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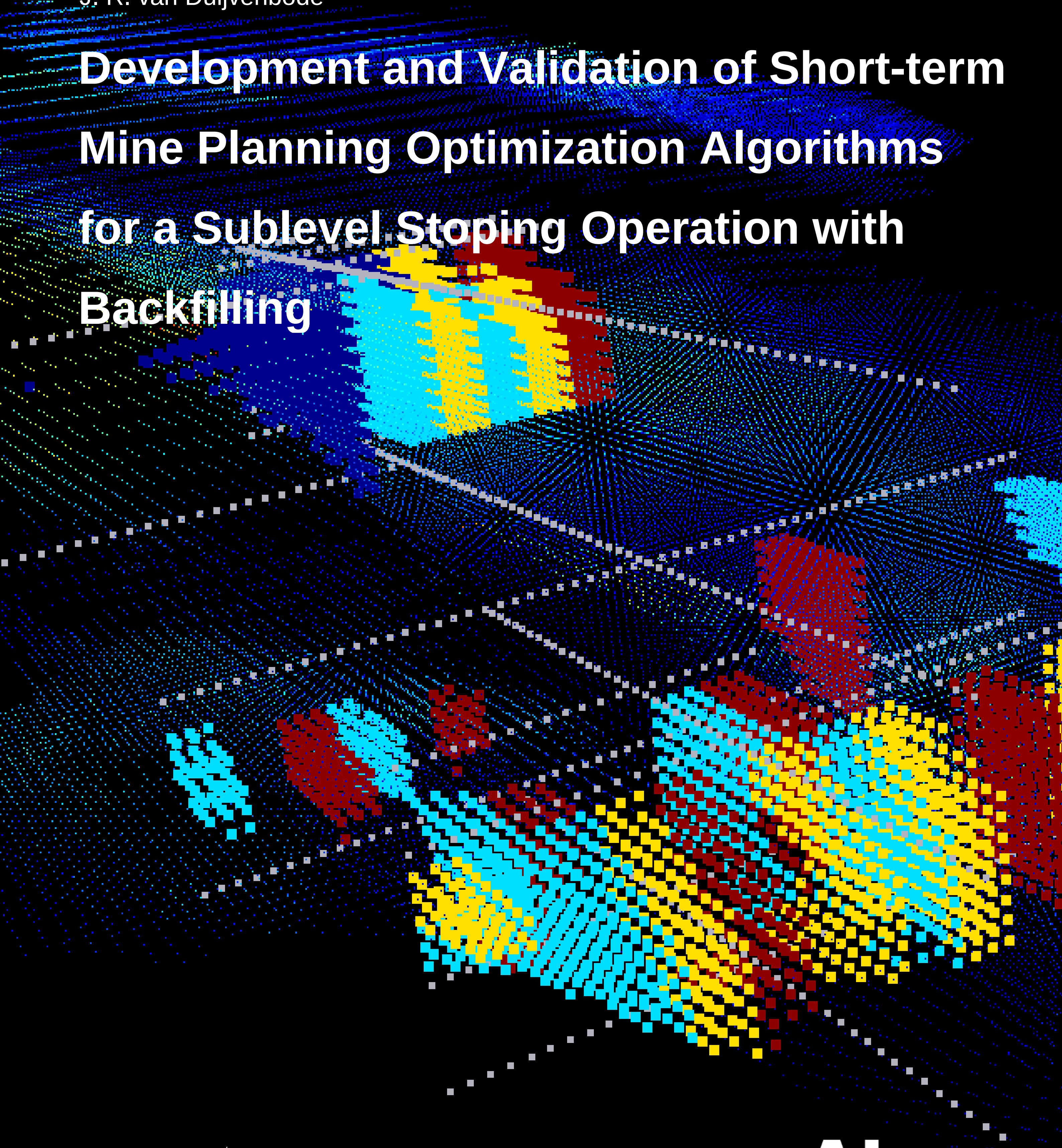
Development and Validation of Short-term Mine Planning Optimization Algorithms for a Sublevel Stoping Operation with Backfilling



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Development and Validation of Short-term Mine Planning Algorithms for a Sublevel Stoping Operation with Backfilling

By

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Abstract

Mining companies desire short-term mine planning optimization since this enables them to schedule major sublevel stoping mining activities like development, drilling, extraction and backfilling. If simultaneous effort is made to reduce the grade deviation resulting from all extracted ore in a certain period, this allows it to finetune the processing operations and meet production targets.

To control the short-term grade deviation, a new mixed integer linear programming model is developed which is able to consider production control constraints. For the scheduler, an objective function is developed which considers all to-be-mined ore and produces the best schedule to reduce grade deviation from combining ore of different locations. Limitations have been set on the availability of this ore. This is necessary as scheduled work must occur in the order of natural sequential transition from development, drilling, extraction and backfilling.

A copper zinc operation is used to show that the periodical grade deviation can be controlled with the model. Furthermore, validation is done to proof the functionality of control constraints. The scheduler proofs that it can create schedules for half-year scheduling horizons and that it can create a better-optimized schedule regarding grade deviation than a model without the grade deviation considerations. The obtained schedules can be used by a planning engineer for detailed shift scheduling.

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List of Abbreviations

%	Percentage
avr	Average
bf	Backfilling
BlastID	Blast Slice Identification
CAF	Cement Aggregate Fill
COS	Cut-Off Slot
Cu	Copper
d	Day
dev	Development
DevID	Development Identification
DH	Drillhole
dr	Drilling
EPS	Enhanced Production Scheduler
ex	Extraction
I-dev	I-Development
km	Kilometer
LHD	Load Haul Dump Machine
LOM	Life Of Mine
LP	Linear Programming
m	Meter
m^3	Cubic Meter
MILP	Mixed Integer Linear Programming
mm	Millimeter
MRE	Mineral Resource Engineering
MSO	Mineable Shape Optimiser
NPV	Net Present Value
RMR	Rock Mass Rating
RQD	Rock Quality Designation
StopeID	Stope Identification
UCS	Unconfined Compressive Strength
VCR	Vertical Crater Retreat
Vol	Volume
X-dev	X-Development
y	Year
Y-dev	Y-Development
Zn	Zinc

All parameters and abbreviations specifically used for the optimization model can be found in chapter 5.

1
Introduction

Production scheduling is challenging for mining companies. They often try to optimize their production by comparing a few scenarios and eventually choose what they believe is the optimum scenario. This gives little, if any, guarantee of optimality and might result in lower profits than possibly achievable. The properties of schedules should resemble many attributes, like a high equipment utilization, high quality products and high profits. Computerized mine planning optimization uses many scenarios, more complex constraints and attributes, and therefore can quickly generate optimal schedules. For open pit mines, several open pit optimization packages exist making computerized scheduling widely available. However, optimization packages are not widely available for underground mines. Among the short-term optimizers found in literature and capabilities of commercial software, no models are available that consider a new mining area in which the stope's development must be completed before the stope can progress through its natural stope life.

Long-term mining plans assume a constant stope grade, but this is not suitable for short-term scheduling. Throughout the stope life, there are different moments that ore is obtained and therefore it is important to look at these individual moments. Such a moment is after each blast in a development drift, if this is in ore or after a stope blast. Stope blasts throughout the stope life can become bigger, because there is more open space available. That means that at the early stages of a stope, less rock is obtained after a blast and send to the processing plant than in a later stage. Different areas in a designed stope also have different metal grades. As some ore is obtained in a later stage, it is important to determine the average grade in each blast. While doing this, it can be discovered that several parts of a stope have no metal at all and that this could be excluded from the short-term schedule, because it is only considered as waste. This could not be done by a long-term optimizer as it considers an average grade for the entire stope.

This research project focuses on the different moments that ore is obtained to minimize deviation of production targets of the processing plant and how this can be successfully implemented in a short-term schedule optimizer for planning of a sublevel stoping operation.

2
Goals and objectives

The first step before analysing the focus of this report is to narrow the extend and define the aim and objectives, research questions and the scope and limitations. Eventually this definition of the research context can be used to better understand the idea behind this research.

2.1 Aim and objectives

The aim of this research is defined as:

“Develop and validate a short-term mine planning optimization algorithm for a sublevel stoping operation with backfilling”

To achieve this result, the following objectives are defined:

- Analyse and identify existing short-term mine planning tools and optimization algorithms, regarding their motivation, implementation and limitations.
- Develop a short-term optimization algorithm including development, drilling, extraction and backfill operations, based on a sublevel stoping operation and a long-term schedule framework.
- Validate the developed short-term scheduler.

2.2 Research question

The above-mentioned research goal and objective brings up the following hypothesis why this research is done:

Hypothesis: ***“****The aid of computational created short-term schedules will benefit short-term planning engineers with minimizing target grade deviation for the processing plant and analysing multiple parameter scenario’s.”*

The main research question based on this hypothesis will be the following:

- Is it possible to develop a flexible short-term scheduler which can validate long-term planning schedules, considering short-term constraints, objectives and targets?

2.3 Thesis outline

This research project is build up in different parts. The report starts with an introduction to the thesis topic in Chapter 1. The current chapter introduces the research context by its aim, objectives and research questions.

Chapter 3 gives a literature review about scheduling and optimization for mine planning by analysing research done in the past and relevant literature for this research. Chapter 4 gives an overview of sublevel stoping. This is the underground mining method which is used as the base for this research and therefore a detailed understanding of this method is of importance for considering short-term details. Chapter 5 contains all information regarding the developed short-term optimizer. The optimization model is described in detail and important considerations are discussed.

Chapter 6 and 7 cover the results of the optimization model. The results are explained and several analyses regarding plausibility are carried out. Chapter 8 describes several cases which are used to validate the model.

Chapter 9, 10, 11 and 12 present a discussion, overall conclusion, recommendations and summary based on the performed research.

2.4 Research scope and limitations

The research is based on a sublevel stoping operation and therefore the considered parameters and constraints are based on this mining method. However, many constraints of the model can be used for multiple mining methods. The model was tested on a copper-zinc deposit and many input parameters were given and constrained by the long-term optimizer result. This output was provided by the MRE Institute.

Important topics that are excluded:

- Production activities in stopes like scaling, charging, blasting, ventilation, support etc.
- The development phase is not divided into multiple stages, commonly associated with development work
- Curing time of backfill
- Effort to minimize the deviation for the target throughput of a processing plant
- Recovery and tonnages are not considered; the model is entirely based on volume units

3

Scheduling and optimization review

Production scheduling can be defined as "specifying the sequence of blocks extraction from the mine to give the highest NPV, subject to variety of production, grade blending and geometric constraints" (Whittle, 1989).

The scheduling horizon over which this happens might differ and three main scheduling horizons can be categorized, long-term, medium-term and short-term. As defined by Osanloo (2008), long-term generally encompasses the whole life of mine (LOM) and can range between 20 - 30 years depending on the mine. Long-term production scheduling focusses on the final and ultimate economic design and shape of the open pit or underground mine. It delineates the economic ore body and evaluates the economic potential of a mineral deposit. The long-term schedule is divided into several smaller time periods between one and five years. A schedule horizon of one to five years is also the base for the medium-term scheduler and divides this time span into one to six-month periods for even more detailed scheduling. Medium-term schedules are more focussed on the specific design of parts of the mine where ore is extracted. Short-term schedules are based on the period of the medium-term schedules and contain even more detailed scheduling. The typical scheduling horizon is between a month and one year. The short-term schedule horizon is broken down into one-day to one-month periods. The schedulers and plans are developed subject to physical, geological, operating, legal, and other policy constraints (Osanloo, Gholamnejad, & Karimi, 2008). This report will focus on short-term planning done by optimizing a sublevel stoping mine operation.

3.1 Optimization focus areas

Mine optimization started in 1965 by Lerch and Grossman, who had a great success in the mine schedule optimization industry. Lerch and Grossman introduced the Lerch-Grossman algorithm for long-term open pit mine design planning. This method was used to find the ultimate pit limit by a graph theory based algorithm. Much recent research in open pit planning is still based on these ideas. The fact that mine optimization started in surface mines is because in the past much more open pits were exploited than underground mines and open pit optimization is significant easier than underground mine optimization due to the complexity of operations (Alford, Brazil, & Lee, 2007).

In the past few years focus has been shifted towards underground mining, because increasingly amount of shallow deposits are exploited and more underground mines have opened. This resulted in progress for optimisation of the design of underground mines (Nehring & Topal, 2007).

The optimization goals for surface mining and underground mining are also slightly different. The design area in surface mines is big and there is often great freedom to optimize pit limits and pushbacks. The focus for surface mines is maximizing the exploitation of the orebody and improving shovel-truck efficiency. The emphasis for underground operations is mainly on maximizing the exploitation of each face and maximizing sequential working on faces to improve productivity (Song, Rinne, & van Wageningen, 2013). The problem is that underground mines often have complicated structures and are restricted with complicated design and operational constraints. Past research has therefore focussed on three main areas. The first area is stope boundary optimization, which is mainly incorporated in the long-term optimization framework. The idea is to maximize the size and economic viability of stopes to generate the highest NPV during the mine life. The second area is production schedule optimization from defined stoping boundaries. Hereby, the stopes from the long-term framework are used and considered for planning in the short-term interval. This is also the focus area in this research. Stope sequencing based on geotechnical considerations is the last area and can be combined with the previous focus areas to resemble more operational conditions. Often these geotechnical considerations are already implemented in short-term underground mining, such as primary and secondary stopes (Alford, Brazil, & Lee, 2007; Nehring & Topal, 2007).

3.2 Optimization techniques in mine planning and scheduling

Up to now, few optimization tools for underground mines are available in comparison to optimization tools for open pits. Therefore, stope optimization and (short-term) scheduling for underground mines is still often done manually. This manual task is beyond the reasonable expectations of a planning engineer (Nehring & Topal, 2007). Various techniques and models are now used and have been developed in the last few years. In mining optimization, the most relevant techniques are linear programming, mixed integer programming, heuristic methods and dynamic or goal programming. Optimization is mathematically represented by an objective function (eq 3.1) which tries to minimize or maximize a function which is subject to several constraints (eq 3.2).

$$Optimizing\ of\ (\max or \min):\quad Z\ = f(X_1, X_2, \ldots, X_n) \tag{3.1}$$

$$Subject\ to:\quad Y_i\ \{>,=,<\}\ g_i\ (X_1, X_2, \ldots, X_n),\qquad i = 1,2,\ldots,m \tag{3.2}$$

Z is the optimal goal, *X* is the decision variable or parameter and *Y* is the constraint (Song, Rinne, & van Wageningen, 2013).

3.2.1 (Mixed integer) linear programming

Linear programming (LP) is a widely used mathematical technique in mine planning and scheduling, but it is not commonly used in underground mines, due to the complicated operations. A LP model has a mathematical structure as described above by equation 3.1 and 3.2, and is adjusted with only linear equations, see equation 3.3 and 3.4 (Song *et al*, 2013; Pochet & Wolsey, 2006).

$$\max or \min \mathrm{Z} = c_1x_1 + c_2x_2 + \ldots + c_nx_n \tag{3.3}$$

$$Subject\ to:\quad \begin{array}{ccccccc} a_{11}x_1 & + & a_{11}x_2 & + \ldots + & a_{1n}x_n & \leq & b_1 \\ a_{21}x_1 & + & a_{22}x_2 & + \ldots + & a_{2n}x_n & \leq & b_2 \\ \vdots & & \vdots & & \vdots & & \vdots \\ a_{m1}x_1 & + & a_{m1}x_2 & + \ldots + & a_{mn}x_n & \leq & b_m \end{array} \tag{3.4}$$

It contains a set of non-negativity restrictions: $x_1, x_2, \ldots, x_n \geq 0$. *N* and *m* define the number of variables and constraints, respectively. *Z* is the objective function value which could be maximized for profit or minimized for costs and x_j are decision variables. The values of x_j are determined by the model and a_{ij} and c_j are constants whose values depend on the LP problem and b_i is the right-hand side constant value. Constraints are for example: extraction sequence, mining equipment and milling capacity, mill feed grades and others. In mine production optimizers, the optimum result is often defined as maximize profits, tonnage or a specific blended ore grade for a specific period (Song, Rinne, & van Wageningen, 2013).

Mixed integer linear programming (MILP) is a combination of integer programming and linear programming. The mathematical form is almost the same, except that MILP allows certain variables which only take on integer values. The mathematical problems become larger and more complex and more constraints are applicable, therefore, MILP is more interesting to use. The solution time of MILP models depends mainly on the number of variables and constraints used in the model and can increase exponentially with the number of variables (Nehring, Topal, & Little, 2010).

In 1995, L.P. Trout formulated a MILP production scheduling model over multiple periods for a sublevel stoping copper ore operation at Mount Isa (Trout, 1995). The objective was to maximize NPV by scheduling the production of 55 stopes over a two-year period at four weekly time intervals. The solution process was interrupted before optimality but it still improved the NPV by 23%. Due to the long process time and lack of important features, the schedule was not implemented at the mine. An important researcher, who did a lot of research similar to this research for short-term scheduling with MILP, is M. Nehring. Nehring and his co-authors continued in several papers on the from Trout's model. They introduced and updated a MILP model from a small conceptual sublevel stoping operation towards a bigger operation. In the first paper (Nehring & Topal, 2007), a MILP model for a small conceptual sublevel stoping operation was presented, with a new constraint which does not allow multiple stopes adjacent to each other, to be exposed open. This was done to give a realistic representation of the operational constraints of a sublevel stoping operation. By comparing the results of a MILP production schedule and a manually generated schedule it is shown that the MILP benefits are significant. In the next paper (Nehring & Topal, 2009), a new MILP model focussed on short-term production scheduling and machine allocation, was created. The objective was to minimize deviation to production targets for each shift across the scheduling horizon. The model scheduled five LHD's and three trucks over 120 shifts by defining all ore-movements and constraining the model with operational constraints such as ore reserves, haulage shaft and machine capacity, but also precedence constraints. The next paper (Nehring, Topal, & Little, 2010) continues from the previous work, but now the model is implemented on a much larger scale to a more realistic operating scenario (50 stopes). This model includes a more realistic grade control to better meet the monthly production targets and a better constant grade plant throughput. It is shown and proved that the new model was applicable to larger stope data sets. In mine scheduling the short-term schedule is normally based on the results of the medium- or long-term schedule and the solution is achieved segregated. Nehring *et al* (2012) made a new model which integrates the short- and medium-term production schedule that considers the interaction between medium- and short-term schedules. It was possible to achieve a better NPV and smoother mill feed grades for the processing plant (Nehring, Topal, Kizil, & Knights, 2012).

3.2.2 Heuristics

A heuristic is an experience-based method to solve problems which have a large size and when the goal is to find a (near-)optimal solution quickly. The objective is to explore the search space and find a solution effectively and efficiently such that a reasonable accurate solution is obtained. It neglects whether the solution can be proven to be optimal. Heuristics are using iterations which depend on the previous step and the model learns which paths to follow for the solution and disregards other paths. O'Sullivan & Newman (2015) used a heuristic approach on a data set from the Lisheen mine, Ireland, to develop an optimization-based decomposition heuristic extraction schedule. A heuristic approach is often used to determine preliminary process designs and in mining this is can be done with genetic algorithms, ant colony optimization and neural networks (O'Sullivan & Newman, 2015).

A genetic algorithm is a directed search algorithm. The conceptual idea of genetic algorithms is based on the mechanics of biological evolution. The algorithm represents complex objects with a vector of simple components, the biological reference used here are chromosomes. Through the generation of random populations, the multidimensional problem can be optimized. It is often used in optimization for geological models of mineral deposits and mine planning and scheduling (Song, Rinne, & van Wageningen, 2013).

3.2.3 Dynamic programming

Dynamic programming is used to optimally solve complex problems by breaking it into smaller subsequent subproblems. The optimization of each stage depends on the previous stage and thus also affects the following stage. The technique is frequently used in mining (Song, Rinne, & van Wageningen, 2013). For example, Dowd and Elvan (1987) used dynamic programming for a scheduling and grade control problem in sublevel open stoping. The algorithm determines the optimal sequence where the sublevel stoping slices should be mined according to their availability and grade (Dowd & Elvan, 1987). It can also be used for stope layout optimization of a block caving operation. Thereby it maximizes the design with respect to the mine limits (Ataee-pour, 2005).

3.3 Existing short-term underground mine planning optimization software

The advantage of underground mine planning optimization is that the planning process is less time consuming and it can guarantee optimal results, which are hardly obtained by manual processes. For optimized scheduling, it is desired that several scheduling phases integrate with each other and that schedules can be adjusted at any time to comply with changing operational conditions. This real-time feedback on schedules can easily be done when schedules are made by mathematical modelling techniques (Nehring & Topal, 2009). In industry, several short-term mine planning tools are available which focussed on the areas mentioned before in Chapter 3.1, namely, production schedule and stope boundary optimization. A list of companies and their software can be seen in Table 3-1 (Matthäus, 2015).

Table 3-1: Overview of underground mine scheduling software capable of optimizing schedules or stope boundaries, after (Matthäus, 2015)

Company	Software	Schedule/planning optimizer	Stope optimizer
GEOVIA	MineSched	X	
	Minex		X
Hexagon Mining	MineSight	X	
Runge Pincock Minarco	XACT	X	
	XECUTE	X	
MineMax	Minemax Scheduler	X	
	Minemax Planner	X	X
	IGantt	X	
Maptek	Vulcan Scheduler	X	X
Mine RP	Mine 2-4D	X	
Datamine	Underground Planning Software		X
	Mineable Shape Optimizer (MSO)		X
	Enhanced Production Scheduler (EPS)	X	X
	Mine5D		X
Deswik	Deswik	X	
ABB	Mine Scape	X	

3.3.1 Application of Datamine

A leading software in mine planning is Datamine. The software (NPV Scheduler, Studio OP) can manage and optimize mining operations for both surface and underground mines and for long-, medium- and short-term planning. For long-term open pit planning it can optimize the strategic pit design through short-term material allocation and operational equipment scheduling. NPV scheduler can be used for pit optimization, pushback generation, cuf-off grade optimization, scheduling, haulage optimization and stockpile management. This software is more focused on long-term and uses algorithms which are based on Lerch-Grossman principles. Studio OP is used for medium- to short-term planning of open pit mines. The program uses optimization to create schedules which will ensure desired financial outcomes or product blend specifications. For underground planning, Datamine has software packages like, Mineable Shape Optimiser (MSO) and Enhanced Production Scheduler (EPS). MSO is used for stope design optimization and can be used with several mining methods, including sublevel stoping. It can maximize the potential mineable ore tonnage by creating stopes which respect the given orebody geometry and design constraints. The solutions from all Datamine software can be linked to EPS. This software is a comprehensive Gantt chart scheduler specifically build for the mining industry. It can optimise the weighted average grade resulting from multiple ore feeds, equipment and employment scheduling (Datamine, 2017).

3.3.2 Application of XACT

XACT is a short-term mine scheduling software from Runge Pincock Minarco (RPM Global). This software is for short-term scheduling and can be used across all mining methods. The program can generate working schedules for the current and next shift, but also for future and beyond. The variables of the schedule can easily be adjusted and therefore it is easy to run multiple scenarios. The software uses linear programming to perform detailed blending of multiple ore feed to achieve specific grade targets (RPM Global, 2017).

3.4 Summary

Among the short-term optimizers found in literature and capabilities of commercial software, no models are available that consider a new mining area in which the stope's development must be completed before the stope can progress through its natural stope life. The implemented scheduler should represent a schedule where decisions are based on the objectives from the short-term scheduling horizon. In this thesis, this feature will be implemented in an optimizer and mixed integer linear programming would be a suitable optimization method to use.

4

Sublevel stoping

This chapter provides background information about sublevel stoping to better understand the considerations and assumptions made in the short-term optimizer. The paragraphs in *italic* at the end of a subchapter contain assumptions or input values for the short-term optimizer described in this report.

4.1 General overview of sublevel stoping operations

Stope mining can be divided into three separate classes, based on the characteristics of the ore and ground conditions of the ore zone. The three classes are unsupported (e.g. sublevel stoping), supported (e.g. cut and fill stoping) and caving (e.g. sublevel caving). Sublevel stoping is characterized as an unsupported method since the roof is sufficient self-supporting due to the favourable ore characteristics. It is usually applied to a relatively steep tabular and regular in shape dipping competent ore body, surrounded by competent host rock. The dip of the footwall should be sufficient (exceeding the angle of repose) to allow broken ore to freely gravitate to the drawpoints for collection (Hartman & Mutmansky, 1987). The focus in this report is on sublevel stoping, an underground mining method.

Stope mining can be described as the removal of an orebody by leaving behind an open space (stope). Several stopes can be separated by pillars. Stoping is used when the country rock is sufficiently strong to not collapse into the stope for limited time. The stope can, depending on rock conditions and resource value, be either left open which means that the pillar is left in place or the stope can be backfilled. When the stope is backfilled, it is possible to mine out the pillar or adjacent stopes, the secondary stope. Pillar recovery is not possible when the stability of surrounding rock masses is not sufficient or cannot be guaranteed (Hustrulid, 1998).

In sublevel stoping the orebody is vertically divided into levels. Sublevel stoping (also referred to as open stoping, longhole stoping or blasthole stoping) takes place at these sublevels, between two levels and the rib and sill pillars. A general layout of sublevel open stoping can be seen in Figure 4-1 (Tunnelbuilder ltd (Atlas Copco Rock Drills AB), 2007).

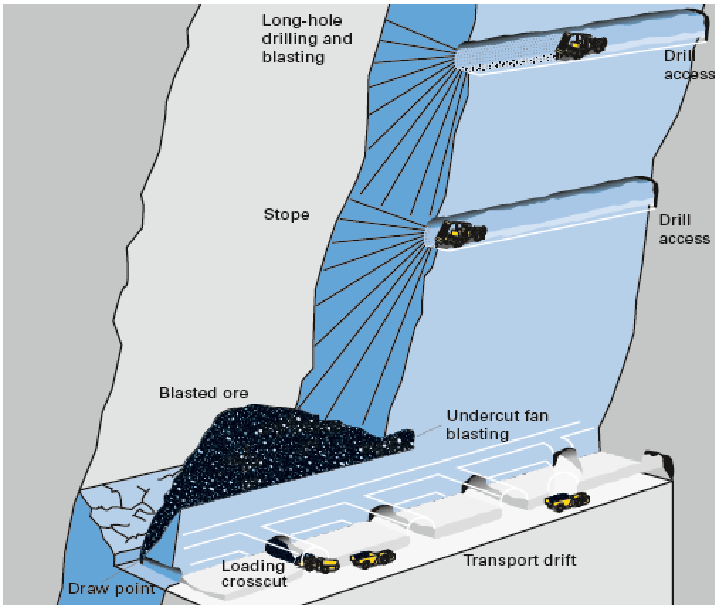


Figure 4-1: Sublevel open stoping layout, after (Tunnelbuilder ltd (Atlas Copco Rock Drills AB), 2007)

4.2 Stope design and consideration

Typical stope design and dimensions depend on various parameters, including the characteristics of the orebody and the stability of the hanging wall. While the mine is in operation, the new stope designs are also influenced by existing development and surrounded filled stopes. The shape of the stope is often adapted to the spatial distribution of the ore. Stope heights typically do not exceed 120 m and can be subdivided in sublevels with a vertical spacing between 20 – 40 m. Stopes have a typical width between 5 – 40 m and the length of a stope varies, depending on the ore body, but generally does also not exceed 40 m (Hamrin, 1980).

While considering the location of stopes, it is important to know the location of existing or planned development. The primary and secondary development of a mine is based on the long-term design framework of the mine and is used to get access to areas of the orebody. This development is required to access multiple stopes. Sublevels might serve as the platform for all tertiary development (drifts, crosscuts and drawpoints) to reach the stopes. This is generally designed for individual stopes and therefore the development for one stope will likely influence the effective mining of another, hence in the schedule and profitability. For this reason, it should be ensured not to lose another stope in the development process of an earlier stope. (Sloane, 2010).

4.2.1 Surrounding filled stopes

When the mine is in full production and different parts of the orebody have been backfilled, it is important to know where the surrounding fillmasses around a stope are located. Often stopes are not fully filled, which may create problems. When the filled volumes differ from calculated volumes, stope over-or-underbreaking can lead to fillmasses being differently positioned than initially thought, see also Chapter 4.7. For designing purposes, it is therefore important to know the fill level of the surrounding stopes and the type of backfill. Curing times and final strengths vary with different types of backfill. For example, the development of paste fill strength increases with time and thus could influence stope design as in later stage extraction might become possible of an adjacent stope (Emad, Mitri, & Kelly, 2015). Another side effect is that the height and type of the backfill to be exposed by mining a stope determines how close and what type of explosive is needed for firing close to filled stopes. When a pillar between stopes is required it also means a loss of mineable ore (Sloane, 2010).

4.2.2 Rock mechanics

Rock mass classification systems and mechanical models provide the key information required for stope design and modelling. The rock mass can be characterized by analysing core logs and based on this, the maximum unsupported spans, support, and reinforcement requirements and estimated rock mass strength can be determined. Rock mass classifications are important to understand the rock mass behaviour and to prevent potential failures (Villaescusa, 2014).

The most common and well-known rock mass characterization systems are the Q (Barton) and RMR (Bieniawski) systems (as cited in Villaescusa, 2014). The RMR and Q system are based on several parameters such as unconfined compressive strength (UCS), rock quality designation (RQD) and number of joints, joint spacing, etc. These parameters correspond to classes and determine the quality of the rock. This can be used as input data for geotechnical modeling (Villaescusa, 2014).

The above described steps and analysis take place during the exploration for new deposits or mining areas before any short-term mining schedule can be made. A long-term schedule and stope optimizer takes several aspects of rock mechanics into account. A stope designer, for example considers the stability of the entire mine and set several parameters as the maximum stope size and drift sizes. Therefore, no further limitations concerning rock mechanics are considered for the short-term optimizer (Villaescusa, 2014).

4.2.3 Design tradeoffs

During the design of stopes, it is tried to get stopes as big as possible, because the bigger the stope dimensions, the higher mining efficiency can be obtained. It is not always possible to take the biggest size due to the stability of the rock which limits the size of the stopes and pillars (Hamrin, 1980).

The number of sublevels in a stope is determined by the stope size, existing development and drillhole length. It is tried to find an optimum between the costs for long drill holes or costs for developing an additional sublevel. A large distance between two sublevels is not always favourable, because the deviation in holes becomes larger the longer a drillhole is. The deviation influences the firing accuracy and results in more dilution or non-mined ore (Lawrence, 1998).

During design, there is a distinction made between primary, secondary and maybe tertiary stopes. This is done because the biggest (geotechnical stable) unsupported open stope is not big enough for complete extraction of an orebody. To extract the secondary and tertiary stopes, the primary stopes must be backfilled with a sufficient strong fill (Lawrence, 1998).

The stopes in this report have a size of 40 m height, 12 m width and either 12, 24 or 36 m length. The long-term optimizer determined these stope dimensions, because it optimized the stope size for maximizing NPV. The stopes consist of one sublevel and therefore only development above and below the stopes is required. There is not yet any existing development or surrounded filled stopes for the first period and therefore the very first activity should be development of the main drifts towards the stopes. Beside the limitation of only four open faces, there are no further geotechnical details considered, which would constraint the model.

4.3 Development phase

After construction of the main access to the orebody or a cluster of stopes, the haulage level is developed where the drawpoints will be located. Following, creation of access towards a stope is the first phase for the production cycle of the stope.

4.3.1 External (stope) development

External development is development located outside a stope and is required to access the internal stope development. Development is a sequential process for making roads and drifts. The development process consists of drilling, charging, blasting, ventilation, mucking, scaling, supporting and surveying. Since this is a labour-intensive job, it is often a high cost item on the balance sheet of mining companies. For this reason, development must be kept at a minimum level, but it cannot neglect the safety considerations or negatively impact effective mining. When creating access to a stope as much as possible existing development is used, before creating any new development. In sublevel stoping, not all development is a loss since most of the development for stopes takes place within the stope. This rock can therefore be sent to the processing plant as ore and it is not lost. It is therefore important to minimize the development in waste (Tunnelbuilder ltd (Atlas Copco Rock Drills AB), 2007).

4.3.2 Internal (stope) development

When access to the stope is reached, internal stope development can occur. The idea of internal development for the crosscut is to construct the loading point for the blasted rock at the bottom of the stope. At the top of the stope internal development of the top level drift must take place in order to drill and blast the stope. Before any blasting in a stope can occur, a cut-off long hole winze or raise needs to be in place. This cut-off long hole winze or raise opens the whole vertical length of the stope including all the sublevels over which the stope stretches. The difference between a winze and raise is that in a winze the opening is driven downwards from one level to another lower level and in a raise the opening is driven upwards from one level to a higher level (Hamrin, 1980). In the case of a winze the hole is blasted in approximately six meter advances with a raise, it is often raise bored. The winze or raise can be used as initial free open area in which the cut-off slot (COS) can be fired, which afterwards creates the open area for the rest of the stope to enable rock to be blasted into during next blasts. The raise or winze normally stretches over the length of the entire stope until six to eight meters above the lower level. These last meters will be simultaneously blasted with the first blast of the stope. Access constraints, orebody width and the fill type of adjacent stopes determine the location of a COS (Villaescusa, 2014). A typical stope development layout which is used in the short-term scheduling optimizer is shown in Figure 4-2.

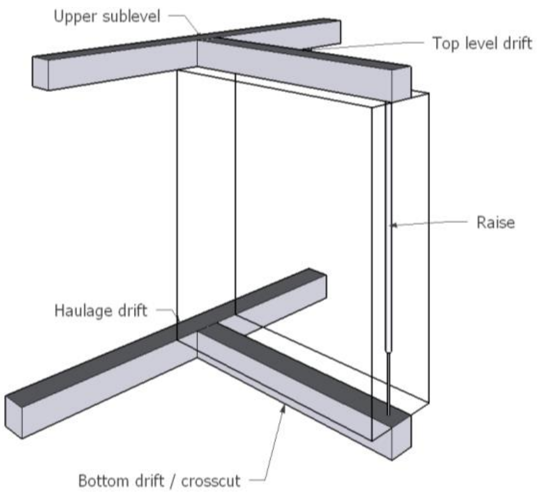


Figure 4-2: A general stope layout with all development items.

The primary development in the model consists of X-direction-development, the secondary development of Y-direction-development and the tertiary development is called internal development (I-dev). The required tertiary development for stopes is developed from the sublevels which serve as a platform. The internal stope development in the short-term scheduler consists of the development of a top level drift for production drilling and a crosscut at the lower level for extracting the ore. At the end of the top level drift is a raise bored until eight meters above the lower level. The ore received from the drifts is considered in the model during the development periods. The ore from the raise is considered in the period in which the first extraction takes place, because this rock remains on the lower level until rock from the first blast is extracted.

4.4 Production drilling and extraction phase

After development work of the stope has been finished, production drilling, followed by extraction are the next phases. Drilling of a stope is done in several drilling rings. After the entire stope has been drilled, charging of one or multiple rings with explosives take place and is followed by blasting. After blasting, the broken ore is extracted to open the void area in the stope again. The charging, blasting and extraction process continues until the full stope is extracted.

4.4.1 Drilling phase and characteristics

Drilling of a stope is one of the most important aspects of sublevel open stope mining and scheduling with the objective to ensure that all ore within the stope boundary can be taken out by blasting. The coverage of waste rock in a blast should be avoided to reduce dilution. Drilling is normally the most time-consuming phase and one stope requires easily several km worth of drilling associated with its high costs (Villaescusa, 2014).

Stopes are typically drilled using a combination of upholes and downholes. Upholes are often only used to drill above the highest sublevel. All other holes are drilled downhole since it is possible to use bigger drill rigs and apply higher drilling forces. Different drill rigs are available for different drilling requirements. Upholes are normally drilled with 89 mm bits. When the holes are bigger it is difficult to load the ANFO since it can fall out. Downholes can be drilled with bigger drill bits. Often 102 mm or 140 mm diameter bits are used (Sloane, 2010). The overall objective of drilling is to drill the least amount of holes as possible but still covering all ore and maintaining the required fragmentation. This is needed for easy mucking of the stope. A smaller drill hole size is better for fragmentation, because more drill holes are needed and more explosive is present. However, this is also more expensive and time consuming. Another drawback of large diameter holes is that it is difficult to follow the designed stope contours, because the fragmentation is bigger. Therefore, a balance has to be found for every stope by optimizing the drilling diameter, amount of meters and required fragmentation (Villaescusa, 2014).

Stope drilling often starts at the highest sublevel with 89 mm upholes used to reach all ore above the top sublevel. From the same level 102 mm or 140 mm downholes are used to drill to the next sublevel below, this is commonly done until the last sublevel. Drilling is performed from the higher levels towards the lower to reduce drilling fluids filling already drilled holes. The bottom of a stope is drilled and blasted in a trough shape, to encourage ore flow at the drawpoints and is called the trough undercut. There are two known approaches which are done in mines in order to establish this. The first approach is to drill downholes from the sublevel above the drawpoint level and from the trough undercut level upholes to shape the stope bottom. With this approach the drill holes intersect. The drawback of this approach is again uphole drilling and charging. Overall this remains an unsafe and difficult procedure. In the second and safer approach, there is not yet a trough undercut level. From the sublevel above the planned drawpoint level, downholes are drilled with additional subdrilling under the planned floor. Subdrilling ensures that the floor for the drawpoint level is better shaped and that the ore can easily be mucked. Since there is no additional free face in the drawpoint drift, the drillhole spacing is smaller to ensure better fragmentation (Sloane, 2010). The decision for which method is based on the sublevels and already existing development.

Drilling is done in rings with a distance between 2 and 4 m apart from each other, this distance is called burden. The drilled holes reach until the stope boundary and the distance from each other at the boundary is the toe spacing (between 3 and 6 m). Toe spacing values are typically 1.5 times greater than the burden values to ensure rock breakage toward a free face, rather than shearing across adjacent holes. Optimized values for the burden and the related toe spacing are critical for the fragmentation, blast damage, and drilling cost. Burden and spacing are a function of the drillhole diameter and the required fragmentation. Since the burden and spacing are depending on the drillhole diameter this is considered the most important blast design parameters in sublevel stoping. An insufficient burden size produces excessive muckpile throw and is thus inefficient. On the other hand, an excessive burden results in coarse fragmentation and tighter muck. Often standard distances are used to reduce any deviations. The only deviation which could occur while planning a big blast is when rings on different sublevels are lined up or when the drill pattern needs to fit into the stope boundaries (Villaescusa, 2014).

For drilling the COS blast holes, a “dice-five” (diamond) pattern is used and this has much more holes than any other blast ring. This is due to the small void area and thus additional holes are required to ensure that the cut-off is effectively blasted (Villaescusa, 2014).

All production drilling is done from the top level until the border with the bottom level. The rings have a burden of four meter and one ring consists of 14 drillholes. It is assumed that one drill hole has an average length of 20 m (half the stope height), which results in 280 m of drilling per ring. A schematic layout of a drill ring assumed in the model can be seen left in Figure 4-3. Figure 4-3 *right, shows the plan view of a 36 m long stope with the drill rings.*

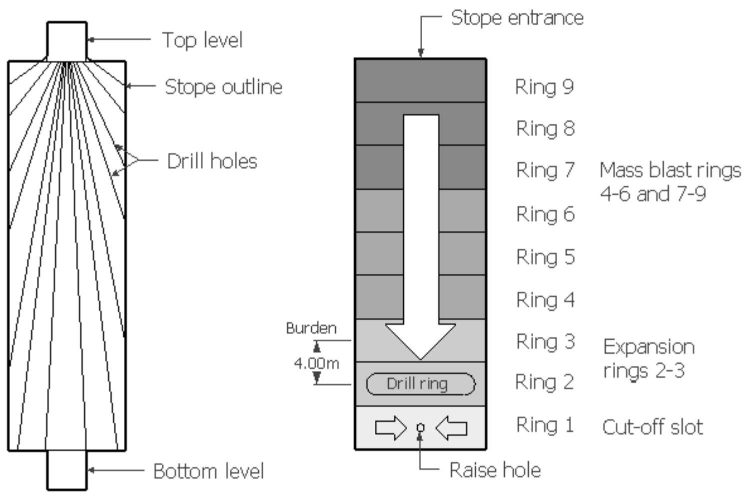


Figure 4-3: Schematic layouts of the front view of the ring blast design of one drill ring (left) and a plan view of the blasting sequence of a stope (right)

4.4.2 Blasting

Blasting is the phase between drilling and extraction of ore. The main goal of this phase and function of explosives are to deliver high energy forces to the orebody what results in fragmentized ore which can be extracted.

The firing sequence is the sequence in which the stope will be blasted, extracted and then blasted again. Three different blasts can be classified during the stope life, which follow up on each other. The first blast of the stope is the cut-off raise or winze, which is expanded into a COS. The COS is enlarged by firing ore into the open area of the raise or winze. After the COS has been taken out and the necessary open space for the stope has been established, the other rings can be taken out. The second blast type is an expansion ring blast. It is used to enlarge the open area to allow bigger mass blasts taking place. An expansion blast is often not bigger than three rings located on the same sublevel, because of the limited open space volume. The last type of blast is the mass blast. Mass blast rings can be blasted once 30 % of the overall stope design is opened.

Mass blasts vary in size, but it must be ensured that the broken ore does not “freeze” the stope due to swelling and that the ore cannot be extracted. Mass blasts progresses along the strike of the orebody retreating from the end of the orebody towards the access point of the ore drive (Villaescusa, 2014).

The blasting pattern used in the model can be seen in Figure 4-3 *on the right side. First, the cut-off slot rock is blasted towards the raise hole. An expansion blast of ring two and three follows which is blasted in the direction of the COS. Eventually two mass blasts take place consisting of three rings each. Regardless the size of the stope the blasting pattern will always be the same and consists of the same four blasts indicated with a blast slice. In the case, a stope is shorter than 36 m, then the last blast slices will have no grade and will not be blasted.*

4.4.3 Extraction phase

Stope extraction is done from the drawpoint level and takes place after blasting and ventilation. The mucked ore is brought to ore passes that are present on the haulage level. Often only LHD’s are used for loading the ore and hauling it to the ore pass. When an orepass is far away from the drawpoint it is more efficient to use truck transport, however, double handling of rock is necessary. The distance from the drawpoint to the orepass also influences the efficiency of the extraction and thus time to extract the stope. Multiple drawpoints increase the extraction efficiency, but require more development (Lawrence, 1998).

Stopes are often loaded by remote controlled LHD’s to protect personnel driving in the stope, because of the danger of falling rocks. The operator of the LHD is not allowed to drive into an open stope. During remote controlling, the operator is positioned at a safe distance. Normally the basement contact zone of the stope has a small footprint and therefore stope entry can be reduced to a minimum. Nowadays many mines operate their LHD’s remote to ensure safe work conditions for the operator and improve efficiency (Villaescusa, 2014).

For the extraction phase, the model only considers the maximum extraction capacity, not specified by a number of machines. The model does not consider any ore passes, driving distance or limitations due to multiple machines on the same level. This is done to simplify the model and not necessary to schedule the moment that ore is extracted.

4.5 Backfill phase and considerations

When the complete stope is empty, the last phase, backfilling can take place. A backfilled stope reduces the open volume underground and improves the geotechnical conditions to enable extraction of secondary or even tertiary stopes when needed. Also, other activities like ventilation and optechs take place when the stope is extracted and before backfilling, but they are not important for the short-term scheduler.

After a stope has been mined out, it is filled to reduce the amount of open volume underground and to prevent caving. Often waste rock and tailings can be used, what helps to reduce the amount of waste disposal. Filled stopes provide alleviation and redistribution of stresses in and around stopes and thus works as superficial, local, and global support (Brady and Brown, 2004 as citied in Villaescusa, 2014). A mine can use different types of backfill for primary and secondary stopes. The preferred type of backfill for primary stopes is cemented paste fill (or CAF, cement aggregate fill), because these stopes need to act as pillars. Cemented paste fill can obtain higher strength compared with other fill types (cemented hydraulic fill) although it is associated with higher costs. Another advantage is that drainage of the backfilled stopes is easier, because the water content is lower than in cemented hydraulic fill. Combinations of rock and paste or hydraulic fill also occur. Rock fill material is hauled to stopes by LHD and all other fill material by pipes from the backfill plant until the stope (Villaescusa, 2014).

After the extraction of all ore of a stope and before filling can start, it is sealed off with a bulkhead. A bulkhead is a wall built to keep the fill material inside the stope. The design of the seal depends on its location but mainly on the type of fill used. A bulkhead is often equipped with a water drainage system to drain excess water off the stope. When a stope is completely sealed, filling starts and in a short period pressure builds up behind the bulkheads. This happens especially at the bulkheads at the drawpoint level. Bulkheads at the drawpoint level also have severe risks, due to falling rocks which can fall out towards the drawpoint during construction. Bulkheads at sublevels can have a slightly weaker design since there is not such high pressure. Filling in sublevel stoping is often done with a delayed backfill system. That means that the stope is entirely filled in one operation without breaks. The backfill curing time depend on the type and amount of backfill used, but often one month is assumed (Tunnelbuilder ltd (Atlas Copco Rock Drills AB), 2007).

An example of primary, secondary and tertiary stope mining can be seen in Figure 4-4**.** When a primary stope has been mined it is filled with CAF fill (paste fill). When the backfilled stope is sufficient strong, the secondary stope can be mined and backfilled. This stope does not have to be as strong as a primary stope, because it will not be exposed anymore at the side of the primary stope and thus another type of backfill can be used. Eventually, a tertiary stope can be mined and backfilled using rockfill (Tunnelbuilder ltd (Atlas Copco Rock Drills AB), 2007).

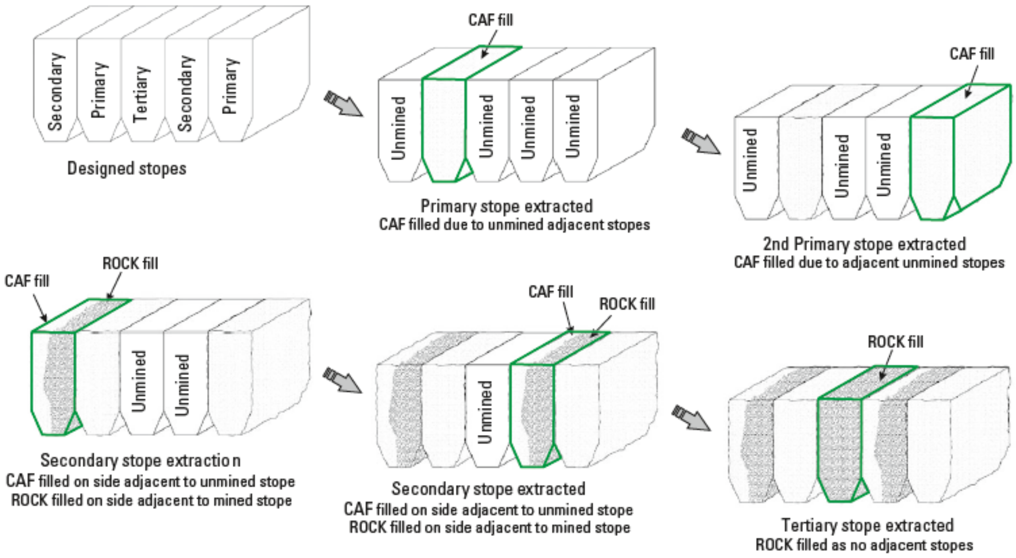


Figure 4-4: Stope extraction and filling sequence at Olympic Dam used to make tertiary stope mining possible, after (Tunnelbuilder ltd (Atlas Copco Rock Drills AB), 2007)

The short-term scheduler does not consider different types of backfill as only the backfill placement rate is important. Curing time is also not considered because this is not necessary to show the application of the model, see also Chapter 9.

4.6 Ventilation

Ventilation of stopes depends on the overall ventilation system used in the mine. This can be over or under pressure and ascensional or descensional ventilation. In sublevel stoping often ascensional ventilation is used to allow for adequate ventilation of all drifts and stopes. If not sufficient air flow can be generated in the stope, auxiliary ventilation could take place. The fresh air is brought to any area where either development or charging takes place and the natural behaviour of air flows the return air to a return air raise. In stopes that means that the fresh air is brought into the stope from the haulage drift and exhausts through the highest sublevel (Tien, 1999).

4.7 Optechs

The size of the final open stope depends on the rock geotechnical conditions, mainly on the blasting performance, because it always gives a different stope boundary contour and changes the designed open space volume. When all blasting rings are fired, the final stope size is obtained and a stope optech can be made. An optech is a digital image of the stope void obtained by firing laser beams into a stope, used for a stope volume survey. The system is brought from the highest sublevel in the stope to make an accurate and detailed image of the stope. It is important to know the stope volume for filling and later stope analysis (Sloane, 2010; Villaescusa, 2004).

5

Proposed scheduling model

The optimization model of this research is based on the characteristics of the sublevel stoping operation described in the previous chapter. This information will be used for the construction of the optimization model which can schedule the sublevel stoping operation.

5.1 General information

The model has been built in MATLAB® R2015b, a mathematical programming software from MathWorks® (MathWorksInc, 2017) and solved with an IBM® ILOG® CPLEX Optimizers *12.7.1*. extension for MATLAB (ILOG, 2017). The model is designed with mixed integer linear programming (MILP) to solve the short-term production schedule problem. It tries to find the optimum extraction sequence with the objective of minimizing deviation from the predefined production targets.

There are four phases proposed for the model which should be scheduled and are described in Chapter 4, namely, development, drilling, extraction and backfilling. These four phases are chosen to represent at considerable detailed level a sublevel stoping operation. This is in accordance with the MRE Institute. Throughout this research the word *activity* is used as a synonym for one or all phases.

The model is restricted by completion targets per half year, over a total time horizon of two years. For each half year, several development drifts must be developed, consisting of X-, Y- and internal-stope-development. Once the internal-stope-development has been finished, the stope progresses through the natural sequential transition of a stope life. This is initiated by drilling and followed by extraction and backfilling. Stopes have a repeating extraction process in which it checks whether the predefined number of blasts slices have been extracted, before commencing the backfilling phase. This sequential order for each stope and its implementation into the MATLAB model can be seen in Figure 5-1.

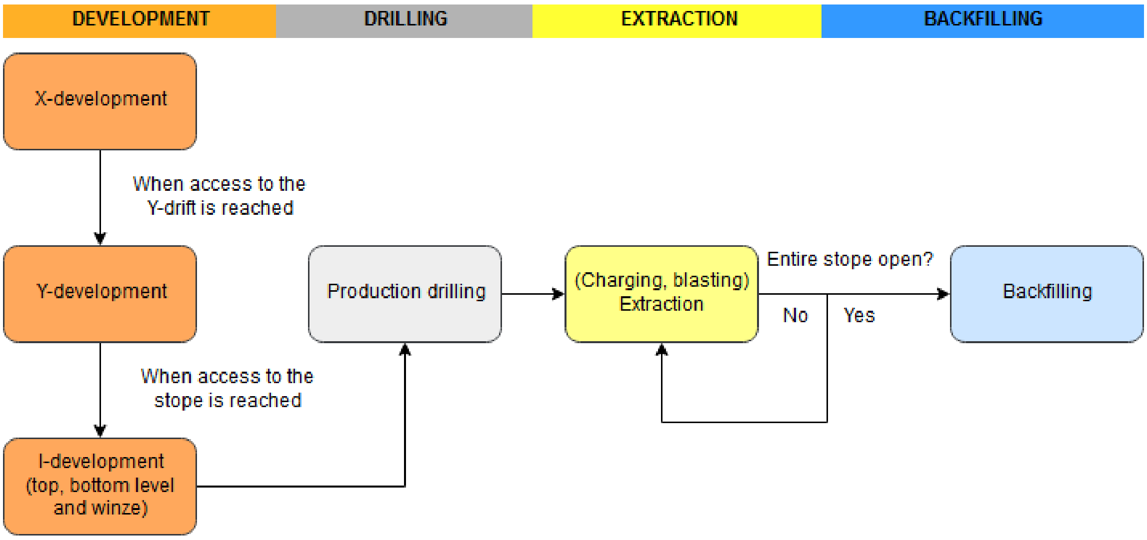


Figure 5-1: Flowsheet representing the typical production phases sequence of the model, including the considered internal phase sequences

5.2 Model assumptions and preparation

The proposed optimization model in this report is based on two things: the blockmodel input for the long-term optimizer and the output of the long-term framework scheduler. This output consists of all development and stopes and these items will be scheduled with the short-term optimizer. This data is received from the MRE Institute.

Normally a long-term mine planning scheduler tries to maximize the NPV by assuming an average grade per stope and only limits the processing parameters to stay within an upper and lower boundary. For a short-term scheduler, it is important to minimize the grade fluctuations and maintain an almost constant grade over the scheduling horizon of all rock that is processed in the processing plant. To better control the grade, stopes are divided into blast slices following the natural firing sequence of a stope. Each blast slice has an average grade suitable for short-term scheduling. The average grade of all blast slices, blasted during a blast, will represent the average grade of the broken ore and thus the ore that is send to the processing plant.

5.2.1 Long-term optimization framework

The long-term optimization model tries to maximize NPV, but also includes geotechnical stability and capacity constraints, among others and therefore this models' output is assumed to be valid. The model's output consists of a set of stopes and development drifts, which are planned to be mined in a two-year time span, consisting of four half year periods. For each development drift and stope is indicated in which half-year period the activity should be commenced and completed. The stopes resulting from the output have an average copper and zinc grade, calculated from the blockmodel.

A summary of the stopes and development items from the long-term optimizer can be seen in Table 5-1. A detailed overview of all stopes and development items from the long-term optimizer can be seen in Appendix A, Table A-1 and A-2. The goal of the short-term model, described in the next subchapter, is to validate this long-term model and to plan it over short-term intervals.

Table 5-1: Summary of the long-term optimizer stopes and development drifts

Half year	Stopes (amount)	Average Cu (%)	Average Zn (%)	Development (m)*
1	9	3.63	0.22	1352
2	10	3.55	0.28	1136
3	9	3.62	0.24	32
4	10	3.58	0.18	40
All	38	3.59	0.23	2560

* The amount of development does not contain internal development

5.2.2 Short-term optimization framework

The short-term model from this report also uses the blockmodel input file, which contains blocks in a specific mining area of the mine with a copper and zinc grade. The mining area represents a small part of a bigger mine. Therefore, it is assumed that while developing and operating this area always simultaneous activity takes place at other parts of the mine. That means that although this model tries to minimize the deviation in grade for the processing plant from this mining area. It can always be blended or compensated with ore from the other parts. Therefore, a bigger deviation than normal in tonnage or grade is assumed to be valid and thus no boundaries are set. It is assumed that all rock that is extracted will be considered in processes of the processing plant and therefore it attributes to grade deviation.

The short-term optimization model starts with calculating an average grade in each stope slice and development item, because a general average grade for a stope is not detailed enough anymore for short-term optimization. It will be assumed that this average grade is the grade percentage for all rock of the extracted volume. The average grade is calculated by summing up all blockmodel blocks (containing a grade) within the development item or stope and diving this by the total number of blocks that should be in the item. This way, also non-existing blocks from the blockmodel are considered for the average grade and it might be possible that several average grades are lower than reality or zero, because there was no "grade-containing block" at all. This is a flaw from the blockmodel, because sometimes at unexpected places within an area surrounded with ore blocks, are no blocks assigned with a grade. Nevertheless, this is assumed to be valid and no effort is put into adjusting this.

If the development item's average grade for copper and zinc is both zero, then it is still considered as rock for the processing plant and as rock that counts for the objective function. If the average grade for both copper and zinc of blast slices is zero, then these slices can be neglected, as mining would be unprofitable. However, if the average grade for a blast slice between two slices, which have a grade is zero, then this cannot be neglected. As this slice must be blasted to follow the natural blasting sequence. All classified extracted rock will be send to the processing plant and there is no difference made between ore and waste within this rock. Therefore, the model will consider all this rock in the objective function.

The above-mentioned steps are all part of the file preparation for the short-term scheduling optimizer. These steps are programmed in MATLAB and the development and stope input file can be automatically constructed from the received output files from the MRE Institute. For a flowsheet representing these steps, see Figure 5-2.

5.3 Input values

In the optimization model the development input file and stope input file, including blast slices, resulting from the file preparation are used. These files consist of all the development or stope items, including several specific individual parameters. For the model, another additional input file is required. The development-relation file is manually created with the aid of a 3D visualization from the stopes and development drifts. Firstly, this file describes the relation between all stopes and the required development to reach this stope. Secondly, it describes the relation between X-, and Y-development and indicates when enough specific X-development is completed to start developing drifts in Y direction. It is assumed that to start internal development both upper and lower Y-development drifts must have reached the point from which the internal development can start.

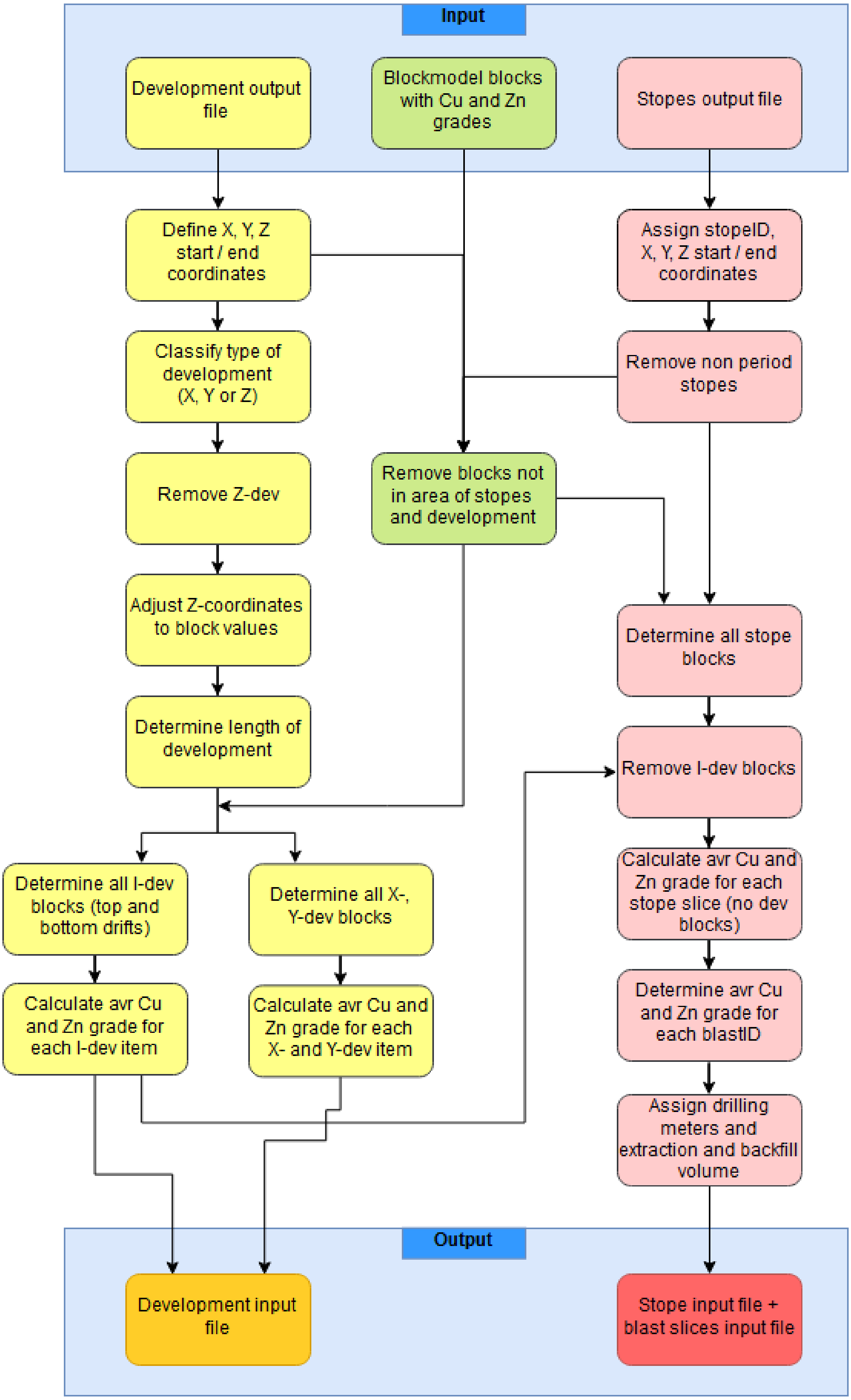


Figure 5-2: Flowsheet representing the model input file preparation steps for the model. It contains different paths and the two output files will be the input for the short-term scheduling optimizer.

There are several parameter inputs required for the model. These values are based on input values from the long-term optimizer and other values have been determined by analysing industry standards and eventually agreed with the MRE institute. The important input values are summarised in Table 5-2. The model applies fixed activity capacities and there is no influence of randomness. Therefore, the model can be classified as deterministic and it should always produce the same output from a given starting condition. The long-term optimizer selected the to-be-mined stopes by means of applying a lower and upper boundary for copper and zinc. The average value of these boundaries is used as a target grade for this research.

Table 5-2: Input parameters used in the optimization model

Parameter	**Value**	**Explanation**
DH	14 DH/ring	The number of drillholes per ring in a stope. The burden between two drill rings is 4 m.
dev machines	3	The maximum amount of development machines available in each period, regardless the type of development
dr machines	2	The maximum amount of production drilling machines in each period
shifts	2 shifts/d	One day consists of two shifts
year	360 d/y	One year consist of 360 days
P	90	Number of periods used to define half a year, 180 days. One period represents two days or four shifts.
dev_shift	9 m/shift	The total development length capacity for all machines per shift combined
dr_shift	600 m/shift	The total drilling length capacity for all machines per shift combined
ex_shift	4500 m^3/shift	The total extraction capacity per shift
bf_shift	1900 m^3	The total backfill placement capacity per shift
Cu target	2.75 %	Target grade for copper
Zn target	0.75 %	Target grade for zinc

All inputs to the MILP model, such as subscript notations, sets, parameters, decision variables and constraints will be shown in the following chapters.

5.4 Subscripts

The MILP model is defined using the following subscript notation:

p	scheduling period: p = 1, 2, 3… P
d	development identification: d = 1, 2, 3… D
	The following subscripts indicate a special selection of *d* identifications:
	dx: X-dev identification: dx = 1, 2, 3… DX
	dy: Y-dev identification: dy = 1, 2, 3… DY
	di: I-dev identification: di = 1, 2, 3… DI
st	stope identification: st = 1, 2, 3… ST
s	blast slice identification: s = 1, 2, 3,… S
	The following subscripts indicate a special selection of *s* identifications:
	s1: blast slice identification from blast #1: $s1$ = 1, 2, 3,… S1
	s2: blast slice identification from blast #2: $s2$ = 1, 2, 3,… S2
	s3: blast slice identification from blast #3: $s3$ = 1, 2, 3,… S3
	s4: blast slice identification from blast #4: $s4$ = 1, 2, 3,… S4

5.5 Sets

To assist in the formulation of constraints and specify activity-specific relations, the following sets are used:

adj_{st}	set of all stopes that are adjacent to and share a boundary with stope *st*
$adjf$	set of all stopes that are adjacent to an existing fillmass and can therefore not be mined simultaneously
$availy_{dx}$	set of all Y-dev drifts that are available after X-dev *dx* is fulfilled
$availi_{dy}$	set of all I-dev drifts that are available after Y-dev *dy* is fulfilled
$availdr_{di}$	set of stopes which are available after I-dev *di* is fulfilled
$avails1_{st}$	set of blast slices that are available after drilling of *st* is fulfilled
$avails2_{s1}$	set of blast slices that are available after extraction of *s1* is fulfilled
$avails3_{s2}$	set of blast slices that are available after extraction of *s2* is fulfilled
$avails4_{s3}$	set of blast slices that are available after extraction of *s3* is fulfilled
$availbf_{s4}$	set of stopes which are available after extraction of *s4* is fulfilled
pb_p	set of time periods that include all periods up to the current period *p*

5.6 Parameters

The following parameters are used for numeric inputs and conditions in the model:

cap_bf_p	total equivalent backfill capacity in period *p*
cap_dev_p	total equivalent development fleet capacity in period *p*
cap_dr_p	total equivalent drilling fleet capacity in period *p*
cap_ex_p	total equivalent extraction fleet capacity in period *p*
$grade\_cu_{d,s}$	equivalent copper grade for development item *d* and blast slice *s*
$grade\_zn_{d,s}$	equivalent zinc grade for development item *d* and blast slice *s*
$mach\_dev_p$	number of development machines available in period *p*
$mach\_dr_p$	number of drilling machines available in period *p*
multiply	multiply factor used to go from development meters to amount of rock (m^3)
res_bf_{st}	backfill reserve for each stope *st*
res_dev_d	development reserve for each development item *d*
res_dr_{st}	drilling reserve for each stope *st*
res_ex_s	extraction reserve for each blast slice *s*
st_bf_{st}	total equivalent backfill requirement for complete backfilling of each stope *st*
$st\_dev_{dx,y,i}$	total equivalent development length for completion of each dev item *dx*, *dy* or *di*
st_dr_{st}	total equivalent production drilling length required for complete drilling stope *st*
$st\_ex_{s1,2,3,4}$	total equivalent extraction amount required for complete extraction of *s1*, *s2*, *s3*, *s4*
target_cu	copper percentage target for the processing plant
target_zn	zinc percentage target for the processing plant

5.7 Decision variables

The MILP model is build up with 13 different variables. These variables are used to define and quantify all phases. The first two continuous variables (a_p and b_p) are only used for the objective function. $DEV_{d,p}$, $DR_{st,p}$, $EX_{s,p}$ and $BF_{st,p}$ are binary variables, which indicate whether an activity in period *p* takes place or not. $Dev_{d,p}$, $dr_{st,p}$, $ex_{s,p}$ and $bf_{st,p}$ are continuous variables, which indicate the amount of activity that takes place in period *p*. $Dr1_{st,p}$, $ex1_{st,p}$ and $bf1_{st,p}$ are commencement variables, which indicate whether the activity has been active in all periods until, and including *p,* or not. The following list summarizes all decision variables:

a_p	sum of ore volume produced above the target grades in period *p*
b_p	sum of ore volume produced below the target grades in period *p*
$DEV_{d,p}$	1 if development of development item *d* takes place in period *p* 0 otherwise
$DR_{st,p}$	1 if drilling in stope *st* and slice *s* takes place in period *p* 0 otherwise
$EX_{s,p}$	1 if extraction in blast slice *s* takes place in period *p* 0 otherwise
$BF_{st,p}$	1 if backfilling of stope *st* takes place in period *p* 0 otherwise
$dev_{d,p}$	amount of development (m) that takes place of development item *d* in period *p*
$dr_{st,p}$	amount of drilling (m) that takes place in stope *st* in period *p*
$ex_{s,p}$	amount of extraction (volume) from blast slice *s* that takes place in period *p*
$bf_{st,p}$	amount of backfill (volume) placed in stope *st* during period *p*
$dr1_{st,p}$	1 if drilling commencement indication is active. In one of the periods in which drilling of stope *st* takes place this variable will be activated 0 otherwise
$ex1_{st,p}$	1 if extraction commencement indication is active. In one of the periods in which extraction of stope *st* takes place this variable will be activated 0 otherwise
$bf1_{st,p}$	1 if backfilling commencement indication is active. In one of the periods in which backfilling of stope *st* takes place this variable will be activated 0 otherwise

5.8 Objective function

The objective function (eq 5.1) tries to minimize copper and zinc grade deviation from the target grade, which is used in the processing plant. It minimizes the sum of copper and zinc ore volume above and below the target grade for each time period *p*:

$$Minimize: \sum_{p} (a_p + b_p) \tag{5.1}$$

5.9 Constraints

Some of the applied constraints of the model are derived from the work of Nehring between 2007 and 2010 and are adjusted to this model. However, there are also additional constraints which have been specially developed for this model to show the required features of the model. The model considers 13 different constraints and a representation of all applied constraints in the constraint coefficient matrix can be seen in Figure 5-3, after (Pourrahimian, 2013).

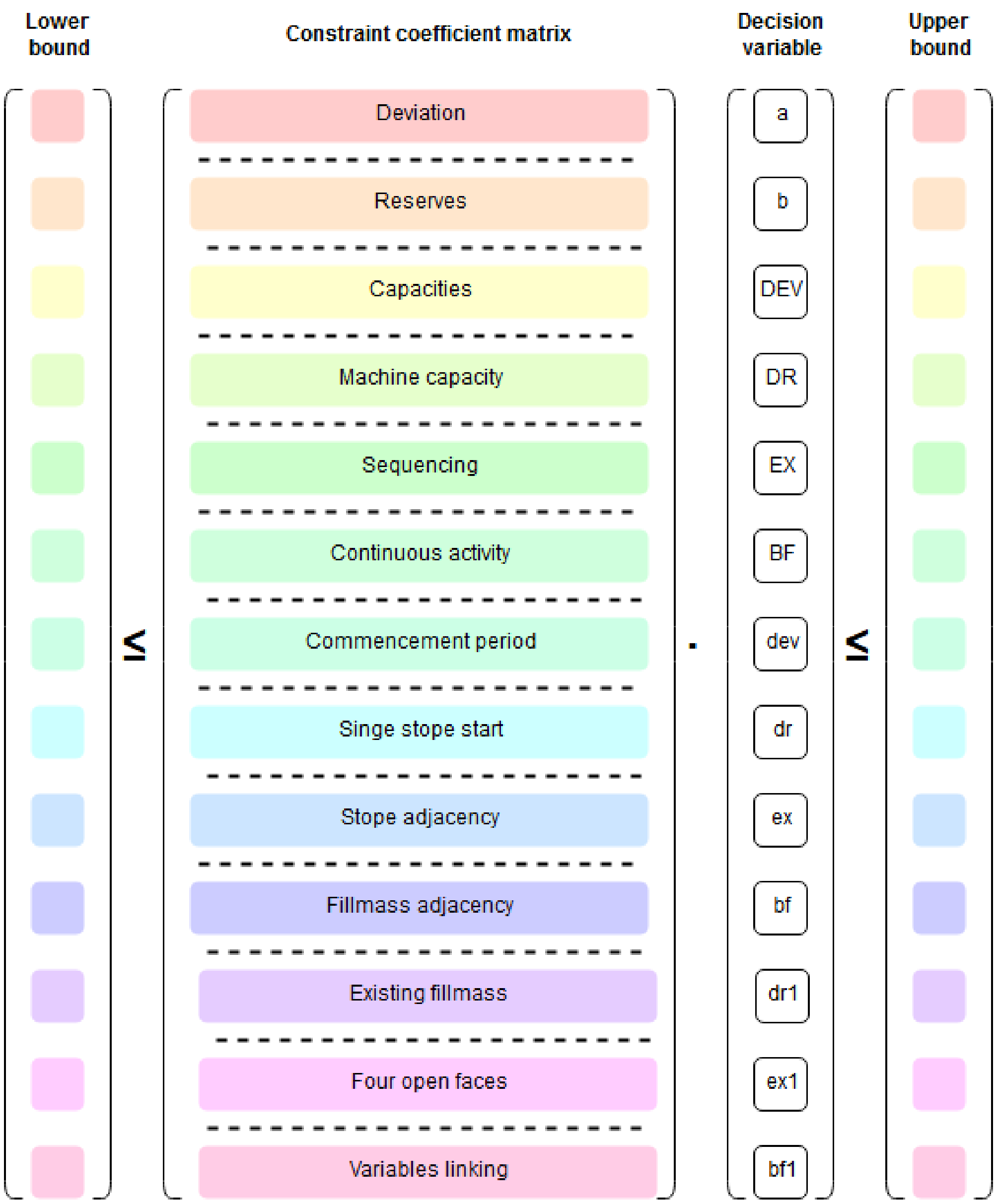


Figure 5-3: The order and representation of the constraints in the constraint coefficient matrix used in the model, after (Pourrahimian, 2013)

Deviation

The deviation constraint (eq 5.2) considers all the ore that is obtained in period p, either from development or stope extraction. The constraint ensures that the ore volume and its respective grade above or below the target grade is balanced by a positive value a_p or negative value b_p to reach a sum of zero.

$$a_p + b_p + multiply \times dev_{d,p} \times \big((grade\_cu_d - target\_cu) + (grade\_zn_d - target\_zn)\big) + ex_{s,p} \times \big((grade\_cu_s - target\_cu) + (grade\_zn_s - target\_zn)\big) = 0 \quad \forall p \tag{5.2}$$

This is the only constraint which contains the decision variables which are used in the objective function. Therefore, there is a close relation between the deviation constraint and the objective function.

Reserves

The reserve constraints (eq 5.3) ensures two things for the development, drilling, extraction and backfill activities. On the one hand it ensures that all activities are completed within the scheduling horizon and on the other hand that each activity cannot exceed its reserve.

$$\begin{aligned} dev&: \sum_{d,p} dev_{d,p} = res\_dev_d \\ dr&: \sum_{st,p} dr_{st,p} = res\_dr_{st} \\ ex&: \sum_{s,p} ex_{s,p} = res\_ex_s \\ bf&: \sum_{st,p} bf_{st,p} = res\_bf_{st} \end{aligned} \tag{5.3}$$

Capacities

The periodical capacity of development, drilling, extraction and backfill cannot be exceeded as enforced by the capacity constraints (eq 5.4).

$$\begin{aligned} dev&: \sum_{d} dev_{d,p} \leq cap\_dev_p \quad \forall p \\ dr&: \sum_{st} dr_{st,p} \leq cap\_dr_p \quad \forall p \\ ex&: \sum_{s} ex_{s,p} \leq cap\_ex_p \quad \forall p \\ bf&: \sum_{st} bf_{st,p} \leq cap\_bf_p \quad \forall p \end{aligned} \tag{5.4}$$

Machine capacity

The machine capacity constraints ensure that the periodical capacity is evenly divided over all activity machines (eq 5.5) and that the individual machine capacity cannot be exceeded (eq 5.6). To achieve this, the model assumes that each machine has the same capacity per specific activity. It also ensures that in periods of planned preventive maintenance the machine cannot be used and thus that the periodical capacity is reduced.

$$dev_{d,p} \leq \frac{cap\_dev_p}{mach\_dev_p} \quad \forall d,p \tag{5.5}$$

$$dr_{st,p} \leq \frac{cap\_dr_p}{mach\_dr_p} \quad \forall st,p$$

$$\sum_{d} DEV_{d,p} \leq mach\_dev_p \quad \forall p \tag{5.6}$$

$$\sum_{st} DR_{st,p} \leq mach\_dr_p \quad \forall p$$

Sequencing

The sequencing constraints ensure that all activities must occur in the order of natural sequential transition from development, drilling, extraction and backfilling (Figure 5-1). In order to mine stope *st* it ensures that the X-development (eq 5.7), Y-development (eq 5.8) and finally I-development (eq 5.9) is completed before drilling takes place. After all blast slices have been drilled (eq 5.10), the first blast and extraction sequence takes place followed by the next blasts and extraction sequences (eq 5.11-5.13). Finally, the backfilling phase can commence when the entire stope is open (eq 5.14). If there is no X-dev in a certain period, then the X-dev constraint is not necessary and the model starts from Y-dev. In some cases, stopes have fewer than four extractions due to the stope size. In this case not all extraction constraints are required. The last backfill constraint will determine if all required extraction is finished before any backfill can start.

- X-dev → Y-dev

$$\sum_{p' \in pb_p} dev_{dx,p'} - st\_dev_{dx} \times DEV_{dx',p=1} - st\_dev_{dx} \times DEV_{dx',p} \geq 0$$

$$\forall dx,p | dx' \in availy_{dx} \tag{5.7}$$

- Y-dev → I-dev

$$\sum_{p' \in pb_p} dev_{dy,p'} - st\_dev_{dy} \times DEV_{dy',p=1} - st\_dev_{dy} \times DEV_{dy',p} \geq 0$$

$$\forall dy,p | dy' \in availi_{dy} \tag{5.8}$$

- I-dev → dr

$$\sum_{p' \in pb_p} dev_{di,p'} - st\_dev_{di} \times DR_{di',p=1} - st\_dev_{di} \times DR_{di',p} \geq 0$$
$$\forall di, p, di | di' \in availdr_{di} \tag{5.9}$$

- dr → ex1

$$\sum_{p' \in pb_p} dr_{st,p'} - st\_dr_{st} \times EX_{s1,p=1} - st\_dr_{st} \times EX_{s1,p} \geq 0$$
$$\forall st, p | s1 \in avails1_{st} \tag{5.10}$$

- ex1 → ex2

$$\sum_{p' \in pb_p} ex_{s1,p'} - st\_ex_{s1} \times EX_{s2,p=1} - st\_ex_{s1} \times EX_{s2,p} \geq 0$$
$$\forall s1, p | s2 \in avails2_{s1} \tag{5.11}$$

- ex2 → ex3

$$\sum_{p' \in pb_p} ex_{s2,p'} - st\_ex_{s2} \times EX_{s3,p=1} - st\_ex_{s2} \times EX_{s3,p} \geq 0$$
$$\forall s2, p | s3 \in avails3_{s2} \tag{5.12}$$

- ex3 → ex4

$$\sum_{p' \in pb_p} ex_{s3,p'} - st\_ex_{s3} \times EX_{s4,p=1} - st\_ex_{s3} \times EX_{s4,p} \geq 0$$
$$\forall s3, p | s4 \in avails4_{s3} \tag{5.13}$$

- ex4 → bf

$$\sum_{p' \in pb_p} ex_{s4,p'} - st\_ex_{s4} \times BF_{bf',p=1} - st\_ex_{s4} \times BF_{bf',p} \geq 0$$
$$\forall s4, p | bf' \in availbf_{s4} \tag{5.14}$$

Continuous activity

The continuous activity constraint (eq 5.15) ensures that once drilling, extraction or backfilling is commenced it will continue this activity in the next period until the activity is finished. This prevents impractical driving distances and redundant driving of machines within the current and next period and yields more geotechnical stability because the open time of stopes is shorter.

$$I\text{-}dev\text{:} \sum_{p' \in pb_p} dev_{d,p'} - st\_dev_{di} \times DEV_{d,p'} + st\_dev_{di} \times DEV_{d,p} \leq 0 \quad \forall d, p \tag{5.15}$$

$$dr\text{:} \sum_{p' \in pb_p} dr_{st,p'} - st\_dr_{st} \times DR_{st,p'} + st\_dr_{st} \times DR_{st,p} \leq 0 \quad \forall st, p$$

$$ex\text{:} \sum_{p' \in pb_p} ex_{s,p'} - st\_ex_{s} \times EX_{s,p'} + st\_ex_{s} \times EX_{s,p} \leq 0 \quad \forall s, p$$

$$bf\text{:} \sum_{p' \in pb_p} bf_{st,p'} - st\_bf_{st} \times BF_{st,p'} + st\_bf_{st} \times BF_{st,p} \leq 0 \quad \forall st, p$$

Commencement period and single stope start

The commencement period constraints (eq 5.16) ensure that when a binary variable of an activity is active this corresponds with the appropriate activity commencement variable and that it therefore can be active.

$$\begin{aligned} dr&: dr1_{st,p} - DR_{st,p} \leq 0 \quad \forall d,p \\ ex&: ex1_{st,p} - EX_{s1,p} \leq 0 \quad \forall st,p \\ bf&: bf1_{st,p} - BF_{st,p} \leq 0 \quad \forall st,p \end{aligned} \tag{5.16}$$

These constraints work together with the single stope start constraint (eq 5.17). It ensure that the indication for commencement can occur once.

$$\begin{aligned} dr&: \sum_p dr1_{st,p} = 1 \quad \forall st \\ ex&: \sum_p ex1_{st,p} = 1 \quad \forall st \\ bf&: \sum_p bf1_{st,p} = 1 \quad \forall st \end{aligned} \tag{5.17}$$

Stope adjacency

The stope adjacency constraints (eq 5.18 and 5.19) ensure that simultaneous activity between all stopes that share a boundary does not occur. This will only count for drilling, extraction and backfill activities. During this period, the stope is open, but it is not necessary that in each period an activity takes place. Therefore, the constraint (5.18) is constructed such, that it considers the first moment of extraction and prevents adjacent activities until the last moment of backfilling. Internal development will take place in the middle of the stope and therefore it is assumed that this could take place at two adjacent stopes. In practice this would be prevented although it is not restricted. The constraint only counts for stopes which have adjacent stope activities.

$$\sum_p ex1_{st,p} - \sum_p \frac{bf_{st,p}}{res\_bf_{st}} + DR_{st',p} + EX_{s',p} + BF_{st',p} \leq 1 \quad \forall p|st', s' \in adj_{st} \tag{5.18}$$

The *ex1* variable selects a period in which extraction from the first slice takes place, but not necessarily means that it is the first period. Therefore, the adjacency constraint has a second component (eq 5.19). This ensures that all other periods, which have been missed by the previous constraint will also be considered for stope adjacency. If only eq 5.19 was used, it could not indicate the periods in which no activity takes place, but which are during the open face time. This would not be sufficient for the adjacency constraint.

$$DR_{st,p} + DR_{st',p} + EX_{s,p} + EX_{st',p} + BF_{st,p} + BF_{st',p} \leq 1 \quad \forall st, s, p | st', s' \in adj_{st} \tag{5.19}$$

Fillmass adjacency

Normally an unmined stope can have activity at two adjacent stopes, if the stope itself undertakes no activity. After a stope has been backfilled and became a fillmass, then activities of adjacent stopes are limited to only one adjacent side. This is required to ensure stability of a fillmass and is ensured by the following constraint (eq 5.20).

$$\sum_{p' \in pb_p} bf1_{st,p'} + \sum_{st' \in adj_{st}} DR_{st',p} + \sum_{s' \in adj_{st}} EX_{s',p} + \sum_{st' \in adj_{st}} BF_{st',p} \leq 2$$
$$\forall st, s, p | st', s' \in adj_{st} \tag{5.20}$$

Existing fillmass

The existing fillmass constraint (eq 5.21) ensures the stability of stopes which have been backfilled in a previous half year. Similarly, the fillmass adjacency constraint, it limits exposure of a fillmass to a single common boundary. This constraint is only applied for fillmasses which have two or more adjacent stopes which will be mined in the considered half year. It does not matter for a fillmass with only one adjacent stope that is mined in the considered half year, because the fillmass will only be exposed at one side. It only considers the extraction and backfill activities at adjacent stopes, because development and drilling will not considerable affect the geotechnical situation of the fillmass.

$$\sum_{s' \in adjf} EX_{s',p} + \sum_{st' \in adjf} BF_{st',p} \leq 1 \quad \forall p \tag{5.21}$$

Four open faces

According to geotechnical limitations it is only allowed to have four open faces in the mine. An open face is defined as a stope where drilling, extraction or backfilling takes place. The four open faces constraint (eq 5.22) will ensure this.

$$\sum_{st} \left(\sum_{p} dr1_{st,p} - \sum_{p} \frac{bf_{st,p}}{res\_bf_{st}} \right) \leq 4 \quad \forall st, p \tag{5.22}$$

Variables linking

The binary variable, which indicate whether an activity in period *p* takes place or not cannot be active without its corresponding continuous variable to be active (eq 5.23).

$$\begin{gathered} dev\colon DEV_{d,p} - dev_{d,p} \leq 0 \quad \forall p, d \\ dr\colon DR_{st,p} - dr_{st,p} \leq 0 \quad \forall p, st \\ ex\colon EX_{s,p} - ex_{s,p} \leq 0 \quad \forall p, s \\ bf\colon BF_{st,p} - bf_{st,p} \leq 0 \quad \forall p, st \end{gathered} \tag{5.23}$$

Similarly, the continuous variable for each activity cannot be active without its corresponding binary variable to be active (eq 5.24).

$$\begin{gathered} dev\colon res\_dev_d \times DEV_{d,p} - dev_{d,p} \geq 0 \quad \forall p, d \\ dr\colon res\_dr_{st} \times DR_{st,p} - dr_{st,p} \geq 0 \quad \forall p, st \\ dev\colon res\_ex_s \times EX_{s,p} - ex_{s,p} \geq 0 \quad \forall p, s \\ dev\colon res\_bf_{st} \times BF_{st,p} - bf_{st,p} \geq 0 \quad \forall p, st \end{gathered} \tag{5.24}$$

Non-negativity and integer

All variables are constructed to be non-negative and specific variables are furthermore forced to be a binary integer variable (eq 5.25).

$$\begin{gathered} all\ variables\ \geq 0 \\ DEV_{d,p}, DR_{st,p}, EX_{s,p}, BF_{st,p}, dr1_{st,p}, ex1_{st,p}, bf1_{st,p} = binary\ integer \end{gathered} \tag{5.25}$$

6

Application of the implemented model

The results obtained after the file preparation described in the previous chapter are based on the considered mine operation, which will be shown in this chapter. The results and overview are necessary as an input for the scheduler and therefore important to understand.

6.1 File preparation

File preparation is the first step required before running the scheduling model. The tables in this chapter only show the results of the file preparation for all items from the first half year. The tables for all other half years can be found in Appendix B. The following input files are generated:

- Development drifts (X- and Y-development)
- Internal development
- Development relation
- Stopes
- Stope slices
- Periodical capacity
- Stope adjacency

6.1.1 Development drifts input

Table 6-1 shows the development input file for the X- and Y-development, which is created from the development output file from the long-term optimizer. This table contains the most important data for the development items from the model. From the X, Y and Z start and end coordinates, the average Cu and Zn grade, type of development and length are calculated. Type 1 development represents X-development drifts and type 2, Y-development drifts. The development input file does not contain the I-development from the stopes since this depends on the length of the stope and is therefore a separate input. See also appendix Table B-1.

Table 6-1: Development input file for the X- and Y-development from half year one

DevID	Xstart	Xend	Ystart	Yend	Zstart	Zend	type*	Half year	Length (m)	Cu (%)	Zn (%)
1	5042	4718	2748	2748	684	684	1	1	324	1.57	0.10
2	5042	4718	2748	2748	640	640	1	1	324	0.95	0.14
3	5042	4862	2748	2748	612	612	1	1	180	0	0
4	5042	4862	2748	2748	568	568	1	1	180	0	0
5	4718	4718	2748	2744	684	684	2	1	4	7.09	0.20
6	4718	4718	2748	2744	640	640	2	1	4	0.00	0.00
7	4718	4718	2748	2764	684	684	2	1	12	5.38	0.27
8	4718	4718	2748	2764	640	640	2	1	12	0	0
9	4754	4754	2748	2732	684	684	2	1	16	9.99	0.60
10	4754	4754	2748	2732	640	640	2	1	16	0	0
11	4754	4754	2748	2776	684	684	2	1	24	8.23	0.96
12	4754	4754	2748	2776	640	640	2	1	24	0	0
13	4826	4826	2748	2632	684	684	2	1	116	2.25	0.04
14	4826	4826	2748	2632	640	640	2	1	116	0	0

* type 1 = X-development, type 2 = Y-development

6.1.2 Internal development input

Table 6-2 shows the input file which contains the average copper and zinc grade from each stopes' internal development, scheduled for the first half year. The represented grade is only obtained from development ore above and under the stope. The received ore from mining the raise is not included, because during raise boring the ore falls on the drawpoint level and this will be extracted after the first blast has taken place. This rock is blended with all ore obtained from the first blast slice. See also appendix Table B-2.

Table 6-2: Internal development input file for half year one, containing the average Cu and Zn grade for each stope from the I-dev

StopeID	Cu (%)	Zn (%)	Half year
248	2.51	0.18	1
250	3.05	0.15	1
252	2.25	0.36	1
256	2.07	0.26	1
291	0.96	0	1
293	3.68	0.11	1
295	3.41	0.08	1
297	4.00	0.08	1
298	4.87	0.03	1

6.1.3 Development relation input

Table 6-3 shows the relation between the required development that must be completed before stope production can commence for the first half year, see Appendix B-3 for all half years. Development drift one until four corresponds with X-development drifts. DevID five until 14 resembles Y-development drifts and the others represent internal stope development drifts. To start drilling at stope 298 it requires that 216 m of X-dev, 28 m of Y-dev and 104 m of I-dev is completed in, respectively, development drift 1 and 2, 13 and 14, and I298. The natural order of mining tells that first the required X-dev needs to be reached before Y-dev can start and, respectively, 28 m of Y-dev before I-dev can commence. If 12 m more development is done at the same drift, then stope 297 is opened and I-dev for stope 297 can start.

Table 6-3: Development relation file which shows the amount of development in meters that must be completed before stope activities can take place for all stopes in half year one

Stope ID	DevID																						
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	I248	I250	I252	I256	I291	I293	I295	I297	I298
248	324	324	0	0	0	0	12	12	0	0	0	0	0	0	104	0	0	0	0	0	0	0	0
250	288	288	0	0	0	0	0	0	16	16	0	0	0	0	0	104	0	0	0	0	0	0	0
252	288	288	0	0	0	0	0	0	0	0	12	12	0	0	0	0	104	0	0	0	0	0	0
256	288	288	0	0	0	0	0	0	0	0	24	24	0	0	0	0	0	104	0	0	0	0	0
291	216	216	0	0	0	0	0	0	0	0	0	0	88	88	0	0	0	0	104	0	0	0	0
293	216	216	0	0	0	0	0	0	0	0	0	0	64	64	0	0	0	0	0	104	0	0	0
295	216	216	0	0	0	0	0	0	0	0	0	0	52	52	0	0	0	0	0	0	104	0	0
297	216	216	0	0	0	0	0	0	0	0	0	0	40	40	0	0	0	0	0	0	0	104	0
298	216	216	0	0	0	0	0	0	0	0	0	0	28	28	0	0	0	0	0	0	0	0	104

6.1.4 Stope input

Table 6-4 shows the most important information from the stopes input file. See appendix Table B-4 for a full overview of all stopes. The volume is the total volume of the designed stope and the designed volume according to the long-term optimizer. Vol_ex is the volume that is extracted during the extraction phase. This is only the volume that is extracted, including the ore from the raise. It also accounts for non-extracted blast slices due to the absence of grades. This will be in more detail explained in the next subchapter. Vol_bf is the volume that must be backfilled. During the backfill phase, the stope at the lower level stope entrance is closed with a bulkhead. This increases the total stope volume and therefore the backfill volume is bigger than the extracted volume. The stope is only backfilled until the top of the stope and not the upper drift itself. When several stopes are designed above each other, the upper drift becomes the new lower drift for the next stope. The average copper and zinc grades from the long-term optimizer are an indication of the grade that can be expected as the average grade for each stope blast. Based on this assumption it can be discussed why stope 291 is appointed as a stope to mine. The average copper and zinc grade is much lower than any other stope and not even close to the target grades. Despite this, the stope is scheduled and not deviated from the schedule, because the model uses the stopes from the long-term optimizer as input.

Table 6-4: Stopes input file from half year one

StopeID	sizeX (m)	sizeY (m)	sizeZ (m)	Cu* (%)	Zn* (%)	Half year	volume (m^3)	vol_ex (m^3)	vol_bf (m^3)	dr (m)
248	36	12	40	1.76	0.08	1	17280	17280	17856	2520
250	36	12	40	6.67	0.55	1	17280	17280	17856	2520
252	36	12	40	8.02	0.66	1	17280	17280	17856	2520
256	36	12	40	6.22	0.46	1	17280	17280	17856	2520
291	36	12	40	0.05	0	1	17280	1920	2496	280
293	36	12	40	0.51	0.02	1	17280	11520	12096	1680
295	36	12	40	1.51	0.05	1	17280	17280	17856	2520
297	36	12	40	3.19	0.08	1	17280	17280	17856	2520
298	36	12	40	4.70	0.09	1	17280	17280	17856	2520

* Average stope Cu and Zn data is shown for information, but not used in the model, because the model requires information per blast

6.1.5 Stope slices input

Table 6-5 shows a part of the stope slices input file which shows the characteristics of each of the four initial determined blasts slices. The amount of blast slices with a grade and volume depends on the length (x size) of the stope and for each blast slice is the average Cu and Zn grade calculated to better indicate grade changes within a stope. The table shows a good example, why it for certain stopes is better not to extract everything. Blast slices two until four of stope 291 contain no copper and zinc and therefore from this point there will be deviated from the long-term schedule plan. This consideration will adjust the designed extraction and backfill volume and it represents improved mining considerations as less waste will be mined. For more details about all stopes, see appendix Table B-5.

Table 6-5: Selection of the stope slices input file

BlastID	Cu (%)	Zn (%)	Vol_ex (m^3)
248_1	4.14	0.25	1920
248_2	2.43	0.11	3840
248_3	1.66	0.06	5760
248_4	0.63	0.02	5760
250_1	3.92	0.32	1920
250_2	4.47	0.46	3840
250_3	6.24	0.54	5760
250_4	9.47	0.70	5760
291_1	0.45	0.00	1920
291_2	0.00	0.00	0
291_3	0.00	0.00	0
291_4	0.00	0.00	0

6.1.6 Periodical capacity input

The periodical capacity input file contains the development, drilling, extraction and backfill capacity for each scheduled period and can be seen in appendix Table B-6. The capacity for each activity has been defined in the models input parameters (Table 5-2). The model assumes constant capacities over the entire scheduling horizon and therefore this file might not be required. Although it is done in the form of a table, because then scheduled maintenance or reduced capacity can be implemented in the schedule. If maintenance is planned in a certain period, it is possible to adapt the overall capacity and number of machines and the model will implement this.

6.1.7 Stope adjacency input

The stope adjacency input file helps with two important things for the model and can be found in appendix Table B-7. It indicates which stopes are adjacent to each stope in the considered period and therefore help with the adjacency constraints. It also indicates all adjacent stopes which became a fillmass in a previous period. Therefore, it indicates the exposure limitation to only one side of those fillmasses.

6.2 Visualization

Visual representations of all development and stopes are designed in SGeMS to graphically validate the feasibility of the produced schedules and constructed input tables and help to understand the natural mine development behaviour. SGeMS is an open-source program for geostatistical modelling (SGeMS, 2017). It has an interactive 3D visualization environment and is therefore suitable for showing the stope blocks as point set data.

6.2.1 Mine operation configuration

Figure 6-1 and Figure 6-2 show two different views of the mining area with the development drifts and stopes. Appendix Figure C-1 and C-2 represent the top view layout of the mine operation including all stopeIDs and devIDs. There is an upper zone with stopes with a Z elevation between 640-680 and a lower zone between 568-608, respectively. Between those two levels a crown pillar is located which ensures geotechnical stability. All stopes from the first half year are in the upper zone, what means that all activities take place in a concentrated area and detailed scheduling is important. For the following periods, the stopes are more spread out, thus less problems with traffic situations will appear, although driving distances between different activities are bigger. The spread of stopes can be explained with the idea of primary, secondary and tertiary stopes. Most of the stopes from half year two until four are single standing stopes considering the stopes from its respective period. Many of the stopes from half year two, three and four can therefore be classified as primary, secondary and tertiary stopes, respectively. The spread of stopes in half year one is limited, because a lot of development has to be carried out and therefore few different locations in the mine can be reached. From half year two onwards, majority of the development has already been completed and therefore a long-term optimizer can better control the behaviour of geotechnical stability and assign stopes further away from each other and mine the intermediate stopes in the following periods.

At several locations development takes place, which is not needed to reach any stope. This is planned by the long-term optimizer and not for compensated in this model, because all planned development is scheduled. This additional development could be used for future access to stopes which are planned after the scheduling horizon of two years. It can be assumed that this development has lower priority and will be scheduled at moments after all required development for stopes is done. This will be further discussed when the schedules are shown. Figure 6-2 shows that almost all X- and Y-development is completed after period two. In period three it is therefore possible to start directly with internal development and with stope mining.

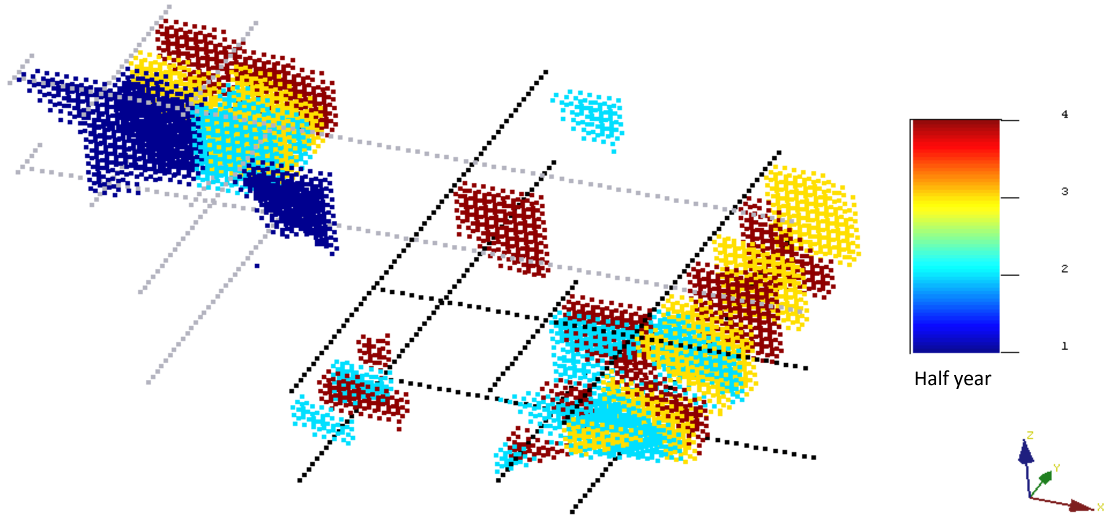


Figure 6-1: Turned view of the stopes and development drifts considered in the model
Note: The grey drifts correspond to the upper two levels and the black drifts to the two lower levels. The color of the stopes represents for which half year the stope is scheduled to be mined

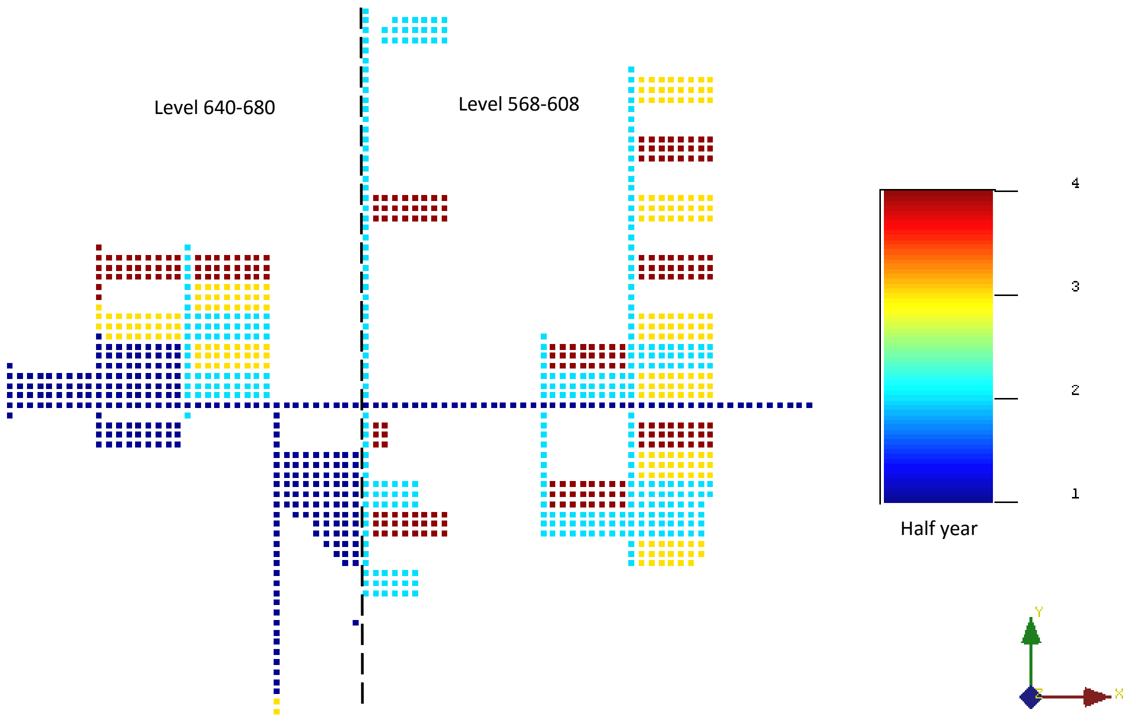


Figure 6-2: Top view of the stopes and development drifts considered in the model
Note: The internal stope development is not shown. There are no stopes located above each other. The color represents for which half year the development or stope is planned.

6.2.2 Half year one

A detailed and closer view on the stopes from the first half year is given in Figure 6-3 and Figure 6-4. Each dot represents a 4 x 4 x 4 m block from the blockmodel and represents a copper and zinc grade. The highest ore grades can be found in stope 250 and 252. These stopes have the highest economic potential and therefore early mining of these stopes will result in an early cash flow. All blocks containing ore in stope 293 are at the upper half of the stope. During extraction of the stope this information could be used to reduce the amount of waste that will be send to the processing plant. Due to the natural gravity flow of rock after a blast, the initial rock could be classified as waste and the later extracted rock as ore.

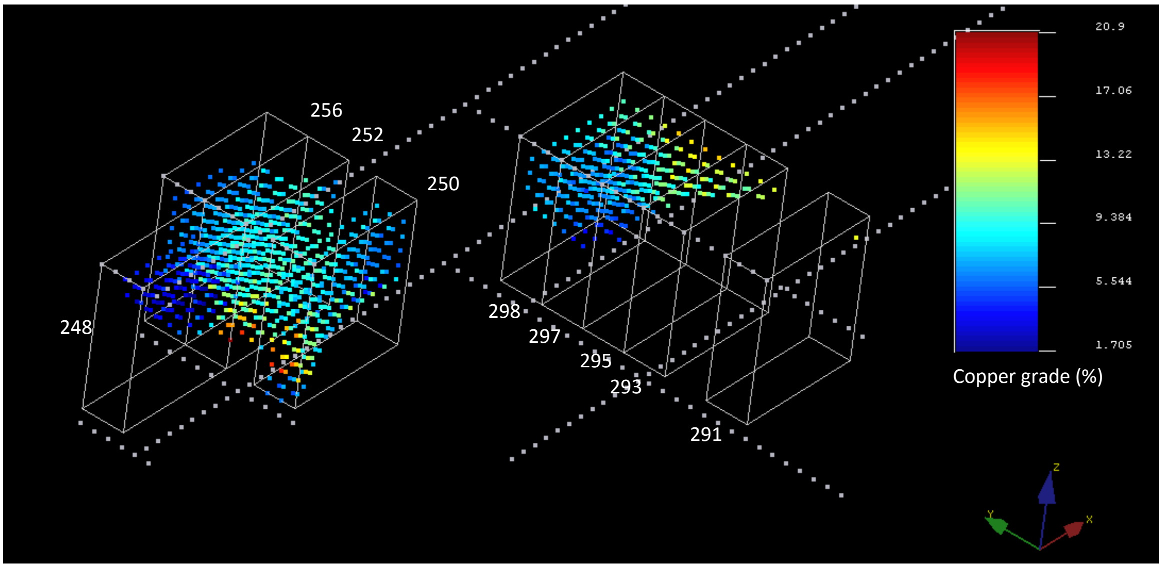


Figure 6-3: Overview of all stopes including copper grade (%) mined in the first half year
Note: The color represents the copper grade (%) in each block from the blockmodel.

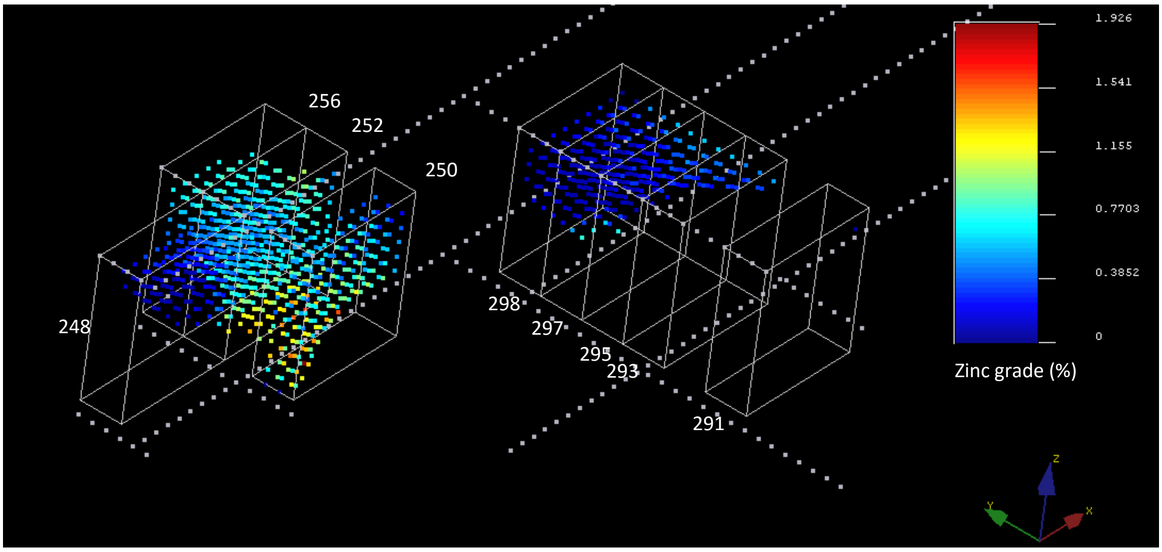


Figure 6-4: Overview of all stopes including zinc grade (%) mined in the first half year
Note: The color represents the zinc grade (%) in each block from the blockmodel.

7

Results

After the model has been appropriately formatted with input values it is solved with the short-term schedule optimizer in MATLAB using CPLEX 12.7.1 on a standard TU Delft office computer with 8 GB RAM. Eventually several analyses regarding the models' objective are performed. The focus of this chapter lies on the discussion of the results of the first half year, but summaries are given for all other half years.

7.1 Short-term schedule from half year one

The model for the first year with all shown parameters is solved in 66003 seconds (18.3 h). It contained 16497 different variables and 35366 constrains. A summary of the model characteristics of all scheduled periods can be seen in Table 7-1.

Table 7-1: Summary of the optimization model variables, constraints and solving time

Half year	Variables	Constraints	Solving time (sec)
1	16497	35366	66003
2	18390	36637	Not possible*
3	14697	28182	Not possible*
4	15870	30791	Not possible*

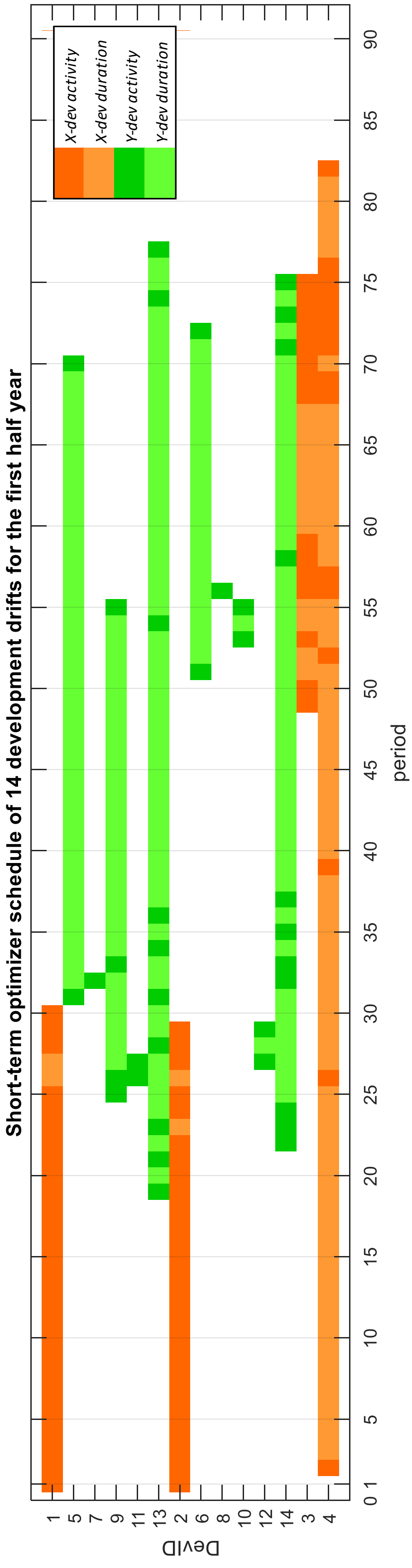
* These models could not be solved, see Chapter 8.2 and 9.2 for more information

A visual representation of the first half year's schedule can be seen in Figure 7-1. The upper schedule shows the scheduled X- and Y-development per drift and period, and the lower schedule the schedule for each stope. In the upper schedule, the orange color is the entire scheduling period for X-development until the activity is completed and the green color for Y-development, respectively. Darker colors indicate the time periods in which any activity takes place in that drift. A development meter indication per drift and period is shown in Table 7-2. From the data, the natural behaviour of tunnel development can be seen. Period one starts with X-development, as was supposed, at the two most upper levels with two development machines. After 216 m meter of tunnel per level is reached in period 18, it is possible to start development in Y-direction in drift 13 and 14 (upper and lower level) in period 19. From period 19 onwards development in X-direction continues and it is finished in period 30. The first stope the model chooses to start developing is at drift 11 and 12. That are short drifts with a very high copper

grade (Table 6-1) and therefore quick access to stope 256 can be reached. Another option for the model was to choose mining any of the stopes at drift 13 and 14, but this would probably have resulted in a larger deviation. In the periods that those stopes are scheduled more production at other locations take place and a lower deviation is obtained. As discussed before, unnecessary required development for the considered period exists. This development (drift 5, 6 and the end of 13, 14) is planned in periods in which no “higher priority” development is planned and in periods in which it can contribute to a lower deviation.
It is seen that the adjacency constraints have been applied well. Stope 252 and 256 are adjacent to each other and share a common boundary (248 also with 252). The adjacency constraint has been constructed such that no drilling, extraction or backfilling can take place when the adjacent stope has commenced extraction until the moment that backfilling is completed. The internal development of stope 256 starts in period 30 and takes 9 periods to complete. Drilling follows from period 39 until 42. In period 43, the first extraction takes place, so no adjacent activity is allowed until period 51, when backfilling is finished. During this period, only the internal development of stope 252 is done. This is allowed according to the phases considered with the adjacency constraint. This appearance will be further discussed in Chapter 9.

At drift 13 and 14 four stopes are adjacent to each other (293, 295, 297 and 298). The mining order of these stopes is therefore important to ensure geotechnical stability. Mining of stopes 293 and 297 or 295 and 298 can be done simultaneously as long as the intermediate stope has not been backfilled. If the intermediate stope is a fillmass, it is only allowed to be exposed at one side. Stope 298 is the first stope to be mined from period 43 until 56. Directly after backfilling is finished, production at stope 297 starts (293 as well in the meantime) and stope 295 remains as an unmined stope. Extraction of stope 295 starts in period 73, after both stopes 293 and 297 have been finished with backfilling in period 72.

There are two single stopes (250 and 291). There are no limitations on scheduling these stopes regarding other stopes. Stope 250 has a very high grade and stope 291 a very low grade, therefore it makes sense mining these two stopes simultaneously to obtain a better average grade for the processing plant. This is indeed the case and they are mined from period 72 until 75.



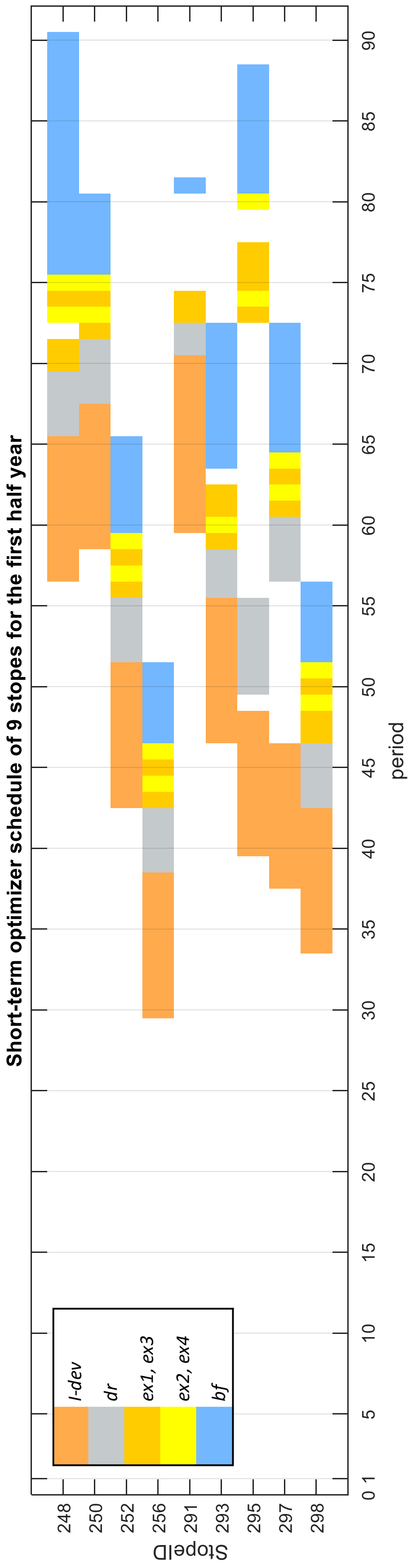


Figure 7-1: Short-term schedule for all development and stope items from the first half year

Table 7-2: Overview which indicates the amount of development meters done in each period for each development drift

Period	DevID 1	5	7	9	11	13	2	6	8	10	12	14	3	4
1	12	0	0	0	0	0	12	0	0	0	0	0	0	0
2	12	0	0	0	0	0	12	0	0	0	0	0	0	12
3	12	0	0	0	0	0	12	0	0	0	0	0	0	0
4	12	0	0	0	0	0	12	0	0	0	0	0	0	0
5	12	0	0	0	0	0	12	0	0	0	0	0	0	0
6	12	0	0	0	0	0	12	0	0	0	0	0	0	0
7	12	0	0	0	0	0	12	0	0	0	0	0	0	0
8	12	0	0	0	0	0	12	0	0	0	0	0	0	0
9	12	0	0	0	0	0	12	0	0	0	0	0	0	0
10	12	0	0	0	0	0	12	0	0	0	0	0	0	0
11	12	0	0	0	0	0	12	0	0	0	0	0	0	0
12	12	0	0	0	0	0	12	0	0	0	0	0	0	0
13	12	0	0	0	0	0	12	0	0	0	0	0	0	0
14	12	0	0	0	0	0	12	0	0	0	0	0	0	0
15	12	0	0	0	0	0	12	0	0	0	0	0	0	0
16	12	0	0	0	0	0	12	0	0	0	0	0	0	0
17	12	0	0	0	0	0	12	0	0	0	0	0	0	0
18	12	0	0	0	0	0	12	0	0	0	0	0	0	0
19	12	0	0	0	0	12	12	0	0	0	0	0	0	0
20	12	0	0	0	0	0	12	0	0	0	0	0	0	0
21	12	0	0	0	0	12	12	0	0	0	0	0	0	0
22	12	0	0	0	0	0	12	0	0	0	0	4	0	0
23	12	0	0	0	0	12	0	0	0	0	0	12	0	0
24	12	0	0	0	0	0	12	0	0	0	0	12	0	0
25	11	0	0	7	0	0	12	0	0	0	0	0	0	0
26	0	0	0	1	12	0	0	0	0	0	0	0	0	12
27	0	0	0	0	12	0	12	0	0	0	12	0	0	0
28	1	0	0	0	0	12	12	0	0	0	0	0	0	0
29	12	0	0	0	0	0	12	0	0	0	12	0	0	0
30	12	0	0	0	0	0	0	0	0	0	0	0	0	0
31	0	1	0	0	0	12	0	0	0	0	0	0	0	0
32	0	0	12	0	0	0	0	0	0	0	0	12	0	0
33	0	0	0	7	0	0	0	0	0	0	0	12	0	0
34	0	0	0	0	0	12	0	0	0	0	0	0	0	0
35	0	0	0	0	0	0	0	0	0	0	0	12	0	0
36	0	0	0	0	0	12	0	0	0	0	0	0	0	0
37	0	0	0	0	0	0	0	0	0	0	0	12	0	0
38	0	0	0	0	0	0	0	0	0	0	0	0	0	0
39	0	0	0	0	0	0	0	0	0	0	0	0	0	12
40	0	0	0	0	0	0	0	0	0	0	0	0	0	0
...														

Period	DevID 1	5	7	9	11	13	2	6	8	10	12	14	3	4
48	0	0	0	0	0	0	0	0	0	0	0	0	0	0
49	0	0	0	0	0	0	0	0	0	0	0	0	12	0
50	0	0	0	0	0	0	0	0	0	0	0	0	12	0
51	0	0	0	0	0	0	0	1	0	0	0	0	0	0
52	0	0	0	0	0	0	0	0	0	0	0	0	0	12
53	0	0	0	0	0	0	0	0	0	4	0	0	12	0
54	0	0	0	0	0	12	0	0	0	0	0	0	0	0
55	0	0	0	1	0	0	0	0	0	12	0	0	0	0
56	0	0	0	0	0	0	0	0	12	0	0	0	12	12
57	0	0	0	0	0	0	0	0	0	0	0	0	12	12
58	0	0	0	0	0	0	0	0	0	0	0	12	12	0
59	0	0	0	0	0	0	0	0	0	0	0	0	12	0
60	0	0	0	0	0	0	0	0	0	0	0	0	0	0
...														
67	0	0	0	0	0	0	0	0	0	0	0	0	0	0
68	0	0	0	0	0	0	0	0	0	0	0	0	12	12
69	0	0	0	0	0	0	0	0	0	0	0	0	12	12
70	0	3	0	0	0	0	0	0	0	0	0	0	12	0
71	0	0	0	0	0	0	0	0	0	0	0	4	12	12
72	0	0	0	0	0	0	0	3	0	0	0	0	12	12
73	0	0	0	0	0	0	0	0	0	0	0	12	12	12
74	0	0	0	0	0	12	0	0	0	0	0	0	12	12
75	0	0	0	0	0	0	0	0	0	0	0	12	12	12
76	0	0	0	0	0	0	0	0	0	0	0	0	0	12
77	0	0	0	0	0	8	0	0	0	0	0	0	0	0
78	0	0	0	0	0	0	0	0	0	0	0	0	0	0
79	0	0	0	0	0	0	0	0	0	0	0	0	0	0
80	0	0	0	0	0	0	0	0	0	0	0	0	0	0
81	0	0	0	0	0	0	0	0	0	0	0	0	0	0
82	0	0	0	0	0	0	0	0	0	0	0	0	0	12

Table 7-3 shows the detailed production schedule from stope 293. A full overview of all stopes from the first half year can be seen in appendix Table D- 1. Development takes nine, drilling three, extraction four and backfilling nine periods, respectively. Backfilling could have been completed four periods earlier, because in four of the periods the model decided to do only 1 m^3 of backfill. This could be interpreted as no activity. The model does this to comply with the continuous activity until completion constraint.

In practice, this would mean that each time, two days no backfill is placed in a stope. This appearance can be interpreted in three ways. First, it is good that backfill placement breaks occur especially in large stopes. This is advantageous for curing of the lower layers to increase the backfill stability. However, it could be discussed whether this is applicable and required for the stope sizes used in this research. Second, it could mean that the model has too much freedom, because it does not consider the open face time of a stope. Additional constraints which limit the open exposure time could be included. And third, it would show the strength of this model. Since the model is free to choose the amount of activity in each period, respectively to all constraints, it would mean that it is also capable of dealing with unplanned maintenance or delays. If a fixed number of periods would be defined for each activity, it would be more difficult to deal with capacity restrictions. A mining engineer would accept this stope schedule as is, but if there are no further restrictions from a processing plant regarding backfill capacity or planning he would choose to continue to backfill at full capacity in each period and backfilling would be finished four periods earlier.

Table 7-3: Production schedule which indicates the amount of activity (m or m³) done in stope 293 in each month

Period	47	48	49	50	51	52	53	54	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72
l-dev	8	12	12	12	12	12	12	12	12	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	800	80	800	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	1920	0	0	0	0	0	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	3840	0	0	0	0	0	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4047	1686	0	0	0	0	0	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1145	1	1142	3799	1	1	2207	1	3799

Chapter 6 discussed the expected stope mining order for half year one. Generally, effort is done to mine high grade ore as soon as possible. Due to the restrictions that all stopes must be finished within half a year, however, the model tries to start production of any stope as quick as possible. The half year one stopes are far away from the begin point of X-development and therefore time limitations could occur and this is a higher weighing factor than searching for the stope with the highest grade. Beside that, the model does not contain any economic consideration, as this is already partly applied in the long-term schedules and all stopes should result in a high economic result for the first half year.

7.2 Grade deviation

The objective function minimized the grade deviation from production target in each period by searching for a combination of activities. The target grades for copper and zinc were 2.75 % and 0.75 %, respectively. The copper and zinc deviation from target can be seen in Figure 7-2. It can be seen that deviation is relatively big and there is an average (absolute) copper deviation of 2.07 % and 0.57 % zinc deviation from production target. The overall average copper grade is 3.01 % during the first half year and zinc 0.19 %. A summary containing all schedule horizons is given in Table 7-4. The average grades indicate that the chosen target grade for copper is too low and too high for zinc, because the average copper grade of all stopes is 3.63 % and zinc only 0.22 %. This confirms that it is almost impossible to reach the target grades. The main contribution to the average copper and zinc grades come from stopes rather than from development.

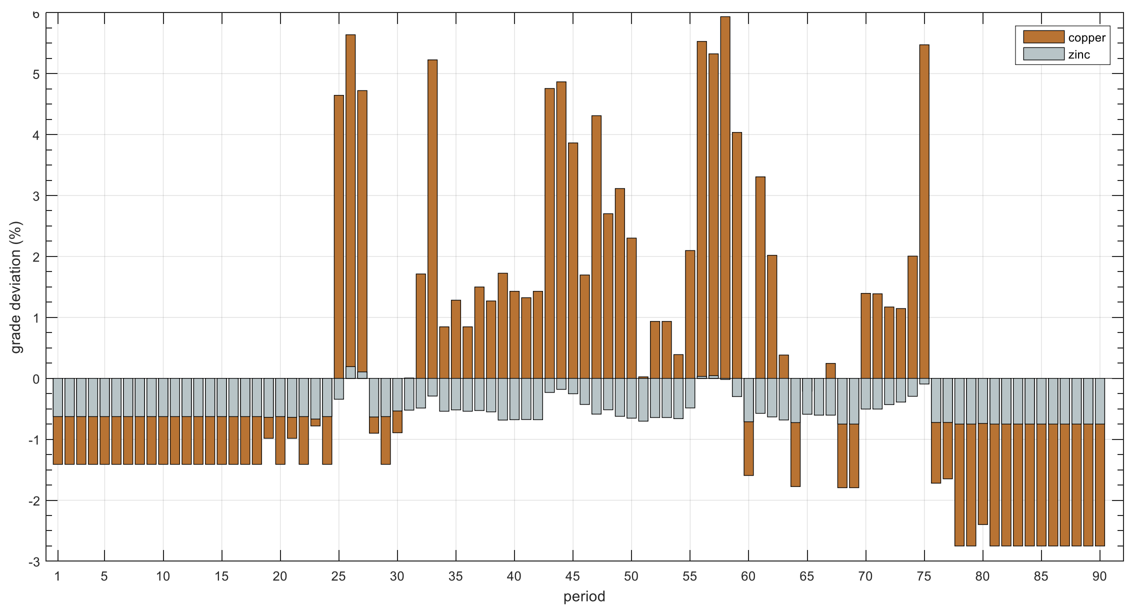


Figure 7-2: Copper and zinc grade deviation from the production target per period

Roughly three trends can be seen in Figure 7-2. From period one until 24, the grade is below the target grade. During this period, only X-development takes place and the average grades for all development drifts is low what explains a negative deviation from the target grade. Generally, this rock should not be send to the processing plant and therefore a deviation of zero should be assumed. From period 25 until 75, the copper deviation is (mainly) positive and zinc deviation (mainly) negative. This is due to the above-mentioned reason regarding the high average grade of the stopes. This period represents the period where most of the stope extraction takes place. From period 76 until 90, no extraction or development takes place, only backfilling. Therefore, the deviation is as big as the target grade and it should not contribute to the average deviation.

Table 7-4: Summary of the Cu and Zn grades for each half year

Half year	Avr absolute Cu deviation (%)	Avr absolute Zn deviation (%)	Avr Cu (%)	Avr Zn (%)	Avr Cu stopes (%)	Avr Zn stopes (%)
1	2.07	0.57	3.01	0.19	3.63	0.22
2	-	-	-	-	3.55	0.28
3	-	-	-	-	3.62	0.24
4	-	-	-	-	3.58	0.18

Based on the deviation analysis for the first half year a new average deviation is calculated. This construction does not consider the development and non-extraction activities for calculating the average grade, in these periods the deviation is zero. Excluding development is better for this specific model, because there are many missing blocks when calculating the average grades of each drift. Normally, the drifts would be classified as waste due to the respective grades. The comparison between including and excluding development and non-extraction periods shows that during most of the extraction periods the copper and zinc deviation is lower than the original values, see Figure 7-3. In some cases, it is still higher, but that is because then development took place in that period and decreased the average grade, hence deviation. The average copper deviation increased to 2.32 % and zinc decreased to 0.46 % (only considering the extraction periods). This analysis is better, because it contains a selection criteria for rock and better represent all rock that is classified as ore and that goes to the processing plant. In case the target grade for copper and zinc in the processing plant would be increased to 3.6 % and decreased to 0.2 %, respectively, then the deviation for both scenarios would be lower.

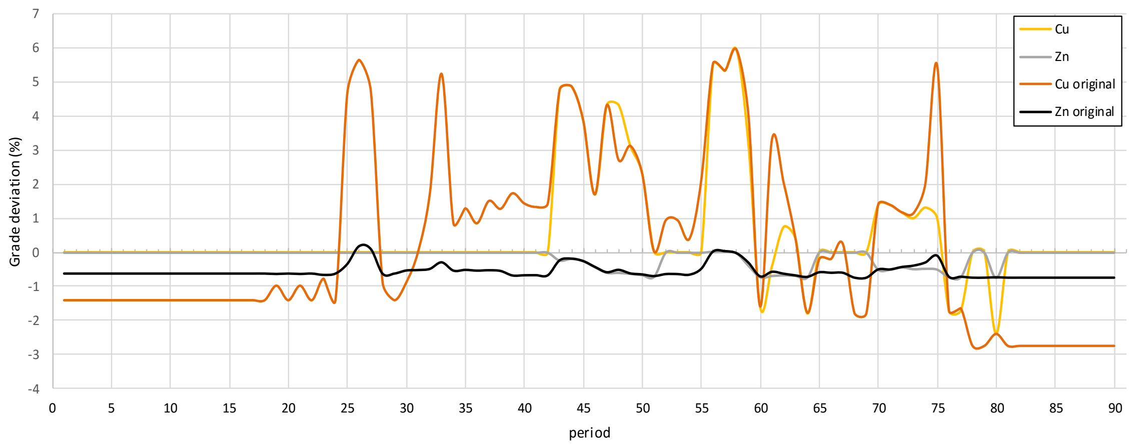


Figure 7-3: Comparison of the grade deviation from including and excluding development and non-extraction period deviations

7.3 Extraction volume analysis

The average grade of each period is calculated with the weighted averages of all contributing rock per period. Figure 7-4 shows the amount of extracted rock per period. In periods with extraction, the extraction rock much more contributes to the total extraction volume than the rock from development. From period one until 42 there is an average of 430 m^3 rock per period what only results from development. From period 43 onwards, the extraction volume increases and there is an average of 3700 m^3 per period. The model does not consider extraction volume deviation and therefore some big differences in extraction volume are observed. For example, during production, it should be tried to spread some of the rock from period 73 until 75 over multiple periods to lower the peaks. Normally, ore can be stockpiled before it enters the processing plant, what also helps providing a constant volume feed for the processing plant.

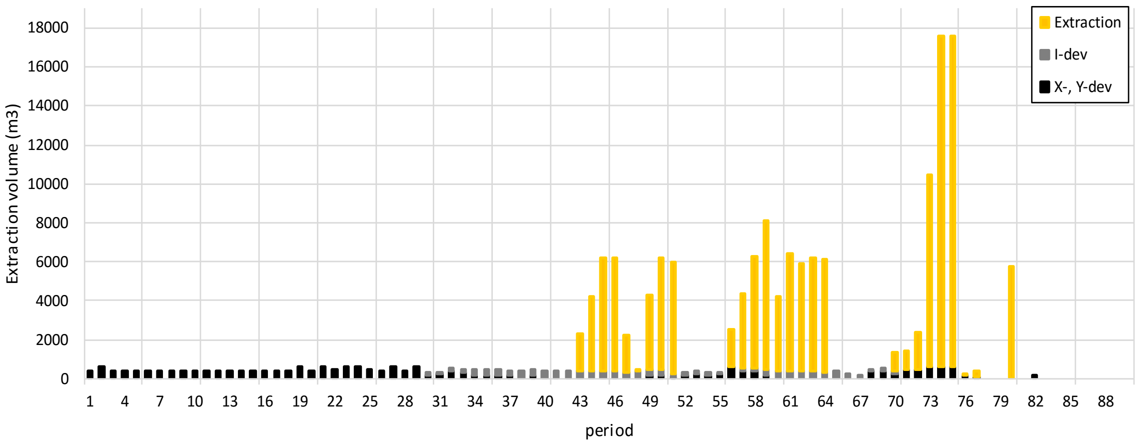


Figure 7-4: Extraction volume of X- and Y-development, internal development and extraction per period

8

Validation

In the following chapter, the short-term optimization model is validated to confirm the model's function and the generated production schedules. Several tests are conducted to show the functionality of individual constraints. First these tests, should confirm the effect of including and excluding constraints and eventually validate the constraints and model. Afterwards, several tests should confirm the flexibility of the short-term optimizer model and show that the model is not only constructed for the described case in this report, but also works with different scenarios.

8.1 Constraint validation

Constraints are applied such that they can enforce certain characteristics in the model. This implies that without these constraints the model has more freedom. Exclusion of one constraint can confirm the functionality for which it is designed and the effect of an exclusion should be seen in the schedule. Table 8-1 shows an overview of the effect of excluding specific constraints.

The solving time for each period takes considerable time. It is found that the half year three model without the existing fillmass constraint can be solved within considerable less time (approximately one hour). Therefore, this adapted model and its solution is used as a reference for validating the functionality of constraints. The schedule for this reference scenario (also called original) can be seen in Appendix E Figure E-1 and can be used as comparison schedule for the shown schedules. This schedule is not entirely good, because normally the existing fillmass constraint prevents stope 388, 402 and 413 to be in production simultaneously. This is not allowed because these stopes share a common boundary with a fillmass. Furthermore, all production control constraints are maintained and the model is valid and suitable for validation.

Each constraint has been separately validated with the above-mentioned reference scenario. The existing fillmass constraint has been validated with a model where no limitation of the capacity was included. The results confirmed the hypothesis of the effect of exclusion. Some of the validations are shown in the following subchapters.

Table 8-1: The effect of constraint exclusion on the model

Constraint	Effect of exclusion
Deviation	While creating the schedules, the model does not consider decision variables which have impact on the objective function. Therefore any feasible solution is shown, regardless the amount of grade deviation.
Reserves	There is no need to complete any of the activities and therefore the model can either do less or more work than required per scheduled activity.
Capacities	The capacity per item and machine is not limited and therefore an activity can be completed within one period.
Sequencing	When the sequencing constraint is not included, the model does not consider the natural transition of the mine operation. That is, all activities can be placed in random periods in the schedule.
Continuous activity	The continuation of activity does not occur in periods directly following each other. It might be possible for example, that there are big delays in the moment of extracting the ore of one blast. This constraint is also required for the adjacency and open face indication of a stope. This functionality is therefore also missing and the schedule cannot anymore guarantee that there are more than four open stopes or that there are activities at two adjacent stopes.
Commencement period and single stope start	The open stope indication does not work without commencement indication. Without the single stope start constraint, it is possible that there are multiple commencement indications. For both it means that the adjacency constraints will not function well.
Adjacency	It is possible that two stopes which share a common boundary are in production simultaneously.
Fillmass and existing adjacency	It is possible that a fillmass has two adjacent open stopes, what in practice would mean that the centred fillmass would become unstable, because all stresses will be centred on the fillmass.
Four open faces	There is no limitation on the number of open faces.
Variables linking	The entire functionality of the model is mixed-up and none of the constraints will do its function.

8.1.1 Deviation

The objective function of the model is based on the a_p and b_p variables. These variables only appear in the deviation constraint. Without this constraint, it can be validated that the model would give a better solution regarding the objective function and that a better-optimized schedule will be obtained.

A comparison of the deviation from the target grade, with the deviation constraint and without the constraint can be seen in Figure 8-1 and a summary in Table 8-2.

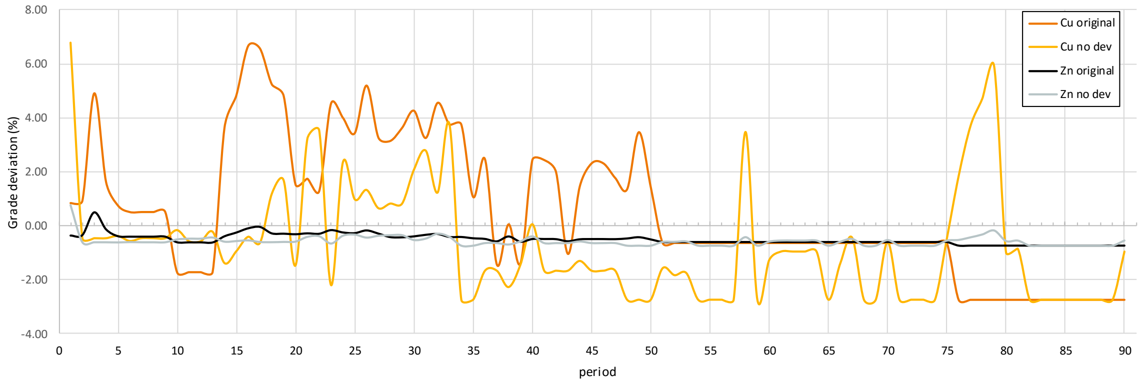


Figure 8-1: Comparison of the grade deviation from including and excluding the deviation constraint for half year 3

The fluctuations for copper with the deviation constraint are bigger, because the average absolute deviation is 2.08 %, however, the non-absolute deviation shows that there is only 0.57 % deviation towards the target grade, hence the average Cu grade is 3.32 %. Without the deviation constraint, an average copper percentage of 3.45 % is obtained. This high grade seems better, because it is a higher average Cu grade for the processing plant, however it should be remembered that it is tried to obtain an average grade near the target grade (2.75 %). For zinc, the compared grades show that better values are obtained from the model with the deviation constraint although every obtained zinc grade is still under the target grade. Therefore, this model can obtain a better schedule regarding the copper and zinc deviation than a model without the deviation constraint. It can be concluded that the deviation constraint works and that the average Cu and Zn grades are closer to the target grade. The model could bring the average Cu and Zn grade 0.13 and 0.08 percentage points, respectively, closer towards the target grade, which is a useful feature for the processing plant.

Table 8-2: Summary of the average grades from including and excluding the deviation constraint for half year 3

	Average absolute deviation (%)	**Average deviation (%)**	**Average grade (%)**
Cu original	2.08	0.57	3.32
Cu no dev	1.87	0.70	3.45
Zn original	0.52	-0.51	0.24
Zn no dev	0.60	-0.59	0.16

8.1.2 Capacity

Figure 8-2 shows the results from excluding the periodical capacity. There are no stopes which have a common boundary. Since the existing fillmasses are not considered, it is possible that all stopes are extracted at the same moment. The schedules show that all activities are scheduled in one period, because there is no limitation on the amount of activity. The sequencing constraint is also not applied correct. The way this constraint is constructed, defines that normally never I-dev, drilling, extraction or backfilling can occur in the first period, because this is prevented by the capacity. However, since suddenly all internal development can be completed at once, drilling can take place in the first period, hence all other activities, too. Future improvements can work around this issue, by designing the constraint differently. However, since this problem normally does not occur, it is not necessary for this model. Despite this fact, it is validated that the capacity constraint works for the model and that it constraints the amount of activity per period.

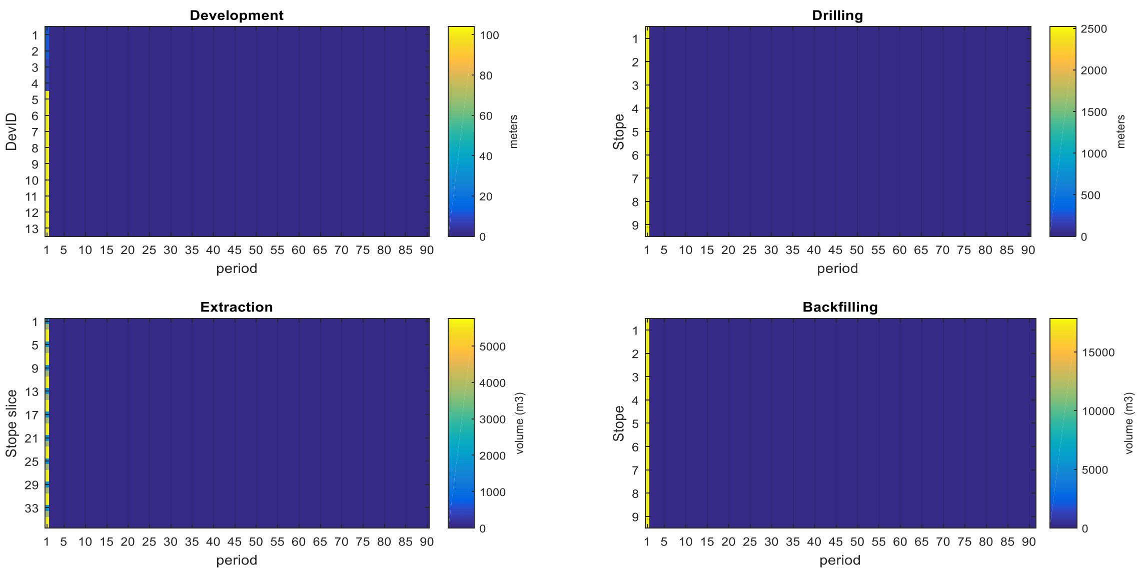


Figure 8-2: Schedules for half year three without capacity constraints

8.1.3 Sequencing

Validation of the sequencing constraint can be seen in Figure 8-3. It shows that each activity is randomly placed and confirms the hypothesis from Table 8-1. For example, stope 504 is drilled before the development drift has been constructed and eventually also backfilled before the stope is extracted. This schedule behaviour proofs that the sequencing constraint works regarding its designed performance and therefore is validated.

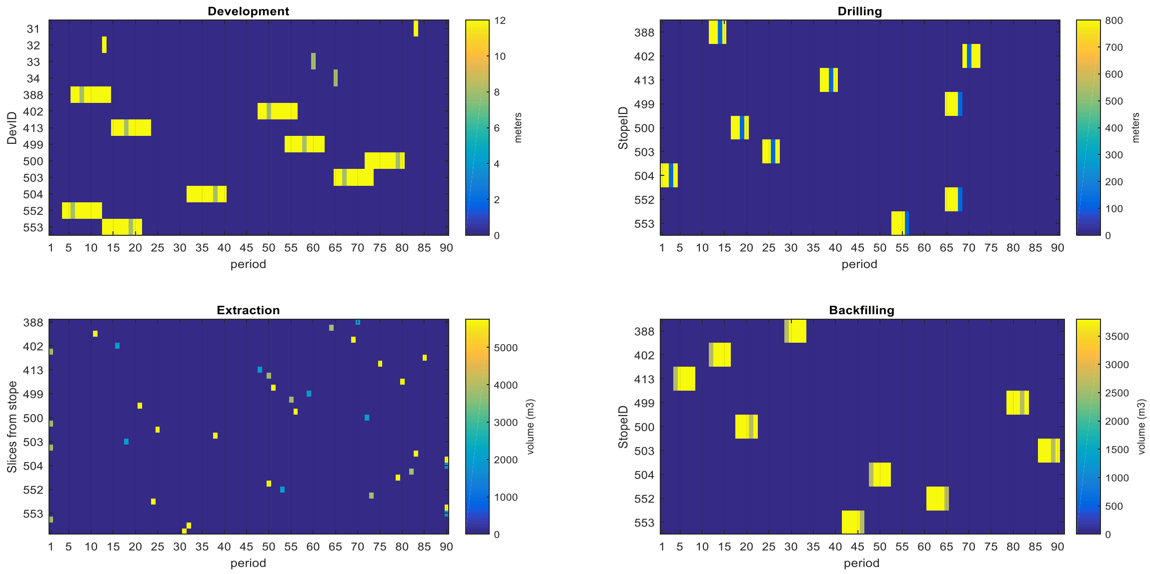


Figure 8-3: Schedules for half year three without sequencing constraints

8.1.4 Continuous activity

The schedules in Figure 8-4, show the exclusion effect of the continuous activity constraint. The activities per item are scattered over different periods, and they do not follow up each other, so the activities cannot be considered continuous anymore. Without the continuous activity constraint, the model is not possible to correctly indicate whether a stope is in production or not. There are stopes which start drilling around period 15 and finalize backfilling around period 90. That means that the entire stope should be considered as an open face during this time. The scattering effect causes that the commencement variables do not function well. This enables the model to schedule two adjacent stopes simultaneously or have more than four open faces. Based on these observations is the functionality of the constraint validated.

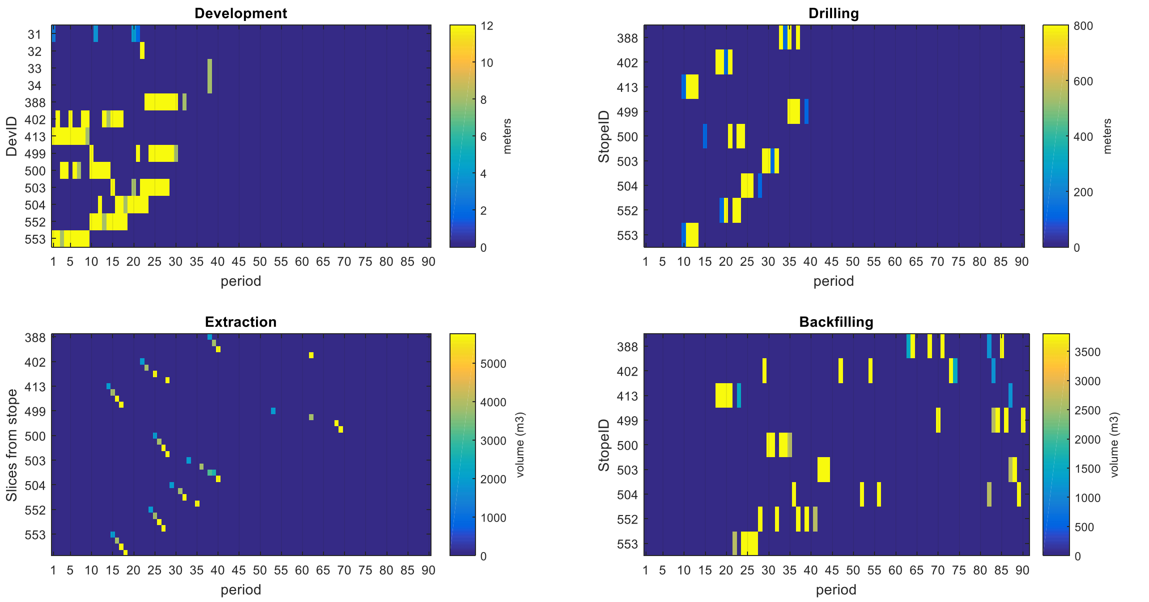


Figure 8-4: Schedules for half year three without continuous activity constraints

8.2 Capacity analysis

The model assumes fixed capacities for each period, but in practise it is known in mines at which moment planned maintenance takes place. This could reduce the overall capacity for certain periods, because a machine is not available. This feature is tested in the model by a manual adjustment of the capacity in a period to one. The capacity is not changed into zero, because as this is not aligned with the model. For example, if drilling takes three periods and the capacity of the drilling machine (number of meters it can drill) in one of these periods is reduced to zero, then the functionality of the model will collide with the continuous activity constraint and the model would give no solution. This is the case as the constraint is constructed like that. By changing the capacity to one, this constraint can still work and it can choose to have a capacity of one in the period of maintenance. In fact, this would mean no activity and confirms the reduced capacity.

The operator's working efficiency also influences the periodical capacity. It could possible increase or decrease the capacity. An increase in capacity means that less time is required to finish the activity, what is an improvement. However, a decrease in capacity (or productivity) means that there is a delay and which influence the schedule negatively. Several tests have been carried out to test the flexibility of the capacity. With a 10 % decrease in the periodical capacity of development, drilling, extraction or backfill the model was still able to schedule the planned development and stopes.

This confirms that the model can be used with different capacity input parameters. It also proved that development is probably the most schedule determining activity. Since the solving time was the longest and a slow development progress in the mine means that there is little time for all stope activities, what tightens the schedule. The reduced development capacity schedule, finish stope development in period 41. This is nine periods later than it is finished in the original model.

8.3 Scheduling horizon analysis

The results shown before in this report are mainly for the first half year. During this period, many development activities must be carried out, which does not apply to the following periods. However, the model should be able to schedule any set of input parameters, hence the other half years. In the third period, it is for example possible that internal development of certain stopes starts directly. In models with a reduced amount of constraints, it is validated that the different starting scenarios for each half year are plausible and work with the model. However, the schedules for period two, three and four for the entire model, as constructed in this research, could not be solved. There were still no results obtained after more than a week of solving.

The hypothesis behind this problem is that since these periods are less constrained there are more possibilities for different schedule combinations. The problem is that all these combinations must be evaluated and an integer solution must be found. This is not successful, because it is very difficult to comply with all binary constraints, see also Chapter 9.2. More than half of the decision variables are binary and are as well linked to a continuous variable. This makes it extremely difficult to find the right combination of values which comply with all implied constraints. CPLEX uses a systematic way to find a solution and since it this tries to minimize the objective function the program starts at the lowest value of the objective value and gradually works towards higher values. The lower values of the objective value correspond with schedules with relatively small deviations, so it reasonable to first search for a solution in this region. The higher values of the objective value correspond with robust and probably "high deviation" schedules. It requires a lot of time before the model is in this region and can search for a feasible solution. Unfortunately, this heavy computational effort did not allow it to obtain solutions for each half year.

8.4 Summary

Validation of the constraints showed that the model is verified to the degree that is needed for the model's intended purpose. Based on the validation of the constraints it can be concluded that the model works and is suitable for different input parameters. The model can consider different start scenarios, regarding stopes and development work. This implies that the model could resemble the reality and schedule sublevel stoping operations.

9

Discussion

The model characteristics and practical considerations, which were used to represent a sublevel stoping operation with backfilling, will be critically reviewed in this chapter relative to real mining situations. However, the assumptions and results can be explained with underpinnings to make a clear conclusion for this research.

9.1 Model characteristics

The grade deviation constraint implies that a processing plant considers a combined copper-zinc concentrate, because copper and zinc are both represented in the same decision variables. Normally, copper and zinc are separated in a two-stage flotation process and produces a separate copper and zinc concentrate with separate production target grades. From a processing plant perspective, it would therefore be better if both copper and zinc were treated separate in the objective function and each of them obtains individual variables. The approach for constructing the deviation constraint in this research is still valid. The a_p and b_p variables are used for both copper and zinc and result in four different terms. Therefore, each term (Cu above, Cu below, Zn above, Zn below) has an individual effect on the objective value and thus their individual values are tried to be minimized. Additionally, copper and zinc occur in the same rock and therefore should be treated jointly.

The short-term scheduler used in this report has shown its capabilities of scheduling activities for two days (four shifts). This shows maturity for weekly scheduling and that it is suitable for mines to plan production. If plans per shift or sub-shift are required, it is possible to go in the so-called ultra-short-term scheduling. XECUTE from RPM Global is a software package that can operate with the granularity of short-term schedules and it is able to plan each shift. Thereby it can consider much more different activities (RPM Global, 2017). When the scheduling horizon becomes shorter, real-time mining information will have influence on schedules. Real-time feedback in scheduling is used for different goals. It is used to adapt the mining location according to the natural spatial distribution of low- and high-grade zones and for real-time indicating differences in ore and waste and changing the destination of each load. In case the period duration considered in this research would represent one shift or day, it should still resemble the schedules obtained in this research. The shorter the period duration is, it will make more sense to implement additional production phases, because the moment of extraction is more accurate. Therefore additional activities like charging, scaling etc. would influence this moment of extraction and real-time production could never be based on the generated schedule.

9.2 MILP solving time analysis

The models for each half year took very long to solve. A similar model from Nehring (2010) with ten stopes has considerable less variables, but much more constraints. However, the solution time is in the range of seconds.

The time for solving a MILP model is highly depending on the number of binary variables, but there is no reasonable possibility to measure the difficulty of a MILP. CPLEX uses a branch-and-bound method to find the optimal solution (ILOG, 2017). For this method, a search tree is constructed in which each node contains a certain number of variables which are set and others which are used (and changed) to search for an integer solution. CPLEX solves a linear programming relaxation at each node of the branch-and-bound tree. Increasing the number of variables increases the amount of possibilities for new nodes and enlarges the branch-and-bound tree. The effectiveness of the branch-and-bound algorithm depends on its ability to prune nodes so that eventually a solution can be found. The large number of variables and constraints drastically increases the time for solving the given model (Klotz & Newman, 2013). It could be discussed whether the proposed model is formulated in the correct way. There are chances that several constraints can be formulated differently, to reduce the solution time. Especially new constraints which disregard the use of the commencement variables would drastically increase the performance of the model. In that case less binary variables and shorter solving times are expected. However, much effort is made to construct different constraints without commencement variables, but without positive result. For this research, it was suitable to construct any constraint formulation which would perform its functionality and therefore no further effort is made to improve formulations since solutions are obtained.

9.3 Practical considerations

There is still flexibility in the generated schedules. In case there are any adjustments made in the schedule it will change the grade deviation, however, it could better resemble true underground mining conditions. The capacities for each phase are adequate to have enough capacity to complete the activities in a reasonable time span. This implies that the model is applicable to implement more constraints.

The adjacency constraint does not consider internal stope development activities at adjacent stopes. If there were an open stope adjacent to a stope where internal stope development activities take place, then a blast would have significant impact on geotechnical stability of the area. Considering this situation, it would mean that only four meters of rock are between the internal development drift and open stope. The danger of falling rock after a blast in the development drift is severe and unsafe working conditions would appear in the open stope.

Drifts from underground mines have restricted sizes and therefore it is important to consider the traffic behaviour at haulage drifts. It is often avoided to have production at two open faces close to each other, because LHD's are often not able to pass each other in drifts. That means that a LHD must wait for the other to pass and results in lower productivity. To consider and account for these situations the model needs much more information and constraints to schedule loading and hauling. However, this would be a scheduling task on its own. If traffic considerations were implemented, such as one stope extraction activity per haulage drift, the schedule would not be very different. This is because the extraction activity in this research only represents an amount of volume mined from the stope and the extraction capacity is the only restriction. Therefore, it does not consider the number of extraction machines or destinations. Additionally, in practice many more machines are driving in the drifts like, drilling jumbos, personnel cars and other support machines.

A defined out of scope topic, which is though important, is that backfill curing time is not included in the model. This is done to simplify the model and it was not needed to show the application of this model. It is expected that implementation of a backfill curing constraint would shift the commencement periods of stope activities from stopes which are adjacent to each other. While curing, it is not possible to do blasting and extraction activities at an adjacent stope, because possibility exists that backfill pours into the adjacent stope. Therefore, this constraint would remarkably change the production schedule and better represent the true underground mining conditions. Implementation of this constraint would probably tighten the schedule of half year one and some backfill must take place after the scheduling horizon and continues in the second half year. During half year three and four, hardly any development takes place and during these scheduling horizons, implementation of the backfill curing constraint would not give problems and all stopes can be completed within the scheduling horizon.

There are hard boundaries set by the long-term optimizer that certain stopes must be finished within a certain half year. Considering this model, it means that the last period of extraction occurs a few periods before period 90 to ensure enough time for backfilling. The next half year continues from the end of the previous half year and often requires several periods before extraction can commence. To ensure a constant throughput for a processing plant it would be better if this “no-extraction-time” would be as short as possible. A more flexible boundary for the moment that stopes should be mined would reduce this no-extraction-time and a more constant ore flow would occur.

Sublevel stoping is the considered mining method for this research. It is tried to incorporate the characteristics of this mining method on scheduling as much as possible and is mainly done by the stope input characteristics. If a different stope input is given with, for example dimensions of a drift and fill operation, it should be possible to adapt the stope slice formulation and resemble the operational mining procedure of drift and fill. Many of the constraints like reserve and adjacency will still be applicable and can therefore be used for many mining methods.

The model in this research could indicate the periods in which each activity takes place during the stope life. It can be discussed whether this is still very useful to know in mines which are already longer in production. Many operational mines have prepared stopes (internal development and drilling) already a long time before the stope starts the extraction phase. Therefore, many stopes are ready for production and the most important decision is to select the combination of stopes which results in the best average grade throughput for the processing plant. However, in the early stages of the mine these mines used a similar scheduling optimization to schedule the production. The mining operation used in this model can either represent an operation in a new mine or a new mining area of an already existing mine. This makes the extend of this model applicable and useful since it is always necessary and important to schedule the entire stope life.

10

Conclusion

This research project focused on optimizing and developing a short-term schedule for an underground sublevel stoping mine based on a long-term plan. The research objectives outlined in Chapter 2 have been achieved with the optimization model and the model can help planning engineers with creating short-term schedules. The following conclusions are drawn from the implementation of the MILP model framework for short-term sublevel stoping production scheduling.

- The presented MILP model can generate production schedules for the stopes and development drifts assigned by a long-term optimization framework. It schedules development, drilling, extraction and backfill activities. Thereby, it can work with different requirements for completing each activity and an extra selection can be made on the amount of blasts required to mine all ore in a stope.
- The model can apply production control constraints which resemble true sublevel stoping mining situations, like (machine) capacities, reserves, activity sequencing, adjacent stopes and existing fillmasses.
- The optimization model can generate short-term production schedules which consider all above-mentioned characteristics and the processing plant objectives of a short-term schedule. It can minimize the periodical grade deviation of copper and zinc towards the target grade for the input of the processing plant.
- It was possible to successfully schedule 23 development drifts and nine stopes over 90 periods for the first half year, in which one period resembles two days or four shifts.
- The MILP model provides a flexible framework in which it is possible to adjust input parameters and periodical capacities to adjust to different mining conditions.
- A comparison of two schedules, in which one used the deviation constraint and one did not, confirmed that the optimizer can obtain a better-optimized short-term schedule regarding grade deviation than a model which does not consider the production target deviation. In the case of the first half year it meant that this improvement brings the average Cu and Zn grade 0.13 and 0.08 percentage points, respectively, closer towards the target grade.

- The model can be used by a planning engineer to construct a production schedule which can be used for detailed scheduling of the work per shift. It is possible to implement more characteristics of the operation since it is shown that the model can deal with sequential activities.
- Validation of the model has proven that the model’s solutions provide valid schedules. Each constraint is therefore individually verified. By using different input parameters, it is proven that the model can be used for each half year.
- The model shows possibility to extend and improve constraints which improves the resemblance of a real mining operation.

11 Recommendations

The model as developed for this research showed positive results and can be used for detailed short-term scheduling. However, it is not ready to be implemented in industry as it is. Further work and research needs to be done to improve the functionality and utilization of this model. Several recommendations for further research are listed below.

- An ultimate and final goal of a short-term optimizer is to have an integrated system with long- or medium-term optimizers to ensure the long-term and short-term capabilities with each its objectives. This research used a fixed input from the long-term optimizer and therefore no interaction between specific short- or long-term considerations is made possible. In future research, the two optimizers could be combined such that the long-term optimizer searches for the combination of stopes that give the highest economic value and the objective of the short-term optimizer is used to select the best combination of stopes for a constant feed grade for the processing plant.
- Additional phases, like charging, scaling, ventilation or bulkhead construction could be implemented in the model in order to schedule these activities and to have a more detailed short-term schedule. Underground drifts have limited space and often limited amount of equipment can be present per drift. These activities require additional machines and equipment, thus no other activity can take place simultaneously. This will delay the time moment that extraction or backfill can commence and is thus important to know. Another way to include this, is by including a minimum number of periods for an activity to ensure time capacity. It is proven that the model can consider sequential activities in a mine. This would also be the case for these additional activities and thus the constraint formulations could be extrapolated to also represent the additional activities.
- As addressed in the discussion, during the deviation calculation the deviation for copper and zinc is combined. For a more constant throughput for both copper and zinc the constructed deviation constraint should be separated in two parts, respectively copper and zinc. However, this requires an additional number of variables of two (above and below) times the number of periods.

- Apart from copper and zinc grade deviation for the processing plant, effort could be made on minimizing the deviation of rock volume for the processing plant. A constant throughput for the processing plant is desired, because it is easier to control the process. Additional constraints could be constructed to ensure this feature.
- The optimization model developed for this research is written in MATLAB. MATLAB's programming language is versatile and extensive documentation and large databases of functions exist. For industrial use, MATLAB is not the most efficient and sophisticated programming language and Python or C++ coding have proven to be more applicable in industry (Aruoba & Fernández-Villaverde, 2013). The level of code writing of the produced scripts can be classified as intermediate. Adjustments in the code by a professional could improve the efficiency of this code and make it suitable for a broader application.
- Results demonstrated that in certain periods the deviation from production target grade is severe. Future improvements could include stockpiles which can be used to blend (stored) lower grade ore with high grade ore, or vice versa. This is normally done in practice and therefore implementation of this better features true processing situations.
- It is several times indicated that there are different flaws in the solution from the long-term framework which have influence on the results of this research. During resource modelling a blockmodel is constructed which contains copper and zinc grades. However, there are doubts about the certainty and extend of this generation. There are many blocks which contain no copper or zinc grade although there is high potential that it should be in a region with ore. Consequently, the average grade for each development and stope item is only based on a few blocks and therefore does not perfectly represent the true grade that will be obtained after mining. The results of the long-term optimizer have been influenced by this lack of information and consequently the results from this research as well. Effort should be made on updating the information from the long-term optimizer if the proposed schedule from this research will be implemented in industry.

12

Summary

In pursuing this research, the literature indicated limitations in the current body of knowledge in optimization of short-term schedules of a sublevel stoping operation with backfilling. Often different aspects of short-term scheduling are considered, for example fleet allocation. To start operation on a new mining area it is essential to know when each stope activity required for a natural sequential transition of a stope will start. The model in this research can indicate this and it is based on a real dataset. The considered activities consist of three different types of development respectively X-, Y- and internal-development and additionally drilling, extraction and backfilling.

Figure 12-1 shows a summary of the workflow of this research. A long-term optimizer defined in which half year specific development drifts and stopes must be mined. This is used as input for the MILP model in this research. The model makes decisions which improve the constant grade throughput of a processing plant by minimizing the grade deviation from copper and zinc resulting from all ore in each period. There are 13 different constraints constructed and applied in the model. Several geotechnical and production control constraints, such as reserves, the amount of open faces, fillmass stability and adjacent activities from stopes with common boundaries are implemented to represent real mining situations.

The objective of this research was to develop a flexible short-term scheduler which can validate long-term planning schedules, considering short-term constraints, objectives and targets. This task has been achieved successfully. This is concluded after the model's application is analysed. It showed that this model can find a better-optimized short-term schedule regarding grade deviation than a schedule which does not consider the production target deviations. The model is validated and able to verify the long-term planning schedule, because it can plan all required stopes and development in the pre-determined time. Therefore, a planning engineer can construct a production schedule in considerable time, which can be used for detailed scheduling of the work per shift.

Furthermore, it is discussed why certain solutions were not achieved and what a practical analysis of the results implies. It can be concluded that several constraints could be extended to better represent mining conditions. For the extend of this research it was sufficient to use the constraints as constructed. However, future research could improve the application of this optimizer by increasing the amount of grade deviation variables and by integrating with a long-term optimizer. Small adjustments in the input parameters should make it possible to use the same model for mining methods others than sublevel stoping. Additional improvements, which will result in a better production control could be done by improving the resource blockmodel, extending the adjacency constraints with development and including stockpiles and backfill curing time.

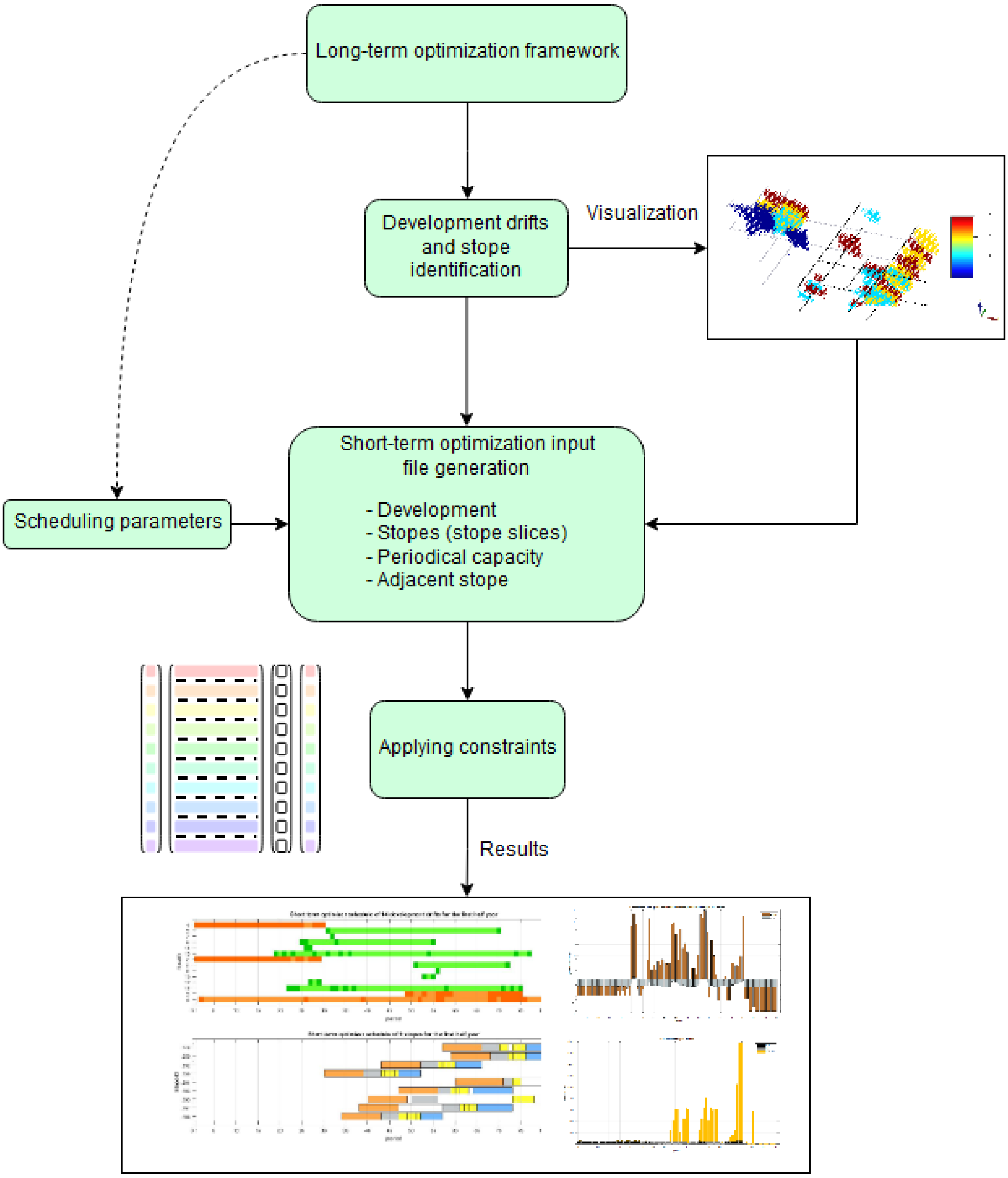


Figure 12-1: Summary of the research method

Bibliography

Alford, C., Brazil, M., & Lee, D. (2007). Optimisation in underground mining. In A. Weintraub, C. Romero, T. Bjørndal , R. Epstein, & J. Miranda, *Handbook Of Operations Research In Natural Resources* (Vol. 99, pp. 561-577). Springer US. doi:10.1007/978-0-387-71815-6_30

Aruoba, S., & Fernández-Villaverde, J. (2013). A Comparison of Programming Languages in Economics, Working Paper No. 20263. *National Bureau of Economic Research*, 19.

Ataee-pour, M. (2005). A critical survey of the existing stope layout optimization techniques. *Journal of Mining Science, 41*(5), 20.

Datamine. (2017). Retrieved June 27, 2017, from http://www.dataminesoftware.com/

Dowd, P., & Elvan, L. (1987). Dynamic programming applied to grade control in sub-level open stoping. *Trans. IMM, 96*, A171-A178.

Emad, M., Mitri, H., & Kelly, C. (2015). State-of-the-art review of backfill practices for sublevel stoping systems. *International Journal of Mining, Reclamation and Environment*, 544-556.

Hamrin, H. (1980). *Guide to underground mining methods and applications.* Stockholm, Sweden: Atlas Copco.

Hartman, H., & Mutmansky, J. (1987). *Introductory Mining Engineering.* New York: Wiley.

Hustrulid, W. (1998). *Underground Mining Methods.* Littleton, Colorado: Society for Mining, Metallurgy, and Exploration.

ILOG, C. (2017). CPLEX, MATLAB extension, Version 12.7.1.

Klotz, E., & Newman, A. (2013). Practical Guidelines for Solving Difficult Mixed Integer Linear Programs. *Surveys in Operations Research and Management Science*, 37.

Lawrence, B. W. (1998). Considerations for sublevel stoping. In R. E. Gertsch, & R. L. Bullock, *Techniques in Underground Mining: Selection From Underground Mining Methods Handbook* (pp. 227-247). Littleton: Society for Mining, Metallurgy, and Exploration, Inc.

Lerchs, H., & Grossmann, I. (1965). Optimum design of open pit mines. *CIM Bull, 58*, 47-54.

MathWorksInc. (2017). MATLAB (R2015b). Ver. 8.6.0.267246.

Matthäus, A. (2015). Underground Mine Planning. *Real-time optimization of extraction and the logistic process in highly complex geological and selective mining settings: Real-Time Mining*, 5-10.

Nehring, M., & Topal, E. (2007). Production schedule optimisation in underground hard rock mining using mixed integer programming. *Proceedings Project Evaluation*, 169-175.

Nehring, M., & Topal, E. (2009). A new short-term production scheduling and machine allocation model using mixed integer programming for application in sublevel stope mining operations. *Proceedings of the 2009 Australian Mining Technology Conference* (pp. 116-133). Brisbane, Australia: Australian Mining Technology Conference.

Nehring, M., Topal, E., & Little, J. (2010). A new mathematical programming model for production schedule optimization in underground mining operations. *The Journal of The Southern African Institute of Mining and Metallurgy, 110*, 437-446.

Nehring, M., Topal, E., Kizil, M., & Knights, P. (2012). Integrated short- and medium-term underground mine production scheduling. *The Journal of The Southern African Institute of Mining and Metallurgy, 112*, 365-378.

Osanloo, M., Gholamnejad, J., & Karimi, B. (2008). Long-term open pit mine production planning: a review of models and algorithms. *International Journal of Mining, Reclamation and Environment, 22*(1), 3-35. doi:10.1080/17480930601118947

O'Sullivan, D., & Newman, A. (2015). Optimization-based heuristics for underground mine scheduling. *European Journal of Operational Research*, 248-259.

Pochet, Y., & Wolsey, L. (2006). *Production Planning by Mixed Integer Programming* (Vol. 1). New York: Springer-Verslag New York. doi:10.1007/0-387-33477-7

Pourrahimian, Y. (2013). *Mathematical programming for sequence optimization in block cave mining.* University of Alberta, Department of Civil and Environmental Engineering, Edmonton, Alberta.

RPM Global. (2017). *XACT*. Retrieved June 27, 2017, from http://www.rpmglobal.com/software/xact/

SGeMS. (2017). SGeMS. Retrieved July 5, 2017, from http://sgems.sourceforge.net/

Sloane, L. (2010). *Sub-level open stoping: Design of the O640, L651 and N659 sub-level open stopes in the 3000 orebody of the Mount Isa Copper Mines, Queensland, Australia.* University of Pretoria.

Song, Z., Rinne, M., & van Wageningen, A. (2013). A review of real-time optimization in underground mining production. *The Journal of The Southern African Institute of Mining and Metallurgy, 113*, 889-897.

Tatiya, R. R. (2013). Underground mining/stoping methods & mine closure. In R. R. Tatiya, *Surface and Underground Excavation: Methods, Techniques and Equipment* (2nd Edition ed., pp. 527-684). London: Taylor & Francis Group plc.

Tien, J. (1999). *Practical Mine Ventilation Engineering.* Intertec Publishing.

Trout, L. (1995). Underground mine production scheduling using mixed integer programming. *Proceedings of the 25th International APCOM Symposium* (pp. 395-400). Melbourne: The Australasian Institute of Mining and Metallurgy.

Tunnelbuilder ltd (Atlas Copco Rock Drills AB) . (2007). *Mining Methods in Underground Mining.* Örebro, Sweden: Welins Tryckeri AB.

Villaescusa, E. (2004). Quantifying open stope performance. *MassMin2004*, (p. 11).

Villaescusa, E. (2014). *Geotechnical Design for Sublevel Open Stoping.* Florida: Boca Raton: CRC Press. doi:10.1201/b16702

Whittle, J. (1989). The Facts and Fallacies of Open Pit Optimization. *Whittle Programming Pty Ltd.* North Balwyn, Victoria.

Appendix A

Table A- 1: Overview of all designed stopes resulting from the long-term optimizer
Note: It is indicated in which half year they must be mined.

X	Y	Z	sizeX (m)	sizeY (m)	sizeZ (m)	density (kg/m^3)	Cu (%)	Zn (%)	Half year
4736	2758	660	36	12	40	3	1.76	0.08	1
4772	2738	660	36	12	40	3	6.67	0.55	1
4772	2758	660	36	12	40	3	8.02	0.66	1
4772	2770	660	36	12	40	3	6.22	0.46	1
4844	2666	660	36	12	40	3	0.05	0	1
4844	2690	660	36	12	40	3	0.51	0.02	1
4844	2702	660	36	12	40	3	1.51	0.05	1
4844	2714	660	36	12	40	3	3.19	0.08	1
4844	2726	660	36	12	40	3	4.70	0.09	1
4808	2758	660	36	12	40	3	7.78	0.47	2
4808	2782	660	36	12	40	3	6.48	0.35	2
4874	2678	588	24	12	40	3	3.70	0.28	2
4874	2714	588	24	12	40	3	5.46	0.29	2
4880	2902	588	36	12	40	3	0.60	0.19	2
4952	2702	588	36	12	40	3	3.09	0.30	2
4952	2758	588	36	12	40	3	2.34	0.24	2
4988	2702	588	36	12	40	3	2.30	0.26	2
4988	2714	588	36	12	40	3	2.30	0.26	2
4988	2770	588	36	12	40	3	1.49	0.21	2
4772	2782	660	36	12	40	3	5.36	0.37	3
4808	2770	660	36	12	40	3	7.50	0.42	3
4808	2794	660	36	12	40	3	6.01	0.29	3
4988	2690	588	36	12	40	3	1.47	0.18	3
4988	2726	588	36	12	40	3	2.04	0.21	3
4988	2758	588	36	12	40	3	1.05	0.15	3
4988	2782	588	36	12	40	3	1.91	0.18	3
4988	2830	588	36	12	40	3	5.19	0.18	3
4988	2878	588	36	12	40	3	2.03	0.16	3
4772	2806	660	36	12	40	3	4.25	0.20	4
4808	2806	660	36	12	40	3	6.14	0.25	4
4880	2702	588	36	12	40	3	5.27	0.25	4
4868	2738	588	12	12	40	3	5.01	0.23	4
4880	2830	588	36	12	40	3	5.15	0.18	4
4952	2714	588	36	12	40	3	1.23	0.11	4
4952	2770	588	36	12	40	3	1.60	0.13	4
4988	2738	588	36	12	40	3	1.32	0.13	4
4988	2806	588	36	12	40	3	3.44	0.14	4
4988	2854	588	36	12	40	3	2.40	0.14	4

Table A- 2: Overview of all development drifts defined by the long-term optimizer
Note: Every first row defines the X, Y and Z start coordinate of the development drift and the second rows represent the X, Y and Z end coordinates.

X	Y	Z	Half year
5042	2750	870	1
5042	2750	608	1
5042	2750	608	2
5042	2750	536	2
5042	2750	682	1
4718	2750	682	1
5042	2750	638	1
4718	2750	638	1
4718	2750	682	1
4718	2744	682	1
4718	2750	638	1
4718	2744	638	1
4718	2750	682	1
4718	2764	682	1
4718	2750	638	1
4718	2764	638	1
4754	2750	682	1
4754	2732	682	1
4754	2750	638	1
4754	2732	638	1
4754	2750	682	1
4754	2776	682	1
4754	2750	638	1
4754	2776	638	1
4826	2750	682	1
4826	2632	682	1
4826	2750	638	1
4826	2632	638	1
5042	2750	610	1
4862	2750	610	1
5042	2750	566	1
4862	2750	566	1
4790	2750	682	2
4790	2744	682	2
4790	2750	638	2
4790	2744	638	2
4790	2750	682	2
4790	2812	682	2
4790	2750	638	2
4790	2812	638	2

X	Y	Z	Half year
4862	2750	610	2
4862	2672	610	2
4862	2750	566	2
4862	2672	566	2
4862	2750	610	2
4862	2908	610	2
4862	2750	566	2
4862	2908	566	2
4934	2750	610	2
4934	2696	610	2
4934	2750	566	2
4934	2696	566	2
4934	2750	610	2
4934	2776	610	2
4934	2750	566	2
4934	2776	566	2
4970	2750	610	2
4970	2684	610	2
4970	2750	566	2
4970	2684	566	2
4970	2750	610	2
4970	2884	610	2
4970	2750	566	2
4970	2884	566	2
4754	2776	682	3
4754	2788	682	3
4754	2776	638	3
4754	2788	638	3
4826	2632	682	3
4826	2624	682	3
4826	2632	638	3
4826	2624	638	3
4754	2788	682	4
4754	2812	682	4
4754	2788	638	4
4754	2812	638	4

Appendix B

Table B- 1: Development input file for the X- and Y-development of all half years

DevID	Xstart	Xend	Ystart	Yend	Zstart	Zend	Type*	Half year	length (m)	Cu (%)	Zn (%)
1	5042	4718	2748	2748	684	684	1	1	324	1.57	0.10
2	5042	4718	2748	2748	640	640	1	1	324	0.95	0.14
3	5042	4862	2748	2748	612	612	1	1	180	0	0
4	5042	4862	2748	2748	568	568	1	1	180	0	0
5	4718	4718	2748	2744	684	684	2	1	4	7.09	0.20
6	4718	4718	2748	2744	640	640	2	1	4	0	0
7	4718	4718	2748	2764	684	684	2	1	12	5.38	0.27
8	4718	4718	2748	2764	640	640	2	1	12	0	0
9	4754	4754	2748	2732	684	684	2	1	16	9.99	0.60
10	4754	4754	2748	2732	640	640	2	1	16	0	0
11	4754	4754	2748	2776	684	684	2	1	24	8.23	0.96
12	4754	4754	2748	2776	640	640	2	1	24	0	0
13	4826	4826	2748	2632	684	684	2	1	116	2.25	0.04
14	4826	4826	2748	2632	640	640	2	1	116	0	0
15	4790	4790	2748	2744	684	684	2	2	4	6.76	0.37
16	4790	4790	2748	2744	640	640	2	2	4	0	0
17	4790	4790	2748	2812	684	684	2	2	60	0.44	0.03
18	4790	4790	2748	2812	640	640	2	2	60	5.46	0.68
19	4862	4862	2748	2672	612	612	2	2	76	0	0
20	4862	4862	2748	2672	568	568	2	2	76	0	0
21	4862	4862	2748	2908	612	612	2	2	156	2.08	0.10
22	4862	4862	2748	2908	568	568	2	2	156	0	0
23	4934	4934	2748	2696	612	612	2	2	52	0.74	0.04
24	4934	4934	2748	2696	568	568	2	2	52	0	0
25	4934	4934	2748	2776	612	612	2	2	24	1.34	0.11
26	4934	4934	2748	2776	568	568	2	2	24	0	0
27	4970	4970	2748	2684	612	612	2	2	64	1.75	0.23
28	4970	4970	2748	2684	568	568	2	2	64	0.83	0
29	4970	4970	2748	2884	612	612	2	2	132	2.40	0.18
30	4970	4970	2748	2884	568	568	2	2	132	2.50	0.17
31	4754	4754	2776	2788	684	684	2	3	8	10.63	1.58
32	4754	4754	2776	2788	640	640	2	3	8	0	0
33	4826	4826	2632	2624	684	684	2	3	8	0	0
34	4826	4826	2632	2624	640	640	2	3	8	0	0
35	4754	4754	2788	2812	684	684	2	4	20	3.88	0.31
36	4754	4754	2788	2812	640	640	2	4	20	0	0

* type 1 = X-development, type 2 = Y-development

Table B- 2: Internal development input file from all half years, containing the average Cu and Zn grade for each stope from the I-dev

StopeID	Cu (%)	Zn (%)	Half year
248	2.51	0.18	1
250	3.05	0.15	1
252	2.25	0.36	1
256	2.07	0.26	1
291	0.96	0	1
293	3.68	0.11	1
295	3.41	0.08	1
297	4.00	0.08	1
298	4.87	0.03	1
318	3.40	0.50	2
321	3.22	0.34	2
326	0	0	2
329	0	0	2
331	0.66	0.46	2
333	2.36	0.25	2
335	0.69	0.15	2
378	0.16	0.02	2
382	0.10	0.02	2
384	0.11	0.03	2
388	1.20	0.16	3
402	3.98	0.44	3
413	2.61	0.31	3
499	1.27	0.15	3
500	0	0	3
503	0.54	0.08	3
504	1.08	0.10	3
552	3.10	0.11	3
553	1.42	0.09	3
554	0.36	0	4
555	2.43	0.31	4
556	0	0	4
557	0	0	4
558	2.32	0.07	4
559	1.25	0.16	4
561	0.83	0.22	4
564	0.34	0	4
568	2.23	0.11	4
572	1.14	0.13	4

Table B- 3: Development relation file which shows the amount of development in meters that must be completed before stope activities can take place for each scheduled period

Half year 1

Stope ID	DevID																						
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**I248**	**I250**	**I252**	**I256**	**I291**	**I293**	**I295**	**I297**	**I298**
248	324	324	0	0	0	0	12	12	0	0	0	0	0	0	104	0	0	0	0	0	0	0	0
250	288	288	0	0	0	0	0	0	16	16	0	0	0	0	0	104	0	0	0	0	0	0	0
252	288	288	0	0	0	0	0	0	0	0	12	12	0	0	0	0	104	0	0	0	0	0	0
256	288	288	0	0	0	0	0	0	0	0	24	24	0	0	0	0	0	104	0	0	0	0	0
291	216	216	0	0	0	0	0	0	0	0	0	0	88	88	0	0	0	0	104	0	0	0	0
293	216	216	0	0	0	0	0	0	0	0	0	0	64	64	0	0	0	0	0	104	0	0	0
295	216	216	0	0	0	0	0	0	0	0	0	0	52	52	0	0	0	0	0	0	104	0	0
297	216	216	0	0	0	0	0	0	0	0	0	0	40	40	0	0	0	0	0	0	0	104	0
298	216	216	0	0	0	0	0	0	0	0	0	0	28	28	0	0	0	0	0	0	0	0	104

Half year 2

Stope ID	DevID																									
	15	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**I318**	**I321**	**I326**	**I329**	**I331**	**I333**	**I335**	**I378**	**I382**	**I384**
318	0	0	12	12	0	0	0	0	0	0	0	0	0	0	0	0	104	0	0	0	0	0	0	0	0	0
321	0	0	36	36	0	0	0	0	0	0	0	0	0	0	0	0	0	104	0	0	0	0	0	0	0	0
326	0	0	0	0	76	76	0	0	0	0	0	0	0	0	0	0	0	0	80	0	0	0	0	0	0	0
329	0	0	0	0	40	40	0	0	0	0	0	0	0	0	0	0	0	0	0	80	0	0	0	0	0	0
331	0	0	0	0	0	0	160	160	0	0	0	0	0	0	0	0	0	0	0	0	104	0	0	0	0	0
333	0	0	0	0	0	0	0	0	52	52	0	0	0	0	0	0	0	0	0	0	0	104	0	0	0	0
335	0	0	0	0	0	0	0	0	0	0	12	12	0	0	0	0	0	0	0	0	0	0	104	0	0	0
378	0	0	0	0	0	0	0	0	0	0	0	0	52	52	0	0	0	0	0	0	0	0	0	104	0	0
382	0	0	0	0	0	0	0	0	0	0	0	0	40	40	0	0	0	0	0	0	0	0	0	0	104	0
384	0	0	0	0	0	0	0	0	0	0	0	0	0	0	24	24	0	0	0	0	0	0	0	0	0	104

Half year 3

Stope ID	DevID												
	31	**32**	**33**	**34**	**I388**	**I402**	**I413**	**I499**	**I500**	**I503**	**I504**	**I552**	**I553**
388	12	12	0	0	104	0	0	0	0	0	0	0	0
402	0	0	0	0	0	104	0	0	0	0	0	0	0
413	0	0	0	0	0	0	104	0	0	0	0	0	0
499	0	0	0	0	0	0	0	104	0	0	0	0	0
500	0	0	0	0	0	0	0	0	104	0	0	0	0
503	0	0	0	0	0	0	0	0	0	104	0	0	0
504	0	0	0	0	0	0	0	0	0	0	104	0	0
552	0	0	0	0	0	0	0	0	0	0	0	104	0
553	0	0	0	0	0	0	0	0	0	0	0	0	104

Half year 4

Stope ID	DevID											
	35	**36**	**I554**	**I555**	**I556**	**I557**	**I558**	**I559**	**I561**	**I564**	**I568**	**I572**
554	24	24	104	0	0	0	0	0	0	0	0	0
555	0	0	0	104	0	0	0	0	0	0	0	0
556	0	0	0	0	104	0	0	0	0	0	0	0
557	0	0	0	0	0	56	0	0	0	0	0	0
558	0	0	0	0	0	0	104	0	0	0	0	0
559	0	0	0	0	0	0	0	104	0	0	0	0
561	0	0	0	0	0	0	0	0	104	0	0	0
564	0	0	0	0	0	0	0	0	0	104	0	0
568	0	0	0	0	0	0	0	0	0	0	104	0
572	0	0	0	0	0	0	0	0	0	0	0	104

Table B- 4: Stopes input file from each half year
Note: The copper and zinc data is the original average grade of the stope. Vol is the original designed volume, vol_ex is the extracted volume calculated from the blasting slices and vol_bf is the volume that must be backfilled.

X	Y	Z	sizeX (m)	sizeY (m)	sizeZ (m)	density (kg/m³)	Cu (%)	Zn (%)	Half year	StopeID	Xstart	Xend	Ystart	Yend	Zstart	Zend	Vol (m³)	Vol_ex (m³)	Vol_bf (m³)	dr
4736	2758	660	36	12	40	3	1.76	0.08	1	248	4718	4750	2752	2760	644	680	17280	17280	17856	2520
4772	2738	660	36	12	40	3	6.67	0.55	1	250	4754	4786	2732	2740	644	680	17280	17280	17856	2520
4772	2758	660	36	12	40	3	8.02	0.66	1	252	4754	4786	2752	2760	644	680	17280	17280	17856	2520
4772	2770	660	36	12	40	3	6.22	0.46	1	256	4754	4786	2764	2772	644	680	17280	17280	17856	2520
4844	2666	660	36	12	40	3	0.05	0	1	291	4826	4858	2660	2668	644	680	17280	1920	2496	280
4844	2690	660	36	12	40	3	0.51	0.02	1	293	4826	4858	2684	2692	644	680	17280	11520	12096	1680
4844	2702	660	36	12	40	3	1.51	0.05	1	295	4826	4858	2696	2704	644	680	17280	17280	17856	2520
4844	2714	660	36	12	40	3	3.19	0.08	1	297	4826	4858	2708	2716	644	680	17280	17280	17856	2520
4844	2726	660	36	12	40	3	4.70	0.09	1	298	4826	4858	2720	2728	644	680	17280	17280	17856	2520
4808	2758	660	36	12	40	3	7.78	0.47	2	318	4790	4822	2752	2760	644	680	17280	17280	17856	2520
4808	2782	660	36	12	40	3	6.48	0.35	2	321	4790	4822	2776	2784	644	680	17280	17280	17856	2520
4874	2678	588	24	12	40	3	3.70	0.28	2	326	4862	4882	2672	2680	572	608	11520	11520	11904	1680
4874	2714	588	24	12	40	3	5.46	0.29	2	329	4862	4882	2708	2716	572	608	11520	11520	11904	1680
4880	2902	588	36	12	40	3	0.60	0.19	2	331	4862	4894	2896	2904	572	608	17280	17280	17856	2520
4952	2702	588	36	12	40	3	3.09	0.30	2	333	4934	4966	2696	2704	572	608	17280	17280	17856	2520
4952	2758	588	36	12	40	3	2.34	0.24	2	335	4934	4966	2752	2760	572	608	17280	17280	17856	2520
4988	2702	588	36	12	40	3	2.30	0.26	2	378	4970	5002	2696	2704	572	608	17280	17280	17856	2240
4988	2714	588	36	12	40	3	2.30	0.26	2	382	4970	5002	2708	2716	572	608	17280	17280	17856	2520
4988	2770	588	36	12	40	3	1.49	0.21	2	384	4970	5002	2764	2772	572	608	17280	17280	17856	2520

Continued on next page.

Table B- 4 continued:

X	Y	Z	sizeX (m)	sizeY (m)	sizeZ (m)	density (kg/m³)	Cu (%)	Zn (%)	Half year	StopeID	Xstart	Xend	Ystart	Yend	Zstart	Zend	Vol (m³)	Vol_ex (m³)	Vol_bf (m³)	dr
4772	2782	660	36	12	40	3	5.36	0.37	3	388	4754	4786	2776	2784	644	680	17280	17280	17856	2520
4808	2770	660	36	12	40	3	7.50	0.42	3	402	4790	4822	2764	2772	644	680	17280	17280	17856	2520
4808	2794	660	36	12	40	3	6.01	0.29	3	413	4790	4822	2788	2796	644	680	17280	17280	17856	2520
4988	2690	588	36	12	40	3	1.47	0.18	3	499	4970	5002	2684	2692	572	608	17280	17280	17856	2240
4988	2726	588	36	12	40	3	2.04	0.21	3	500	4970	5002	2720	2728	572	608	17280	17280	17856	2520
4988	2758	588	36	12	40	3	1.05	0.15	3	503	4970	5002	2752	2760	572	608	17280	17280	17856	2520
4988	2782	588	36	12	40	3	1.91	0.18	3	504	4970	5002	2776	2784	572	608	17280	17280	17856	2520
4988	2830	588	36	12	40	3	5.19	0.18	3	552	4970	5002	2824	2832	572	608	17280	17280	17856	2520
4988	2878	588	36	12	40	3	2.03	0.16	3	553	4970	5002	2872	2880	572	608	17280	17280	17856	2520
4772	2806	660	36	12	40	3	4.25	0.20	4	554	4754	4786	2800	2808	644	680	17280	17280	17856	2520
4808	2806	660	36	12	40	3	6.14	0.25	4	555	4790	4822	2800	2808	644	680	17280	17280	17856	2520
4880	2702	588	36	12	40	3	5.27	0.25	4	556	4862	4894	2696	2704	572	608	17280	17280	17856	2520
4868	2738	588	12	12	40	3	5.01	0.23	4	557	4862	4870	2732	2740	572	608	5760	5760	5952	840
4880	2830	588	36	12	40	3	5.15	0.18	4	558	4862	4894	2824	2832	572	608	17280	17280	17856	2520
4952	2714	588	36	12	40	3	1.23	0.11	4	559	4934	4966	2708	2716	572	608	17280	17280	17856	2520
4952	2770	588	36	12	40	3	1.60	0.13	4	561	4934	4966	2764	2772	572	608	17280	17280	17856	2520
4988	2738	588	36	12	40	3	1.32	0.13	4	564	4970	5002	2732	2740	572	608	17280	17280	17856	2520
4988	2806	588	36	12	40	3	3.44	0.14	4	568	4970	5002	2800	2808	572	608	17280	17280	17856	2520
4988	2854	588	36	12	40	3	2.40	0.14	4	572	4970	5002	2848	2856	572	608	17280	17280	17856	2520

Table B- 5: Blast slices input file for all stopes of each half year

Half year 1				Half year 2			
Blast_ID	**Cu (%)**	**Zn (%)**	**Volume (m³)**	**Blast_ID**	**Cu (%)**	**Zn (%)**	**Volume (m³)**
248_1	4.14	0.25	1920	318_1	7.09	0.33	1920
248_2	2.43	0.11	3840	318_2	7.52	0.38	3840
248_3	1.66	0.06	5760	318_3	7.75	0.43	5760
248_4	0.63	0.02	5760	318_4	8.21	0.60	5760
250_1	3.92	0.32	1920	321_1	4.63	0.20	1920
250_2	4.47	0.46	3840	321_2	5.33	0.26	3840
250_3	6.24	0.54	5760	321_3	6.54	0.36	5760
250_4	9.47	0.70	5760	321_4	7.80	0.46	5760
252_1	8.28	0.78	1920	326_1	5.91	0.53	1920
252_2	8.08	0.79	3840	326_2	3.96	0.32	3840
252_3	8.69	0.73	5760	326_3	2.80	0.18	5760
252_4	7.24	0.47	5760	326_4	0	0	0
256_1	7.54	0.52	1920	329_1	5.32	0.22	1920
256_2	7.63	0.57	3840	329_2	5.11	0.27	3840
256_3	6.62	0.50	5760	329_3	5.75	0.33	5760
256_4	4.45	0.32	5760	329_4	0	0	0
291_1	0.45	0	1920	331_1	0.83	0.36	1920
291_2	0	0	0	331_2	0.90	0.35	3840
291_3	0	0	0	331_3	0.86	0.21	5760
291_4	0	0	0	331_4	0.16	0.03	5760
293_1	2.04	0.07	1920	333_1	4.90	0.50	1920
293_2	1.14	0.03	3840	333_2	4.88	0.52	3840
293_3	0.31	0.01	5760	333_3	3.10	0.34	5760
293_4	0	0	0	333_4	1.27	0.04	5760
295_1	4.40	0.15	1920	335_1	1.05	0.14	1920
295_2	2.89	0.09	3840	335_2	1.32	0.15	3840
295_3	1.03	0.03	5760	335_3	2.17	0.23	5760
295_4	0.35	0.01	5760	335_4	3.63	0.35	5760
297_1	6.65	0.19	1920	378_1	0	0	1920
297_2	4.90	0.12	3840	378_2	1.34	0.15	3840
297_3	3.14	0.07	5760	378_3	2.24	0.25	5760
297_4	0.95	0.02	5760	378_4	3.75	0.42	5760
298_1	7.08	0.16	1920	382_1	0.87	0.10	1920
298_2	5.87	0.13	3840	382_2	1.97	0.24	3840
298_3	5.06	0.09	5760	382_3	2.29	0.26	5760
298_4	2.77	0.04	5760	382_4	3.01	0.32	5760
				384_1	1.67	0.17	1920
				384_2	1.41	0.17	3840
				384_3	1.45	0.26	5760
				384_4	1.51	0.21	5760

Continued on next page.

Table B- 5 continued:

Half year 3				Half year 4			
Blast_ID	**Cu (%)**	**Zn (%)**	**Volume (m^3)**	**Blast_ID**	**Cu (%)**	**Zn (%)**	**Volume (m^3)**
388_1	7.48	0.48	1920	554_1	5.97	0.25	1920
388_2	6.61	0.44	3840	554_2	5.06	0.20	3840
388_3	5.10	0.37	5760	554_3	4.34	0.20	5760
388_4	4.08	0.30	5760	554_4	3.03	0.19	5760
402_1	6.25	0.27	1920	555_1	6.23	0.24	1920
402_2	6.45	0.31	3840	555_2	6.28	0.24	3840
402_3	7.47	0.41	5760	555_3	6.16	0.26	5760
402_4	8.65	0.56	5760	555_4	5.99	0.26	5760
413_1	4.83	0.21	1920	556_1	6.35	0.20	1920
413_2	5.54	0.26	3840	556_2	6.13	0.23	3840
413_3	6.51	0.32	5760	556_3	4.96	0.27	5760
413_4	6.23	0.31	5760	556_4	4.64	0.27	5760
499_1	0	0	1920	557_1	5.75	0.26	1920
499_2	0.47	0.04	3840	557_2	4.64	0.22	3840
499_3	1.31	0.16	5760	557_3	0	0	0
499_4	2.80	0.34	5760	557_4	0	0	0
500_1	1.47	0.15	1920	558_1	5.98	0.18	1920
500_2	1.79	0.19	3840	558_2	5.86	0.20	3840
500_3	2.31	0.25	5760	558_3	5.37	0.18	5760
500_4	2.14	0.21	5760	558_4	4.19	0.16	5760
503_1	1.44	0.16	1920	559_1	3.02	0.31	1920
503_2	1.17	0.14	3840	559_2	2.34	0.24	3840
503_3	0.91	0.14	5760	559_3	1.23	0.12	5760
503_4	0.99	0.16	5760	559_4	0.91	0	5760
504_1	2.26	0.19	1920	561_1	1.53	0.18	1920
504_2	1.99	0.16	3840	561_2	1.63	0.16	3840
504_3	1.76	0.17	5760	561_3	1.59	0.12	5760
504_4	1.89	0.19	5760	561_4	1.61	0.10	5760
552_1	3.95	0.13	1920	564_1	1.47	0.15	1920
552_2	4.43	0.14	3840	564_2	1.54	0.15	3840
552_3	5.26	0.19	5760	564_3	1.22	0.12	5760
552_4	6.04	0.22	5760	564_4	1.21	0.13	5760
553_1	1.34	0.15	1920	568_1	2.68	0.09	1920
553_2	1.80	0.17	3840	568_2	3.10	0.12	3840
553_3	2.34	0.19	5760	568_3	3.27	0.14	5760
553_4	2.11	0.13	5760	568_4	4.09	0.18	5760
				572_1	1.24	0.09	1920
				572_2	1.23	0.07	3840
				572_3	2.41	0.13	5760
				572_4	3.56	0.22	5760

Table B- 6: Overview of the development, drilling, extraction and backfill periodical capacity and the number of available development and drilling machines per period

period	**development capacity (m)**	**drilling capacity (m)**	**extraction capacity (m^3)**	**backfill capacity (m^3)**	**development machines (#)**	**drilling machines (#)**
1	36	1600	17000	3800	3	2
2	36	1600	17000	3800	3	2
3	36	1600	17000	3800	3	2
…	…	…	…	…	…	…
…	…	…	…	…	…	…
88	36	1600	17000	3800	3	2
89	36	1600	17000	3800	3	2
90	36	1600	17000	3800	3	2
91	0	0	0	0/70000	0	0

Table B- 7: Overview of all stopes adjacent to each other, considering the corresponding half year
Note: All adjacent stopes from previous half years are an adjacent fillmass.

Half year	StopeID	248	250	252	256	291	293	295	297	298	318	321	326	329	331	333	335	378	382	384	388	402	413	499	500	503	504	552	553	554	555	556	557	558	559	561	564	568	572
Half year 1	248			X																																			
Half year 1	250																																						
Half year 1	252	X			X						X																												
Half year 1	256			X																	X	X																	
Half year 1	291																																						
Half year 1	293							X																															
Half year 1	295						X		X																														
Half year 1	297							X		X																													
Half year 1	298								X																														
Half year 2	318																					X																	
Half year 2	321																				X	X	X																
Half year 2	326																																						
Half year 2	329																															X							
Half year 2	331																																						
Half year 2	333																	X																	X				
Half year 2	335																									X										X			
Half year 2	378															X			X					X															
Half year 2	382																	X							X										X				
Half year 2	384																									X	X									X			
Half year 3	388																																						
Half year 3	402																																						
Half year 3	413																														X								
Half year 3	499																																						
Half year 3	500																																				X		
Half year 3	503																																						
Half year 3	504																																						
Half year 3	552																																						
Half year 3	553																																						
Half year 4	554																														X								
Half year 4	555																													X									
Half year 4	556																																						
Half year 4	557																																						
Half year 4	558																																						
Half year 4	559																																						
Half year 4	561																																						
Half year 4	564																																						
Half year 4	568																																						
Half year 4	572																																						

Appendix C

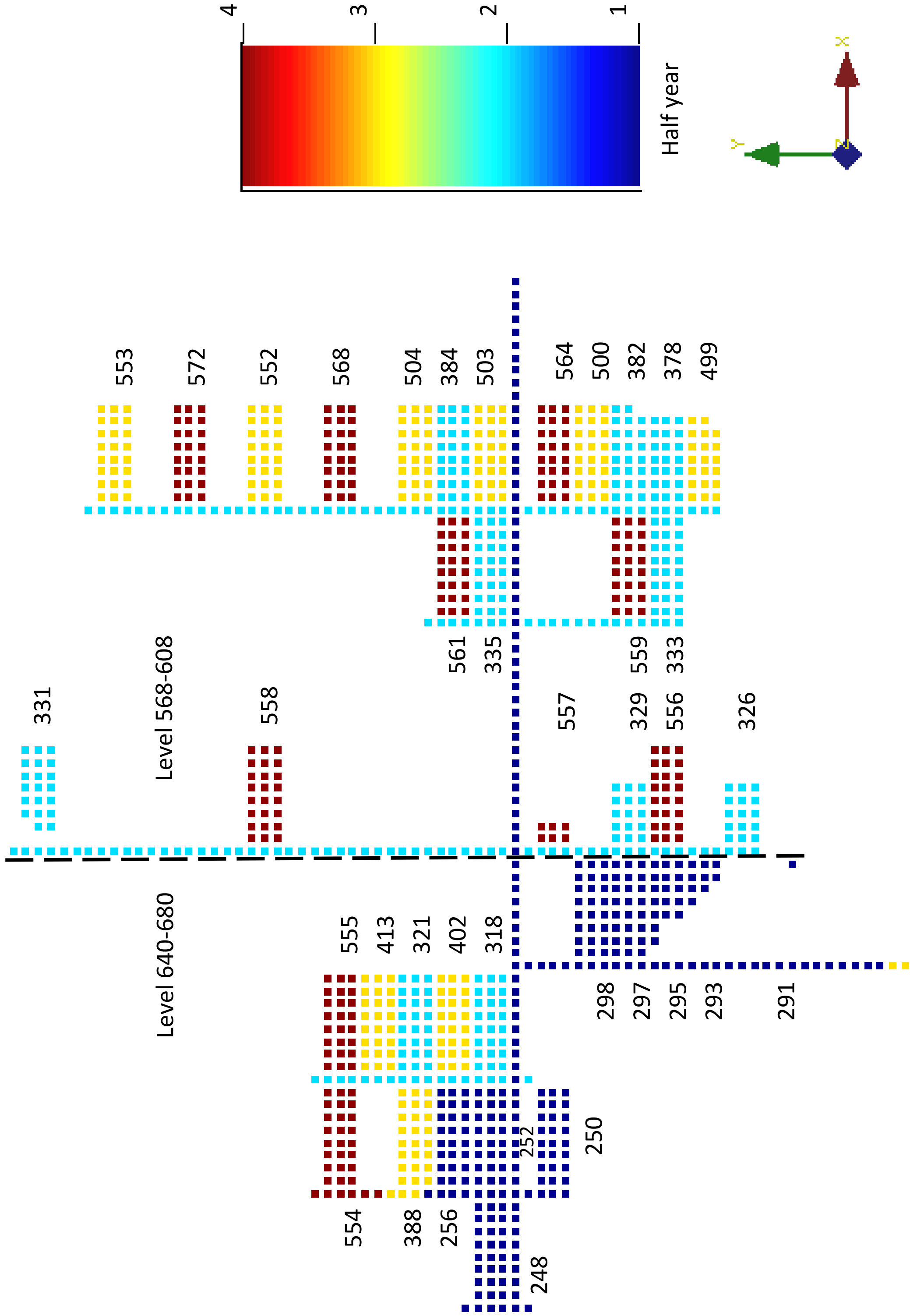


Figure C- 1: Top view of the stopes and development drifts
Note: The stopeID for each stope is given adjacent to the stope. The color represents for which half year the development or stope is planned.

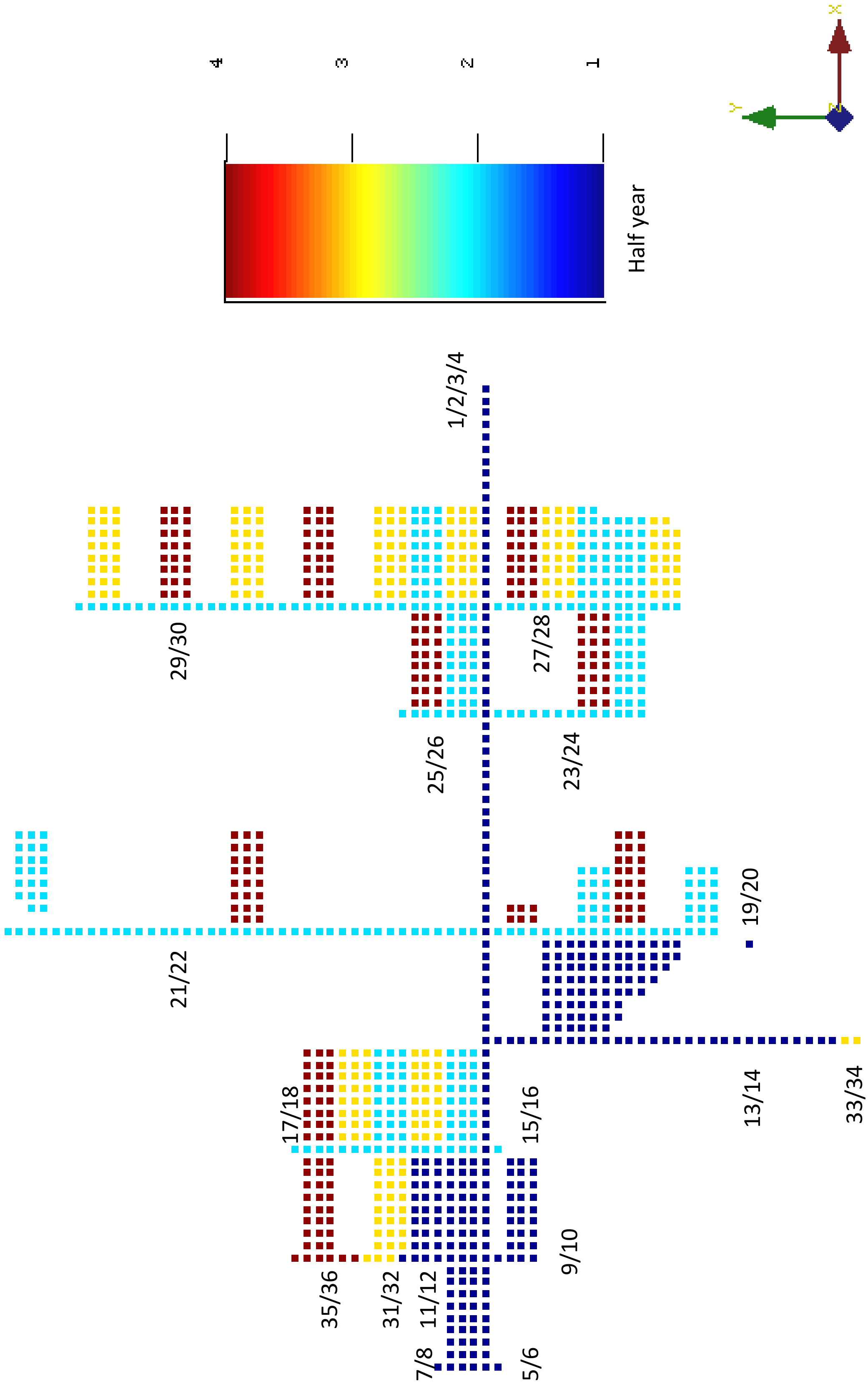


Figure C- 2: Top view of the stopes and development drifts
Note: The devID for each stope is given adjacent to the development drift. The color represents for which half year the development or stope is planned.

Appendix D

Table D- 1: Overview of the amount of activity (m or m³) that is done in each period for internal development, drilling, extraction and backfilling of each stope

248

Period	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75	76	77	78	79	80	81	82	83	84	85	86	87	88	89	90
I-dev	12	12	12	12	12	12	12	12	8	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	800	120	800	800	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	0	937	983	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3840	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1140	1	1	1	1	1303	1	1	3799	3799	1143	1	1	3800	2864

250

Period	59	60	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75	76	77	78	79	80
I-dev	12	8	12	12	12	12	12	12	12	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	800	120	800	800	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	0	1920	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3840	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0
ex4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	2660	3799	3799	3799	3799

252

Period	43	44	45	46	47	48	49	50	51	52	53	54	55	56	57	58	59	60	61	62	63	64	65
I-dev	12	12	12	12	12	12	12	12	8	0	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	120	800	800	800	0	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	0	1920	0	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3840	0	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0	0
ex4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3800	3800	3800	3800	2655	1

256

Period	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51
I-dev	12	12	12	12	12	12	12	8	12	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	800	800	120	800	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	0	1920	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3840	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0
ex4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3800	2656	3800	3800	3800

Continued on next page.

Table D- 1 continued:

291

Period	60	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75	76	77	78	79	80	81
I-dev	12	12	12	12	6	12	12	1	1	12	12	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	0	0	279	1	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	0	280	1640	0	0	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	2496

293

Period	47	48	49	50	51	52	53	54	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72
I-dev	8	12	12	12	12	12	12	12	12	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	800	80	800	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	1920	0	0	0	0	0	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	3840	0	0	0	0	0	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4047	1686	0	0	0	0	0	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1145	1	1142	3799	1	1	2207	1	3799

295

Period	40	41	42	43	44	45	46	47	48	49	50	51	52	53	54	55	…	…	…	73	74	75	76	77	78	79	80	81	82	83	84	85	86	87	88
I-dev	12	12	12	12	12	12	12	8	12	0	0	0	0	0	0	0				0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	0	800	1	800	118	1	800				0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0				1920	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0				0	3840	0	0	0	0	0	0	0	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0				0	0	5480	1	279	0	0	0	0	0	0	0	0	0	0	0
ex4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0				0	0	0	0	0	0	0	5760	0	0	0	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0				0	0	0	0	0	0	0	0	1	3799	3799	1	1	2657	3799	3799

297

Period	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52	53	54	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72
I-dev	12	12	12	12	12	12	8	12	12	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	800	800	800	120	0	0	0	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1920	0	0	0	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3840	0	0	0	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0	0	0	0
ex4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3798	2658	1	3799	3799	1	3799	1

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Table D- 1 continued:

298

Period	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49	50	51	52	53	54	55	56
l-dev	12	12	12	12	12	12	12	8	12	0	0	0	0	0	0	0	0	0	0	0	0	0	0
dr	0	0	0	0	0	0	0	0	0	800	800	800	120	0	0	0	0	0	0	0	0	0	0
ex1	0	0	0	0	0	0	0	0	0	0	0	0	0	1898	21	0	0	0	0	0	0	0	0
ex2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3840	0	0	0	0	0	0	0
ex3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0	0
ex4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5760	0	0	0	0	0
bf	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3800	2656	3800	3800	3800

Appendix E

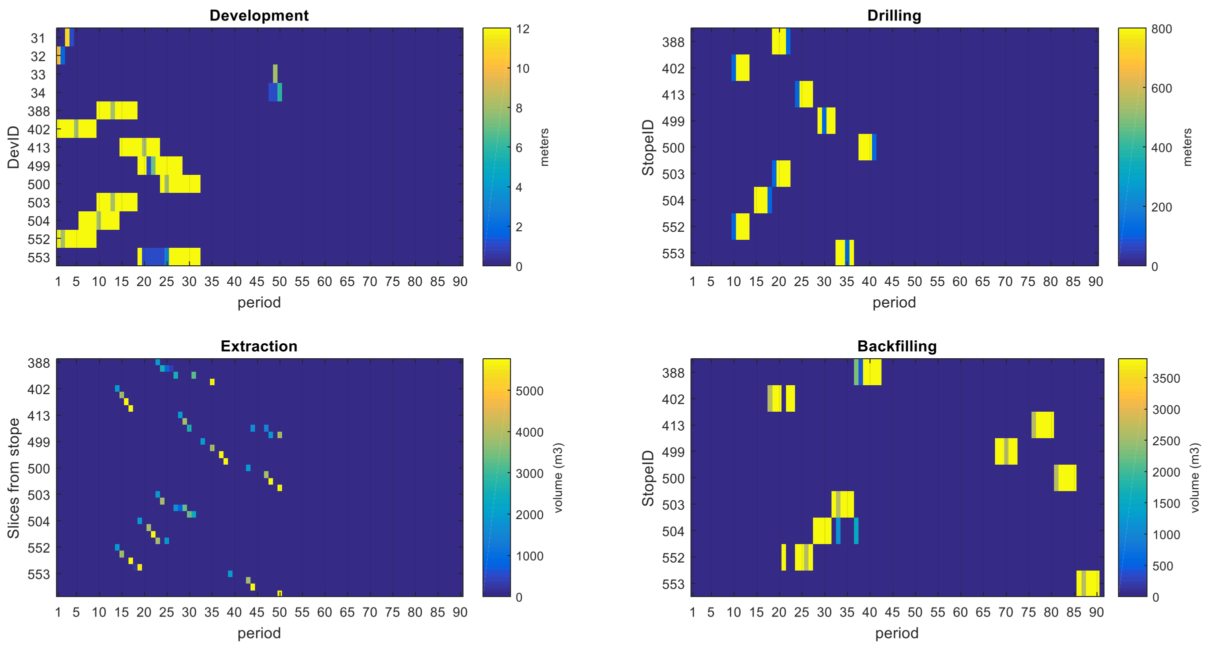

Figure E- 1: Short-term schedule for half year three, reference scenario