process of estimating mineable ore resources is a very important subject for the mine planning and design phase of a mining project. This process is a core component of any surface mining project, affecting the life of the project, the annual production rate of the mine, the life of the appropriate mineral processing or other facility, and the overall viability of the project. The role of reserve estimation is also important in setting sustainable targets in the context of managing the mining industry's largest asset (Roumpos and Papacosta, 2013).

Any resource estimation process is iterative, requiring a good understanding of the underlying geological science, good resource evaluation practices, and knowledge of the relevant literature including company reports and maps, to recognize and learn from previous experiences. This information provides references to deposits to guide the modeling and estimation process (Chanderman et al, 2017).

2.5 Inverse Distance Weighting (IDW) Method

Currently, 3D geological modeling adopts various interpolation methods, such as IDW interpolation, kriging interpolation, triangulation with linear interpolation, and cubic spline interpolation. Among these methods, the IDW interpolation method is the most widely used spatial interpolation method in 3D geological modeling. This method is also widely used in the visual analysis of geospatial data, modeling of rain areas, evaluation of geological resources, evaluation of geothermal resources, drawing of an urban geological maps, construction of geological models, GIS-based pollutant analysis, interpolation of small data sets, and many other research fields (Liu et al, 2021).

The IDW method is the most widely used spatial interpolation method in 3D geological modeling and directly determines the accuracy and reliability of 3D geological models. With the strong development of smart geology, the demand for large-scale geological modeling is growing rapidly. Large-scale geological modeling refers to the formation of a geological model for a large-scale research area that typically covers hundreds or thousands of square kilometers. The expansion of the modeling scope has brought many problems because the model must cover several geological units, reflect large changes in terrain, and cover a data (Liu et al, 2019).

The main characteristic of this method is that all points on the earth's surface are considered to be interdependent based on distance. Therefore, the calculation of the height in an area depends on the height of the surrounding data points. The height of the interpolation point is related to the height of the surrounding reference point. This relationship is a function that is inversely proportional to the distance to each reference point, raised to a power which is usually a square or cubic as shown in Formula 2.1 (Achilleos, 2012).

$$Wj = \frac{\frac{1}{di^n} \times q_1}{\sum i=n \frac{1}{di^n}} \quad \text{...(2.1)}$$

Where:

Wj = Estimated weight

di = Distance

n = Rank, q_1 = surrounding points

2.6 Pit Optimization

Open pit mining is an important common mining method where mineral deposits are mined through openings. The shape of the mining area at the end of the mining operation must be designed before starting the operation. Based on the designed final pit limit, mining operational parameters such as width, length, pit depth, way of opening the line, waste disposal location, stripping allowance, mine life, mineable ore tonnage, waste tonnage, and production schedule can also be determined (Zeyni et al, 2011).

Optimal pit boundaries are usually designed with the use of block models. The geological block model, which presents the reserves as a combination of many small blocks, is determined by the Inverse Distance Weighting (IDW) method or the geostatistical method. The economic block model is then calculated by applying costs, prices, and other parameters to each block. In this model the ore block has a positive value, the waste block has a negative value and the air block, and the block above the surface topography has a zero value. Most of the optimal pit boundary methods use an economical block model to determine pit boundaries. The methodology is to find a combination of blocks with the maximum economic value in the current economic and technical conditions (Zeyni et al, 2011).

The goal of any optimal open hole design algorithm is to determine the final pit limit of the ore body and the associated grade and tonnage, which will maximize several economic and/or technical criteria while meeting practical operational requirements. Since the advent and widespread use of computers, open pit mine designs have been implemented by applying different methods and various algorithms, all with the same goal of maximizing the overall mining profit within the designed pit limits (Djilani, 1997).

The first step in any optimization problem is to determine the optimization criteria. For pit design, there are several criteria, namely technical, geological, economic, or a combination of the three. The most commonly used criteria are economic such as maximum profit, maximum metal extraction, maximum net present value, and optimal mine life. Based on this amount, the most widely accepted is the variance of maximum profit. However, ore bodies can be mined at a range of limit values each of which will yield the same amount of metal for different tonnages of ore mined or at least over of practical values. Based on the time value of money, optimization results always more profitable to be mined at a higher cap rate in the early years and then at a decreasing cap rate during later years. So, the optimization criteria must be the maximum net present value rather than the maximum total profit (Djilani, 1997).

2.7 Lerch-Grossman (LG) Method

LG's 3D pit design algorithm has been used for more than 30 years for open pit design. It is well known and has been implemented in commercial software (eg Muir, Whittle, and MaxiPit). It wasn't until the 1990s that other efficient network flow algorithms were developed (eg Push-Relabel and Pseudoflow). These algorithms can theoretically solve pit optimization problems more efficiently and some have been implemented commercially (eg MineMax uses push-relabel) (Muir, 2004).

LG's optimization algorithm finds the maximum closure of a weighted directed graph, in this case, the vertices represent the blocks in the model, the weights represent the block's net profit, and the arc represents the mining constraint (usually slope). Thus, the algorithm solves very special cases of linear programming or network flow problems. LG's basic algorithms have been used for more than 30 years in many feasibility studies and many producing mines. Hochbaum (1996, 2001, 2002) has extended the LG algorithm with the pseudo flow concept in the formulation of the network flow problem. This problem formulation refines the basic Lerch-Grossman algorithm with a structured strategy to determine the next series of arcs to be processed (Muir, 2004).

Before attempting an optimization, it is essential to thoroughly understand the grade distribution and geometry of an orebody. This can be achieved by producing a block model of the deposit via such programs as datamine. On cross-sections of this model, a manual interpretation and pit design can be accomplished which will give a general indication of the geological and mineable reserves and the ultimate stripping ratio. From this, realistic estimates of mining and processing costs can be made plus breakeven and operational cut-off grades (Annels, 1991).

The Lerch and Grossmann method progressively generates lists of related blocks, like the branches of a tree, in which branches are flagged as 'strong', if the total of their block values is positive, or 'weak', if they contain waste blocks. The program then searches for structure arcs where some part of a strong branch underlies a weak branch. If found, the two branches are restructured by combining them into one (which may be either strong or weak) or breaking a portion of one branch and adding it to the other (Annels, 1991).

The procedure is repeated until no structure arc goes from a strong branch to a weak one. All strong branches are then combined to form the ultimate pit design. A listing of the ore and waste blocks within the optimal pit limits is then produced. The program has the facility to produce a series of pits each down to a different pit bottom and thus it simulates mining with incremental push-backs of the pit limit. Crosssections in three orthogonal planes can now be produced showing, for each block, the phase number in which it will be mined (Annels, 1991).

The first optimization method strictly for the general case was proposed by Lerch and Grossmann (1965). This method overcomes the limitations of traditional pit designs and can be proven to always produce an optimal solution. The Lerch-Grossmann algorithm is based on Graph Theory (Djilani, 1997).

The Graph Theory approach was developed by Lerch and Grossmann (1965), for determining the optimum pit limit based on the construction of the maximum graph closure. The Lerch Grossmann algorithm converts the three-dimensional block grid in the ore body model into a directed graph. Each block in the grid is represented by a vertex which is assigned a mass equal to the net present value of the corresponding block. The vertices are connected by arcs so that the connection leading from a given vertex to the surface determines the set of vertices that must be removed if that vertex (block) is to be mined. A simple two-dimensional example is shown in Figure 2.7 (Djilani, 1997).

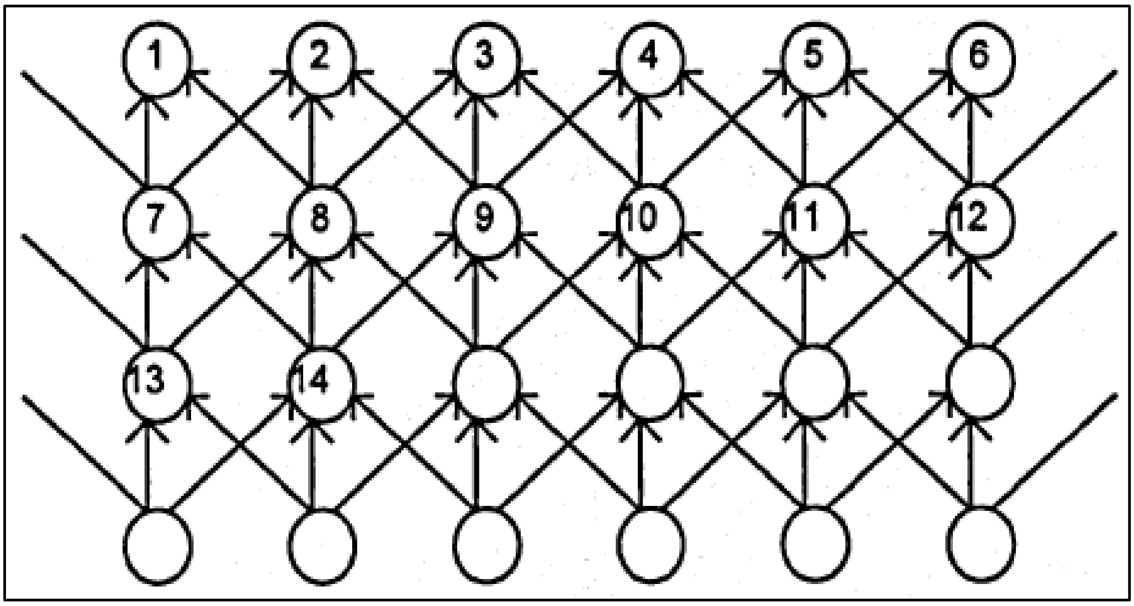


Figure 2.7 Directional graphs represent a 2D deposit model, nodes represent blocks and arcs define mining boundaries (Djilani, 1997)

The vertex connected by an arc away from the vertex is called the successor of the vertex, that is, vertex y is the successor to vertex x if there is an arc that is directed from x to y. The set of all successors of x is denoted rx. For example, in Figure 2.7, rX9 = {X2' X3 , ~ }. The closure of the directed graph, which consists of the vertex set X, is the vertex set Y C X so that if x E Y then rx E Y. Y = {X1, X2, X3, X4, X5, X8, X9, X10} is trending chart close. The closing value is the mass sum (revenue value) of the nodes in the closure. Each closure defines a possible hole, the closure with the maximum value defines the optimal hole (Djilani, 1997).

This method is the only method that can be rigorously proven, mathematically always leads to the correct optimal solution. However, several new methods published recently also make similar claims but remain to be independently verified (Djilani, 1997).

The 3D Lerch and Grossmann method has some drawbacks in that it allows only the consideration of a single attribute (the economic value of the block) and that it precludes the separate consideration of different metals in a polymetallic ore body or the use of ore blending. Constant economic parameters are used for all blocks in the model whereas mining costs and metal prices may vary with time as the pit deepens. Detailed knowledge of the order in which blocks are to be mined would be necessary to allow for this. Finally, all blocks that can be economically mined are included in the optimal model and high grading is precluded (Annels, 1991).

2.8 Pit Design

Open pit mine design is described in several stages consisting of designing (planning) a scheme or a set of alternative schemes, followed by evaluation and selection of the optimal scheme. The most economical final pit design depends on factors that the mining engineer cannot control such as the geometric outline of the ore body, distribution of ore within the ore body, topography, maximum allowable slope angle, etc (Jiskani, 2017).

Mining design is a part of the planning of the mining stages as a very important factor is determined before the actual mining plan begins. Design is the determination of detailed and definite requirements, specifications, and criteria as well as techniques to achieve the goals and objectives of activities as well as the technical sequence of their implementation. In the mining industry, there is also a mine design that includes activities loading, hauling, compacting, and many more (Hustrulid et al, 2013). The illustration of open pit design and it’s design parameters are shown in Figure 2.8 (Jiskani, 2017).

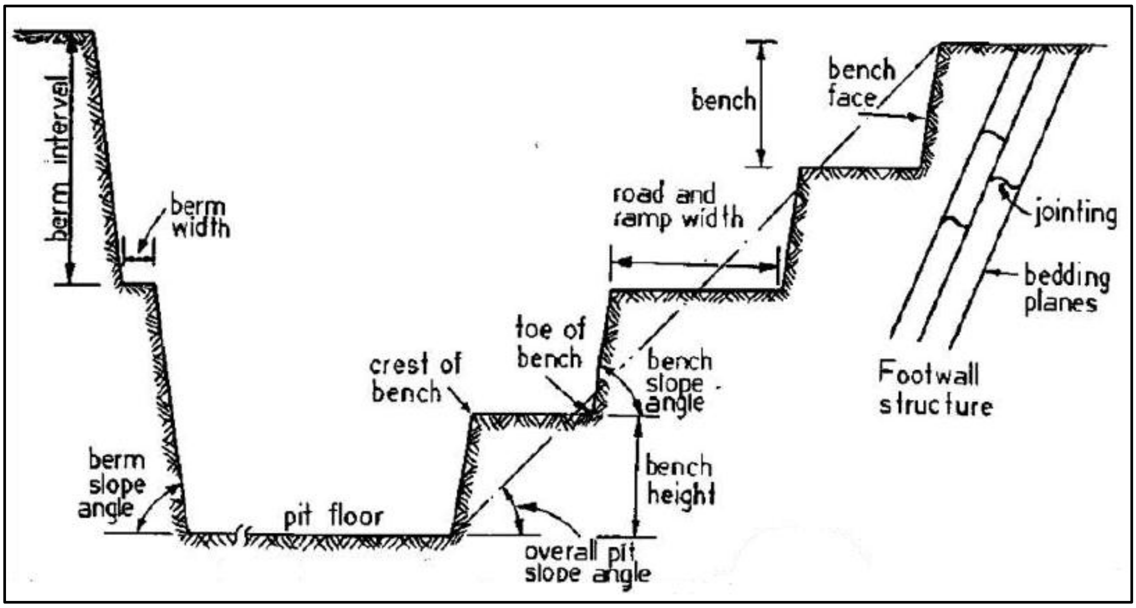


Figure 2.8 Pit design components (Jiskani, 2017)

One thing that needs to be considered in pit design is the stripping ratio. The stripping ratio is a critical parameter in pit design and scheduling. The stripping ratio refers to the ratio of the volume of overburden (or waste material) that needs to be removed to the tonnage of ore recovered (Jiskani, 2017). A lower stripping ratio means that less waste has to be removed to expose the ore for mining which generally results in a lower operating cost (Altiti et al, 2020). The major types of stripping ratios are overall, instantaneous, and break-even. The stripping ratio is obtained from Formula 2.2 (Hustrulid et al, 2013).

$$\text{Stripping ratio} = \frac{\text{Overburden volume}}{\text{Ore tonnage}} \quad \text{..............................(2.2)}$$

Pit design parameters shown in Figure 2.8 are (Jiskani, 2017):

1. Bench, is a ledge that forms a stage of operation where minerals or waste materials are mined from the face of the bench.
2. Bench height, is the vertical distance between the top and bottom surfaces of the bench. This is influenced by equipment size, mining selectivity, government regulations, and safety.

3. Bench width, is the distance between the peak and the foot. So, as each bench is mined, it will be cut back towards the limit of the overlying bench until it reaches a distance equivalent to the specified berm width.
4. Face angle, is the angle of the bench face with the horizontal.
5. Crest, is the top point of the bench face.
6. Toe, is the lower point of the bench face.
7. Bench slope/bank angle, is the horizontal angle from the line connecting the end of the bench toe to the bench crest.
8. Berm, is a horizontal shelf or ledge in the remaining ultimate pit wall slope to increase slope stability in the opening and increase safety. The berm interval, berm width and slope angle are determined by geotechnical investigations.
9. Overall pit slope angle, is the angle measured from the crest to the toe. This is the wall angle of the open pit, measured between the horizontal line and the imaginary line connecting the toe to the crest. It is determined by rock strength, geological structure, and water conditions. The overall pit slope angle is affected by the width and slope of the haul road.
10. Haul road, is a haul road used for access during the life of the pit. It is usually added to the pit design once the depth of the pit bottom has been established.

    The width of the ramp shown in Figure 2.9 is designed based on the calculation of the road geometry according to Hustrulid et al (2013) in the following equation Formula 2.3 (Hustrulid et al, 2013).

    $$L_{min} = N \times W_t + (n + 1) \times (1/2 \times W_t) \quad \text{......................(2.3)}$$

    Where:

    Lmin = Min road width (ramp) (m)

    N = Number of paths

    Wt = Width of dump truck (m)

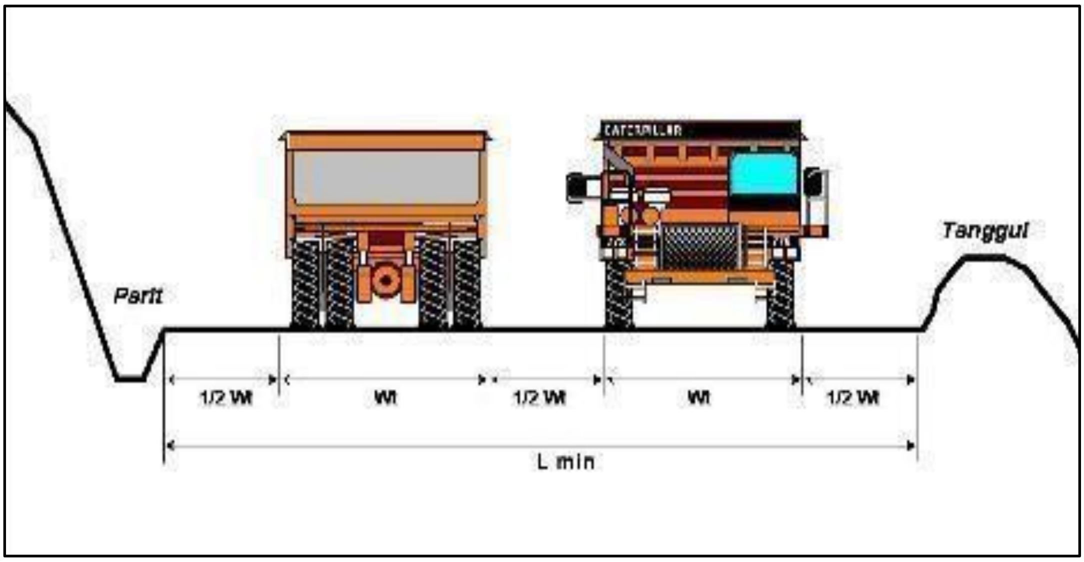


Figure 2.9 Mine road components (Hustrulid et al, 2013)

2.9 Production Scheduling

Production scheduling is a very important part of the mining business (Djilani, 1997; Meagher et al, 2014). Production scheduling can be either long-term or short-term depending on the duration of the scheduling period. Short-range scheduling is an extension of the depletion sequence on a daily, weekly or monthly basis, whereas long-range scheduling is concerned with annual plans and includes ore reserves, stripping ratios, and capital investment (Djilani, 1997).

The main objective of short and long-term mine planning (scheduling) in open pit operations is to maximize the profits realized in each mining period and throughout the life of the mine. In general, the production scheduling procedure is to determine which blocks must be moved in each mining period subject to mining constraints to maximize the NPV of the mine (Djilani, 1997).

Planning is an ongoing activity throughout the life of the mine. Plans are made (Hustrulid et al, 20130. The life of a mine is the time calculated from the number of reserves divided by the production of mines per year. Mine life is strongly influenced by the number of reserves that can be mined and the level of production per year. The life of the mine is made not too fast or too long, depending on the company's ability to determine the level of production. Too low a production level means that the profits obtained will be long (the payback will be long), while the production level is too high, the investment costs can be too large (Djilani, 1997).

The basic objectives or goals of extraction planning are (Hustrulid et al, 2013):

1. To mine the ore body in such a way that for each year the cost to produce a kg of metal is a minimum, i.e. a philosophy of mining the best ore in sequence.
2. To maintain operation viability within the plan through the incorporation of adequate equipment operating room, haulage access to each active bench, etc.
3. To incorporate sufficient exposed ore insurance to counter the possibility of misestimation of ore tonnages and grades in the reserve model. This is particularly true in the early years which are so critical to economic success.
4. To defer waste stripping requirements, as much as possible, and yet provide relatively smooth equipment and manpower build-up.
5. To develop a logical and easily achievable start-up schedule with due recognition to manpower training, pioneering activities, equipment deployment, infrastructure, and logistical support, thus minimizing the risk of delaying the initiation of positive cash flow from the venture.
6. To maximize design pit slope angles in response to adequate geotechnical investigations, and yet through careful planning minimize the adverse impacts of any slope instability, should it occur.
7. To properly examine the economic merits of alternative ore production rates and cut-off grade scenarios.
8. To thoroughly subject the proposed mining strategy, equipment selection, and mine development plan to 'what if' contingency planning, before a commitment to proceed is made.

There are two kinds of production planning which correspond to different periods (Hustrulid et al, 2013):

1. Operational or short-range production planning is necessary for the function of an operating mine.
2. Long-range production planning is usually done for feasibility or budget studies. It supplements pit design and reserve estimation work and is an important element in the decision-making process.

In guiding the planner, Couzens (1979) has proposed the following five planning commandments or rules (Hustrulid et al, 2013):

1. We must keep our objectives clearly defined while realizing that we are dealing with estimates of grade, projections of geology, and guesses about economics. We must be open to change.
2. We must communicate. If planning is not clear to those who must make decisions and to those who must execute plans, then the planning will be either misunderstood or ignored.
3. We must remember that we are dealing with volumes of earth that must be moved in sequence. Geometry is as important to a planner as arithmetic.
4. We must remember that we are dealing with time. Volumes must be moved in time to realize our production goals. The productive use of time will determine efficiency and cost-effectiveness.
5. We must seek acceptance of our plans such that they become the company's goals and not just the planner's ideas.

2.10 Sequence Design

An important issue in mine planning and design is determining the mining sequence that will optimize certain criteria, especially net present value. Ideally, the maximum net present value should also be a criterion for optimal open pit design (Djilani, 1997). An illustration of the secant section is shown in Figure 2.11 (Jiskani, 2017).

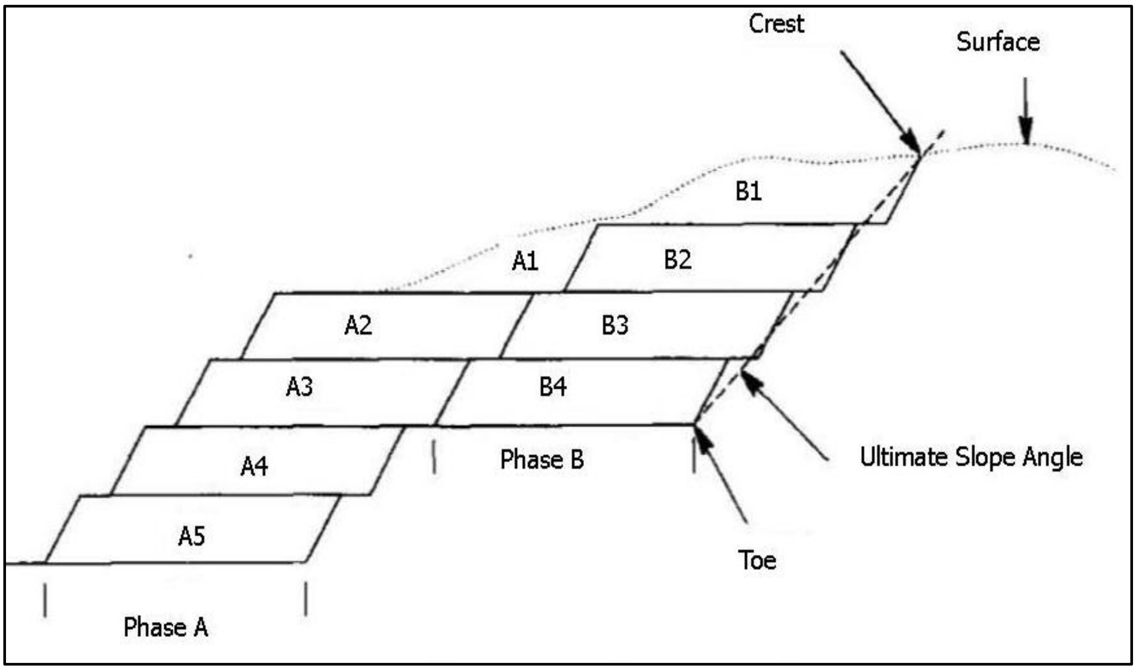


Figure 2.10 Sequence cut illustration (Jiskani, 2017)

Pushback (mining cut or mining phase) is a part of ultimate pit limit used in mine planning for two reasons (Nogholi, 2015):

1. To guide for block sequencing.
2. To improve the practicality of mining operations.

Block sequencing is an attempt to find the sequence in which blocks should be removed over a certain period time. Unlike the ultimate pit limit problem which determines which block to remove, the block sequencing problem considers the removal period. Therefore, block sequencing determines which block to remove from the ultimate pit limit and when (Nogholi, 2015).

Whittle (2011) identified two outcomes in designing pushback, there are the best case, in which there are many pushbacks, and the worst case, where there is no pushback. If the production schedules obtained based on the best case and worst cases are different in terms of economic value, then different numbers of pushbacks must be examined to achieve an optimum number of pushbacks (Nogholi, 2015).

As a practical approach, nested pits are used to design pushbacks. Nested pits are explained in the Lerchs and Grossmann (1965) paper and are obtained by parametric analysis of the Lerch-Grossmann algorithm. In the parametric analysis of the LG algorithm, reducing the price of the commodity leads to obtaining smaller pits. These smaller pits indicate the production phase for long-term production sequencing (Nogholi, 2015).

Although nested pits show the sequences for mining and tend to maximize NPV, they have the main drawback referred to as the gap problem. The gap problem means that the sizes of all nested pits are not equal. Hence, sometimes there is a significant difference between two consecutive pits in terms of tonnage and ore content. In other words, there is no controlling the nested pit size and shape in the nested pit approach (Nogholi, 2015).

An illustration of mining pushback is shown in Figure 2.12 (Hustrulid et al, 2013).

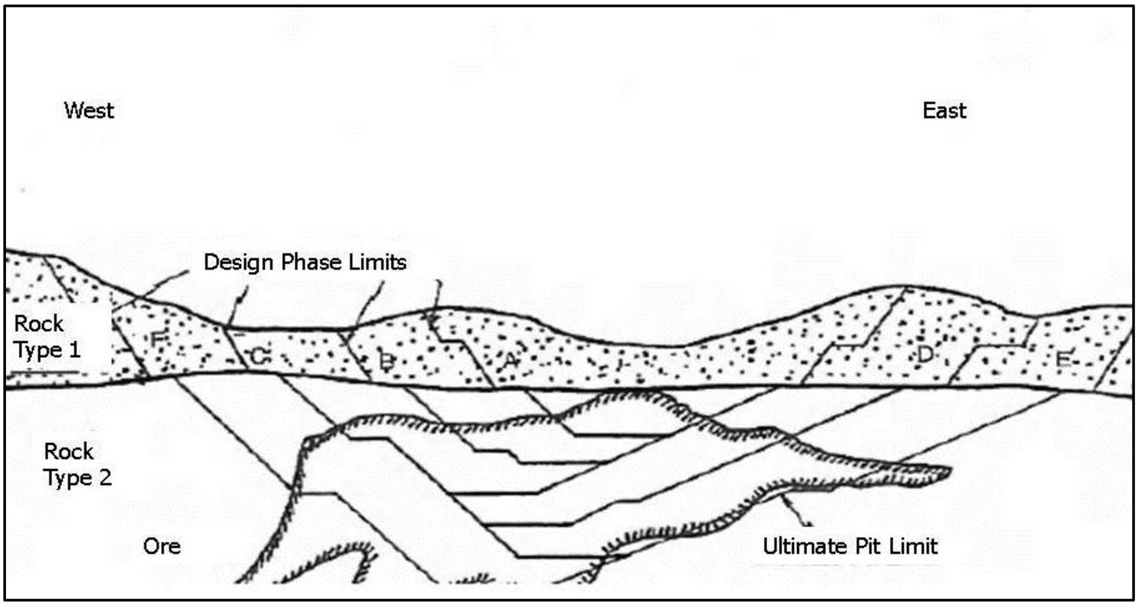


Figure 2.11 Mining pushback illustration (Hustrulid et al, 2013)

2.11 Equipment Productivity

Mining equipment in a mining operation is vital to support target production by management. Equipment that is often used in mining operation is excavators, wheel loaders, and dump trucks. One type of equipment can also serve more than one activity, such as an excavator, besides functioning as an excavation, can also be used as equipment of fit or even as a dump truck for a certain distance (Franicha et al, 2017).

Every construction project is a unique undertaking. Although similar work may have been performed previously, no two projects have identical job conditions. The pace, complexity, and cost of modern construction are incompatible with trial-and-error corrective actions as the work proceeds. Therefore, planning is undertaken to develop appropriate courses of action and to prepare strategies for meeting potential problems (Peurifoy et al, 2013).

Project work elements are defined in physical terms as volume of stripping, soil excavation, rock excavation, embankment, or waste material. This is a project material take-off or quantity survey. The takeoff effort must calculate not only the total quantity of material to be handled but must segregate the total quantity of material based on factors affecting productivity (Peurifoy et al, 2013).

Some key points as considerations in terms of using heavy equipment are as follows (Franicha et al, 2017):

1. The decision in terms of using heavy equipment based on a scenario that the equipment must provide an income greater than expenses incurred (including costs of operation/ownership). Knowledge of heavy equipment should also be controlled by an engineer, both the latest information regarding the development of latest equipment as well as the ability to vote on the right heavy equipment.

2. The relationship between costs with objectives is the heavy equipment must be able to increase work capacity and minimize costs.
3. Problems that may arise and should be planned, such as the expenditure for purchase or maintenance of equipment, the cost of surveillance (periodic), skilled operators, and training needed for workers.

Several stages in determining the number of equipment needed are:

1. Working efficiency

   The calculated production must be adjusted by an efficiency factor. Longer hauling distances usually result in better driver efficiency. Driver efficiency improves as haul distances increase out to about 8,000 ft, after which efficiency remains constant. Other critical elements affecting efficiency are bunching, equipment condition, load and dump area congestion, fueling and maintenance, operator breaks, work rules, and project layout (Peurifoy et al, 2013).

2. Swell factor

   The swell factor figures for each material classification differ according to the type of material itself as shown in Table 2.1 (Peurifoy et al, 2013).

Table 2.1 Swell factor of various materials (Peurifoy et al, 2018)

Materials	Average weight (kg/m^3)		%Swell	Swell factor
	Bank	Loose		
Clay, dry	1,600	1,185	35	0,74
Clay, wet	1,780	1,305	35	0,74
Earth, dry	1,660	1,325	25	0,80
Earth, wet	1,895	1,528	25	0,80
Earth and gravel	1,895	1,575	20	0,83
Gravel, dry	1,660	1,475	12	0,89
Gravel, wet	2,020	1,765	14	0,88
Limestone	2,610	1,630	60	0,63
Rock, well blasted	2,490	1,565	60	0,63

Materials	Average weight (kg/m^3)		%Swell	Swell factor
	Bank	Loose		
Sand, dry	1,542	1,340	15	0,87
Sand, wet	1,600	1,400	15	0,87
Shale	2,075	1,470	40	0,71

The swell factor can be calculated by Formula 2.4 (Peurifoy et al, 2018).

$$\text{Swell factor} = \frac{\text{Loose dry unit weight}}{\text{Bank dry unit weight}} \quad \text{.............................(2.4)}$$

3. Fill factor

The fill factor is the ratio of the actual loose volume of material in the bucket compared to the bucket's rated-heaped capacity. It varies based on the type of material being handled and the type of excavator. Fill factors represent percentages greater than or less than the rated-heaped capacity. The load in the bucket is adjusted based on material type by multiplying the rated-heaped capacity by the material fill factor. To validate fill factors, one should, when possible, conduct field tests based on the weight of material per bucket load. Fill factors for hydraulic hoe buckets are shown in Table 2.2 (Peurifoy et al, 2018).

Table 2.2 Fill factors for hydraulic hoe buckets (Peurifoy et al, 2018)

Material	Fill factor (%)
Moist loam/sandy clay	100-110
Sand and gravel	95-110
Rock-well blasted	60-75
Rock-poorly blasted	40-50
Hard, tough clay	80-90

4. Cycle time

Cycle time is the time it takes for a tool to carry out certain activities from start to finish and is ready to start again. The circulation time has a very important influence on the production of work tools because the circulation time becomes a variable in the calculation of the number of ritases that can be carried out in one working hour. The smaller the circulation time, the greater the amount of productivity that will be produced. Mechanical devices work according to a certain pattern which in principle consists of several components of movement in one cycle of cycle time (Erwanda et al, 2021).

5. Equipment productivity

   Tool productivity is the ability of heavy equipment to complete work which is calculated at one time and is influenced by time capacity, the time factor, cycle, and pro-duction correction factor (Erwanda et al, 2021). Mining operational cost, mined material quantitative, and so on are heavily influenced by equipment productivity. With a higher quantity of material produced per time (hour), the equipment cost per unit is also getting low. Otherwise, when it is low productivity, the costs per unit of working equipment are increasingly high. Therefore productivity is a very important role in the management of the mining equipment (Franicha et al, 2017).

6. Match factor

   The equipment compatibility factor is the ratio of the truck arrival rate to the loading service time (the level of work suitability of a dump truck and excavator). The purpose of using the match factor is to determine the number of matched dump trucks to serve one excavator unit. The tool compatibility factor ratio is an important productivity index in the mining industry (Erwanda et al, 2021).

2.12 Cash Flow Analysis

The mining industry is a very risky industry when compared with the other industries because decision-makers must consider so many uncertain inputs in the mining industry. Therefore, economic analysis and engineering economy are extremely important for the evaluation of a new mine or an operating mine (Güyagüler and Erdem, 2011).

Financial and economic analysis of investment projects is usually carried out using discounted cash flow (DCF) analysis techniques. DCF analysis is a technique used to obtain economic and financial performance criteria for investment projects (Herbohn and Harrison, 2002).

The DCF method assumes that the current value range of the company at the valuation date is equal to the present value of future cash flows to shareholders of the company. Firm value is a combination of two factors, this is due to the limitations of the financial projection period (Janiszewski, 2011):

1. Present value of cash flows (total present value of dividends that the company can pay to shareholders and/or additional capital injections made by shareholders).
2. The residual value of the company, namely the company's discounted value generated from the cash flows generated after the projection period.

The discount rate is a function of the risk inherent in business and industry, the degree of uncertainty regarding the projected cash flows, and the assumed capital structure. Generally, discount rates vary across businesses and industries. The greater the uncertainty about the projected cash flows, the higher the appropriate discount rate and the lower the current cash flow value (Janiszewski, 2011).

Cash flow analysis is simply the process of identifying and categorizing the cash flows associated with a proposed project or action, and estimating of their value. For example, when considering the establishment of a plantation, this will involve identifying and forecasting the cash outflows associated with planting trees (e.g. costs of buying or renting land, buying seedlings, and planting seedlings), maintaining the plantation (such as costs of fertilizer, labor, etc.). In addition, it is necessary to estimate cash inflows from plantations through thinning and timber sales at the final harvest (Herbohn and Harrison, 2002).

In the field of company investment, most projects are characterized by long-term and large capital expenditure. To reduce projects' investment risk and make the correct decisions for enterprises, it is necessary to there are some investment decision-making methods, such as Net Present Value (NPV), Internal Rate of Return (IRR), Payback Period (PBP), and so on (Güyagüler and Erdem, 2011).

1. Net present value (NPV)

   The NPV method as an investment appraisal or capital budgeting technique shows how an investment project affects the wealth of the company's shareholders in terms of PV. The calculation of NPV is shown in Formula 2.5 (Jory et al, 2016).

$$NPV = CF_0 + \frac{CF_1}{(1+R)^1} + \frac{CF_2}{(1+R)^2} + \ldots + \frac{CF_n}{(1+R)^n} \qquad (2.5)$$

   Where:

   CF = cash flow

   R = discount rate

   1,2,…n = the last period in which cash flows are expected

   The decision criteria of this method are (Güyagüler and Erdem, 2011):

   a. NPV < 0, investment is not profitable/not feasible.
   b. NPV = 0, marginal investment.
   c. NPV > 0, profitable/feasible investment.

2. Internal rate of return (IRR)

   The internal rate of return is the interest rate such that the amount of discounted net cash flows is zero. If the interest rate were equal to the IRR, the net present value would be exactly zero. IRR cannot be determined by algebraic formula, but must be approached by trial and error (Herbohn and Harrison, 2002).

The IRR is the highest interest rate the project can support and still break even. A project is considered economically useful if the internal rate of return is greater than the cost of capital. If this is the case, the project can support a higher interest rate than it is, and still deliver positive returns (Herbohn and Harrison, 2002).

3. Payback period (PP)

   Payback analysis is another use of the present worth technique. It is used to determine the amount of time, usually expressed in years, required to recover the first cost of an asset or project (Blank and Tarquin, 2012). The payback period (PP) is the number of years for the project to break even, i.e. the number of years in which the discounted annual net cash flows must be added up before the amount becomes positive (and remains positive for the remainder of the project's life) (Herbohn and Harrison, 2002).

2.13 Sensitivity Analysis

The sensitivity analysis is a study of how the output of a system is affected by its inputs. In many applications, the system in question involves one or a series of mathematical models coded using computer software that simulates the desired system function in the real world. Such mathematical models can be data-driven (also called statistical), directly mapping inputs to outputs. Factors in sensitivity analysis may include model parameters, tight variables, constraints and initial conditions, structural model configuration choices, assumptions, and constraints. Output can include any function of the model response, including objective functions such as production or cost functions in cost and benefit analysis, etc. (Razavi et al, 2020).

Sensitivity analysis is necessary to evaluate the probable effects of project parameters on economic outputs that can be used for recognizing the most critical factors and in this way, prediction of extreme optimistic and pessimistic conditions would be possible (Orae et al, 2011; Anas et al, 2019). The sensitivity analysis of Net Present Value (NPV) based on economic parameters could be a key consideration for companies to anticipate future events due to changes in economic parameters (Anat et al, 2019).

Sensitivity analysis involves the process of determining how the distribution of all possible returns for a particular project is affected by changes in one particular input variable, which is done by estimating the NPV of pessimistic, most likely and optimistic values of each variable. There is only one variable at a time that changes and is analyzed while the other variables are constant (Herdianto and Daryanto, 2019).

There are two general methods used to determine the variables at the pessimistic and optimistic levels. The first method is done by taking certain values for the estimated variables as an extension and pro forma forecasting technique, with the values taken representing predictable events, either through a statistical approach, expert judgment or management opinion, or with the limits of ability consideration. The second method is to use a more mechanistic approach, where the level of the variable is chosen without reference to the trend of values in the future, but by setting the level of the variable value higher or lower than the value that is most likely to occur (Herdianto and Daryanto, 2019).