THESIS

CONCEPTUAL PLANNING OF NORTH BLOCK NICKEL ORE MINE PT PACIFIC ORE RESOURCES, BOMBANA REGENCY, SOUTHEAST SULAWESI PROVINCE

Compiled and submitted by:

M. FAHMI
D111 18 1018

MINING ENGINEERING UNDERGRADUATE STUDY PROGRAM
FACULTY OF ENGINEERING
HASANUDDIN UNIVERSITY
GOWA
2023

LEGALIZATION

CONCEPTUAL PLANNING OF NORTH BLOCK NICKEL ORE MINE PT PACIFIC ORE RESOURCES, BOMBANA REGENCY, SOUTHEAST SULAWESI PROVINCE

Compiled and submitted by

M. Fahmi
D111181018

Has been defended in front of Examination Committee established for the Completion of the Mining Engineering Undergraduate Study Program
Faculty of Engineering Hasanuddin University
On August 18th, 2023
and declared eligible

Approved by,

Supervisor,	Co-Supervisor,
Dr. Aryanti Virtanti Anas, S.T., M.T.	Rizki Amalia, S.T., M.T.
NIP. 197010052008012026	NIP. 199205042019016001

Head of Study Program,

Dr. Aryanti Virtanti Anas, S.T., M.T.
NIP. 197010052008012026

LEMBAR PENGESAHAN

CONCEPTUAL PLANNING OF NORTH BLOCK NICKEL ORE MINE PT PACIFIC ORE RESOURCES, BOMBANA REGENCY, SOUTHEAST SULAWESI PROVINCE

Disusun dan diajukan oleh

M. Fahmi
D111181018

Telah dipertahankan di hadapan Panitia Ujian yang dibentuk dalam rangka Penyelesaian Studi Program Sarjana Program Studi Teknik Pertambangan Fakultas Teknik Universitas Hasanuddin Pada tanggal 18 Agustus 2023 dan dinyatakan telah memenuhi syarat kelulusan

Menyetujui,

Pembimbing Utama,	Pembimbing Pendamping,
Dr. Aryanti Virtanti Anas, S.T., M.T.	Rizki Amalia, S.T., M.T.
NIP. 197010052008012026	NIP. 199205042019016001

Ketua Program Studi,

KEMENTERIAN PENDIDIKAN, KEBUDAYAAN, RISET, DAN TEKNOLOGI UNIVERSITAS HASANUDDIN DEPARTEMEN TEKNIK PERTAMBANGAN FAKULTAS TEKNIK

Dr. Aryanti Virtanti Anas, S.T., M.T.
NIP. 197010052008012026

STATEMENT OF AUTHENTICITY

The undersigned below;
Name : M. Fahmi
Student ID : D111181018
Study Program : Mining Engineering
Degree : Bachelor (S1)

Declared that my bachelor thesis titled

Conceptual Planning of North Block Nickel Ore Mine PT Pacific Ore Resources, Bombana Regency, Southeast Sulawesi Province

Is my own writing and is not a claim of other people's writings and that the thesis that I wrote is really the result of my own work.

All information written in the thesis that comes from other authors has been given credit, namely by citing the source and the year of publication. Therefore, all writing in this thesis is fully the responsibility of the author. If there are any parties who feel that there are similarities in the title and/or findings in this thesis, the authors are ready to clarify and take responsibility for all risks.

All data and information obtained during the process of writing the thesis, which will be published by the author in the future must obtain approval from the supervisor.

If in the future it is proven or can be proven that part or all of the parts of this thesis are the work of other people, then I am willing to accept sanctions for such actions.

Makassar, August 18th 2023

Declared by,

M. Fahmi

ABSTRACT

M. FAHMI. *Conceptual Planning of North Block Nickel Ore Mine PT Pacific Ore Resources, Bombana Regency, Southeast Sulawesi Province* (supervised by Aryanti Virtanti Anas and Rizki Amalia)

PT Pacific Ore Resources is a nickel laterite mining company located in North Kabaena, Bombana Regency, Southeast Sulawesi Province. The company has divided the work area into two blocks, namely the South Block which is the focus of the production process, and the North Block which is still in the exploration and mining planning stages. This study aims to make a conceptual mine planning in the North Block to determine optimal mining boundaries, pit design, production scheduling, cash flow, and sensitivity analysis. Research using quantitative methods. Data processing is assisted by Micromine v.2021.5 and Microsoft Excel software. The reporting results of laterite nickel ore resources are 262,996 tons. The optimal pit limit is in pit shell 6 with an NPV of USD 961,991. The results of the evaluation of ore reserves in the pit design were 216,771 tons with an overburden of 683,255 BCM so a waste dump design was carried out that could accommodate up to 685,763 BCM of waste material. The mine haul road design results include new roads and existing roads which have a road length of 687.31 meters respectively 5315.42 meters. The ore reserves based on sequence design sequentially from sequence 1 to 6 are 23,237, 43,007, 41,249, 42,173, 42,891, and 27,777 tons, while the overburden layer is 129,279, 100,781, 119,065, 117,674, 113,629 and 100,053 BCM. The digging-loading equipment assigment from sequences 1 to 6 is 3 units each, while the hauling equipment is 16, 21, 22, 22, 22, and 16 units. The cashflow analysis results showed an NPV value of USD 1,050,838.64, an IRR of 37.94%, and a PBP of 70 days. The sensitivity analysis results show that changes in nickel prices affect the NPV the most compared to changes in operating and capital costs.

Keywords: Mine Planning, Nickel Laterit, Investment Analysis

ABSTRAK

M. FAHMI. *Perencanaan Konseptual Tambang Bijih Nikel Blok Utara PT Pacific Ore Resources, Kabupaten Bombana, Provinsi Sulawesi Tenggara (dibimbing oleh Aryanti Virtanti Anas dan Rizki Amalia)*

PT Pacific Ore Resources merupakan perusahaan tambang nikel laterit yang terletak di Kabaena Utara, Kabupaten Bombana, Provinsi Sulawesi Tenggara. Perusahaan ini memiliki pembagian area kerja menjadi dua blok, yaitu Blok Selatan yang merupakan fokus proses produksi dan Blok Utara yang masih dalam tahap eksplorasi dan perencanaan penambangan. Penelitian ini bertujuan untuk membuat perencanaan konseptual tambang pada Blok Utara dengan ruang lingkup penentuan batas penambangan optimal, merancang pit, penjadwalan produksi, analisis aliran kas dan sensitivitas. Penelitian menggunakan metode kuantitatif. Proses pengolahan data dibantu dengan perangkat lunak Micromine v.2021.5 *dan* Microsoft Excel. *Hasil pelaporan sumberdaya bijih nikel laterit adalah 262.996 ton.* Pit limit *optimal berada pada* pit shell *6 dengan NPV sebesar USD 961.991. Hasil evaluasi cadangan bijih pada desain pit adalah 216.771 tons dengan lapisan tanah penutup yaitu 683.255 BCM sehingga dilakukan desain* waste dump *yang dapat menampung material* waste *hingga 685.763 BCM. Hasil desain* mine haul road *mencakup* new road *dan* existing road *yang memiliki panjang* road *masing-masing 687,31 Ddan 5315,42 meters. Cadangan bijih berdasarkan desain sekuen secara berurut dari sekuen 1 hingga 6 adalah 23.237, 43.007, 41.249, 42.173, 42.891 , dan 27.777 tons, sedangkan lapisan tanah penutup sebesar 129.279, 100.781, 119.065, 117.674, 113.629 dan 100.053 BCM. Pemilihan alat gali-muat dari sekuen 1 hingga 6 adalah masing-masing 3 unit, sedangkan alat angkut adalah 16, 21, 22, 22, 22 dan 16 unit. Hasil analisis aliran kas menunjukkan nilai NPV sebesar USD* 1,050,838.64, *IRR sebesar 37,94%, dan PBP sebesar 70 hari. Hasil analisis sensitivitas menunjukkan bahwa perubahan harga nikel yang paling mempengaruhi NPV dibanding perubahan biaya operasional dan biaya investasi.*

Kata kunci: Perencanaan Tambang, Nikel Laterit, Analisis Investasi

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

LIST OF ABBREVIATION AND SYMBOL MEANING

Symbol/Abbreviation	Meaning and Description
Ha	Hectares
NPV	Net Present Value
IRR	Internal Rate Of Return
PBP	Payback Periode
ROR	Rate Of Return
IUP	*Izin Usaha Pertambangan*
α	Slope Angle/Gradient
H	Height
USGS	United States Geological Survey
NNP	Nearest Neighbor Point
IDW	Inverse Distance Weighted
SNI	*Standar Nasional Indonesia*
LCM	Loose Cubic Meter
BCM	Bank Cubic Meter
LOM	Life of Mine
SR	Stripping Ratio
U	Utility
CT	Cycle Time
i	Interest Rate
PWC	Present Worth of Cost
PWB	Present Worth of Benefit
Km	Kilometer
m^2	Area
Ni	Nickel
$	US Dollar
T	Tonnage
wmt	Wet Metric Ton
RAF	Revenue Adjustment Factor
LGO	Low Grade Ore
MGO	Medium Grade Ore
HGO	High Grade Ore

Symbol/Abbreviation	Meaning and Description
MF	Match Factor
EBD	Earning Before Tax
WACC	Weighted Average Cost Of Capital
DCF	Discounted Cashflow
Fe	Iron
COG	Cut-off Grade
m^3	Meter Cubic
m	Meter
masl	Meter Above Sea Level
CAPEX	Capital Expenditure
OPEX	Operating Expenditure

LIST OF APPENDICES

PREFACE

Bismillahirrahmanirrahim. The thesis with the title Conceptual Planning of North Block Nickel Ore Mine PT Pacific Ore Resources, Bombana Regency, Southeast 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 Pacific Ore Resources, Larolanu Village, North Kabaena District, Bombana Regency, Southeast Sulawesi Province, and the Mine Planning and Valuation Laboratory, Mining Engineering Department, Faculty of Engineering, Hasanuddin University.

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 Pacific Ore Resources for the research location. The author would also like to thank Mr. Muhammad Ihsan, S.T. as Head of Mine Engineer, and Mr. Afri Ifthihar, 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 Mr. Enos Paembonan, S.T and Mr. Budiman Yusuf as Geologist, Mr. Burhanuddin as General Affair Supervisor 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, Miss. Rizki Amalia S.T. as the Co-Supervisor. Thanks very much also the writer express to Mr. Dr. Eng. Ir Muhammad Ramli, M.T. as an examiner and Mrs. Dr. Eng. Rini Novrianti Sutardjo Tui, S.T., M.BA., M.T. as an examiner and 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, 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. Abd. Malik and Mrs. Alm. Hamida 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.

Makassar, 18th August 2023

M. Fahmi

CHAPTER 1
INTRODUCTION

1.1 Background

PT Pacific Ore Resources is a mining company in Southeast Sulawesi. This company holds a mining business permit (IUP) to carry out nickel ore mining activities in Larolanu Village, North Kabaena District, Bombana Regency, Southeast Sulawesi Province, covering an area of 2,672 Ha based on Decree 583/DPM-PTSP/VII/2018. Currently, the division of the company's work area is divided into two blocks, namely the North and South Blocks. The South Block is the focus of the mining production process. In this block, there are several pits namely Pit Kratos and Pit Luna. Meanwhile, the North Block is still in the mining exploration and planning stage.

Planning is determining the requirements for achieving the target activities and the technical sequence of implementing various activities to achieve the desired goals and objectives. The concept of mine planning includes planning and designing a mine to obtain and transport minerals of economic value (Sasongko, 2009). Planning series of steps are undertaken in varying amounts of detail, depending on the precision, economic action, or decision being sought. Elements of the mine planning cycle are developing the mine design and managing the layout of the mine (depth, stripping ratio, ore grade, selling price, spacing, etc.); determination of heavy equipment; production scheduling; estimation of mining costs and construction of financial models (Malli et al., 2015).

Determination of mining pit boundaries is an initial step that needs to be carried out before designing mine designs and mining directions in the mine planning stage (Sidiq and Pusvito, 2017). Determination of mining pit boundaries is an attempt to determine the best mine boundaries and determine optimum reserves that provide the best profit margins (Rifandy and MP, 2018). The purpose of determining pit boundaries is to determine the ultimate opening limit of the ore body and the associated grade and tonnage, which will maximize some economic or technical criteria while meeting practical operational requirements (Djilani, 1997).

The design of the mining pit is made to determine the shape, size, and volume of mineral deposits so that the design process depends on the shape and direction of distribution of a mineral to be mined so that minerals with such a wide distribution lead to a wide mining pit design as well (Hidayat et al., 2018). Mine design is needed to estimate or predict an area of potential ore resources to be developed into a mining pit location (Adnannst et al., 2015).

After the mining reserves are known, the next stage is a production planning, which is in the form of production scheduling and sequence planning activities (Rifandy and MP, 2018). The mine production schedule is stated in a certain period, including data on tonnage, overburden, and total material removal from the mine. The basic principle of production scheduling is to maximize NPV (net present value), and ROR (rate of return) (Bargawa, 2008). Mining sequence design is designing mining shapes (mineable geometries) to mine out reserves starting from the initial entry limit to the final pit boundary. Mining sequences are mining forms that indicate how a pit will be mined from the initial point of entry to the final shape of the pit. Sequence design or mining stages divide the ultimate pit into smaller and more manageable planning units, this will make complex three-dimensional mine design problems simpler, at this stage, the time element has begun to be incorporated into the design mining due to mining sequences (pushback) has begun to be considered (Bargawa, 2008).

The final stage of the mine planning process is the appraisal of the mining investment. Mining investment analysis is a systematic step taken to evaluate the profit potential of a mining project (Sari et al., 2018). Mining investment feasibility assessment is carried out by determining economic indicators (financial/economic model) such as net present value, internal rate of return (IRR) which takes into account the equivalent concept, and payback period which does not take into account the equivalent concept. It is very important in investment analysis that the project evaluation criteria do not by themselves provide investment decisions, but only provide guidelines for making decisions. This is because, in the process, mining activities require very large capital. Good financial planning will make the possibility of loss smaller and the amount of profit and return on capital can be estimated (Gentry and O’Neil, 1984).

Based on this description, this research was conducted to carry out conceptual planning in the North Block, starting from determining optimal mining boundaries (ultimate pit limit), mine design and reserve calculation, production scheduling, mining pushback design, cash flow analysis, and sensitivity analysis.

1.2 Problem Statement

The North Block is an area in PT Pacific Ore Resources that has not been mined requiring a mine conceptual planning study in the form of determining pit boundaries (ultimate pit limit), mine design and reserve calculation, production scheduling, mining pushback design, selection of heavy equipment, analysis of mining investment and sensitivity. Therefore, the formulation of the problem of this research are:

1. How are the shape of the most profitable ultimate pit limit.
2. How is the mine design of the North Block pit and how much are the evaluated laterite nickel reserves based on the mine design created.
3. How many mining pushbacks design can be formed.
4. How many digging-loading and hauling equipment are needed for each sequence to production the reserve.
5. How are the results of mining investment analysis obtained.
6. How is the sensitivity analysis of the obtained NPV based on changes in nickel prices, operating cost expenditure and capital expenditure.

1.3 Research Objective

The research conducted has the following objectives:

1. Determine the ultimate pit limit based on technical and economic parameters.
2. Create a mine design and evaluate nickel reserves based on the results of optimization studies with several geotechnical parameters.
3. Design a mining pushback design based on the production scheduling.
4. Calculate the number of digging-loading and hauling equipment needed for mining operations in each sequence.
5. Conduct mining investment analysis with NPV, IRR, and PBP parameters based on the mine planning result.

6. Conduct NPV sensitivity analysis based on changes in nickel prices, operation cost expenditure, and capital expenditure.

1.4 Research Benefits

This research is expected to be useful for various parties as follows:

1. This research is useful for the development and application of insights, and knowledge possessed by researchers, as well as the application of technological advances in mine planning.
2. The company will receive the results of a conceptual design for a work area that has not been mined so that it becomes a consideration in planning.
3. The academic field will get literature regarding mine planning, especially mining conceptual planning.

1.5 Research Scope

This research aims to undertake long-term mine planning with a focus on mine design and economic analysis. The research scope includes several relevant aspects, such as:

1. Mineral resources analysis: Estimating available mineral resources based on the block model resulting from resource modeling. This will form the basis for efficient mine exploitation planning.
2. Mine design planning: Planning the necessary technical infrastructure for long-term mining operations, namely pit, road access, and waste dump. The goal of this planning is to ensure optimal and sustainable infrastructure to support mining activities.
3. Mine Scheduling: Conducting scheduling to find out the life of mine (LOM). This will be the basis for creating a sequence of each mining period.
4. Economic Analysis: Conducting an economic analysis related to investment and long-term returns with several variables, namely capital expenditure, operational expenditure and interest rates.

CHAPTER 2
LITERATURE REVIEW

2.1 Mine Planning

Mine planning consists of three stages, namely geological modeling, long-term planning, and short-term planning. Long-term planning is the initial planning made before exploitation activity. Long-term planning is often directed at strategic planning or feasibility studies. Short-term planning, namely operational planning to achieve plans that have been set in long-term planning (Sasongko, 2009). Mine planning is a circular process or iteration as shown in Figure 1. Elements of the mine planning cycle are creating a mine design and managing mine layout (depth, stripping ratio, ore grade, selling price, distance, etc.); strategic planning and development; detailed long-term mine plan or feasibility study; machine equipment selection; mine development phase; yearly planning, monthly planning and daily planning schedules; production scheduling; estimation of mining costs and construction of financial models (Malli et al., 2015).

After the geological model for coal/mineral resources is created, then the block model is made. Creating a block model is the stage of dividing the resource area into smaller blocks. After that, it is adjusted to the production scheduling plan and the mining equipment used. Based on the block model, the number of resources can be estimated numerically. The next stage is pit optimization by considering technical factors; safe mine slope angle, level, and local conditions, and consideration of economic factors; selling prices of mining commodities, mining costs, and financial obligations of mining companies to the government (Sasongko, 2009).

Pit optimization is to determine the ultimate pit limit, where the mine boundary is used as a spatial boundary in the calculation of mineable reserves. Once the mineable reserves are known, the next stage is a production planning, namely in the form of mine stage planning (pushback) activities, mine sequences, and mine production scheduling. The final stage of the mine planning process is the assessment of reserves by determining economic indicators such as net present value, internal rate of return, and payback period (Sasongko, 2009).

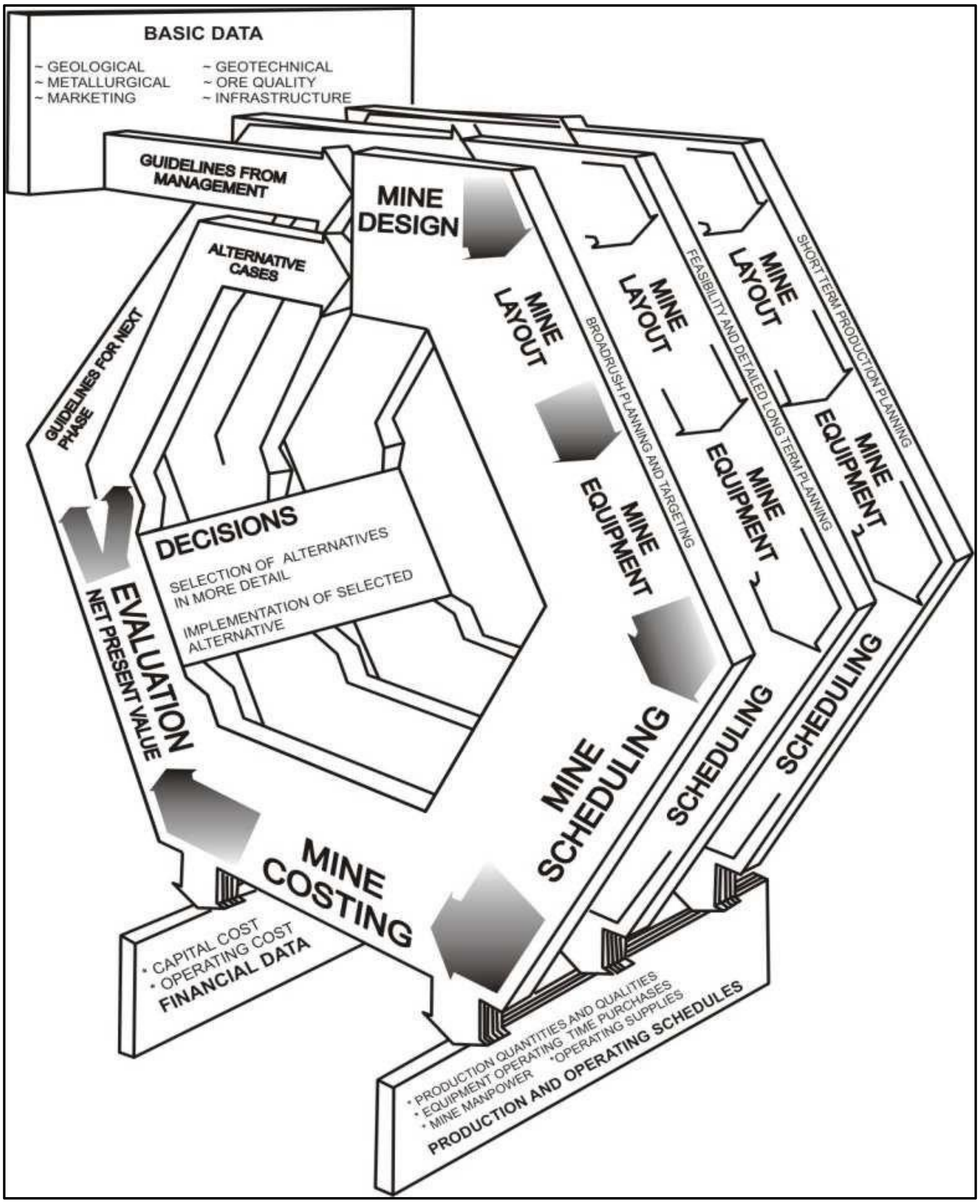

Figure 1 Phases of mine planning cycle and related element (Malli et al., 2015)

Things that need to be considered in making a mine planning are (Hustrulid et al., 2013):

1. Determination of pit boundaries

   Determination of pit limits is determining the pit limit final of the mining process, where a mine planner must be able to plan how much mineral will be mined, but in determining this pit limit, time and cost have not been taken into account.

2. Pushback design

   In mining geometry design, pushback design is an important stage because at this stage the determination of pit limits is made into even smaller parts, making it easier to work on, and in designing the three-dimensional shape of the mine it becomes even easier.

3. Production scheduling

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

4. Tool selection

   Once the production to be achieved is known, the next stage is the selection of the tools to be used in the mining activity. In addition to selecting tools for production, tools 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 target to be achieved and the selection of equipment to be used. In addition, at this stage, the amount of working time and work shifts required to achieve the planned production targets can also be determined.

2.2 Pit Optimization

Pit optimization is an effort to determine the ultimate pit limit and determine the optimum reserves that provide the best profit margin. Ultimate pit limit boundaries define what can be economically mined from a particular deposit by identifying which blocks should be mined and which should be left in the ground. An effort to identify blocks to be mined, an economical block model is created from the geological model. This is done using production and process costs and commodity prices at current economic conditions (i.e. current costs and prices). Then using the value of the economic block, each positive block is examined further to see if its value can pay for the removal of the waste block above it. This analysis is based on a break even calculation that checks whether the undiscounted profit obtained from a particular ore block can pay for the undiscounted cost of mining the waste block. The analysis is based on the breakeven calculation that check if undiscounted profits obtained from a given ore block can pay for the undiscounted cost of mining the waste blocks (Dagdelen, 2001).

Pit limit determination is an iterative process with the aim of maximizing profit. In mine planning, the determination of pit limits is the initial step that needs to be carried out before carrying out the design of the mine design and mining direction. In determining the pit limit, a method that uses the Lerch-Grossmann mathematical algorithm is generally used where this method is good enough to be applied to almost all deposits (Sidiq and Pusvito, 2017).

The goal of any optimal open hole design algorithm is to determine the ultimate opening limit of the ore body and the associated grade and tonnage, which will maximize some 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 common goal of maximizing overall mining profits within the designed pit boundaries (Djilani, 1997).

The first step in any optimization problem is to define optimization criteria. For pit design, there are a number of criteria, namely technical, geological and economic matters or a combination of the three. The most commonly used criteria are economics 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 variant of maximum profit. However, ore bodies can be mined at a range of cut-off values which will each (at least over practical values) yield the same amount of metal for the different tonnages of ore mined. Based on the time value of money, it is always more profitable to mine at a higher cap rate in the early years and then at a decreasing cap rate over the following years. So, the optimization criterion should be the maximum net present value rather than the maximum total profit (Djilani, 1997).

2.3 Lerchs-Grossman Algorithm

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

The Lerchs-Grossman (1965) optimization algorithm finds the maximum closure of a weighted directed graph, in which case vertices represent blocks in the model, weights represent net profit of blocks, and arcs represent mining constraints (usually slope). The algorithm solves very special cases of linear programming or network flow problems. Since the problem is a special subset of general linear programming problems, it can only be expected that algorithms specifically designed to solve such a subset might be more computationally efficient. The basic Lerchs-Grossman algorithm has been used for more than thirty years in many feasibility studies and for many producing mines (Muir, 2007).

Despite the optimal nature (Lerchs and Grossman, 1965), this algorithm has the drawbacks of method complexity, long computation time and difficulty in combining the slope of the pit variables. This method converts the economical block model of the deposit into a directed graph, which is a simple diagram consisting of a set of nodes, or vertices, and a set of connecting arcs (lines with directions) used to show the relationships between nodes. Each block is represented by a node; each node is assigned a mass equal to the net value of its corresponding block. The nodes are connected by arcs in such a way as to represent mining, or access, constraints. These arcs indicate which blocks must be removed before certain blocks can be mined. Figure 2 shows a directed graph for a simple two-dimensional example where the inclination angle of the pit is 45° and the blocks are square. In this example to mine block 10, it is first necessary to delete blocks 2, 3 and 4.

In graph theory notation, the vertex is denoted x_I and the arc connecting the vertices x_i and x_j is denoted (x_i,x_j), the order of which determines the direction of the arc. If the set of all vertices is denoted X and the set of all arcs is denoted A, then the graph G = (X, A) is defined as the set of all vertices X together with the set of all arcs A. It is said that a vertex x_j becomes the successor of vertex x_i if there is an arc with an initial extremity at xi and its terminal extremity is in x_j. The set of all successors from node x_i is denoted by Γx_i. For example, in Figure 2 the set of all successors to node number 18 is $\Gamma x_{18} = \{x_{10}, x_{11}, x_{12}, x_2, x_3, x_4, x_5, x_6\}$.

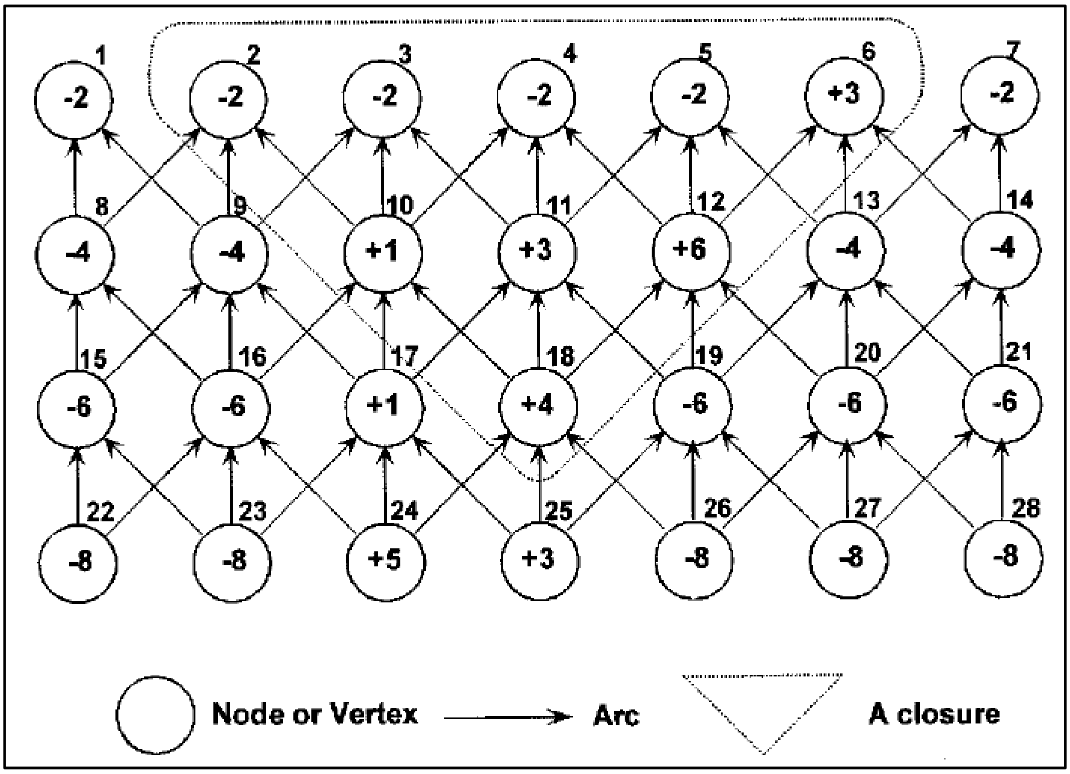

Figure 2 Directed graph representing a vertical section (Khalokakaie et al., 2000)

A node set is a graph closure if successors of all vertices in the set are also included in the set, i.e., if the set of blocks, represented by nodes, satisfies the access constraints for all blocks in the set. Thus, vertices 2, 3, 4, and 10 are closures. The closures is defined as a subset of vertices $Y \subset X$ such that if $x \in Y$, $\Gamma x \in Y$. The closing value is the sum of the masses of the knots in it. The optimal open hole is determined by closing with the maximum value. Thus, the algorithm involves finding the maximum closure of the graph representing the ore body block model (Khalokakaie et al., 2000).

The shape of the ore deposit is represented as a model block containing the ore grade or yield block. The Lerchs-Grossman algorithm determines the shape of the pit by identifying the above blocks that must be excluded to provide access to each block in the block model. The first outcome of pit optimization is to determine the primary pit that provides the highest possible undiscounted overpayment between net revenue and total operating costs, without considering scheduling constraints or discounts. A layered pit shell analysis is then used to determine the optimal discounted pit. Layered pit shells are sequences of ultimate pits that are generated by raising the price of a commodity around its base price value. The optimal pit provides the highest net present value, taking into account all operational scheduling constraints and recurring capital cost (Micromine, n.d.).

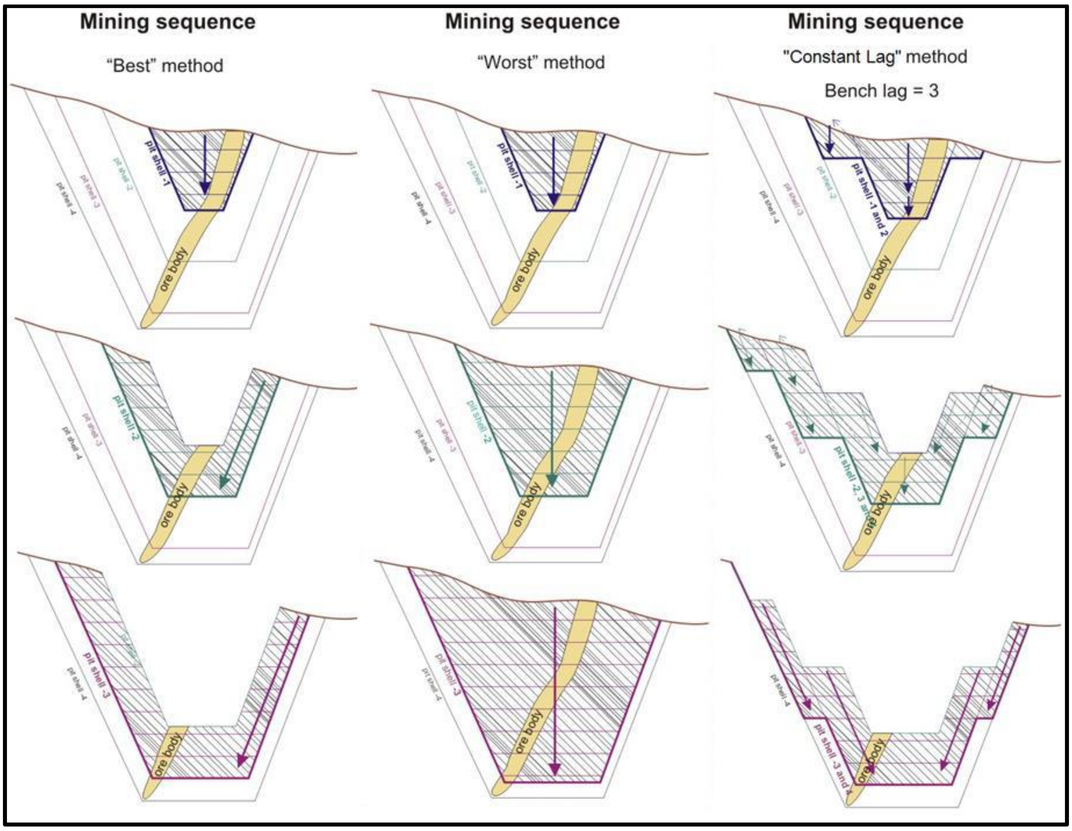

Figure 3 Best and worst case and constant lag mining scenarios
(Micromine, 2014)

Three methods are used to determine the mining sequence for the pit shells. The best method assumes that the pit shells will be mined consecutively (initially the first one, then the second one, etc.). The worst method assumes that each pit shell will be mined completely from top to bottom, without taking any other pit shells into consideration. Constant lag assumes that the pushbacks will be mined in consecutive order, as per the best case scenario, but also considers the number of benches that need to be mined on each pit shell. If the lag is set to 0 (zero), each pushback is completed before the next is started, so the analysis works exactly like the worst case. Obviously, increasing the bench lag takes the mining sequence closer to the best case (Micromine, 2014).

2.4 Mine Design

The mine design consist of bench geometry pit, mine road or ramp, reserve calculation, and disposal design.

2.4.1 Bench geometry

The elements that must be considered when designing a mine pushback pit design are the geometry of the steps, including slope, bench width, bench height, and operational entrance (ramp) whose dimensions are determined by: (a) heavy equipment used (especially digging and transporting equipment), (b) geological conditions, (c) physical properties of the rock, (d) the desired separation selectivity between ore and waste, (e) production rate and (f) climate. The bench height is the vertical distance between the horizontal levels in the pit; the width of the bench is the horizontal distance of the floor where all excavation, loading and drilling-blasting activities are carried out; and the slope of the bench is the slope angle of the bench. The height limit of the bench is strived according to the type of loading device used so that the top part is reached by the loading device (Hustrulid et al., 2013).

Factors that affect the geometry of the bench include (Novian, 2008):

1. Production

   The aim of making levels is to get nickel deposits according to the desired production. The dimensions of the bench to be made need to consider the amount of planned production.

2. Rock condition

   The condition of the rock determines which equipment should be used. The dominant rock conditions include rock strength, swelling factor, rock density and geological structure. Distance and excavation height need to be taken into account in estimating the width and height of the bench.

3. Production equipment

   The production equipment to be used must be adjusted to the desired production capacity and according to the material to be worked on.

The bench dimension that takes into account these factors has good working conditions which will affect work efficiency. Figure geometry levels can be seen in Figure 4.

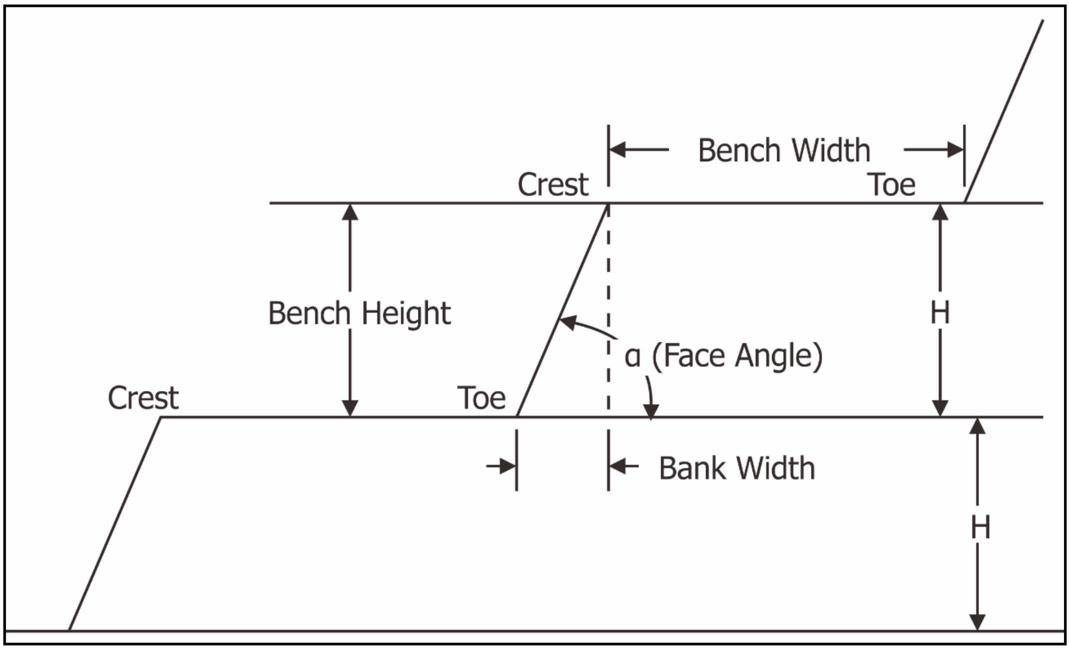

Figure 4 Parts of a bench geometry (Hustrulid et al., 2013)

Bench width is the horizontal distance measured from the end of the floor level to the back of the floor level. There is no standard equation for calculating the minimum bench width. The minimum bench width is strongly influenced by (Novian, 2008):

1. Equipment types and dimensions.
2. The working position of the equipment that is operating on the same floor.
3. The width of the excavating material.

The working bench is a level where mining or quarrying takes place. The part being excavated from the working bench is called the cut. After the cut is mined, a safety bench will be formed. Figure work levels can be seen in Figure 5 a safety bench may also be called a catch bench and may refer to a berm. The function of this safety bench or berm is to collect soil material that falls from the bench above and hold it so that fatal avalanches do not occur.

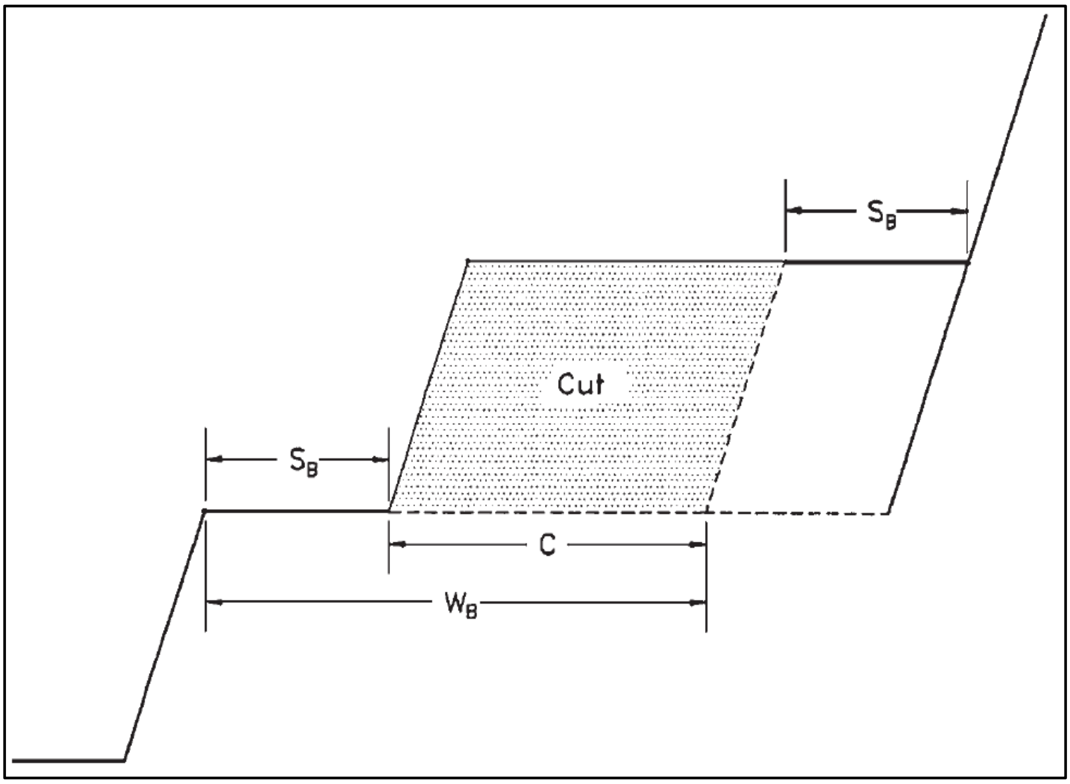

Figure 5 Parts of working bench (Hustrulid et al., 2013)

At catchment levels, material domes (berms) are usually found along the crest. The domes will form a channel between the domes and the toe to catch falling rock. The impact zone is the area affected by falling material (Hustrulid et al., 2013). The catch bench geometry can be seen in Figure 6. The width of the catch bench is adjusted to the height of the bench. Based on studies conducted by mining experts it is recommended that the higher the bench, the wider the catch bench is required.

Single slope is a slope formed by one level or formed by crest and toe. The overall slope is the slope formed by the entire level. This slope is measured from the top crest to the last toe of the mining front. The slope is strongly influenced by rock characteristics and blasting activities. In pit operations, controlling slope angle is usually done by marking the location of the desired crest using a small flag. The shovel operator was instructed to dig until the bucket reached the location of the flag (Hustrulid et al., 2013).

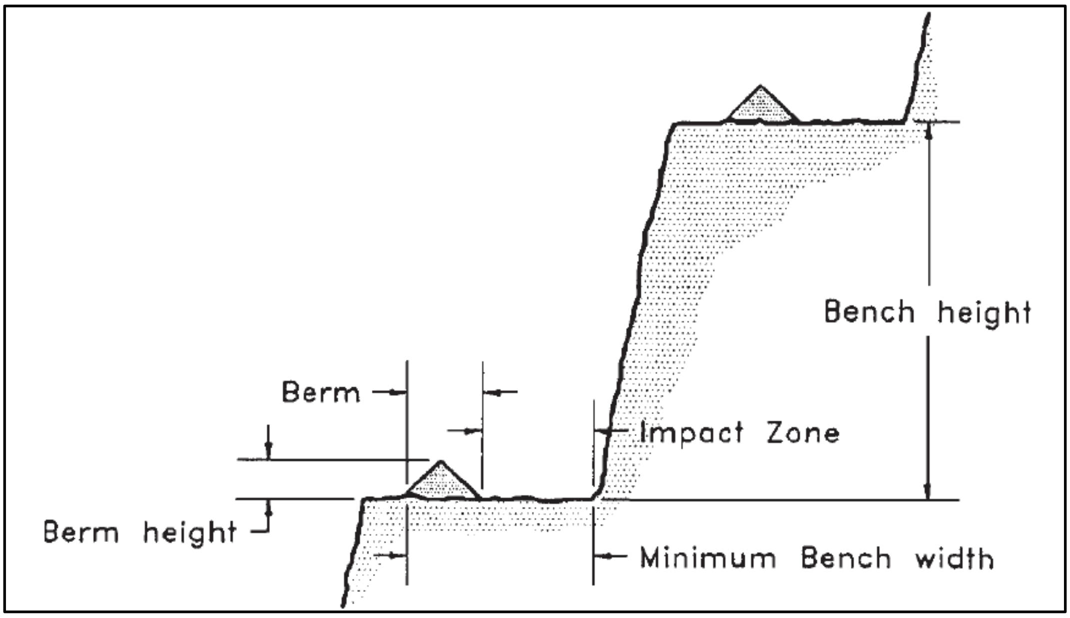

Figure 6 Catch bench geometry (Hustrulid et al., 2013)

2.4.2 Ramp

Ramps (or declines for underground) are often used in mines to transport ore, waste, materials, and personnel (Haviland and Marshall, 2015). Ramp is a road that is used in the mining pit area (bench) and will be used in accordance with the direction of the mining progress. The following are the parameters for making a ramp design based on (Hustrulid et al., 2013):

1. Berm width, namely the distance between the toe of the upper slope (toe) and the head of the lower slope (crest) designed at the same elevation.
2. Overall slope height, is the total height of the slope from the topographical surface to the lowest depth of the mine design (pit bottom).
3. Overall slope, is the total angle from the slope to the bottom depth of the mine design (pit bottom).

The ramp width shown in Figure 7 is designed based on road geometry calculations according to Hustrulid et al (2013) in below.

$$\mathrm{Lmin} = \mathrm{N} \times \mathrm{Wt} + (\mathrm{n} + 1) \times (1/2 \times \mathrm{Wt}) \tag{1}$$

where,

Lmin = minimum width of ramp (m)

N = number of lanes

Wt = dump truck width (m)

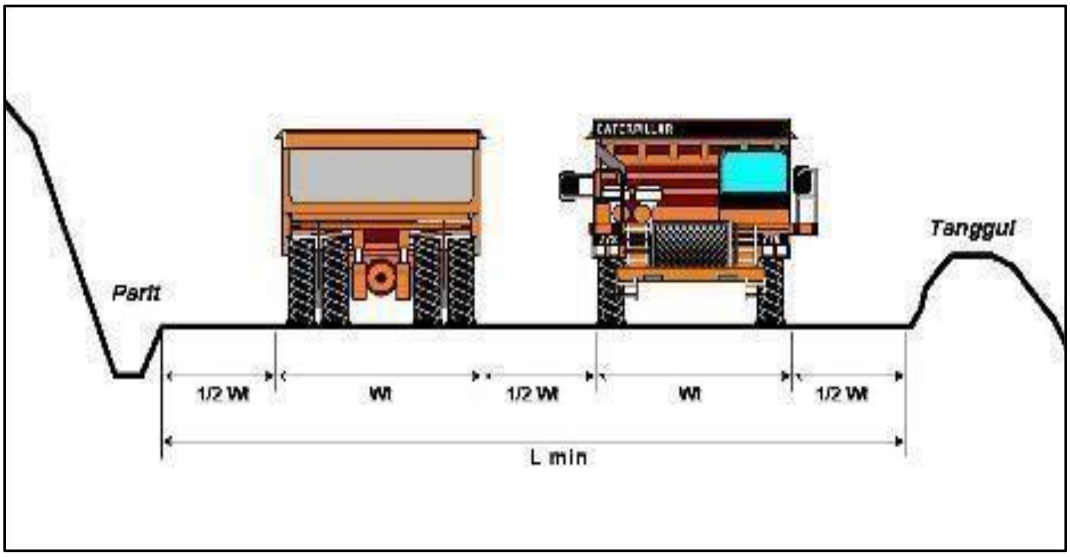

Figure 7 Parts of mine road (Hustrulid et al., 2013)

2.4.3 Reserve

Reserves are mineral deposits whose size, shape, distribution, quantity and quality are known and which are economically, technically, legally, environmentally and socially mineable at the time the calculation is made (Sinclair and Blackwell, 2002).

1. Calculation of reserves

Recoverable reserves are determined from a subset of local estimates (reserves actually recoverable by the planned mining procedures) and serve as a basis for financial planning. Calculation of reserves must take into account certain requirements, including (Sinclair and Blackwell, 2002):

a. A reserve estimate must accurately reflect the geological conditions and characteristics/nature of the deposit of minerals.
b. A reserve model to be used for mine planning must be consistent with the mining method and mine planning techniques to be applied.
c. A good estimate and must be based on actual data that is processed/treated objectively. The decision to use or not to use data in interpretation must be taken with clear and consistent guidelines. There should be no different weighting of data and it must be done on a solid basis.

Reserve calculation methods are used according to the shape and direction of distribution of reserves, including (Sinclair and Blackwell, 2002):

a. Cross section method

Cross section method is more suitable for deposit types that have sharp contacts such as tabular forms (bedding or veins). The exploration pattern (drill) is generally regular and is located along the cross-sectional line, but the case of deposits to be mined underground generally has a less regular drill pattern.

b. Polygon method (area of influence)

This method is generally applied to deposits that are relatively homogeneous and have simple geometries. The grade of an area within a polygon is estimated by the data value in the middle of the polygon, so this method is often called the area of influence method. The weakness of this method is that it does not take into account the layout (space) of data values around the polygon and there is no definite limit to how far the sample values affect the distribution of space.

c. USGS circular 891 method

The United States Geological Survey (USGS) system is an extension of the block system and the usual volume calculations. The USGS system is considered suitable to be applied in the calculation of coal reserves because this system is aimed at measuring minerals in the form of layers (tabular) which have thickness relatively consistent slope of layers.

d. Triangle method

This method is used to estimate parameters and is also used to calculate reserves. The calculation equation in this method is almost the same as the polygon method, except that in the triangle method the data points are used to represent the parameters of the entire triangular area, while the polygon method uses data points that are in the middle of the polygon area.

e. Block system method

Computer modeling to represent mineral deposits is generally carried out using a block model. The dimensions of the block model are made according to the mining design, which is the same size as the bench height. All parameters such as rock type, quantity of minerals, and topography can be modeled in block form. The parameters representing each regular block are obtained using common interpretation methods, namely NNP, IDW, or Kriging.

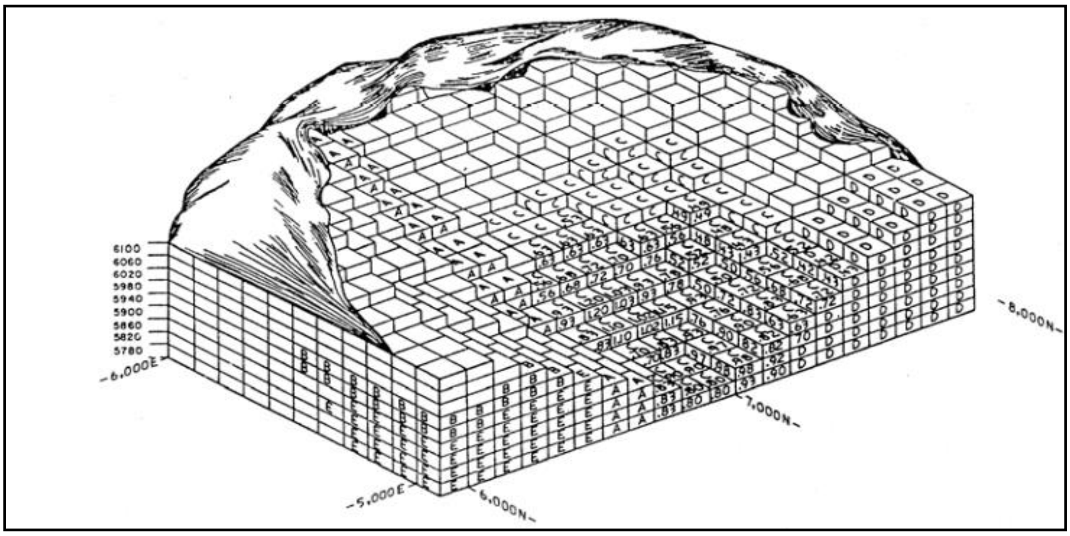

Figure 8 Reserve calculation with block model method (Sinclair and Blackwell, 2002)

2. Resources and Reserves Classification

Classification of mineral resources and reserves is a process of collecting, filtering, and processing data and information from a mineral deposit to obtain a brief description of the deposit based on the criteria Komite Cadangan Mineral Indonesia (2017) the classification of mineral resources and reserves divided in to several parts:

a. Hypothetical mineral resources are mineral resources whose quantity and quality are obtained based on estimates at the review survey stage.
b. An inferred mineral resource is a mineral resource whose quantity and quality are obtained based on the results of the prospecting stage.
c. Indicated mineral resources are mineral resources whose quantity and quality are obtained based on the results of the general exploration stage.
d. Measured mineral resources are mineral resources whose quantity and quality are obtained based on the results of the detailed exploration stage.
e. Probable reserves are indicated mineral resources and some measured mineral resources whose level of geological confidence is still lower, based on a mine feasibility study, all related factors have been fulfilled, so that mining can be carried out economically.

f. Proved reserves are measurable mineral resources based on a mining feasibility study, all related factors have been fulfilled, so that mining can be carried out economically.

2.4.4 Disposal design

Disposal is an area in an open pit mining operation that is used as a place to dispose of low-grade materials and/or non ore materials. Disposals are usually made in ex-mining pits or quarry mining. Disposal planning broadly consists of two parts, namely technical and economic aspects. The technical side of disposal planning includes design, allocation of dozer thrusters, and the age of a disposal. The economic side is the cost of a disposal which includes the operating costs of the equipment and the cost of using disposal reinforcement materials, namely civil materials (Fajrin et al., 2019). A good disposal design should be made by following a predetermined production plan and following predetermined geometry rules or geotechnical parameters, so that the design can accommodate production and is safe to implement in the field (Hardianti and Halim, 2021).

The disposal design is very important for economic calculations. The location and form of disposal will affect the number of truck shifts, operating costs and the number of trucks in a fleet required. In general, the area required for disposal ranges from 2-3 times that of the mining area (pit). This is based on considerations including:

a. Loose material expands 30 - 45 % compared to in situ material.
b. The slope angle for a dump is generally gentler than the pit.
c. Material generally cannot be stacked as deep as the pit.

The types of disposals commonly applied in mining using open cast mining are divided into three types, namely finger disposal, semi induced disposal and induced flow disposal (PT Vale Indonesia Tbk, 2018):

1. Finger disposal

   Finger disposal is a disposal made advanced with the help of a dozer. This type of disposal has the characteristics of a height of less than 15 meters with a gentle slope of less than 40°. Continuity of civil material is needed as the basis for the dump truck so that avalanches do not occur.

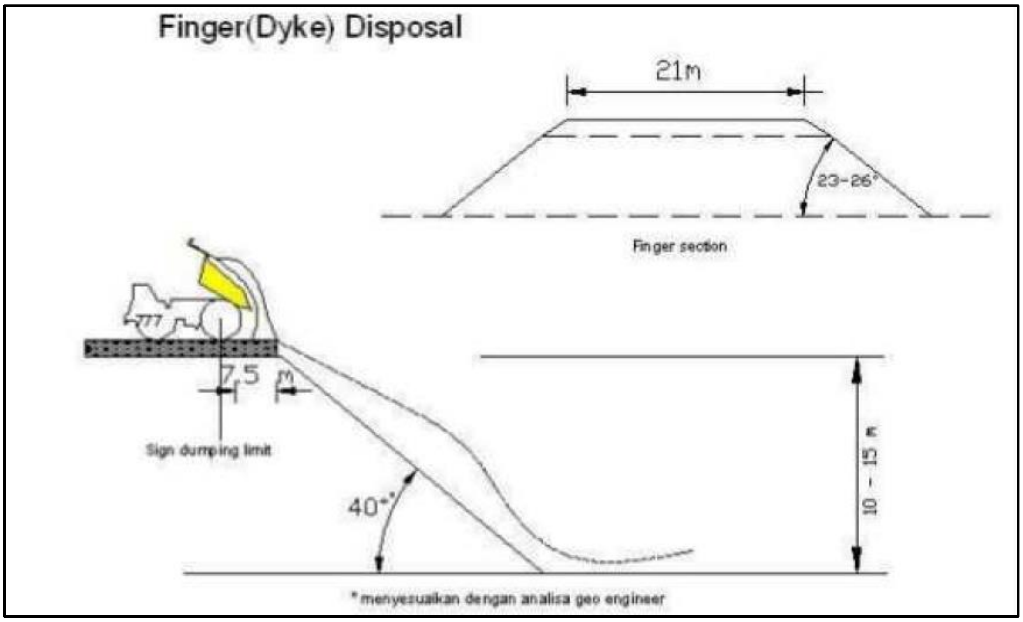

Figure 9 Finger disposal design (PT Vale Indonesia Tbk, 2018)

2. Induced flow disposal

   Induced flow disposal is a disposal type that utilizes a height difference of > 15 meters for dumping material, with a tilt angle of 50° to a maximum of 70°. This type of disposal is built on original, stable soil (original, in the blue zone area or on an area recommended by the geotechnical engineer). This disposal is also equipped with a backstop as a mount (bund wall) half the height of a truck wheel tire which is located at the end of the crest as shown seen in Figure 10.

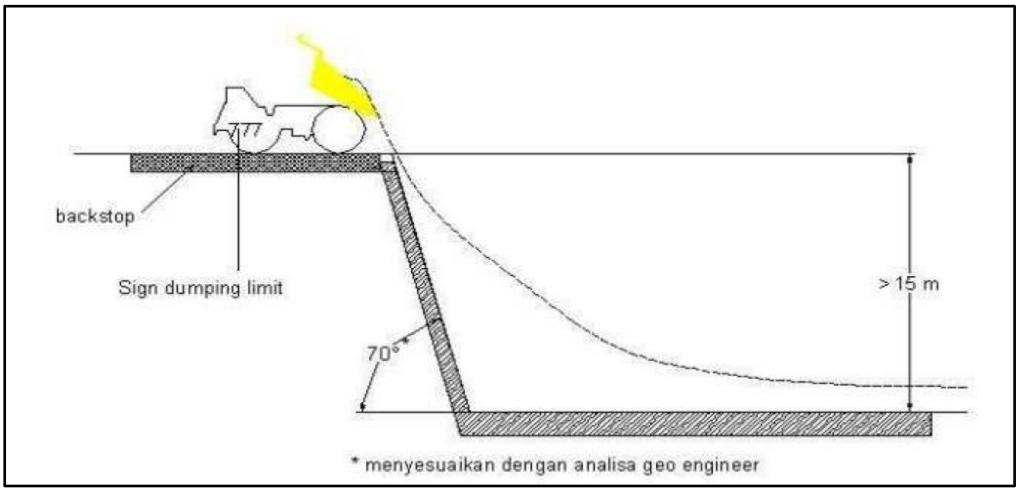

Figure 10 Induced flow disposal design (PT Vale Indonesia Tbk, 2018)

3. Semi induced disposal

   Semi induced flow disposal, generally the same or similar to induce flow but trucks can only dump at a certain distance that is allowed, namely 12.5 m from the original crest. After that, the overburden is pushed by the dozer until the end of the crest. The crest to the toe is 30 meters with a slope between 26°-36°. Semi induce flow requires the help of civil material on the truck bed that will support the soil to increase the carrying capacity of the soil to prevent subsidence. Due to the greater slope, this type of disposal requires fewer dozers than Finger Flow. However, the dozer drive limit on this type of disposal has not moved forward.

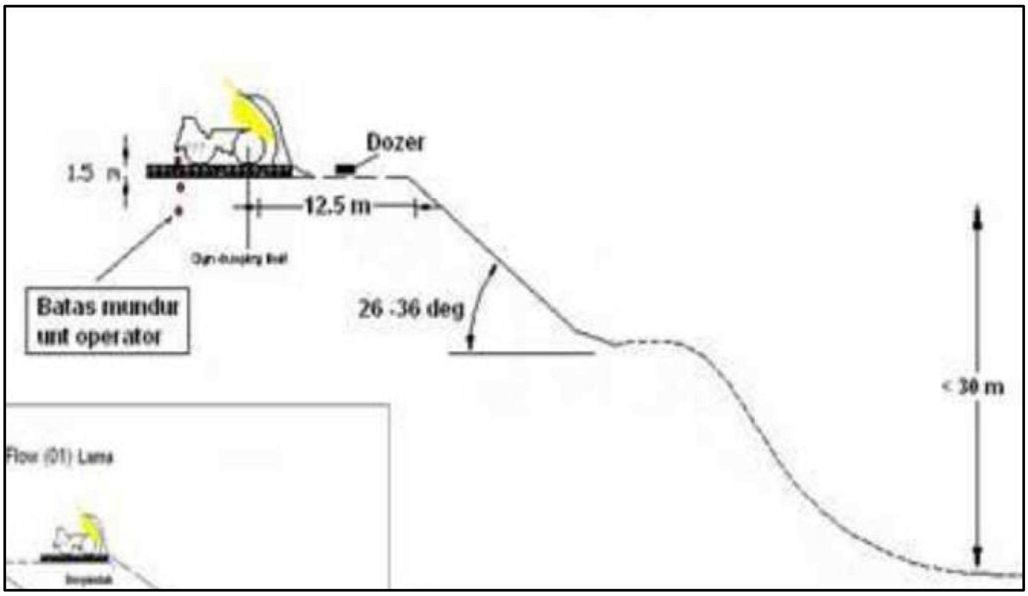

Figure 11 Semi induced flow disposal (PT Vale Indonesia Tbk, 2018)

2.5 Mine Scheduling

Mine scheduling is process of determining the step of mining blocks and simulating their movement while collecting minerals in each time period to fulfill predetermined targets. Mine Scheduling often includes grade, tonnage and waste considerations in each period and it is these elements that make the scheduling problem more complex than mining ore to the amount required. The development of a practical mining schedule is an iterative process in which an outline of the holes with the highest NPV cannot be determined unless the block values in the model are known. Input data for open pit mine scheduling include (Ricciardone and Chanda, 2001):

1. A block model or mining database, representing the economic boundaries of the ore body.
2. Target amount of material to be mined per period, including grade and tonnage of ore and waste.
3. Definition of the timeframe for the mine plan (days, weeks, months or years).

There are various methods for solving long-term production scheduling. Each of these methods to solve long-term production scheduling problems use one of the following strategies (Pouresmaieli et al., 2017):

1. Mining pits are determined and then a production schedule is obtained using mathematical programming to maximize net present value. In the big pits, after setting the pit limit, some pushback is gained. Then, the mining stage in pushback is planned using mathematical techniques.
2. Mining pit limits and production schedule are determined simultaneously and the algorithm uses this strategy.

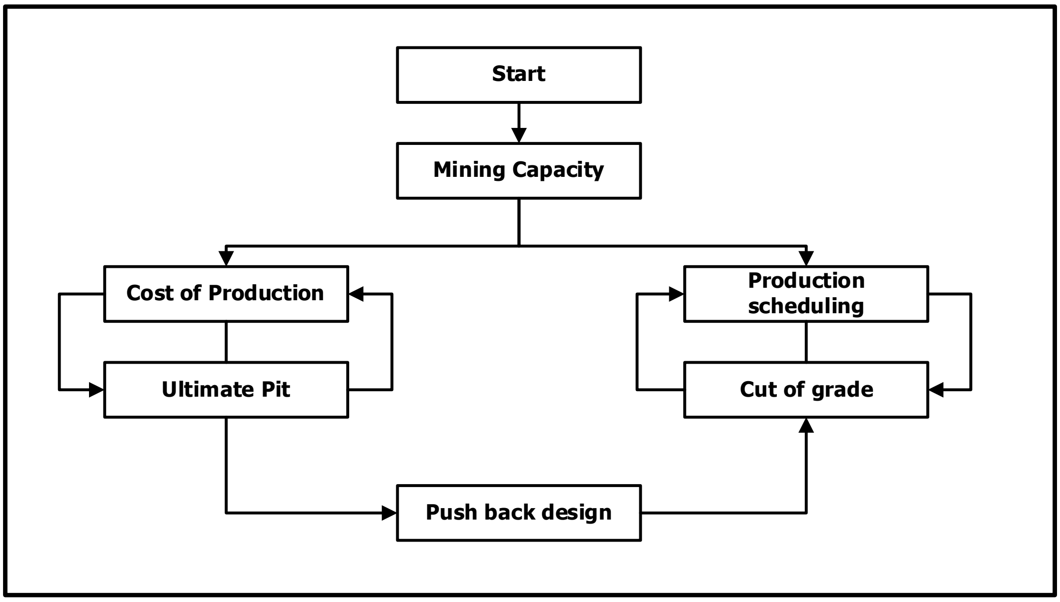

Figure 12 Open pit mine scheduling variable (Pouresmaieli et al., 2017)

The long-term production scheduling of a mine is to determine the timing and sequence of extraction of ore and waste blocks from different mining points so that taking into account various existing operating constraints, the highest economic value is obtained for the mine. Providing this program is very important for mining engineers. The four main parameters that affect long-term mine scheduling are shown as cycles in Figure 12 which interact with each other. The four parameters are the ultimate pit, annual production scheduling, costs, and grades (Pouresmaieli et al., 2017).

Figure 12 shows that in order to determine ore expansion as well as pit boundaries, a cut-off grade must be determined. After determining the pit boundaries, the mine production schedule is determined. As seen in this cycle, each variable value cannot be achieved unless the previous variable value has been obtained. This process is a multivariable optimization process and requires simultaneous completion (Pouresmaieli et al., 2017). The time to complete work is greatly affected by the number of statutory holidays, and work days lost due to climate and scheduled and unscheduled breakdowns. The calculation of work (production) targets is carried out using Equation 2 (Hanafi et al., 2020).

Working target = working volume (lcm)/effectivity time (hour) (2)

Production scheduling during the life of the mine (LOM) is a very important and decisive part of the open pit system mining business and is related to managing the effective flow of funds. LOM production scheduling functions to determine the amount and quality of ore and waste material excavated from the mine over time, to maximize the net present value (NPV) of the mine (Hakim et al, 2020). Mine life (life of mine) is the time calculated from the amount of reserves divided by mine production per year. Mine life is strongly influenced by the amount of reserves that can be mined and the level of production per year. The calculation of mine age can be calculated using Equation 3 (Pranata and Yulhendra, 2021).

$$LOM\ (year) = \frac{Reserve\ (ton)}{Production\ (\frac{ton}{year})} \qquad (3)$$

Life of mine is made neither too fast nor too long, depending on the company's ability to determine production levels. Too low a production level means that the profits will take a long time (the return on investment will take a long time), while the production level is too high, the investment costs can be too large so that the company's financial capacity is likely to be unable to handle it (Pranata and Yulhendra, 2021).

2.6 Pushback Design

The first step in the engineering planning process is to break down the entire pit reserve into more manageable planning units. These units are commonly called sequences, expansions, phases, working pits, slices or pushbacks. Initially a fairly rough division covering a period of several years may be used. This is an early attempt to link mining geometry with ore distribution geometry (Hustrulid et al., 2013).

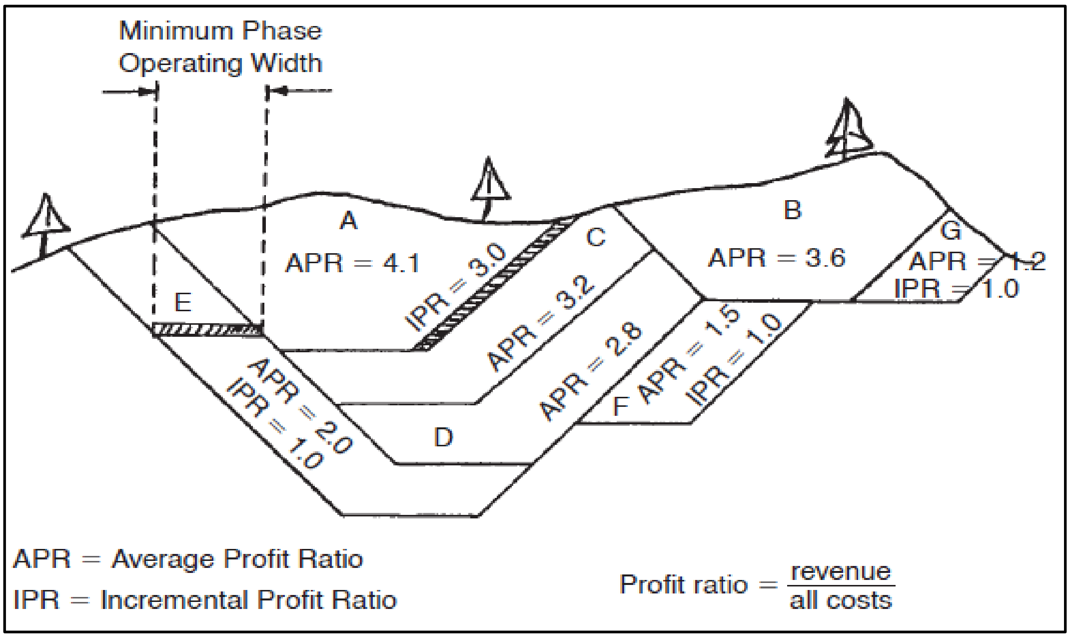

Figure 13 Pit order in descending order of value (Mathieson, 1982)

The planning phase should start with mining the portion of the ore body that will generate the maximum cash flow. Subsequent phases are sequenced in terms of their cash flow contribution. Finally, the final pit limit was reached. Figure 13 shows a two-dimensional representation of the phases used to extract the ore reserves Mathieson (1982) in Hustrulid et al., (2013). The extraction sequence progresses from the phase that has the highest to the lowest average profit ratio (APR). In this case they proceed in alphabetical order A to G (Hustrulid et al., 2013).

$$Profit\ ratio = \frac{Revenue}{All\ cost} \tag{4}$$

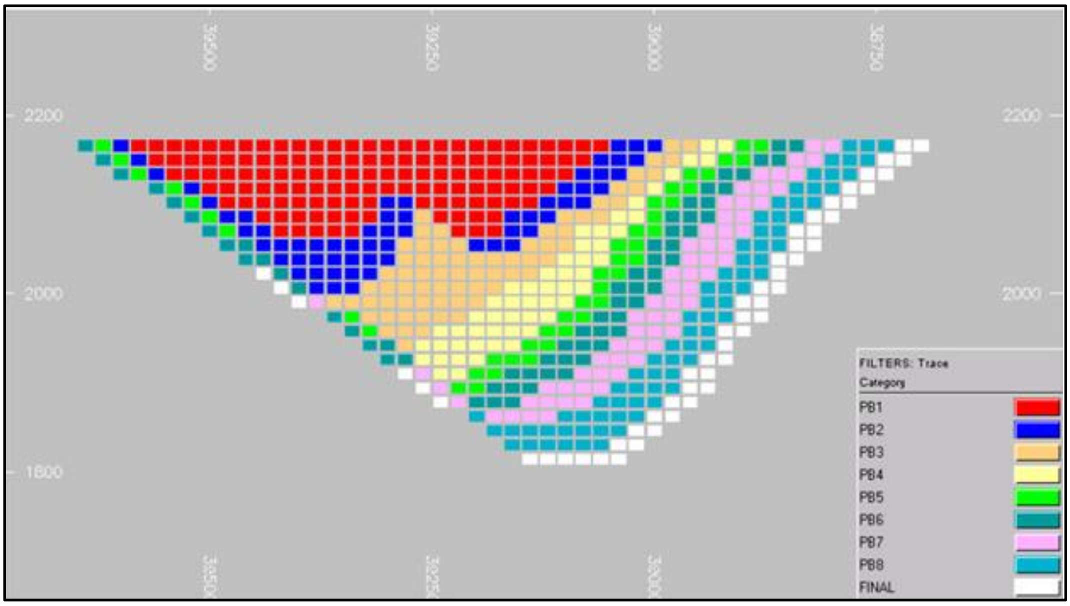

Figure 14 Open pit mine pushback (Smith, 2013)

Figure 14 shows a model cross-section of a typical open pit mine block. Exploitation begins at the surface and material is removed gradually which is called pushback. These are exploited sequentially before reaching the ultimate hole. The final hole cover shows the boundaries of the mine after the exploitation of the mineral resource has been completed (Arteaga et al., 2014).

In open pit mining the objective is to remove the ore from the ground and bring it to the surface. However, ore is generally located in the deeper parts of the pit, therefore the start of each pushback takes into account most of the waste extraction. Once the ore extraction begins, the ore and waste are removed from the mine location by maintaining a relationship called the stripping ratio (Arteaga et al., 2014).

The time parameter needs to be taken into account in designing pushback because time is a very influential parameter. Well designed mining stages will provide access to all work areas and provide sufficient work space for the operation of mining work equipment. There are several factors that need to be considered when planning a pushback, such as geology, geotechnical factors, haul road design, economics, heavy equipment selection, hydrology, production targets, and environmental issues (Reza, 2018).

2.7 Equipment Selection

The basic method of scheduling equipment in the open pit mining industry is to determine the equipment needed to maintain the desired tonnage and then schedule what equipment is needed to do the work. In the event of a breakdown, production is maintained by employees by taking another machine that is on standby or a tool that has just been repaired. To properly schedule and measure the entire equipment fleet requires knowledge of a number of factors related to machine availability and usage. In this case, the following terms are used (Hustrulid et al., 2013):

1. Availability
2. Utilization
3. Work Efficiency
4. Operations Efficiency

Utilization is key performance indicator of equipment and it is a tool commonly used for decision-making by management in mine operations. Production rates are very sensitive to equipment availability and utilization. Therefore, serious efforts must be made by mining companies to achieve and maintain higher levels of availability and utilization for capital-intensive equipment. By looking at the related literature, field visits, and investigations at the case study mines, various events in the operation of the BELT equipment were identified, and the classification used in the definition of the concepts of availability and utilization is shown Figure 15 (Mohammadi et al., 2015).

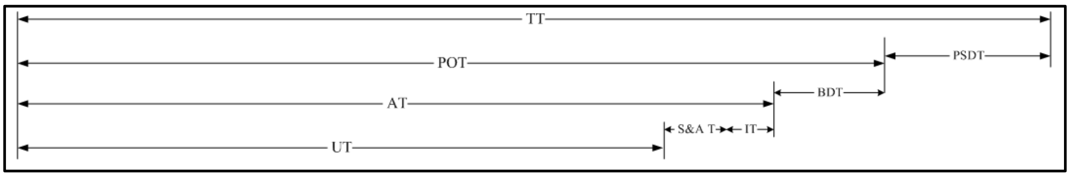

Figure 15 Classification of total time by different types of time (Mohammadi et al., 2015)

Where, TT is total time, PSDT is planned downtime or unscheduled time for operation, POT is planned uptime or loading time, BDT is breakdown time, AT is available time, IT is stand-by time, S&A T setting and adjustment, and UT is utilization time. Various forms of availability are determined depending on their application and time duration considerations, such as operational availability and default availability (Mohammadi et al., 2015).

2.7.1 Operational availability

Operational availability is associated with the operation of equipment or systems. This can be represented by the number of hours in one period that the machine is fit to work. However, there are some inconsistencies in the definition and use of the term within a period, namely some researchers propose a time-based approach that considers this period based on scheduled and unscheduled operating times. Other researchers have proposed loading time-based approaches that consider this period based on planned or scheduled uptime. The main difference between the two approaches lies in the consideration of the total time that can be accounted for evaluating operational availability. Mathematically, the difference between these two approaches can be understood by Equations 5 and 6.

$$A_0 = \frac{AT}{TT} = \frac{TT - (PSDT + BDT)}{TT} \tag{5}$$

where, A_0 is operational availability

$$A'_0 = \frac{AT}{POT} = \frac{POT - BDT}{POT} \tag{6}$$

where, A'_0 is operational availability based on loading time

2.7.2 Availability inherent

Availability inherent (A_i) is associated with the inherent characteristics of the equipment or its parts. This ignores downtime due to other sources not directly attributable to the equipment design and generally beyond the designer's control. Therefore, it is recommended to assess design characteristics during the design process which can be used as an important tool for developing optimal preventive maintenance schedules, parts management, and replacement strategies. Mathematically A_i can be expressed as follows:

$$A_i = \frac{MTTF}{MTTF + MTTR} \tag{7}$$

where,

MTTF = average time to breakdown,

MTTR = average time to repair.

2.7.3 Utility (U)

Utility or what can be called work efficiency signifies the productive use of available hours. An appliance may be available but may still be out of service during the available hours due to irregular and out-of-work conditions. The working efficiency of the tool can be expressed as the ratio of UT and AT. Mathematically, it can be stated in Equation 8.

$$U = \frac{UT}{AT} \tag{8}$$

2.7.4 Production cycle time

Production cycle time is the time required to perform a work cycle. The cycle time of the digging equipment consists of loading until leaving and until it is ready to be reloaded, while the circulation time of the conveyance consists of the time of arrival until the time of dumping and until it spots, the cycle time of the tool loading and unloading can be calculated by Equations 9 and 10.

$$CT = C1 + C2 + C3 + C4 \tag{9}$$

where,

CT = cycle time (second)

T1 = digging time (second)

T2 = swing time with the bucket filled (second)

T3 = dumping time (second)

T4 = swing time with the empty bucket (second)

while the calculation of the dump truck cycle time is as follows:

$$CT = T1 + T2 + T3 + T4 + T5 + T6 \tag{10}$$

where,

CT = cycle time dump truck (minute)

T1 = maneuver to loading time (minute)

T2 = loading time (minute)

T3 = running time with the vessel filled (minute)

T4 = maneuver to dumping time (minute)

T5 = dumping time (minute)

T6 = running time with the empty vessel (minute)

2.7.5 Bucket fill factor

The bucket fill factor indicates how much of the available space in the bucket is used. This is the percentage of bucket capacity that is actually filled with material. Mathematically, it is expressed as:

$$Bucket\ fill\ factor\ = \frac{Material\ volume\ in\ bucket}{Bucket\ capaticies} \tag{11}$$

The bucket fill factor depends on the size and shape of the bucket, the digging ability of the material (moving and filling the bucket), fragmentation (particle size, shape and distribution of the material in the bucket), the angle of swing of the material over the bucket, the skill of the operator and more.

2.7.6 Material swell factor

Mine material after being excavated expands and its original volume increases. The material swelling factor is defined as the swell factor (m^3) of the same material weight before and after blasting/digging as:

$$Swell\ factor\ = \frac{Material\ volume\ before\ digging}{Material\ volume\ after\ digging} \tag{12}$$

Material swell factor can variation between 0.6-0.9, depending on the nature of the material (stickiness, moisture content), fragmentation (shape, size and distribution of material), and others.

2.7.7 Production index

The production index (PI) is another indicator in the mining industry that functions to control daily operations. Rai (1992) termed it, as production efficiency in respect of the studies performed by him on shovel-truck and dragline operations in surface mines. The PI for each material rig can be understood as the ratio between the actual production and the potential production of the equipment during its operating period. Misra (2006) stated that production efficiency of any excavator has two components: machine operating efficiency and the job management efficiency. However, the PI did not indicate the reason for the low production. Mathematically, it can be expressed as (Mohammadi et al., 2015):

$$Production\ Index\ = \frac{Actual\ production}{Potential\ Production} \tag{13}$$

To calculate the production potential of the Central Mine Planning and Design CMPDI (2000) in Mohammadi et al., (2015) formulate it in Equation 14.

$$O = \frac{BC \times f \times TT \times A \times U \times 3600}{CT} \times m \tag{14}$$

where,

O = potential Production per periode (m^3)

BC = bucket capacities (m^3)

f = bucket fill factor

TT = time total (hour)

A = availability (minute)

U = utility

CT = cycle time (second)

m = the other factor such positioning, travelling, etc)

After obtaining the production potential value per period or the productivity of the equipment, the number of digging equipment and transportation equipment can be determined. Calculation of the required number of excavators/loaders is done by Equation 15 and 16 (Peurifoy et al., 1985):

$$Number\ of\ digger = \frac{ore\ tonnage\ /\ waste\ volume\ on\ sequence}{(daily\ digger\ productivity\ x\ working\ days)} \tag{15}$$

$$Number\ of\ hauler = \frac{ore\ tonnage\ /\ waste\ volume\ on\ sequence}{(daily\ hauler\ productivity\ x\ working\ days)} \tag{16}$$

2.8 Mine Investment Analysis

Mining investment analysis is a systematic step taken to evaluate the profit potential of a mining project (Sari, et al., 2018). Mining investment feasibility assessment is carried out by determining economic indicators (financial/economic model) such as net present value, internal rate of return (lRR), and payback period. This is because in the process, mining activities require very large capital. Good financial planning will make the possibility of loss smaller and the amount of profit and return on investment can be estimated (Gentry and O’Neil, 1984).

The NPV method defines the difference between the present value of all cash flows and the investment and gives realistic results. When compared to other evaluation methods, NPV is considered and preferred as a more realistic and reliable tool in project evaluation and thus mining investment decisions are largely associated with project NPV (Malli et al., 2015).

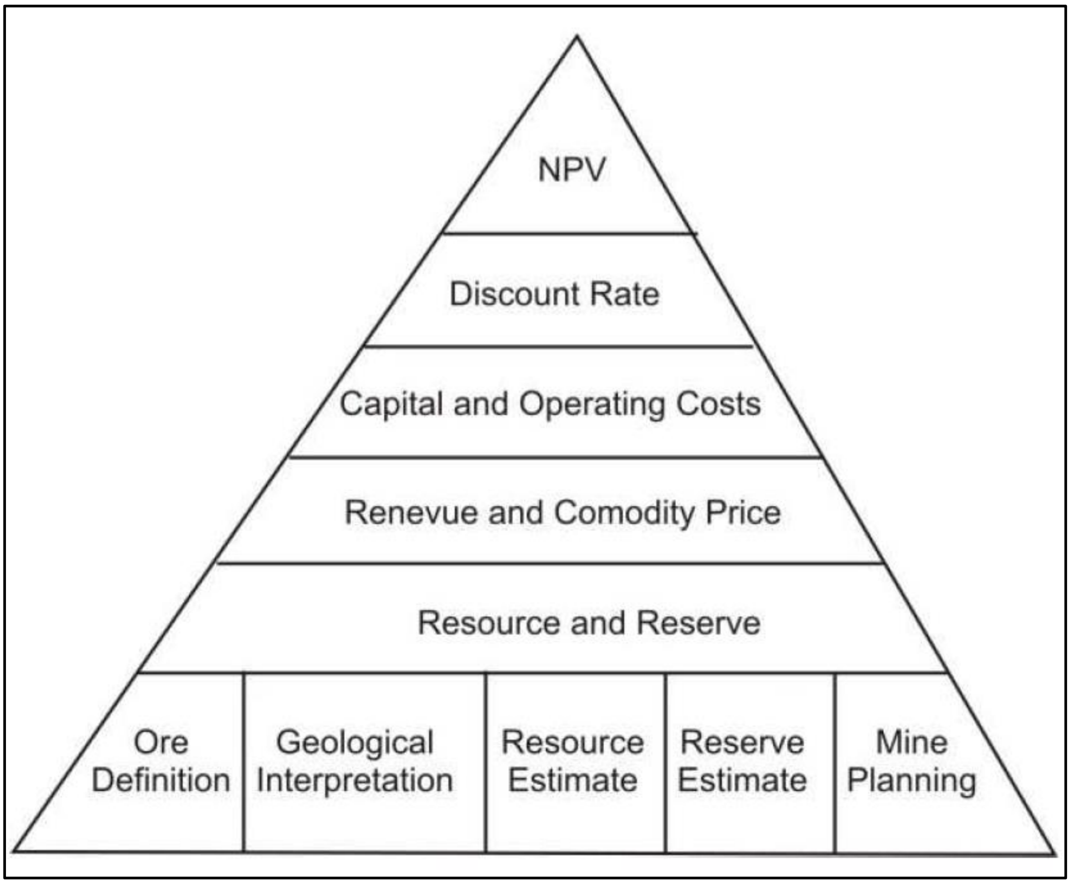

Figure 16 Mine value chain (Malli et al., 2015)

The construction of a financial model requires accurate estimates of revenues and costs. The estimation of revenues and costs includes a lot of uncertainty because the uncertainties affect the estimated values and they form the value chain. Therefore, inputs must be analyzed to optimize the overall mine process. Value chain optimization must be carried out well from the initial stage to the final process to identify high risk areas and eliminate their impact on profit maximization. Value chain evaluation is an interdisciplinary process and the interdisciplinary components of the value chain are geology, geomechanics, mining and metallurgical engineering. It relates to every stage from exploration through feasibility studies, through to grade control, mineral processing and marketing. A simple demonstration of the mine value chain process and the uncertainty nodes considered for NPV estimation is presented in Figure 16.

2.8.1 Net present value

The term present value (PV) simply represents an amount of money at the moment (r = 0) which is equivalent to some sequence of future cash flows discounted at a certain interest rate. In other words, this technique recognizes the time value of money and provides a current calculation of the amount equivalent in value to a series of future cash flows (Gentry and O'Neil, 1984).

Present value calculations are most often performed to determine the present value of an income-producing property, such as an existing mining operation. If future annual cash flows can be estimated, then by selecting an appropriate interest rate, the present value of the property can be calculated. This value should provide a reasonable estimate of the price at which the property can be bought or sold (Gentry and O'Neil, 1984).

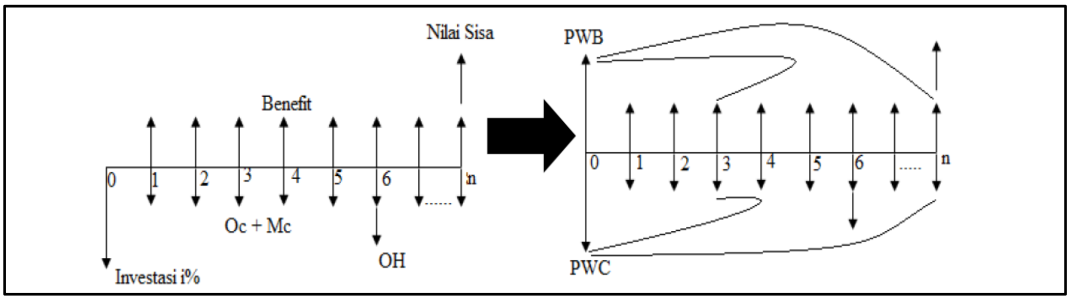

Figure 17 Graph of initial condition and present condition

Net present value is a method of calculating net value at present take into account the concept of equivalence (Sari et al., 2018). The present assumption explains that the initial time of calculation coincides with the time the evaluation was carried out, or in the 0 year period in calculating the investment cash flow (Zakri and Saldy, 2020). The NPV method is basically moving cash flows that spread throughout the investment period to the initial investment time (t = 0) or present conditions, of course by applying the concept of money equivalence. Cash flow consists of cash-out or cash-in. Cash flow for which only benefits are calculated is called present worth of benefit (PWB), whereas if only cash-out (cost) is taken into account it is called present worth of cost (PWC). Meanwhile, the NPV values were obtained from the PWB-PWC (Zakri and Saldy, 2020). NPV calculation can be done with Equation 17.

$$NPV = \sum_{t=0}^{n} \frac{At}{(1+k)^t} \quad (17)$$

where,

k = discount rate

At = cashflow at periode t

n = the last period in which cash flow is expected

2.8.2 Internal rate of return

The internal rate of return (IRR) is the interest rate/rate of return that can make the NPV of a project zero. IRR is used to find out how much interest is earned so that the NPV of incoming projects is the same as the NPV of outgoing projects. Thus, if the IRR value of a project has been determined, then the IRR value can be compared with interest rates elsewhere (bank interest rates). The greater the project IRR value obtained than the minimum IRR value, the more feasible the project is to run (Valent et al., 2018).

Calculation of the internal rate of return can be done with the following Formula 18 (Purnatiyo, 2016):

$$IRR = rr + \frac{NPV_{rr}}{NPV_{rr} - NPV_{rt}} x(rt - rr) \quad (18)$$

where,

rr = discount rate (r) is lower

rt = discount rate (r) is higher

NPV = net present value

The decision criteria of this method are (Zakri and Saldy, 2020):

1. NPV < 0, unprofitable investment.
2. NPV = 0, marginal investment.
3. NPV > 0, profitable investment.

2.8.3 Payback period

Payback period (PBP) is the initial cost recovery period. The faster the return, the alternative is more attractive compared to other alternatives. The advantages of the payback period method are that it is easy to use and calculate, it is useful for choosing which investment has the fastest recovery period, the capital recovery period can be used as a predictor of future uncertainties, and the fastest recovery period has less risk than the recovery period. relatively longer (Rachadian et al., 2013). The weakness of the payback period is to ignore changes in the value of money from time to time, ignore cash flows after the capital recovery period is achieved, ignore the residual value of the process and often trap the analyst if the cost of capital or credit interest is not taken into account in cash flows which causes the business to be illiquid (Rachadian et al., 2013).

The payback period can be calculated using Formula 19 (Rachadian et al., 2013):

$$Payback\ period = \mathrm{n} + \frac{a-b}{c-b}\ x\ 1\ year \tag{19}$$

where,

n = last year the amount of cash flow has not been able to cover the initial investment capital.

a = initial investment amount.

b = the cumulative amount of cash flows in the nth year

c = cumulative sum of cash flows in year n + 1

2.8.4 Sensitivity analysis

Sensitivity analysis is an analysis used to determine the effect of the output model related to the variation of parameters used. If a small change is made to the parameters entered, it results in a relatively large change in the output model, then the model is said to be sensitive to parameters. This analysis makes it possible to recognize which parameters have the most influence on the model. The possible consequences of these changes can be known and anticipated beforehand if a sensitivity analysis is carried out. Sensitivity analysis is a technique used to determine how the impact of differences in the values of the independent variables on certain dependent variables. This analysis calculates the rate of change in system output with respect to changes in parameters at system input (Alvarez, 2009).

One method that is often used in making sensitivity analysis is the sobol method. This method is a standard method of variance-based sensitivity analysis for evaluating complex systems and it has an accurate variance-based sensitivity index with a large sample size. However, this method cannot be applied to models with a large number of input parameters (Mai et al., 2016).

To find out how sensitive a decision is to changes in the factors that influence it, every decision in engineering economics should be accompanied by a sensitivity analysis. This analysis will provide an overview of the extent to which a decision will be strong enough to deal with changes in influencing factors. In the mining business, parameters that can make the mining business sensitive to losses include a decrease in the selling price of mining minerals, changes in the value of stripping ratios, increases in production costs such as equipment rental prices and increases in diesel prices, as well as a decrease in the value of the rupiah exchange rate against the dollar. considering that not a few of the needs for mining equipment use the dollar exchange rate (Valent et al., 2018).

Overall, the sensitivity analysis provided insights into the potential risks and uncertainties associated with the mining project, allowing for better decision-making and risk management. It highlighted the importance of considering the fluctuations in the price of nickel, operating expenditure, and capital expenditure when evaluating the feasibility and profitability of the project (Shafira et al., 2023).