DOKUZ EYLÜL UNIVERSITY
GRADUATE SCHOOL OF NATURAL AND APPLIED SCIENCES

DEVELOPMENT OF OPEN PIT MINE PLANNING STRATEGIES BASED ON THE QUALITY DISTRIBUTION OF RAW MATERIAL QUARRIES

by
Reha SEZEN

January, 2017
İZMİR

DEVELOPMENT OF OPEN PIT MINE PLANNING STRATEGIES BASED ON THE QUALITY DISTRIBUTION OF RAW MATERIAL QUARRIES

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in Mining Engineering, Mining Operation Program

by

Reha SEZEN

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İZMİR

M.Sc THESIS EXAMINATION RESULT FORM

We have read the thesis entitled **"DEVELOPMENT OF OPEN PIT MINE PLANNING STRATEGIES BASED ON THE QUALITY DISTRIBUTION OF RAW MATERIAL QUARRIES"** completed by **REHA SEZEN** under the supervision of **PROF. DR. ÇAĞATAY PAMUKÇU** and we certify that in our opinion it is fully adequate, in scope and in quality, as a thesis for the degree of Master of Science.

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DEVELOPMENT OF OPEN PIT MINE PLANNING STRATEGIES BASED ON THE QUALITY DISTRIBUTION OF RAW MATERIAL QUARRIES

ABSTRACT

In this project, the excavation works planned for Cankurtaran Limestone field of Batisöke Cement Industry Co. Inc., and the works to supply raw materials in qualities and quantities required by the Cement Plant have been explained.

The Cement Factory of Batisöke Cement Industry Inc. satisfies the limestone need required for cement production from the Cankurtaran Field located in the north of the factory. The Cankurtaran field is located about 1 km north of Batisöke Cement Factory. The percent MgO content in the chemical composition of the limestone is determinative in raw material production for cement production within the defined limits. Determination of the distribution of raw materials that can be used in the production of cement within the boundaries of the defined license and determination of the appropriate quality raw material reserves constitute the subject of this project

Within the scope of the project, percent MgO distribution has been determined up to plus 300 m. elevation using percent MgO data obtained from 58 drilling works and 51 829 265 ton operable limestone reserves have been determined within the limits of production activity.

The studies to determine in what ratio the raw material that would meet the planned cement production capacity expansion of Batısöke Cement Industry Co. Inc. will be supplied from the Cankurtaran field in order to determine the quality of the limestone below 300 m elevation have been commenced within the scope of this project. Determination of the quality distribution of the limestone below 300 m elevation according to the MgO contents and determination of the operable reserves within the operating permit limits considering the mining technique have been done with the help of the SURPAC program.

Keywords: Plant planning, 3d mining planning, size and quality distribution

HAMMADDE OCAKLARININ KALİTE DAĞILIMINA GÖRE AÇIK İŞLETME PLANLAMA STRATEJİLERİNİN GELİŞTİRİLMESİ

ÖZ

Bu projede Batısöke Çimento Sanayi T.A.Ş.'ne ait Cankurtaran Kalker sahasında planlanan kazi calismalari ile Çimento Fabrikasının ihtiyacı olan kalitede ve miktarda hammadde temin edilmesine yönelik çalışmalar anlatilmistir.

Batısöke Çimento Sanayi T.A.Ş.'ne ait Çimento Fabrikası, çimento üretiminde gerekli olan kireçtaşı ihtiyacını fabrikanın kuzeyinde yer alan Cankurtaran Sahasından temin etmektedir. Cankurtaran sahası, Batısöke Çimento Fabrikasının yaklaşık 1 km kuzeyinde yer almaktadır Tanımlanan sınırlar içerisinde çimento üretimine yönelik olarak hammadde üretiminde belirleyici olan, kireçtaşının kimyasal bileşenindeki percent MgO, içeriğidir. Tanımlanan ruhsat sınırları içerisinde çimento üretiminde kullanılabilecek kalitede hammadde dağılımının tespit edilmesi ve uygun kalitedeki hammadde rezervinin belirlenmesi bu projenin konusunu oluşturmaktadır.

Proje kapsamında 58 adet sondaj çalışması sonucu elde edilen yüzde MgO verileri kullanılarak artı 300 m kotuna kadar yüzde MgO dağılımı belirlenmiş ve üretim faaliyeti yapılabilecek sınırlar içerisinde 51 829 265 ton işletilebilir kalker rezervi tespit edilmiştir.

Batısöke Çimento T.A.Ş' nin çimento üretim kapasitesinde planlamış olduğu artışı karşılayabilecek hammaddenin Cankurtaran sahasından hangi oranda temin edilebileceğinin belirlenmesi amacıyla 300 m kotu altındaki kireçtaşının kalitesinin belirlenmesine yönelik çalışmalar bu proje kapsamında gerçekleştirilmiştir. 300 m kotu altındaki kalkerin MgO içeriklerine göre kalite dağılımının tespit edilmesi ve madencilik tekniği gözetilerek işletme izin sınırları içerisindeki işletilebilir rezervin belirlenmesi SURPAC programı yardımıyla yapılmıştır.

Anahtar kelimeler: Tesis tasarımı, 3 boyutlu maden planlaması, boyut ve kalite dağılımı.

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CHAPTER ONE
INTRODUCTION

1.1 General

Constraints on the planning and design processes employed by surface mining operations vary in distinct ways from other commodities mined by both hard and soft rock open pit methods. During the surface mining operations planning and design can be limited The unique waste handling needs, including tailings disposal, of contemporary mining requires specific planning considerations (Anderson, 1999).

It is the purpose of this research to analyze and document a conventional hard rock, metal mine planning system, and contrast this with the unconventional mine planning system used by mines. Systems activity models of both the conventional and unconventional systems are developed in support of documenting and contrasting the two systems (Anderson, 1999).

Constraints unique to mine planning are identified and their impact on the mine planning system are documented. The impacts of challenging waste handling and storage requirements and a uniquely prescriptive regulatory environment defining mineable ore are identified as key constraints (Anderson, 1999).

The research concludes with a proposal for a new planning system to better support the planning of mines. The proposed system respects the unique waste management considerations in planning and revisits the current regulatory approach to ensuring resource recovery. The proposed system is compatible with traditional approaches to economic analysis in open pit planning, and with emerging best practices to manage technical and economic uncertainty, improve project optimization, and develop robust mine plans (Anderson, 1999).

1.2 Organization of the Thesis

In this study, the region which is within the license limits of the Cankurtaran field is modeled and the excavation plan is modeled according to the quality distribution.

With this context, chapters can be explained in details. Chapter one provides a brief introduction to the area of research and the research objectives of this thesis. In Chapter two, a comprehensive literature reviews on sampling, block modelling and reserve estimation methods. Chapter three, describes the location of the scholastic modeling and simulation in the literature. In addition, probability distributions, use of simulation results, and interpretation of simulation results are explained in this chapter. In Chapter four, the excavation plan is shown and discussed in detail according to the quality of the Cankurturan field simulated using a scholastic model. The conclusion is summarized in Chapter five.

CHAPTER TWO
RESOURCE ESTIMATION

2.1 Definition of Reserve

Classification is an important initial step for modelling. Each group of material are defined, grouped and valued by classification. Exploration and feasibility are two processes those take place after (Anderson, 1999).

In order to estimate the real value of a mining company, quantity of unextracted material has to be figured out. Amount of unreached valuable mineral can be simply named resource while the amount of extractable mineral is called reserve. There has been a continuous complexity between various mining associations about finding uniform descriptions (Anderson, 1999).

Ore is a metalliferous mineral, or an aggregate of metalliferous minerals, more or less mixed with gangue, which from the standpoint of the metallurgist can be treated at a profit (Evans, 2013).

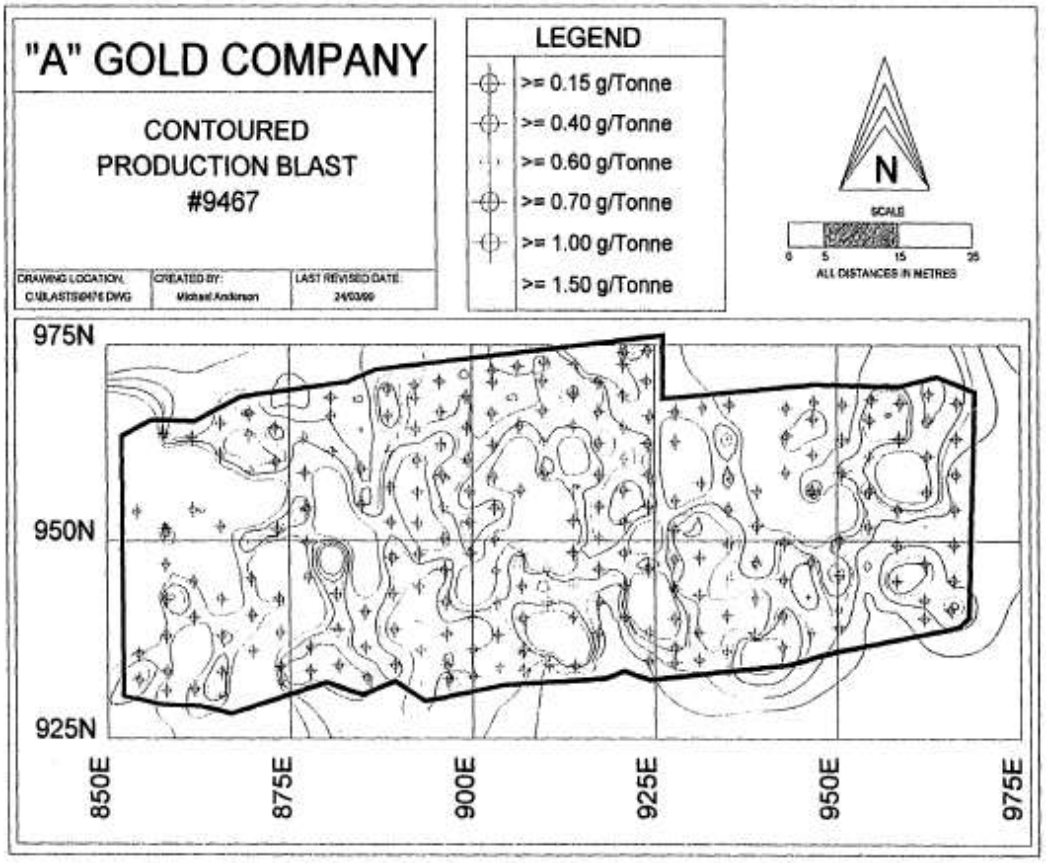


Figure 2.1 Contoured blasthole grades (Anderson, 1999)

Another way of classificating the material would be sub classing it as resource or reserve. Both the ore and mineralized terms can be considered as initial words to explain resource and reserve and to better material categorizing. Resource and reserve can be separated from each other via their economically valued material content. Resource is a term that covers many concepts starting from the smallest mineable material to whole deposit. Mine ability is determined by economic, mining, metallurgical, marketing, environmental, social and governmental factors (Haystead, 1993). Reserve points to financially vital part of the material which stays in the mineable part of material as well as feasibility assessments (Anderson, 1999).

Reserve and resource can also be furtherly classified by some of their contrasts. As an example reserve can easily categorized as probable and proven. In contrast, resource can be classified as under tree headings such as indicated, measured and inferred. Further movement can be maintained with the hostage of drilling data as well as

geologic stability. Advancing in a mine operation can only continue depending on good geologic stability and drilling data. When the material quality is hesitant poor drilling occurs. On the other hand well drilling applies in the areas where drilling data gives continuous high grade thus more confidence.

Figure 2.2 shows a schematic of the basic classification system used by the Australian Institute of Mining and Metallurgy (AUSIMM) and the Society of Mining, Metallurgy and Exploration £SME in the United States. Since 90s Canadian Institute of Mining and Metallurgy focused on a new classification system which can be seen in Figure 2.2 and a brand new system released which is almost the same as what is shown in Figure 2.2. Ore and mineralized were two terms which used to describe reserve and resource on a simplistic level. However they have been disappeared and not been existing anymore on the schematic (Anderson, 1999).

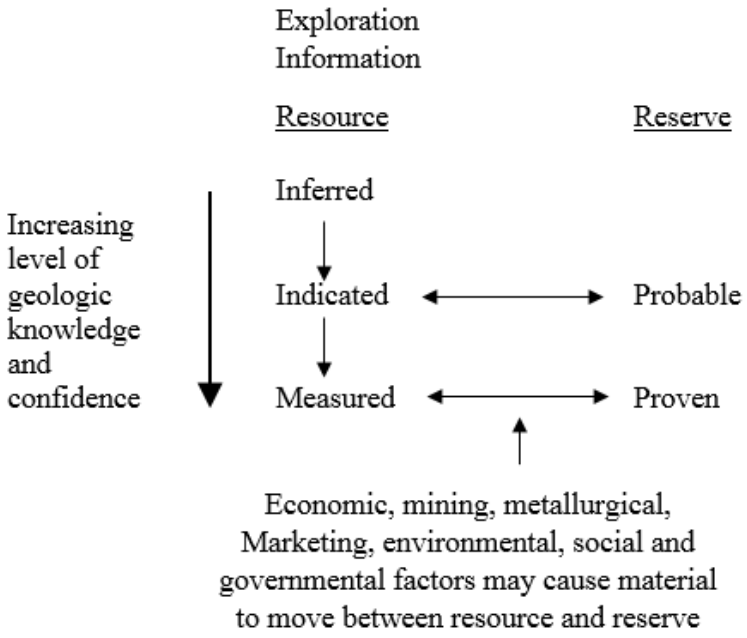


Figure 2.2 Classification System

Fresh techniques of classification has been occurred in recent years. These methods are expected to provide more control over resource and reserve estimation. A further approach has been brought by Diehl & David (1979) that not only helps to enter a more sophisticated geostatistical level of estimation but also would combine a few kits those

became available with the introduction of more advanced geostatistical estimation methods. To explain the minimization of estimation errors these techniques will be well discussed in further parts of the chapter. Moreover, mentioned techniques are also useful at measuring estimation errors where they bring a confidence level. Therefore Diehl & David came up with the idea of using this confidence level on spotting measured, indicated and inferred materials (Anderson, 1999).

Making sharp predictions have become easier through improved geostatistical methods are for mining businesses' hence predicting the resource and reserve values. Needless to say industry wide seeking for better techniques will be continuing (Anderson, 1999).

2.2 Block Model

One characteristic of block model method is its easiness when it comes to assigning various items like lithology, density and grade. Via the method various computer software's can be teamed and deposit can be displayed as arrays in series. This allows for faster more efficient modeling and the use of more complex estimation methods that were not previously viable (Blackwell & Sinclair, 2002).

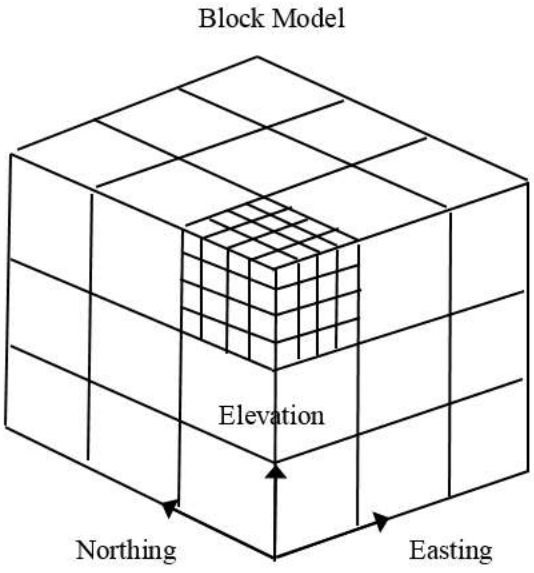


Figure 2.3 Block model concept

Figure 2.3 shows a property which carries a big amount of material. In this case it is possible to dividing it into smaller blocks which would be considered as homogenous. The size of these smaller blocks is typically of a size that is considered suitable for selection as either ore or waste given the available equipment and mining techniques. As block size decreases unique parts are appear to be more visible. Nonetheless it would not be efficient for computers to work with parts which are far from selected mining method and subdivided. It would also be pointless to create blocks suitable for a small shovel unless the mining technique and equipment supports taking that block out adjacently (Anderson, 1999).

Estimation error magnitude can be a reflection of estimated material volume which has already been mentioned earlier in this chapter. As the volume gets higher it is more likely to get a larger amount of error. It is naturally expected that there would be more material as well as identified sample points in greater blocks. To get an average grade which exposes the whole deposit it is much efficient to take the average grade in once instead of taking the average grade for a small size shovel (hand size) deposit. Figure 2.1 displays a blast pattern which has the assumptions below (Anderson, 1999).

Block size/direction: 1m in size and runs east/west direction, also 1m in size and runs north/south direction

Block heights: Block height is 5m which equals to the bench height

Blocks centers are also assumed as homogenous (along the block) and they have also taken into consideration in calculations

2.3 Sampling and Logging Procedures

Accuracy does get affected by two things such as the nature and the sample information. The more accurate information maintained the less chance of fail or incorrect. What establishes the vital steps of property evaluation can be counted as geological logging, sampling, assaying as well as giving essential data required for

modelling. Needless to say, data collection process must be operated with a high level of professionalism as carefulness and accuracy plays a major role of estimation and advance of mine. Time, care and money spent on this part of the process will contribute greatly to obtaining an accurate model of mineralization (Owens, 1993).

2.3.1 Sampling

Some typical methods of sample collecting are chip sampling, core drilling, and channeling. Amongst these core drilling probably the most common and advantageous one. It is a comparatively low cost method ($100 to $600 per meter) that can be used in both underground and open pit mines Hastings (1996).

In order to succeed further geological modelling large diameter core drilling (diamond drilling 50mm) one method that can be operated as it delivers correct results in sampling and also geological figures can be transferred in a computer. Enough number When percussion drilling or reverse circulation is applied an insecurity may occur in sampling results due to unprofessional usage e.g. when water enters the drilling hole (Anderson, 1999).

Mine exploration is one area that diamond core drilling is widely applied during sample collecting Holes are meant to be drilled with the right angle to hit the areas where high mineralization emerges. Orebody shape and its features can be figured out through drilling holes. Exploration process must bring representative sampling which is really crucial for operating the sampling work accurately. Therefore drilling has to be done in various levels of orebody as well as doing tests where high grade ore is laying. As the more mineralization happens the more changes occur in density, grade and metallurgical features. This gives a good explanatory about necessity of representative sampling in terms of modelling the orebody properly. Since the size of the samples provided by diamond core drilling are very small there is the danger that they may not be representative of the area through which they intersected (Rossi & Parker, 1993). There is also necessary operational procedures to be followed for analyzing the drillhole samplings (Anderson, 1999).

The number of sufficient drillings may vary depending on a few parameters. Features of deposit is one of those which directly affects spacing in a drilling process. As an example while coal might require drilling with 100m grids this number decreases to 10m grids if the deposit is diamond. Geological steadiness can only be maintained with appropriate drilling (Anderson, 1999).

To figure out various geological characteristics of deposit (density, grade distribution, roughness of material, metallurgy) sufficient amount of sampling should be done. Knowledge of these properties is essential during the feasibility stage of the project and will help to determine the size of the reserve (Roscoe, 1993).

In sample collecting blasthole cuttings is one spot which preferred when grade control is the purpose. In a blasting process the main idea is catching the desired fragmentation by using less explosives and blastholes hence drilling has never been a process of maintaining continuous and variable grade. Bames (1989) introduced an interesting model for examining the tradeoff between the blasthole spacing required to obtain adequate fragmentation versus the geostatistical data requirements for suitable ore/waste classification. The premise is that by spending more money on blasting, a better model of ore/waste classification can be established, returning a greater revenue from better grade control. This approach would be extremely useful in deposits where the mineralization is very erratic (Anderson, 1999).

Springett (1993), discusses how blasthole "sample acquisition and preparation is often of lesser quality than that used for exploration purposes despite the fact that the results will be used to make a unique and irrevocable decision." One common mistake many companies make focusing on preparation and sampling at the very early stages of mining such as evaluation or when material has just fed to mill. However a better approach would be spending resources around grade control step which is the stage that material is been separated as ore and waste.

The actual acquisition of sample material and the partitioning of this material into an appropriate size for assaying is represented by a series of comminution and splitting

stages in Figure 2.4. The comminution stages are represented by the horizontal segments of the black line while the splitting stages are represented by the vertical segments of the black line. The dashed line in Figure 2.4 is called the ‘safety line’. This line represents the hypothetical boundary for the smallest acceptable sample size before unacceptable errors begin to become present. Therefore all sampling stages (vertical line segments) should take place to the left of the line in this diagram. If sampling draws the line segment to the right of the line, then this shows evidence of an unsafe sample reduction scheme (Anderson, 1999).

Gy (1979), discusses the importance of the sample size in relation to the particle size through a concept called the ‘safety line’. Figure 2.4 shows a diagram from Gy that illustrates the ‘Safety Line’ concept.

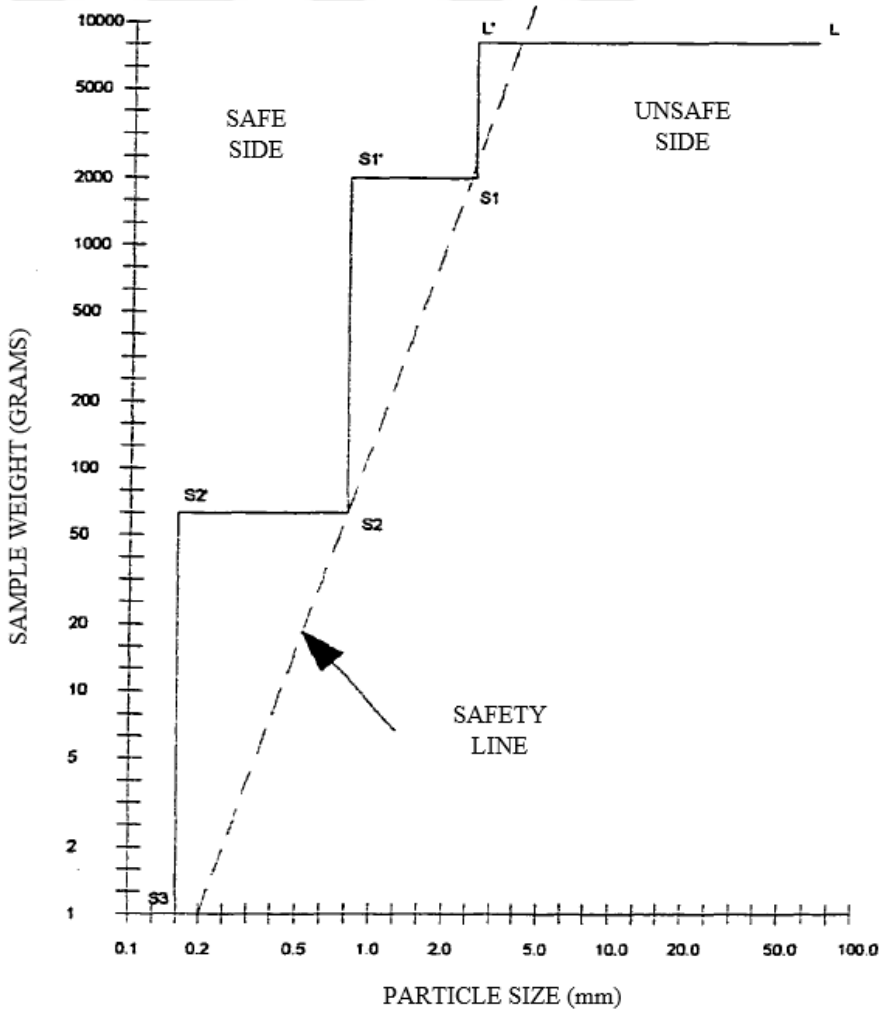


Figure 2.4 Gy's safety line (Gy, 1979)

To make this more understandable Equation 2.1 is given. Also the safety line drawn to help getting the main idea as it is just a generic line.

$$M_s \geq 125000\ d^3 \tag{2.1}$$

In Equation 2.1 Ms shows the sample mass (grams) while d is particle size (cm). Equation 2.2. below is shorter, although is more difficult.

$$\sigma^2 = \frac{Cd^3}{M_s} \tag{2.2}$$

$$\sigma^2 = variance \tag{2.3}$$

$$C = mlfg \tag{2.4}$$

g = granulometric distribution factor

f= form factor

l = liberation parameter

m = mineralogical parameter

In Equation 2.3 the C can be determined by the test center. Then once the variance is chosen applying it on Equation 2.2 particle size and sample size will be correlated. In the same manner, given a particular sample size and particle size, the variance of the sample can be found (David, 1977).

There is several ways to collect samples. They might sometimes be collected just around the blasthole however this may cause suspicious results thus it becomes really hard to get truthful results. Further in this article mainly three different sampling techniques will be conversed. None of these methods can produce samples of the quality obtained through diamond core drilling (Anderson, 1999).

In the very first method a 100mm pipe will be employed in cuttings and which 10 times as long as the pile in terms of length. Cuttings are very likely to be moisturized

which stay around and inside the pipe and they will reshape the sample and then all will be ground to the desired sample size.

In the second method a bucket will be employed in the process to catch early coming cuttings as soon as they leave the hole. Then process will continue as in the first one and samples will be ground to get needed size. With the third method, a 0.3m wide trench is cut through the pile from one side to the other using a hand shovel and this material is collected, ground and riffled (Blackwell, 1995).

This initial field sampling is almost always conducted on the wrong side of the safety line and is usually unavoidable due to the large amount (tons) of cuttings that a large diameter deep blasthole can produce. However, by selecting a representative section of the cuttings and following Gy's safety line for the remaining comminution and sampling stages, sampling error can be kept to a minimum (Anderson, 1999).

It is not uncommon for the grade to be biased by the size distribution. This problem might be handled by assaying one size fraction that is representative of the average grade, but water from drill dust suppression and lubrication, and/or ground water washing through the cuttings will likely remove the fines and (usually) reduce the grade of the remaining cuttings (Blackwell, 1995).

2.3.2 Logging

Geology is a highly vital part of mineral estimation and its value should not be underestimated. Inaccurate results are very likely to occur if geological boundaries and constraints are not considered appropriately. In fact, some resistance to new geostatistical methods is based on the belief that geostatisticians omit geological information when forming a reserve model and rely solely on some complex formula to arrive at an answer (Champigny, 1993).

The importance of geology is a function of the scale of both the geological features and the mining method that will be used to extract the ore (Anderson, 1999).

2.4 Reserve Estimation Methods

The final source of error as listed in the initial quote of this chapter lies in the estimation method chosen. Unfortunately, there is no clear ‘best’ method to use for every situation. Each of the methods described in this chapter have their own advantages and disadvantages which make them better or worse in different situations (Anderson, 1999).

2.4.1 Conventional Methods

2.4.1.1Polygonal Estimation

The polygon method of estimation is usually performed at some point during the initial stages of exploration. It is a very simple method that assigns each block the grade (or other property being estimated) of the closest sample, or 'nearest neighbor (Anderson, 1999).

The main advantage of this method is that it is quick and simple. Although originally earned out by hand, a computer is usually required as drawing polygons by hand on sections or plans and calculating areas and volumes is extremely tedious and time consuming. Another advantage is in its handling of clustered samples. Clustered data automatically form smaller polygons and thus samples are weighted less. The main disadvantage is that this method may assign high grades to a large number of blocks skewing the estimated metal content for the orebody. It is most likely that repeat sampling and assaying of the highest 5percent outliers of a lognormal distribution would result in substantially different grades. Polygon estimates also tend to produce realizations that are spotted in appearance and lack transition from block to block. Figure 2.5 shows grades estimated using the polygon method for the example blast in Figure 2.1 (Anderson, 1999).

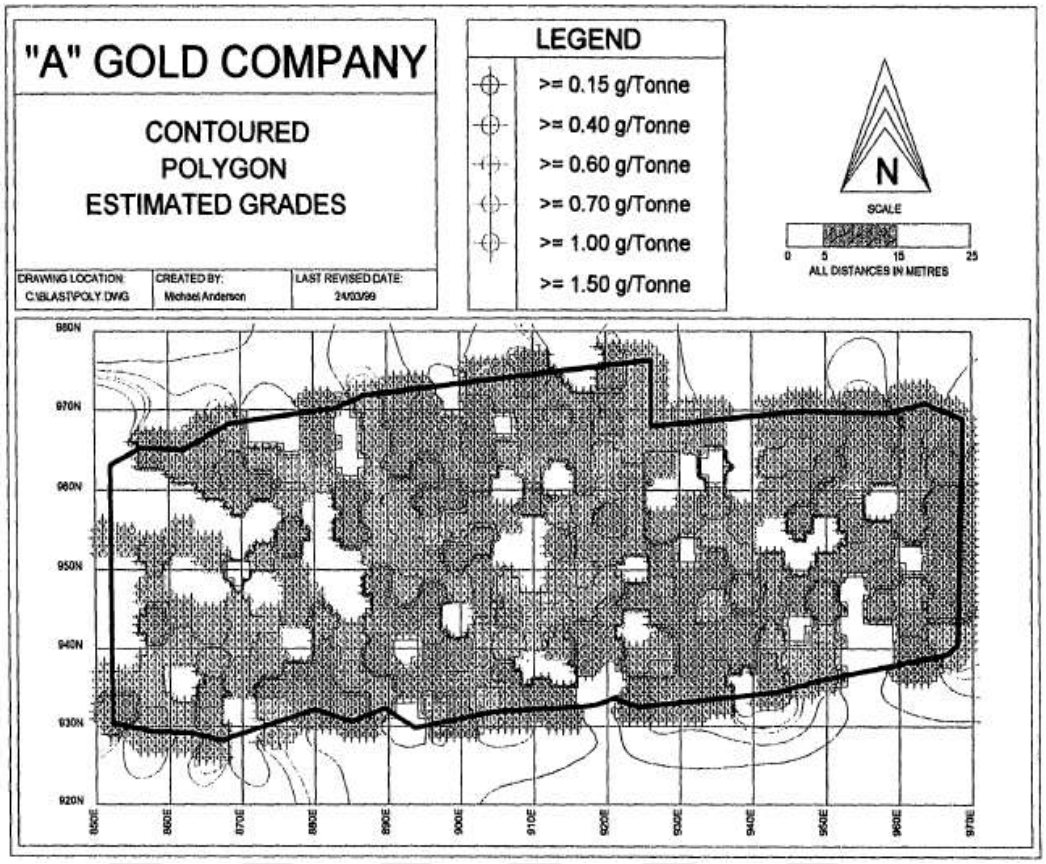


Figure 2.5 Polygon estimation method realization (Anderson, 1999)

Figure 2.5 is quite spotted in appearance and each 'spot' contains identical estimates within it. However, if estimation is controlled by boundaries and there are samples on a fairly regular grid such as in the blast of Figure 2.1 then the distribution of values within the estimated blast matches the original distribution quite well. Figure 2.7 shows a histogram of the estimated polygon grades. The similarity between the histogram of Figure 2.7 and the one in Figure 2.6 is expected since the estimation uses identical values to that of reality. Due to the regularly spaced blasthole data each original value represents an equal volume in the estimate. Where samples are irregularly spaced, the polygon method naturally declutters the data (Anderson, 1999).

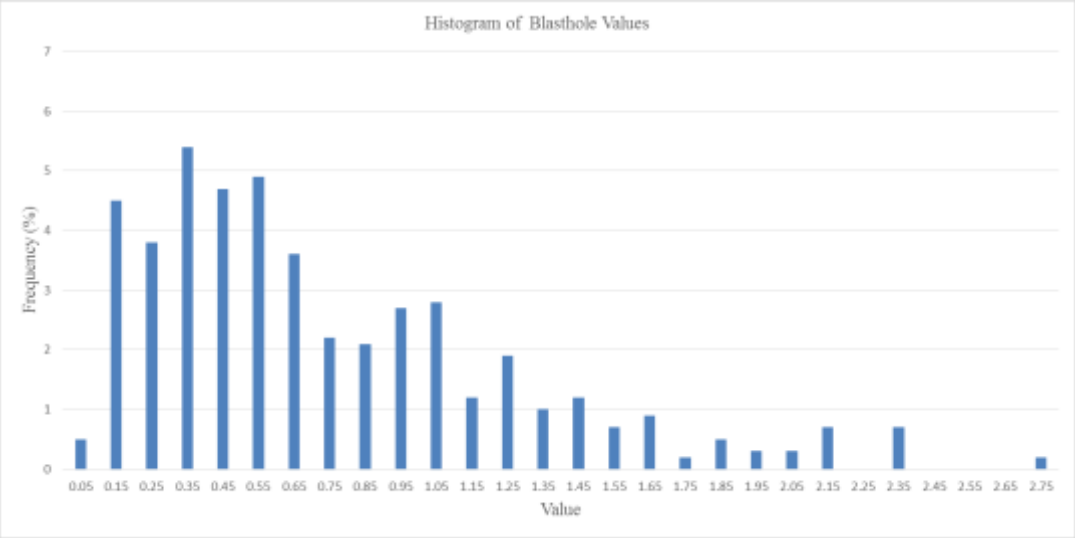


Figure 2.6 Histogram of blasthole values

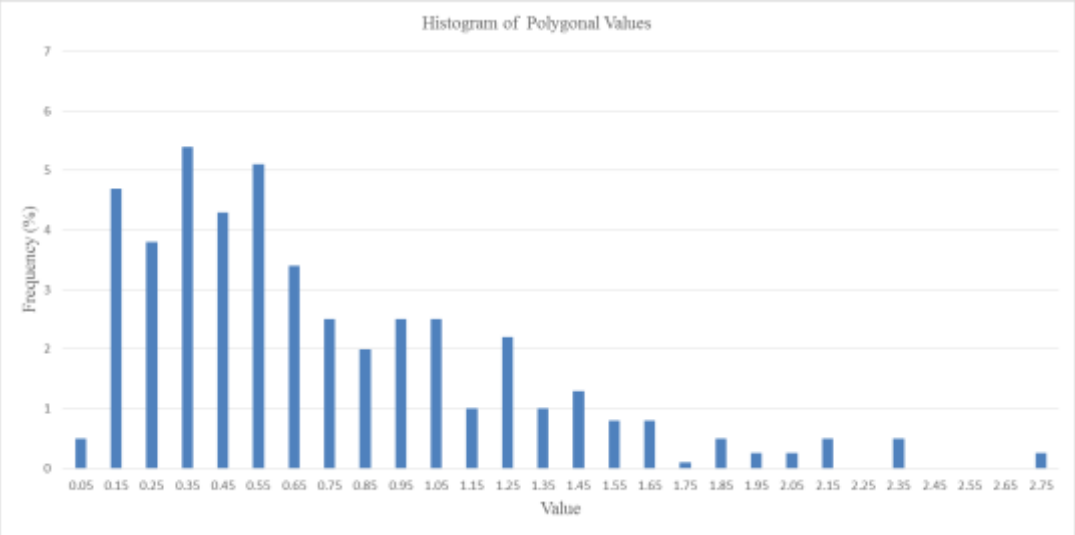


Figure 2.7 Histogram of polygonal values

2.4.1.2 Square of Inverse Distance

With this method, the mine planner simply specifies a range in each direction to search for samples. Search ranges are norm ally determined by a variogram or from examination of the host rock geology and observation of the nature of mineralization. The search range must also be sufficient to gather enough close samples but not include too many distant ones. The quadrant or octant search method is helpful in accomplishing this and ensuring that samples from all round the block are used in the block estimate. (Anderson, 1999).

The inverse distance method then uses the selected samples within the search range to estimate the block. Samples are given weights that are inversely proportional to their distance from the center of the block being estimated. The following equation gives the formula used to calculate the block grade using the inverse distance method. The value of 'n' in the equation is the number of samples within the search range (Anderson, 1999).

$$Block\ Grade = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i^{power}}}{\sum_{i=1}^{n} \frac{1}{d_i^{power}}} \tag{2.5}$$

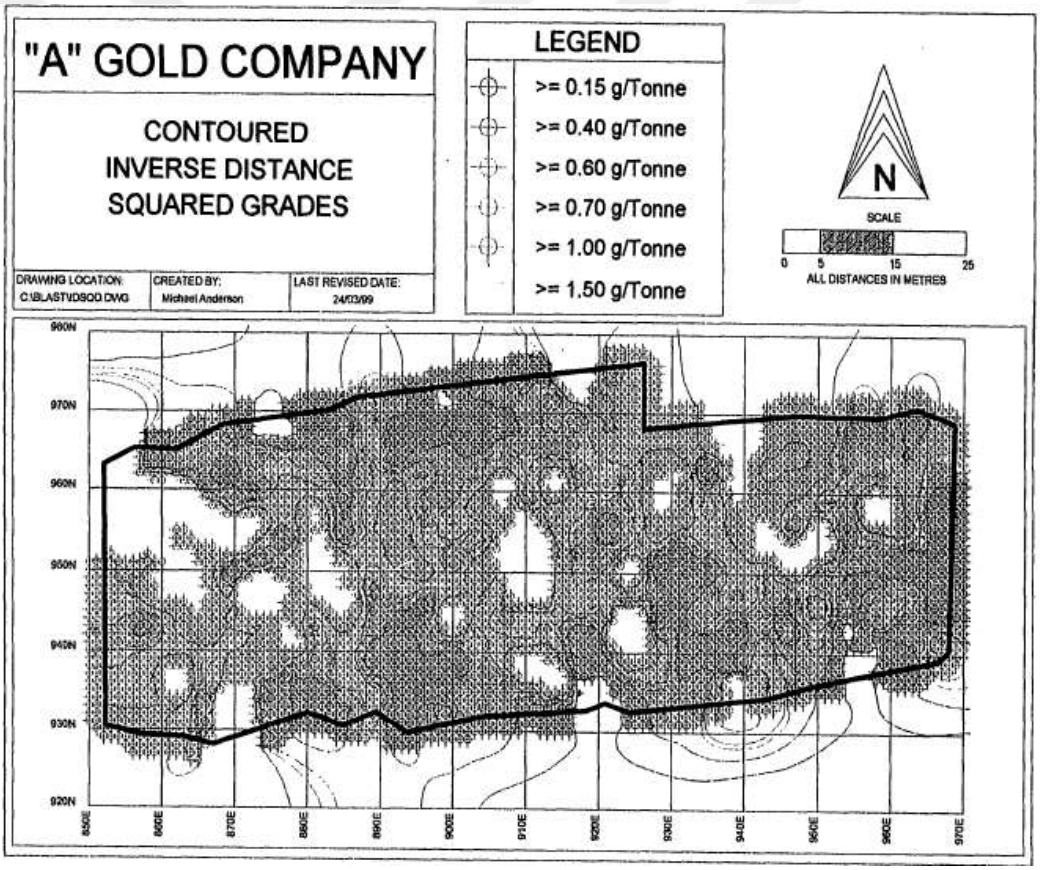


Figure 2.8 Inverse distance squared estimation method realization (Anderson, 1999)

The value of the power is usually two or three. The selection of the power may appear to be somewhat arbitrary, but is based on the type of mineralization and geology of the area in question. As the value of the power gets higher, the estimate approaches that of the nearest neighbor method (i.e. the closest sample receives a weight much greater than that of any other sample). If the value of the power is one, then the weights are linearly assigned and estimates produce a more smoothed realization. Figure 2.8 shows the example blast when estimated using inverse distance squared method (in this case with a power of 2, search radius (range) of 15 m and 1 sample per quadrant) (Anderson, 1999).

An advantage of the inverse distance method over that of the polygon method is that overall error variance is usually less because it uses more than one sample to arrive at an estimate. This provides more confidence in each estimate. The main drawback of this method is that it does not automatically handle clustered samples. As well, by using more than one sample to make an estimate, smoothing is bound to occur and the extreme values of the data set will not be represented. This is clearly shown by comparing the low grades in Figure 2.8 with those of Figure 2.11 and noting the rounded contour boundaries in Figure 2.8. Figure 2.9 shows the effect of smoothing on the estimates. The histogram shows a smoothed appearance as values are averaged together and thus closer to the mean value (Anderson, 1999).

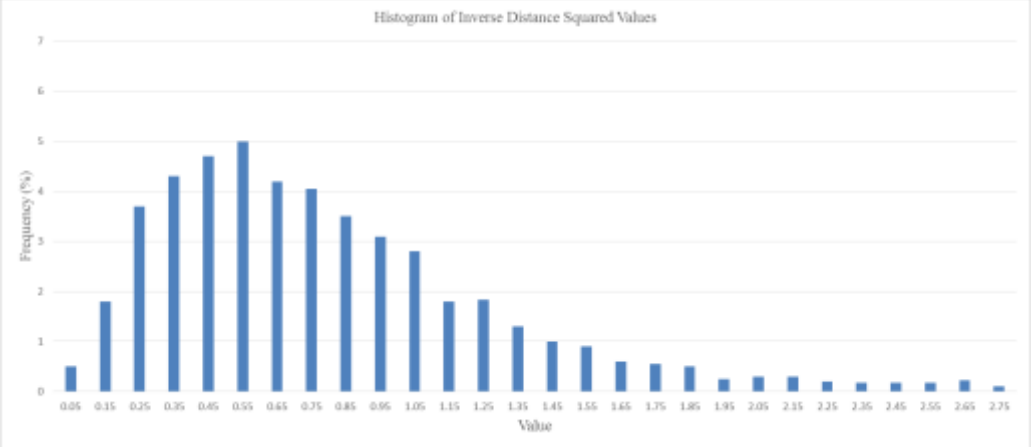


Figure 2.9 Histogram of inverse distance squared values

Historically, inverse distance power methodology resolved many of the problems to be faced later by geostatistics. Typical examples (Gauthier & Gray, 1971) include;

- Desurveying, to find the co-ordinates of samples in three dimensional space
- Micromaps, to define areas/volumes of similar rock type
- Quadrant and octant searches, to find the closest samples and avoid clustering
- Compositing samples to suitable lengths; and elliptical rotation of coordinates
- Parallelopipedic search volumes to allow for anisotropy (Anderson, 1999).

2.4.2 Geostatistical Methods

Champigny (1993) provides a brief history of the development of geostatistics, which was formalized by G. Matheron in 1965, but was based on random function theory first used by B. Matem in the forestry industry around 1962. Early work on what became geostatistics is attributed to D.G. Krige and H.S. Sichel working on South African gold mining projects in the late 40's and early 50's. Although geostatistics initially only encompassed the simple and ordinary kriging estimation methods, it has grown and developed from the late 60's onwards to encompass many different kriging approaches and the newer simulation methods. Unfortunately, the early lack of acceptance in the English speaking world resulted in inappropriate terminology when translated from the original French by non-mining personnel. The terms 'simple kriging' (not augmenting the kriging matrix to have the sum of sample weights be unity) and 'grading' (change in variogram range from plan to section due to the large directional sample length) are examples (Anderson, 1999).

Geostatistical methods make a great deal of use of the variogram (or other continuity measure such as corellogram) to describe random error (nugget) and regional continuity (range and sill-nugget). The advantage of this is that the amount of estimation error involved with each estimate can be quantified and minimized. Geostatistical methods also allow 'negative weights' to produce values that are higher or lower than the extreme values among the samples selected to perform the estimation. Another main advantage is that clustered samples are declustered

automatically, and the method further accounts for the size and shape of the block being estimated. (Anderson, 1999).

There are many different kriging variations such as disjunctive kriging, rank kriging, lognormal kriging, multi-Gaussian kriging and Bayesian kriging. The following sections will cover two kriging methods that are important and relevant to the explanation of sequential indicator simulation in the following chapter. For more information on other geostatistical methods the reader is referred to the book "GSLib: Geostatistical Software Library and User's Guide" by C. Deutsch and A. Joumel, 1998.

2.4.2.1 Ordinary Kriging

Estimation using ordinary kriging is more complicated than the traditional methods mentioned previously. Unlike the other methods that assign weightings to samples based purely on their proximity to the point being estimated, ordinary kriging uses the results from the variogram to weight the samples appropriately (Anderson, 1999).

As mentioned previously, covariance values can be obtained from the modeled variogram by subtracting the variance from the sill value. These covariance values are then used to obtain the weightings using the matrix equation shown in Equation 2.6. (Anderson, 1999).

$$[C_{ss}]x[weightings] = [C_{SB}] \tag{2.6}$$

$$[weightings] = [C_{SS}]^{-1}x[C_{SB}] \tag{2.7}$$

The sample to block covariance can be expressed as a linear combination of the sample to sample covariances. Multiplying both sides by the transpose of the sample to sample covariance matrix yields the weightings. In the equations shown, C_{ss} is the sample to sample covariance and C_{sb} is the sample to 'block shape/size1 covariance. The sample to sample covariance matrix handles the declustering of the samples while

the sample to block vector handles the sample geometry with respect to the block and the block size and shape (Anderson, 1999).

Figure 2.10 shows grades estimated using ordinary kriging. From Figure 2.10 one of the main disadvantages of ordinary kriging, the over-smoothing of grades (also typical of least squares estimates) can be seen. The weightings given by solving the previous matrix equation by definition minimizes the estimation error. This is the main advantage of kriging as one of the obvious goals of estimation is to reduce the error associated with any estimate. However, in doing so, the method tends to create block estimates that are more of a moving average, thus smoothing high grade areas and lower grade areas and losing the extremes found in practice. This can produce values which collectively overestimate tonnage and underestimate the average grade of ore, or vice versa depending on the relative value of the cut-off grade to the mean grade (Anderson, 1999).

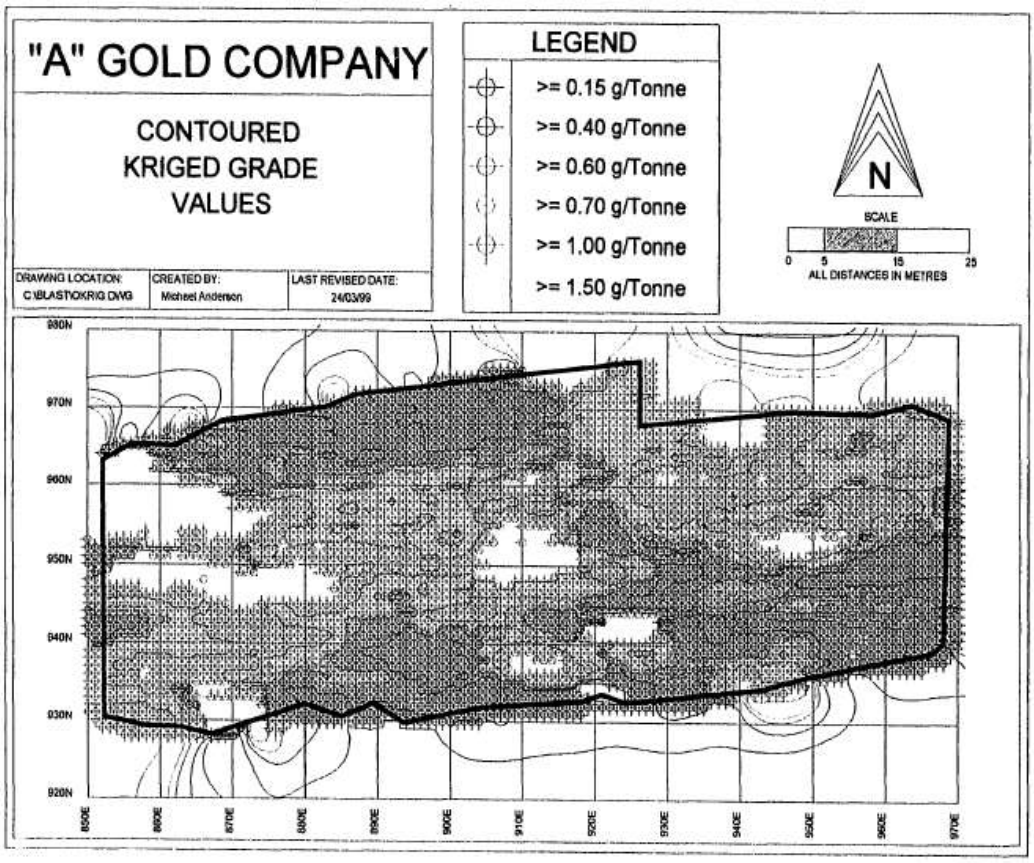


Figure 2.10 Ordinary kriging estimation method realization (Anderson, 1999)

By including the anisotropy as defined by the variogram, the smooth grade contours of inverse distance power estimates (Figure 2.13) clearly show a distinct east/west continuity in Figure 2.10. The variogram parameters used for the ordinary kriging estimation are shown in Table 2.1. (Anderson, 1999).

Table 2.1 Variogram thresholds and parameters

Azimuth	Range 1	Range 2	C_0	C_1	C_2
86	6	20	0.25	0.05	0.7

The histogram from ordinary kriging shows less grade smoothing than that evident in the inverse distance squared method, and that the mode of the distribution has moved to the rig h t in both estimates, and is a result of grade smoothing. In Figure 2.6 and Figure 2.7 the mode is at 0.35 g/ton while in Figures 2.14 and 2.11 the mode= has moved to 0.55 g/ton (Anderson, 1999).

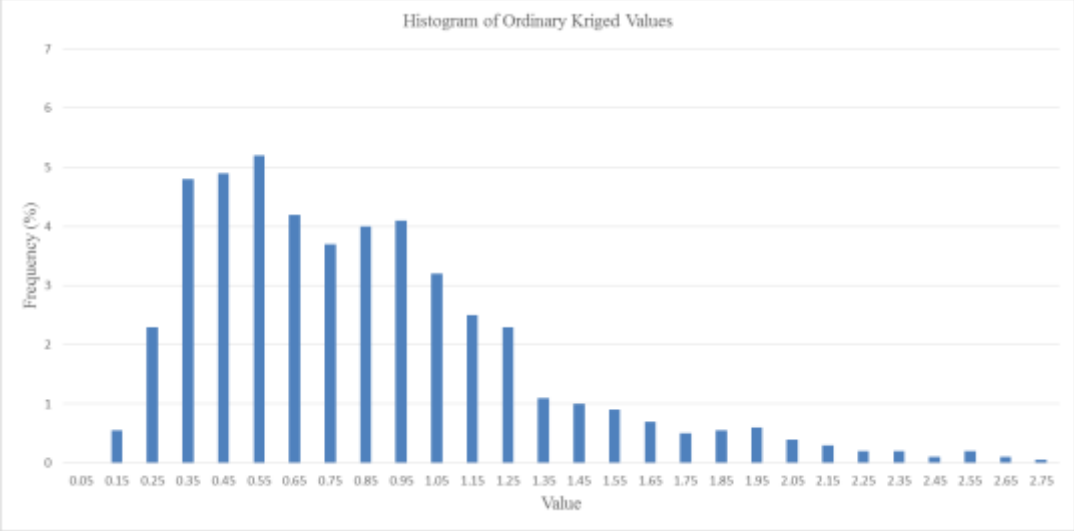


Figure 2.11 Histogram of ordinary kriged values

2.4.2.2 Multiple Indicator Kriging

The polygon, inverse distance and ordinary kriging methods are all more or less appropriate for estimating a mean value for a mineral deposit, but the mining engineer requires accurate values for the tonnage and grade above some economic cut-off grade. Kriging attempts to estimate the local mean of each block and does a satisfactory job

of reducing the error of each estimate. The polygonal method does an acceptable job of estimating the global distribution of grades if high-grade outliers are reduced, while the inverse distance squared method serves as a good intermediary between ordinary kriging and polygonal estimation methods. Occasionally, it is more important that other characteristics of the data set are reproduced besides the mean grade. Multiple indicator kriging is a method that attempts to determine the local distribution of grades and obtain the expected value from the distribution. This is a useful technique in deposits that have highly skewed grade distributions (Anderson, 1999).

Multiple indicator kriging is a simple variation on kriging where the kriged estimate for each block is not the grade, but the probability that the block's grade is below a certain threshold or cutoff. The weightings for each sample are calculated in exactly the same manner as for ordinary kriging. However, instead of multiplying the weightings by each sample's grade they are multiplied by 1 if the sample is below the threshold and 0 if it exceeds the threshold. Typically, various thresholds are used in order to find a probability distribution for the grade of the block (Anderson, 1999).

If for example, five thresholds were calculated, each one with an increasing grade value, then the indicator kriging results would produce five different probabilities. These probabilities would be the probability that the grade of the block is below the corresponding threshold value. A mine planner could then use the average grade for all the samples within each threshold and multiply it by its probability of occurrence. The sum of grade and probability, plus the probability of exceeding the upper threshold times the average grade above that threshold, provides the block grade estimate according to probability theory. The major advantage of this method is that it creates estimates that follow the local probability distribution. Another main advantage of multiple indicator kriging is that different variograms can be used with each threshold. Therefore, the method achieves realistic modeling of high grades having short ranges (little continuity) plus high nuggets (poor repeatability of sample grades), versus low grades with long ranges and low nuggets. This is advantageous in deposits where mineralization is erratic, as this method will not create realizations as smoothed as with ordinary kriging. The histogram shown in Figure 2.12 shows how the multiple

indicator kriging method produces a distribution very similar to that of ordinary kriging. However, it better represents the low-grade material with the mode being at 0.45 g/ton (Anderson, 1999).

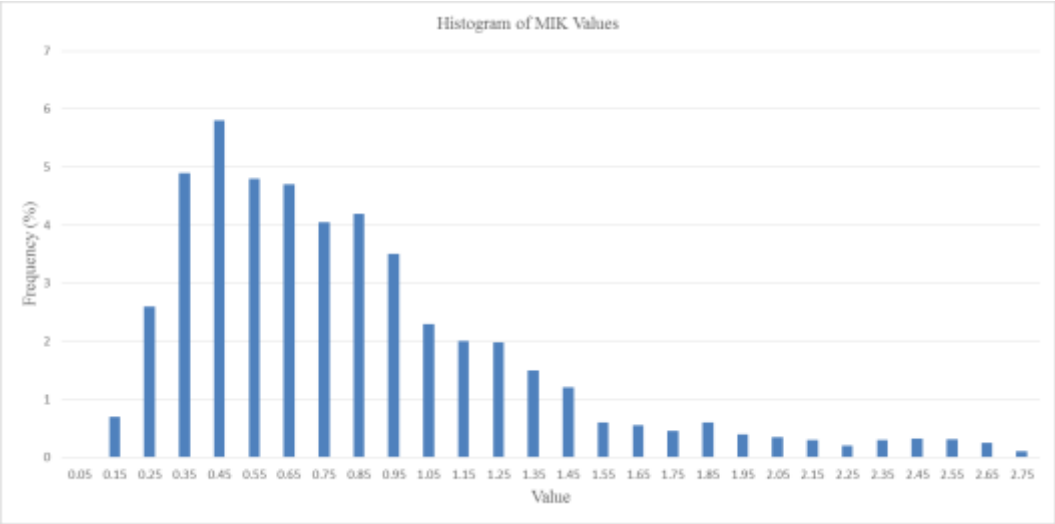


Figure 2.12 Histogram of MIK values

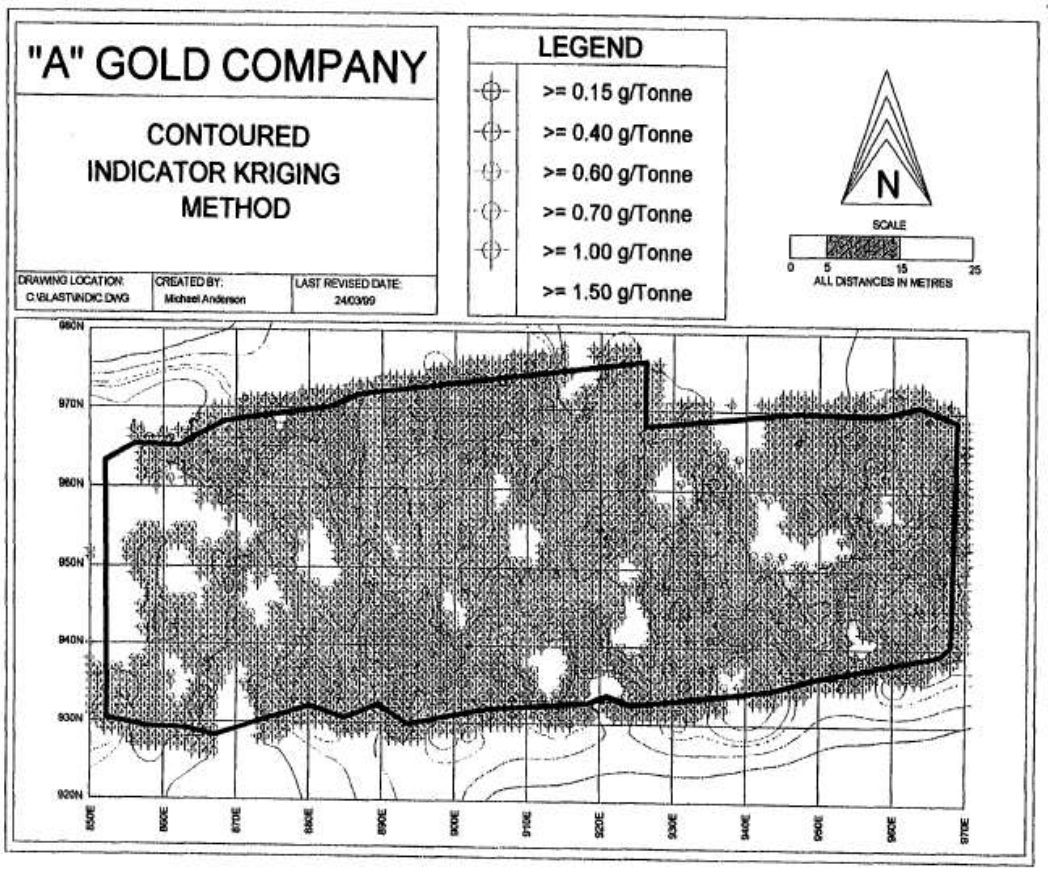


Figure 2.13 Multiple indicator kriging estimation method realization (Anderson, 1999)

Figure 2.13 shows the original blast from Figure 2.1 when estimated using multiple indicator kriging. The use of separate variogram s for various grade groups has emphasized local north-south trends which can be seen in Figure 2.1, the contour plot of the original data. The indicator variogram parameters used are shown in Table 2.2. (Anderson, 1999).

Table 2.2 Indicator variogram thresholds and parameters

Indicator Threshold	**Azimuth**	**Range 1**	**Range 2**	**C_0**	**C_1**	**C_2**
0.15	86	10	261	0.357	0.330	0.313
0.34	86	13	244	0.750	0.226	0.399
0.43	86	12	249	0.346	0.246	0.408
0.57	86	12	246	0.291	0.291	0.419
0.78	86	13	241	0.304	0.350	0.347
1.22	86	11	235	0.302	0.355	0.343
2.48	94	14	159	0.344	0.396	0.259
3.91	94	13	119	0.256	0.578	0.166
9.38	94	10	60	0.260	0.655	0.085
14.75	-	12	25	0.504	0.356	0.140

Figures 2.1, 2.10, 2.11, 2.12 and 2.13 provide a visual understanding of the interpolation of block estimates from sample data. The polygonal method may provide numerically adequate grades, but visual performance of the estimator is poor. Without structural input, the inverse distance power estimates are both visually and numerically inferior. The simple description of continuity available from a single variogram distorts the visual appearance of the ordinary kriging results. Only with the more complex description of continuity provided by multiple indicator kriging are block grade estimates both visually and numerically acceptable (Anderson, 1999).

CHAPTER THREE
STOCHASTIC MODELING AND SIMULATION

Within the last fifteen to twenty years many new methods have been developed that attempt to improve the estimation of unsampled points in space. These newer methods usually focus on modeling one particular aspect of the true data set while minimizing the error associated with that characteristic. The reality is that no method can reproduce the true data set and the error involved depends greatly on the individual entering the parameters and performing the estimation. Therefore, one more point should be added to the quotation at the beginning of Chapter 2; *'experience at estimation is vita*/'. (Anderson, 1999).

In the late 80's the U.S. Environmental Protection Agency performed an experiment in which 12 different well-respected geostatisticians were given the same sample data set and information regarding contaminants on a property and asked to create a model of the contaminant values. These experts came up with 12, quite different models. The study showed that estimation is not an exact science and should be handled with caution. It demonstrated that any individual's attempt to produce one realization does not necessarily produce the most likely realization (Isaak's & Srivastava, 1989).

Unfortunately, all of the classical estimation methods mentioned so far produce their own individual realizations. It would be more useful if equally probable realizations could be produced, and sequential indicator simulation can be used to accomplish this. The benefit of simulation for the mine planner is that used to delineate ore or waste with a higher level of confidence. Sequential indicator simulation is based on work by Joumel (1974) on Monte Carlo simulation and probability kriging (Alabert, 1987), as well as spiral search theory work by Gomez-Hemandez and Srivastava (1990).

3.1 Probability Distributions

Sequential indicator simulation can be viewed as a variation on multiple indicator kriging. As mentioned in the previous chapter, multiple indicator kriging determines the local probability of exceeding a series of threshold values. Thus, one of the initial requirements for simulation is knowledge of the global distribution that the estimation is to follow such that the threshold values can be selected (Anderson, 1999).

Once a sample data set has been collected through exploration drillhole sampling or through blasthole sampling, the first step is to examine the probability distribution of the data set. The probability distribution has several uses apart from helping to decide on threshold values and defining the cumulative distribution function. If representative sampling has been conducted throughout the area of mineralization, the probability distribution can be used to gain insight into the approximate amount of material above certain grades. In combination with the grade-tonnage curve, it gives an indication as to how fluctuations in the metal price might affect the project's profitability (Anderson, 1999).

The threshold values are chosen such that the entire distribution is represented and grades in between thresholds can be linearly interpolated.

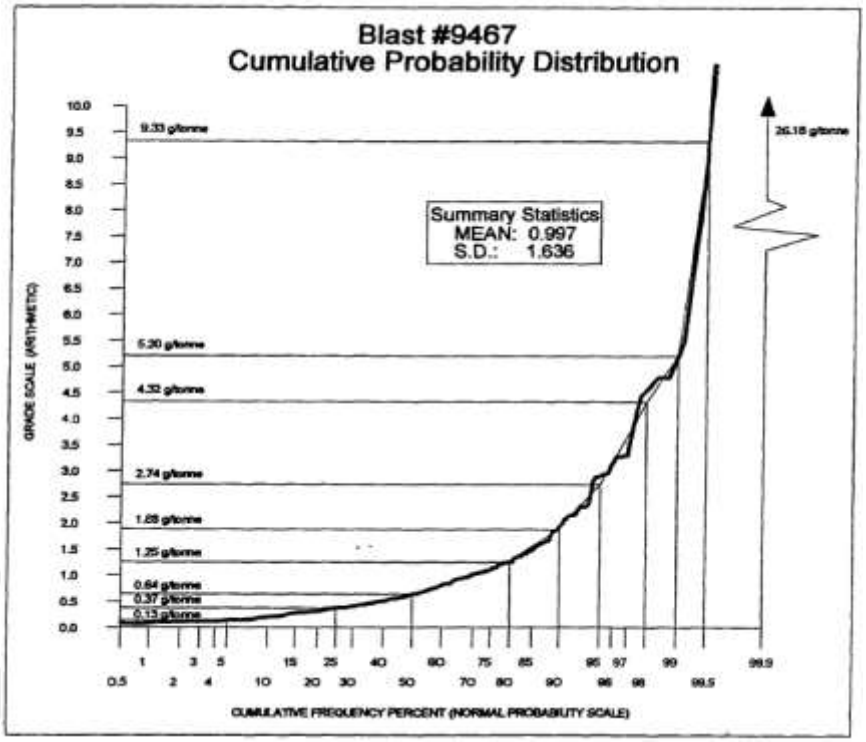


Figure 3.1 Cumulative probability distribution (Anderson, 1999)

Figure 3.1 shows a cumulative probability distribution for the blasthole data collected for the example blast in Figure 2.1. The probability plot shown in Figure 3.1 has a 'lognormal' shape that is common to many mineral deposits. A large percentage of the material has a relatively low grade (70 percent below 1.00 g/ton in this example) while a small percentage of the data is in the high-grade range. Gold grades in this upper range (from 70 to 99.9) vary from 1.00 g/ton to 26.18 g/ton. Therefore, if this company currently has a cutoff grade of 1.00 g/ton and for some reason has to increase the cutoff grade to 1.50 g/ton, approximately 15 percent of the total mineralized deposit would change from ore to waste (Anderson, 1999).

3.2 Use of Simulation Results

After the simulations have been completed, the mine planner is left with multiple answers for each block in the model. Methods of combining the simulations and classifying the material as ore or waste using these multiple answers must be investigated, and three techniques will be discussed in this section. It should be noted

that for the example blast used thus far, in order for material to be classified as ore it must have a gold grade greater than or equal to 0.5 g/ton (Anderson, 1999).

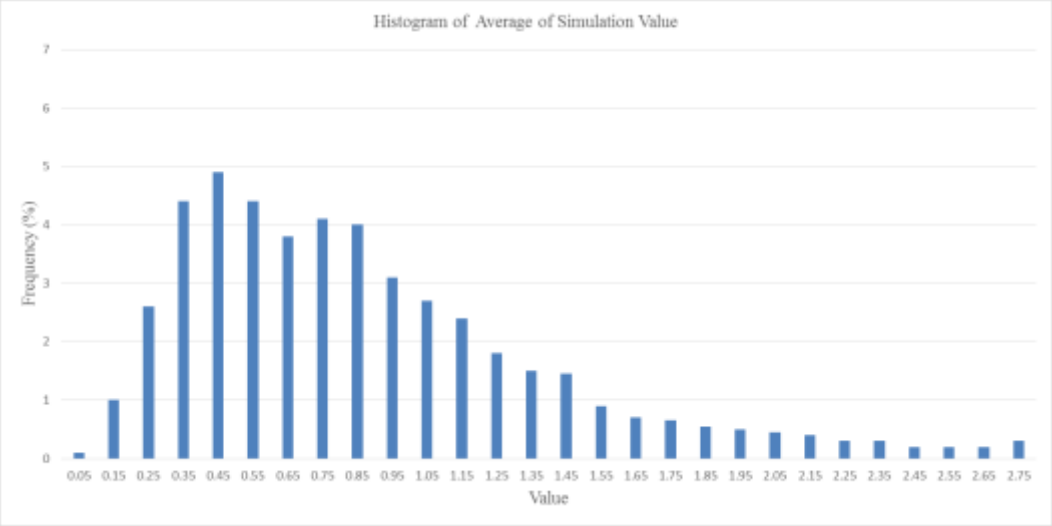


Figure 3.2 Histogram of average of simulations

3.2.1 Grade-Tonnage Curves

To further examine similarities between the methods examined thus far, one can look at a grade-tonnage curve for the example blast. Figure 3.3 shows a grade tonnage curve for all of the methods. The purpose of a grade-tonnage curve is to outline the quality and amount of material available above a certain grade (Anderson, 1999).

Grade, tonnage and 'ounce' curves are important in understanding the economic strength of a mine. To produce such curves a series of cut-off grades, often converted to revenue, are applied to the block model of any volume from an individual blast to an entire orebody. At a zero cut off, the grade is the average grade of the volume, and the tonnage the total tonnage in the volume. As not all material can be profitably mined, a cut off is applied, and only the material above cut off is included in the calculation of ‘ore’ grade and tonnage. As the cut off increases, the ore tonnage is reduced, but the average ore grade increases (Anderson, 1999).

Thus for the grade-tonnage curve of Figure 3.3, the “Sim Avg Tonnage” value refers to the “individual simulation average” ore tonnage available for all of the simulations at a particular cut-off. Conversely, the “Avg Sim Tonnage” refers to the ore tonnage available when all of the simulations are averaged together to produce the “average of simulations”, and then the cut-off is applied to the realization formed (Anderson, 1999).

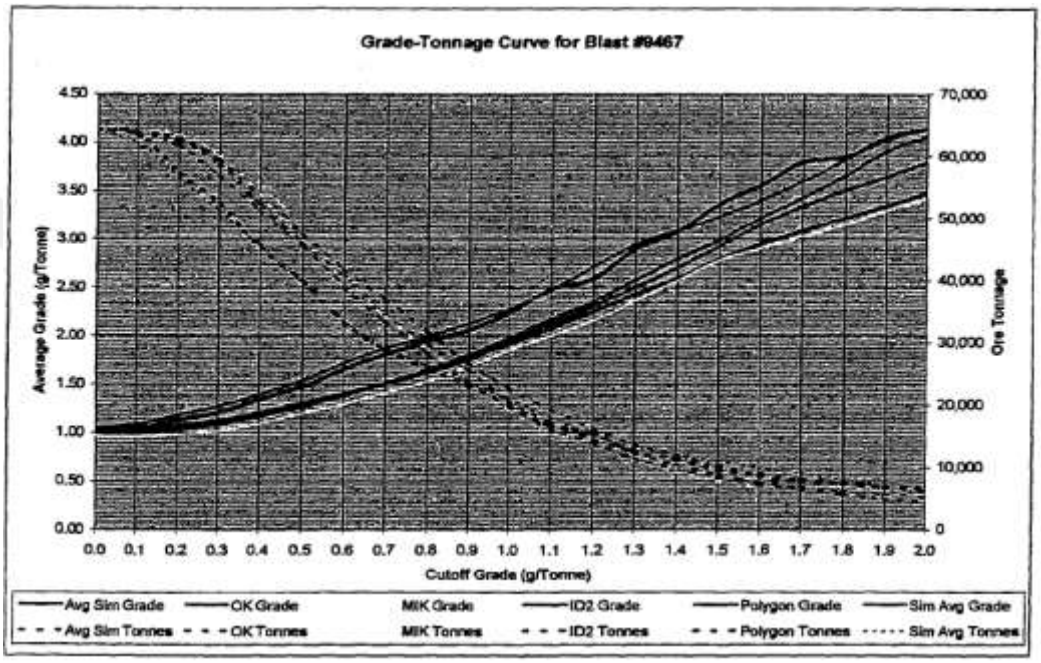


Figure 3.3 Grade-tonnage curve for different estimation methods

From Figure 3.3 a number of conclusions can be made. As seen by the histograms for each estimation method, the inverse distance squared, ordinary kriging, multiple indicator kriging and average of simulations all yield relatively similar and smoothed pictures of the distribution of grades. This is reflected in the grade-tonnage curve. The blue (Average of Simulations) and yellow (Multiple Indicator Kriging) lines are very similar as expected. The red (Ordinary Kriging) line is fairly similar to the blue and yellow except at the high-grade range where it loses some similarity. This is expected since one of the main goals of indicator kriging is to control the influence of high-grade areas of the deposit The inverse distance method (green) line is also shown for comparison and gives a result a little closer to the polygonal method, but still within the group being described (later referred to as ‘Group 2’) (Anderson, 1999).

The black lines (Polygonal Method) are different from those of the other methods. This is an important feature to notice since the polygonal method reproduces the histogram in a much more reliable fashion (i.e. the grade-tonnage curve is more representative of the samples) even though it usually shows a higher variability from true grades than the other methods when mining is actually carried out on individual blocks (Blackwell, 1995). One of the advantages of simulation is that it reproduces this same distribution of sample grades and therefore the red line (Individual Simulation Average) follows the black line very closely. These two realizations are referred later as ‘Group 1' (Anderson, 1999).

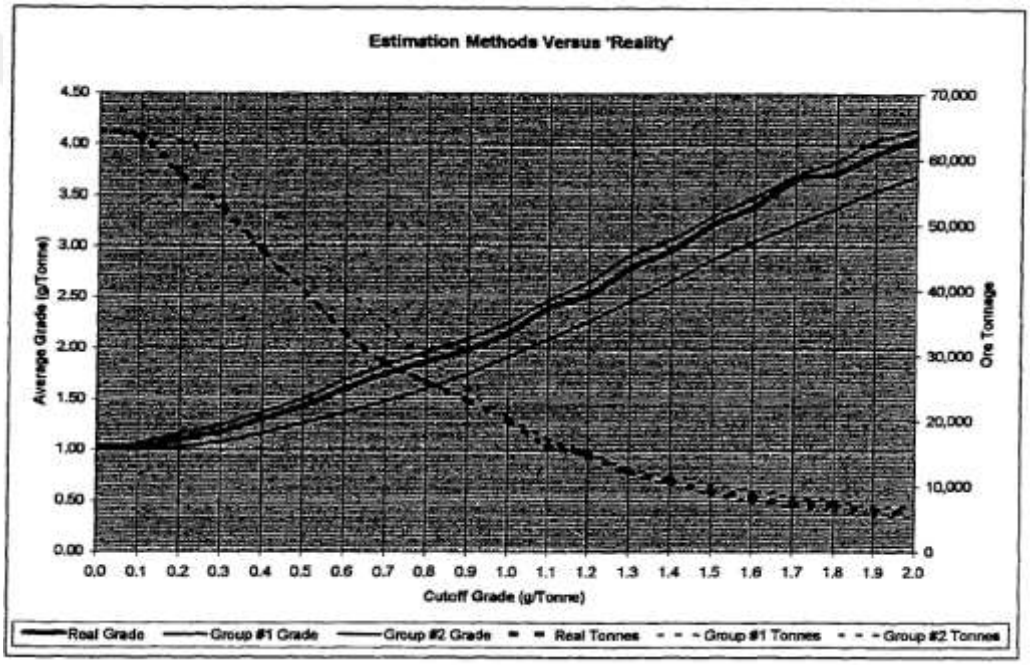


Figure 3.4 Grade-tonnage curve for sample data and estimation groups

Figure 3.4 illustrates one of the main advantages of simulation. This advantage is its ability to replicate the results of other more traditional methods. The polygonal method provides realistic grade-tonnage curve results as the histogram matches that of sample reality very closely. As is evident in Figure 3.3, sequential indicator simulation can be used to form accurate grade-tonnage curves, and the polygonal and individual simulation average data are combined due to their similarity as the red line, to make 'Group T, and clearly follow the black sample data curves in Figure 3.4. Unfortunately,

as seen in Figure 2.11 and as will be shown in Chapter 5, the polygonal method is not very accurate at predicting grades on a local level (Anderson, 1999).

For this, multiple indicator kriging is a proven method for estimating the grade at individual locations accurately. Figure 3.3, as well as the similarity between Figures 2.13 and 3.6, shows that simulation results can be used to replicate those of multiple indicator kriging. This group ('Group 2' includes ordinary kriging, inverse distance power method, average of simulations and multiple indicator kriging) is shown by the green lines in Figure 3.4 (Anderson, 1999).

The main advantage of simulation, as will be outlined in the rest of this chapter, is its ability to incorporate the random variability of grade values into the decision making process (Anderson, 1999).

3.2.2 Averaging

Perhaps, the most obvious method of combining the simulations and classifying the material as ore or waste, is simply to average the simulations together and compare the average grade of each block to the cutoff grade of 0.5 g/ton (average of simulations). Figure 3.6 shows the realization produced by averaging all of the simulations. As evident in both the appearance of Figure 3.6 and the histogram of Figure 3.2, the results are quite comparable with multiple indicator kriging. Both methods produce a realization showing a smoother transition from high grade to low grade material. As mentioned in section 3.3, the average o f a number of equally probable realizations gives the expected value, and obtaining this is the goal of multiple indicator kriging (Anderson, 1999).

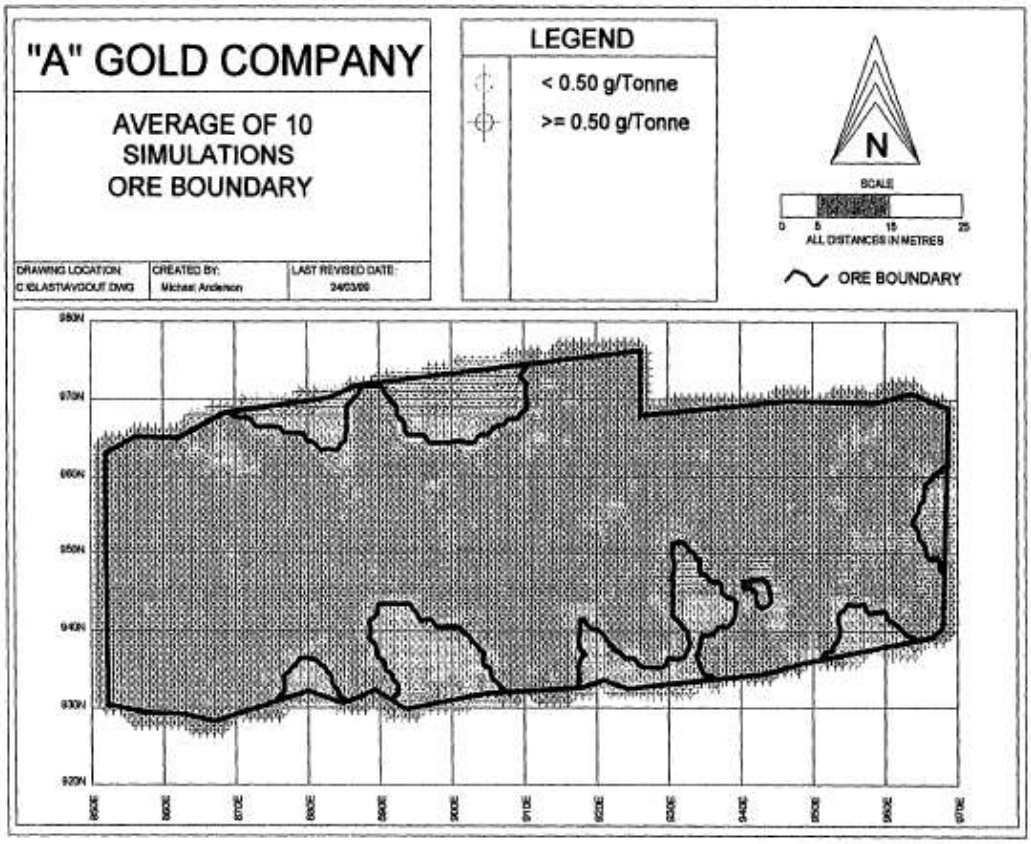


Figure 3.5 Digging boundary for averaging method (Anderson, 1999)

Figure 3.5 shows how the blast looks when all of the simulations are averaged and then the 0.5 g/ton cutoff applied. The resulting classification is much the same as would be formed by using multiple indicator kriging. The advantage of using simulation, is that once this classification has been carried out using the averaging method and the ore and waste boundary has been formed the effects of applying this same boundary to each of the simulations in turn, can be examined. This allows the mine planner to determine the expected variability in average grade that the mill can expect, the amount of internal dilution and ore loss (Anderson, 1999).

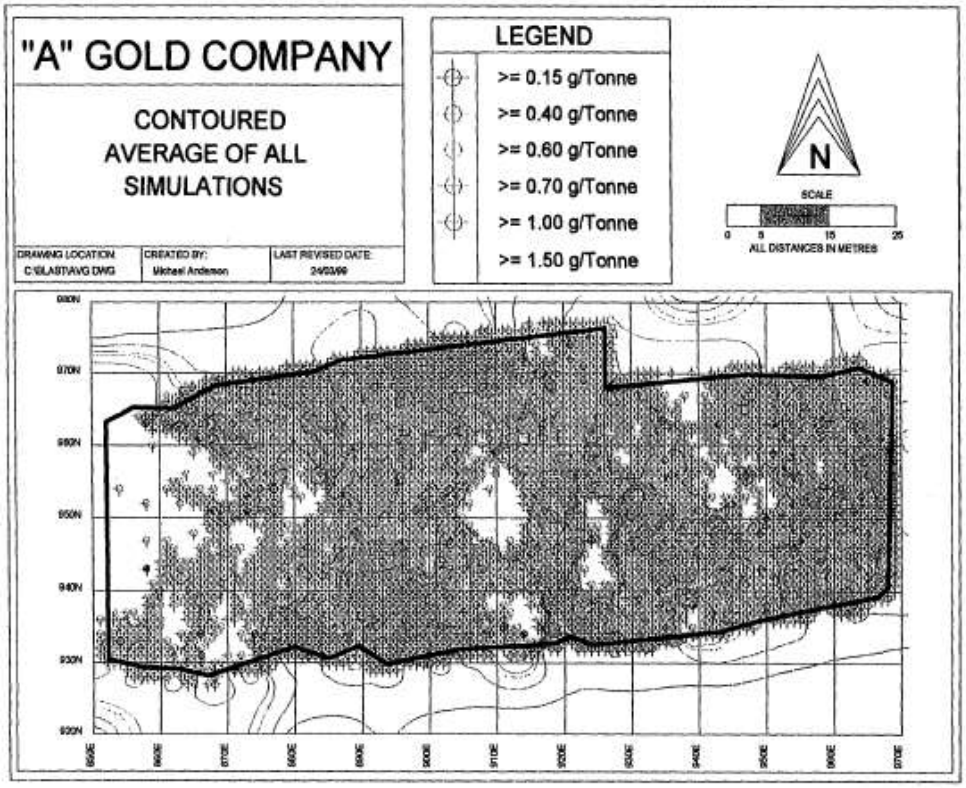


Figure 3.6 Average of all simulations (Anderson, 1999)

Table 3.1 lists the average grade, internal dilution and ore loss within the ore polygon shown later in Figure 3.5. Also shown, is the range of these same values based on the individual simulations. From the values in Table 3.1 some general conclusions can be made. Although, averaging the simulations and multiple indicator kriging may give accurate polygon grades, the amount of internal dilution and ore loss to expect is not accurate. Simulation provides the mine planner with a tool for examining the variability in grade to expect at the mill as w ell as the true amount of internal dilution and ore loss to expect (Anderson, 1999).

Table 3.1 Simulations for ore grade, dilution and ore loss

	Ore Grade (g/tonne)	Internal Dilution (%)	Ore Loss (%)
Average of Simulations	1.224	10.49	2.82
Simulation Minimum	1.163	26	4.39
Simulation Maximum	1.328	33.37	6.6
Individual Simulation Average	1.224	28.87	5.64

3.2.3 Probability

Another method of classification is to look at the probability of each block being ore or waste. This method reduces the effect of outliers and gives the mine planner a tool for establishing confidence that what is designated as ore is ore.

The probability is calculated by counting the number of times that a particular block's estimate is above the cutoff versus the total number of simulations performed. For example, if there are ten estimates for each block and a particular block has six of the ten estimates above the cutoff then there is a 60 percent chance that the block is ore. Typically, if there is a 50 percent chance or greater o f a block being ore then it is classified as ore. Figure 3.7 shows classification of the blast using a 50 percent probability cutoff (Anderson, 1999).

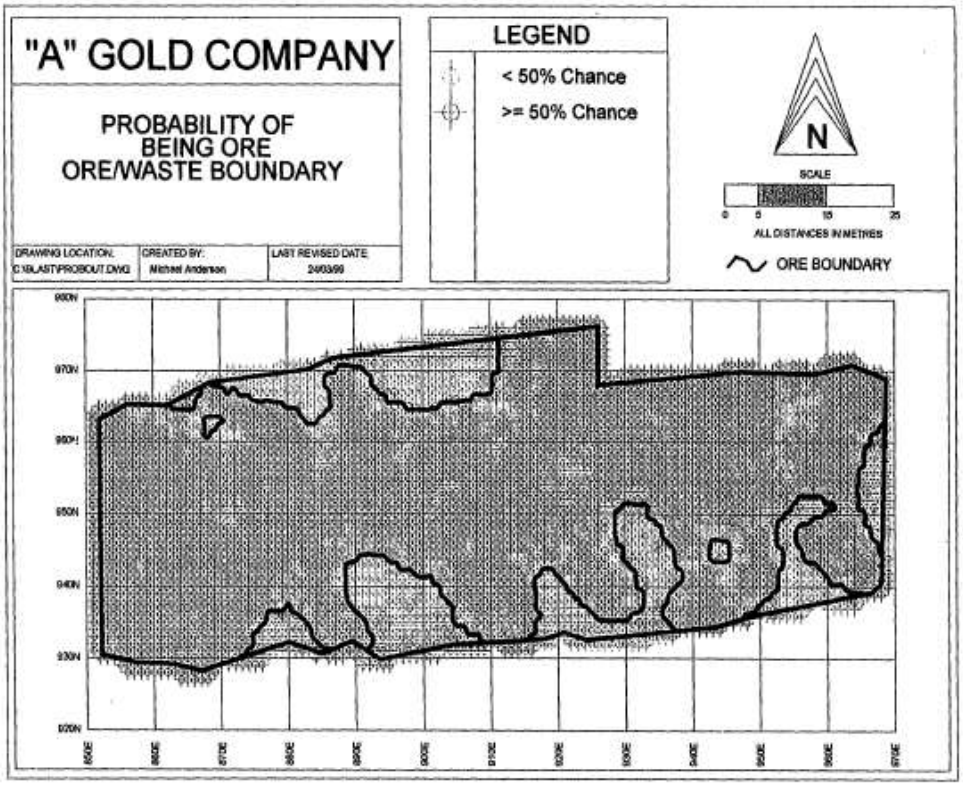


Figure 3.7 Digging boundary for probability method (Anderson, 1999)

This method is very similar to that of averaging the results yet extreme values are not given the influence that averaging allows them. If the mine planner wants to be more or less conservative and allow or deny marginal material in the ore polygon then the required probability can be adjusted without altering the overall cutoff grade. For example, if the mine has limited milling capacity, then raising the required probability to 60 or 70 would help to ensure that what enters the mill is more likely to be ore (Anderson, 1999).

Figure 3.8 shows a realization outlining the probability of each block being greater than or equal to the cutoff grade of 0.5 g/ton.

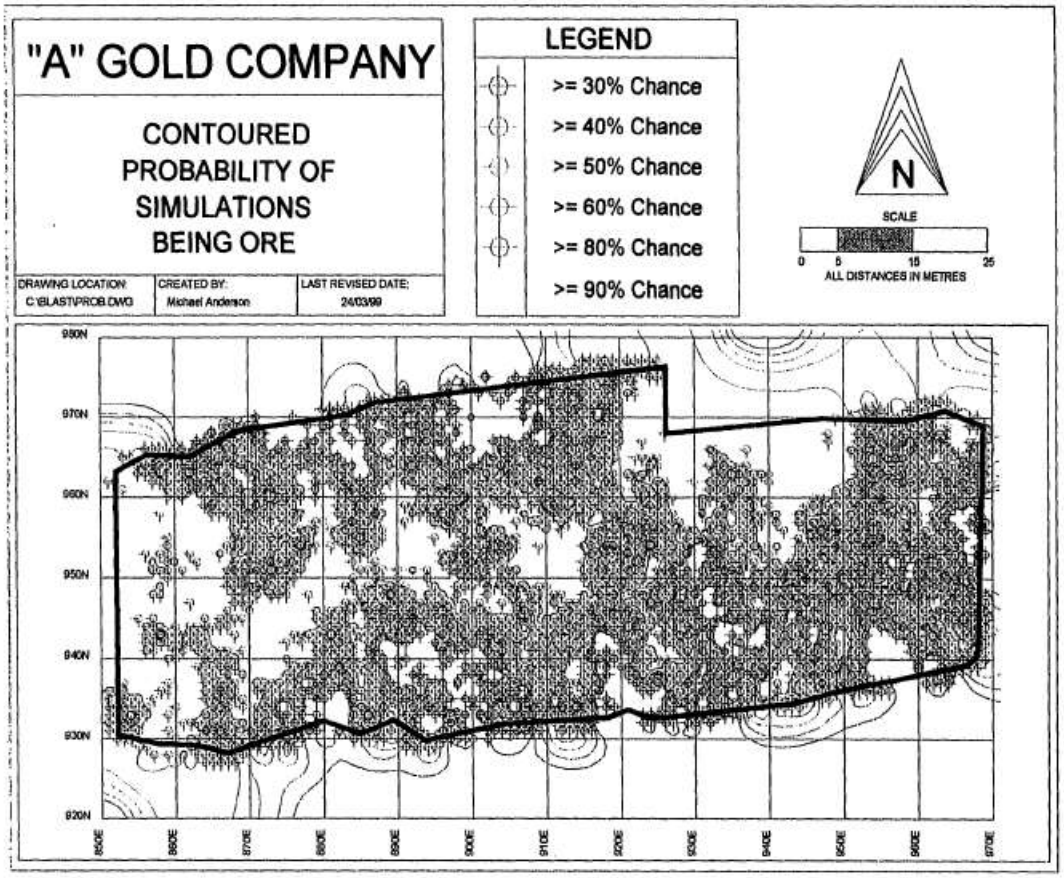


Figure 3.8 Probability of being ore (Anderson, 1999)

In an erratic deposit, estimates can vary greatly between simulations. Therefore it is sometimes important to limit the effect that high-grade or low-grade outliers can have. For example, if a particular block has eight of its ten estimates below the cutoff and yet one or two of the remaining estimates are sufficiently high to bring the average grade above 0.5 g/ton then the averaging method designates this block as ore. The probability method on the other hand informs the mine planner that there is a large probability that the block is waste and should be classified as such. Figure 3.9 shows a grade-tonnage curve for different probabilities. For Figure 3.9, the cut-off grade has been held constant at 0.5 g/ton but the required confidence for a block to be called ore is allowed to change (Anderson, 1999).

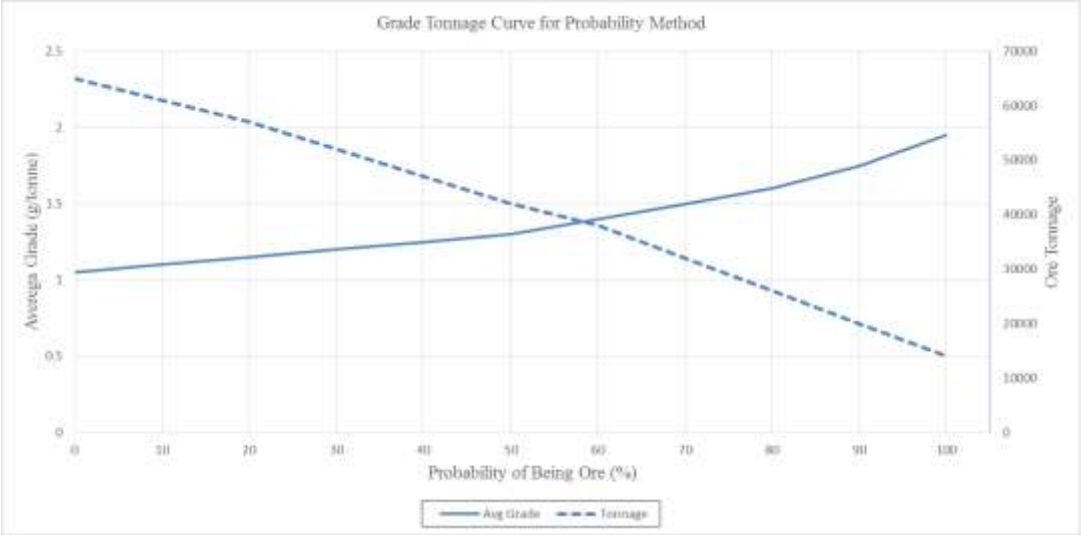


Figure 3.9 Grade-tonnage curve for different confidence levels

Figure 3.9 provides the mine planner with some confidence as to what is available in the blast. For example, the mine planner can tell that there is at least 12,725 tons of ore available at an average grade of 1.96 g/ton. Likewise, the mine planner can see that by applying a 50 percent confidence of being ore there will be 44,088 tons of ore at an average grade of 1.34 g/ton (Anderson, 1999).

3.2.4 Loss Function

While these methods are valid for classifying ore, the mine planner may wish to make more complete use of the information available. There are a number of estimates for each block, and therefore the planner can gain insight into the variance of each block estimate. This information can be used to classify each block in a manner that will minimizes the loss experienced by the company from any misclassification of material (Anderson, 1999).

Kriging by nature, attempts to minimize the error of the estimate and in doing so assumes that the penalty for misclassification of ore or waste is the same (i.e. underestimation is as bad as overestimation). However, the loss to a company, monetarily, for processing waste material (overestimation) is not necessarily the same

as the loss associated with sending ore to the waste dump (underestimation) (Anderson, 1999).

The mine planner can use the simulation estimates to minimize the loss that the company would experience from the misclassification of that material. This idea of minimizing loss is known as the loss function concept (Srivastava, 1987).

The loss function is defined by the mining operation based on costs and revenues from milling waste and not milling ore, and is a useful tool for influencing material flow to meet company or project goals/objectives, and as such will differ between companies with different objectives (Anderson, 1999).

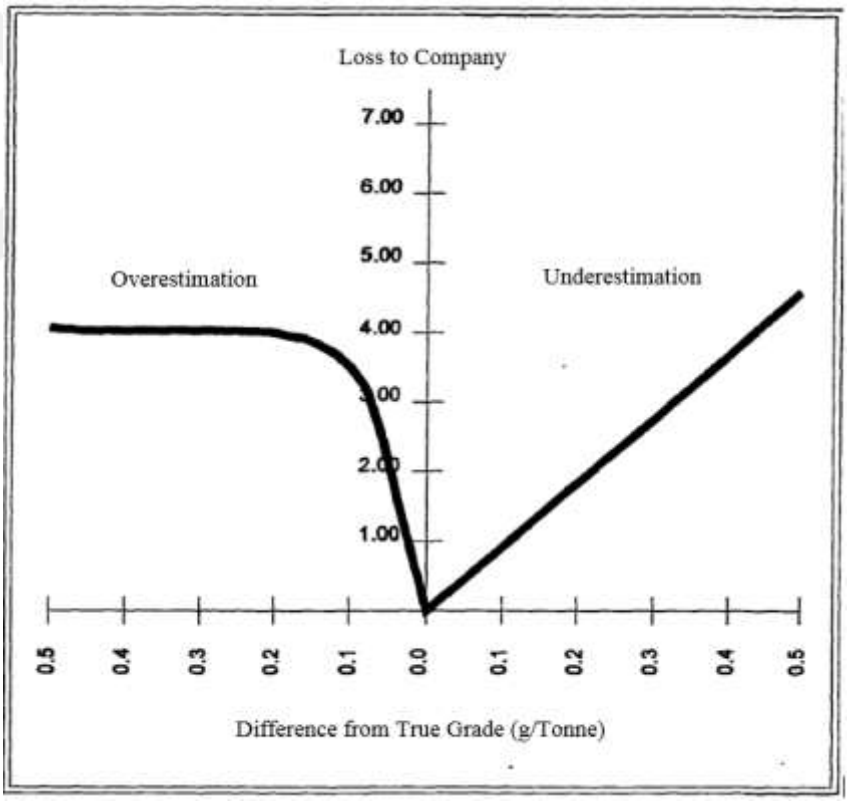


Figure 3.10 Example loss function (Anderson, 1999)

Figure 3.10 shows an example o f a loss function that will be used with the original blast shown in Figure 2.1. The loss function in this case is based on monetary loss and the desire o f a small mining company to concentrate their mining efforts on high-grade material. Overestimation, shown on the left side, is represented by a function

that rises steeply up to the milling cost and then plateaus. The right hand side of the curve is for underestimation and is based on lost revenue. Therefore, slightly sub-marginal material going to the mill must be penalized more than marginal ore going to the waste dump (Anderson, 1999).

In summary, sending waste to the ore process (overestimation) causes a loss equal to the cost of processing minus the revenue provided by the lower grade contained in the waste. Sending ore to the waste dump (underestimation) causes a loss due to the revenue that would have been provided by the ore material (Anderson, 1999).

For example, one block in the simulated blast area has the ten simulated values shown in Table 3.3. The average grade of these ten values is 0.547 g/ton, which is above the cutoff of 0.5 g/ton and six of the ten values are above cutoff suggesting again that this block is ore (probability of 60 percent). Simulation by simulation, if the mine planner assumes that each of these ten estimates is the true grade, the loss experienced by the company for misclassifying the block can be derived (Anderson, 1999).

Table 3.2 Example loss function

Simulation #	**Grade (g/tonne)**	**Misclassification**	**Underestimation Loss (U)**	**Overestimation Loss (O)**
1	0.8769	WASTE (U)	$3.07	
2	0.5805	WASTE (U)	$1.41	
3	0.1465	ORE (O)		$4.06
4	0.7135	WASTE (U)	$2.30	
5	0.296	ORE (O)		$4.06
6	0.3501	ORE (O)		$4.06
7	1.235	WASTE (U)	$4.29	
8	0.5126	WASTE (U)	$0.52	
9	0.1345	ORE (O)		$3.21
10	0.6234	WASTE (U)	$1.75	
Total			$13.34	$15.39

Table 3.3 illustrates how, by summing the loss for misclassifying the block as waste, the loss to the company is $13.34. The loss experienced in misclassifying the

block as ore, is $15.39. Since the loss for misclassification as waste is less than misclassification as ore, the mine planner should classify this block as waste, and not 'ore' as the other two methods suggest (Anderson, 1999).

This result informs the mine planner that the safest route to minimize loss is to send this block of material to the waste dump. Mathematically, the simulated grades that suggest the block is ore are not sufficiently high to show a large amount of revenue, while the simulated grades that identify the block as waste indicate processing this material would be costly (Anderson, 1999).

It should be noted that the objective of the loss function is not to quantify the expected amount of loss due to misclassification. Its purpose is to serve as a tool for classifying material according to a company's view of the relative potential costs (losses) associated with misclassification (Anderson, 1999).

The loss function method can be applied to other facets of mining such as how many rock bolts to use in an underground drift. The cost associated with putting in too many bolts is merely the cost of additional bolting (i.e. linear function). The costs associated with insufficient bolting may be things such as cleanup, lost production or even loss of life. The function defining the costs associated with these items may not be very clear and is most probably not linear. Therefore, the loss function that the company derives for this situation need not be the exact costs associated with overestimation or underestimation for the number of bolts but more a function outlining the degree of acceptable risk associated with overestimation or underestimation (Srivastava, 1987).

It should also be noted that while this thesis discusses the loss function within the context of there being only one cutoff and two classifications, ore and waste, it can also be used to classify material in situations where there are many classifications. Typically, these include multiple marginal ore stockpiles (in case metal prices rise or better technology reduces costs), and whether the material is acid generating or neutralizing (for environmental control) (Anderson, 1999).

In a situation where a mill and a heap leach cut off were applied, two loss functions would be necessary. The first loss function applied would be as previously illustrated for the higher of the two cutoffs (in this case the mill cutoff). If material was deemed as ore using this loss function then there would be nothing further to classify. However, if it were deemed waste, then it would be subject to analysis using the second loss function with the lower cutoff grade. If the material is classified as 'ore' using this second loss function then it is heap leach material as opposed to waste (Anderson, 1999).

A realization for the example blast of Figure 2.1 showing the classification of material as ore or waste is shown in Figure 3.11. This realization exhibits a more conservative boundary showing the mine planner that the loss function did as expected, limiting marginal material from being classified as ore (Anderson, 1999).

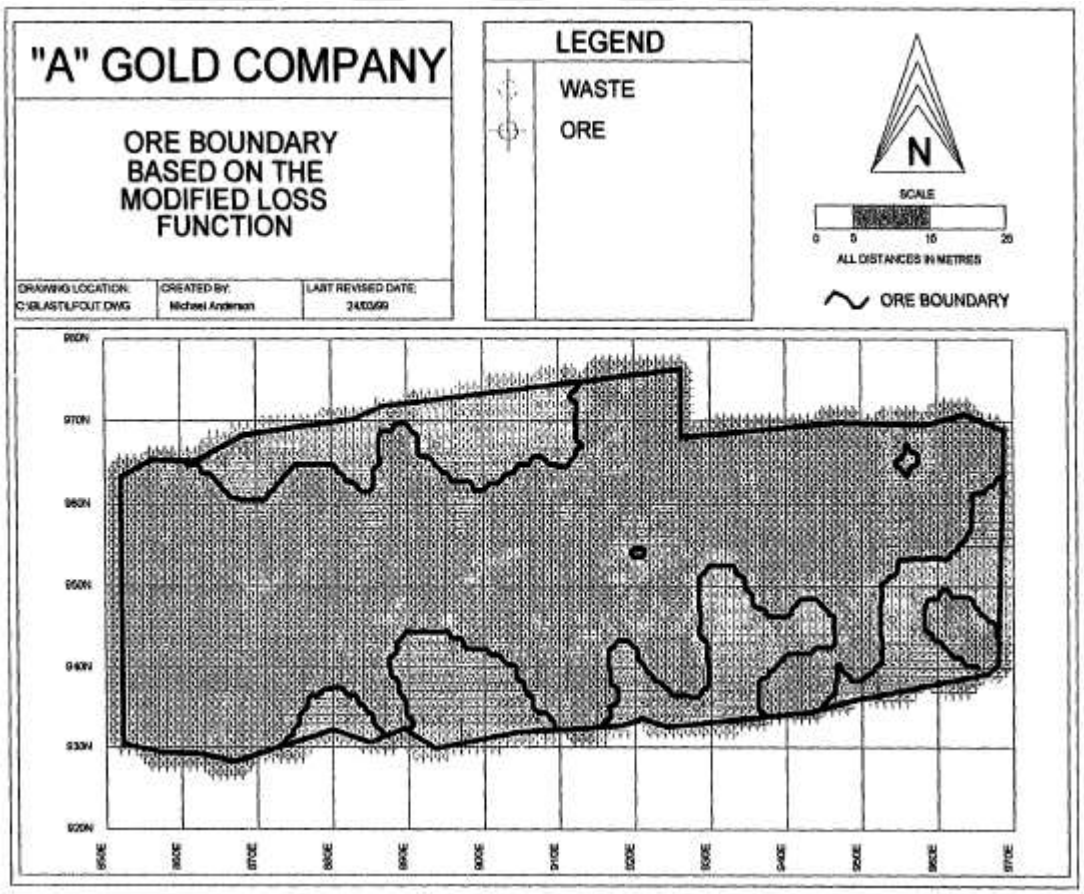


Figure 3.11 Digging boundary for loss function method (Anderson, 1999)

CHAPTER FOUR

CASE STUDY- DETERMINATION OF QUALITY DISTRIBUTION AT AN EXAMPLE QUARRY

4.1 Geographical Situation

The study area is located near Söke District of Aydin Province. The Cankurtaran limestone field is located nearly 1 km north of BATISÖKE Cement Factory as the crow flies.

Within the Cankurtaran Limestone Field located the adjacent South and North zones under the responsibility of Batısöke Cement Co. Inc. Production activity is carried out in accordance with the open pit operation technique in the south zone and the north zone, which are located adjacent to each other, on the Cankurtaran limestone field. The map showing the locations of the Batisöke Cement Factory and the South and the North zones is shown in Figure 4.1 (Onur, 2014)

4.2 Vegetation

It is seen that the region is widely covered with maquis shrublands and olive groves. Apart from this, pine forests are seen on high peaks. And on alluvial plains cotton is widely grown.

4.3 Climate

Mediterranean climate conditions predominate in the study area. In the region, summers are hot and dry, while winters are warm and rainy. The highest precipitation in the area is in April, May, September, October and November, and is usually in the form of rain.

According to the yearly average meteorological data;
Annual average temperature: 17.4 ° C
Annual average precipitation: 94, 3 mm / month
Annual average relative humidity: 58percent



Figure 4.1 North field and south field (Google Earth, 2016)

4.4 Geology of the Field

There exist metamorphic rocks belonging to the Menderes massif, Selçuk nappe and Lycian nappes, and Miocene and younger sedimentary and volcanic rocks around Söke (Figure 4.2).

4.4.1 The Menderes Massif

The Menderes massif, which is structurally the undermost of the study area, is represented by the Bayındır formation consisting of mica-schists and Kayaaltı formation formed as a result of a metamorphism of a thick carbonate platform (Erdoğan & Güngör, 1992 ; Güngör & İşintek, 2013).

4.4.2 The Bayındır formation (Pzş, Pzm)

The Bayındır formation consists of mica schists, metacuversites, phyllites and marbles. At the top of the formation are the quartz-rich metachic rocks, mafic metatuffs and chloritridin schists. At the top of the formation are the quartz-rich metaconglomerates, mafic metatuffs and chloritridin schists. The Bayındır formation also contains gray-black marble, dolomite and calcschist intercalations. The marbles are generally gray, dark gray and occasionally white in color, and are usually composed of large-sized calcite and dolomite crystals. Production is made from these marbles in the Cankurtaran Field (Güngör & İşintek, 2013).

4.4.3 The Kayaaltı formation (TRJKk)

The Kayaaltı formation is composed of yellow, white, coarse crystal marbles, dark gray colored dolomitic marbles and dolomites at the bottom. The upper part of the formation is represented by gray colored medium-thin-layered marbles, and these levels contain meta-bauxite lenses. The uppermost parts of the Kayaaltı formation consist of thin layered gray - white colored marbles containing chert band and ovules.

These levels of the formation do not let any crops for new licence fields (Güngör & İşintek, 2013).

4.4.4 The Selçuk nappe (Ks)

Above the metamorphic rocks of the Menderes Massif is the tectonic contact along a matrix of garnet micaschists in various sizes, including Selçuk nappe containing mafic volcanite, serpentinite, marble and dolomite blocks; and the corps belonging to this tectonic level are located to the north of the Cankurtaran Field (Güngör & İşintek, 2013).

4.4.5 The Lycian nappes

The Lycian nappes are composed of red metapelite and metapsammites at the bottom and dolomites at the top within the area between Söke and Kuşadası and appear along the tectonic contact on the Menderes massif and the Selçuk nappe (Güngör & Erdoğan, 2001). There are no rocks belonging to the Lycian nappes in the study area (Güngör & İşintek, 2013).

4.4.6 Miocene - The Söke Formation

The Söke formation consists of rough conglomerates and lacustrine limestones. The thickness of this stack, which consists of these kinds of rocks and transitive to each other in the lateral and vertical directions, is approximately 250 meters (Sümer et al., 2013). The conglomerates are greyish-brown, massive, inclusion body assisted, poorly sorted and well compacted. Within these conglomerates are intercalations of sandstone and laminated mudstones. It forms the granular components of gneiss, marble, quartzite, schist, recrystallized limestone and metaserpantinite conglomerates. Above the conglomerates is the stack of sandstone, mudstone, limestone and bituminous shales. There are layers of lignite within this stack. The limestones are represented by white micritic limestones and contain claystone and lignite intercalations in thin layers. The gastropod covered by the Söke formation has been

determined to be micromorphic (Ünay & Göktaş, 1999) and sporomorphic fossils age to be Early-Mid Miocene (Akgün & Akyol, 1999). The Söke formation covers the metamorphic rocks of the Menderes massif, Selçuk nappes and Lycian nappes along an angled disharmonic contact. The Davutlar conglomerate is found along a disharmonic contact over the Söke formation (Güngör & İşintek, 2013).

4.4.7 Miocene -Davutlar Conglomerate

Davutlar Conglomerate consists of blocky conglomerates, pebbly sandstones and sandstones and approximately has 200 metres of thickness. The unit begins with a massive conglomerate level at the bottom where marble blocks predominate, and the grain size shrinks upward and passes to pebbly sandstones and cross-bedded conglomerates. The conglomerates are beige-yellow colored, massive and cross-bedded, well compacted, grain-supported, well-moderately rounded, and coarse-grained in the lower sections. The lower contacts of the conglomerate layers are abraded and contain sandstone intercalations. The conglomerates predominantly consist of marble and recrystallized limestones, dolomites, quartzcists, mica schists, serpentinite and metaboxites (emery). The grain size is usually between the conglomerate and the block but contains larger sized pieces than the block. The Hisartepe volcanite, which is 12 millon years old (Sümer et al., 2013), cuts the Davutlar conglomerate and Kuşadası. This relationship suggests that the Davutlar Conglomerate may be Middle-Late Miocene.

Two drillings on the firebreak in the Cankurtaran Field have commenced on the Davutlar conglomerate and continued at the conglomerate forming this formation of 65.2 m to 203 m. The first drilling has shown that under the Davutlar conglomerate are the serpentinites of Selçuk nappe between 65.2-108.5 meters and the Menderes massif after 118 meters. In this area, the contact between the Davutlar conglomerate and the marbles of the Menderes massif (Pzm), which is still produced, is a normal faultline of a high angle in the northeast direction (Güngör & İşintek, 2013).

4.4.8 Miocene – The Kuşadası Formation

The Kuşadası Formation (Unay & Göktaş, 1999; Gürer et al., 2001), which does not let any crops in the area of investigation, has been formed from sandstones, mudstones and limestones. The lower part of the formation is composed of sandstones containing thin conglomerate intercalations. Mudstones are gray colored and alternate with beige colored limestones. The total thickness of the formation is approximately 115 meters. Fossil coverage and contact with the Hisartepe volcanics indicate that the formation is Middle-Late Miocene. Fevzipaşa formation covers the Kuşadası formation with a disharmonic contact (Güngör & İşintek, 2013).

4.4.9 Miocene -Hisartepe Volcanites

Hisartepe volcanites (Ercan et al., 1986) consist of trachyandesitic, latit, rhyodacite lava domes and are exposed between Davutlar and Söke along a northeast-southwest line. In the volcanites, column joints specific to the dom structure are widely observed. The mineral composition of volcanites consists of plagioclase, augite, hornblende and sanidine. These fenkrists are in a floating position in glassy and very fine grained matrix. This unit cuts the Davutlar conglomerate and Kuşadası Formation in the form of volcanic domes, dykes and intrusive masses (Güngör & İşintek, 2013).

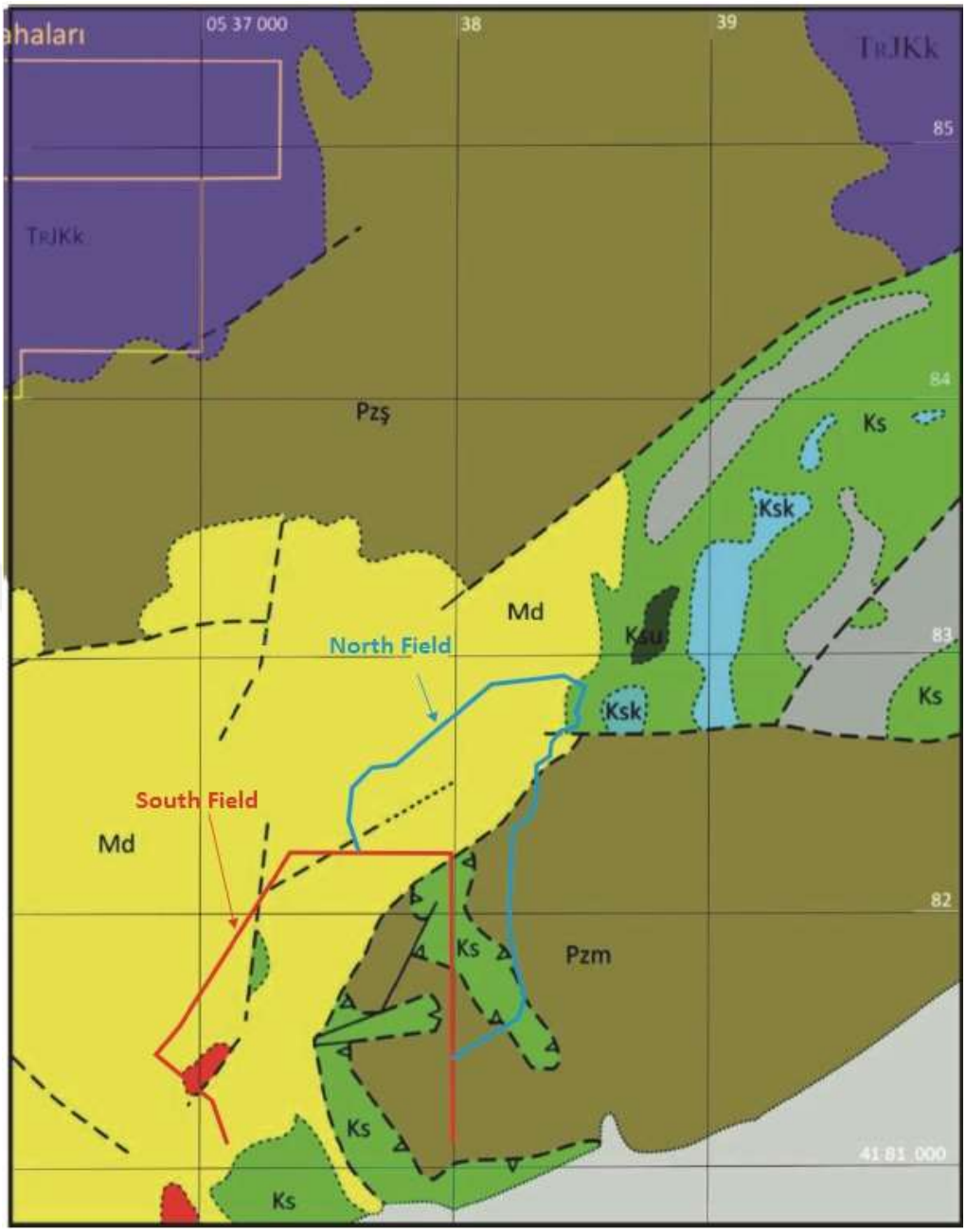


Figure 4.2 Geological map of the Cankurtaran limestone field and its surroundings

4.5 Drilling Studies

The entire area covering the southern and the northern zones is expressed as the Cankurtaran Limestone Field. In this context, evaluation of the data obtained from 58 core drillings has been carried out to model the quality values of the Cankurtaran Field. According to the chemical analysis results of the cores obtained from the drillings, percent MgO content has been taken as the most important reserve determination parameter. The total number of cores within the scope of the study is 5598 pieces. A total of 5598 drilling data have been subjected to statistical evaluation. Distribution graph of percent MgO content is given in Figure 4.3. (Onur, 2014)

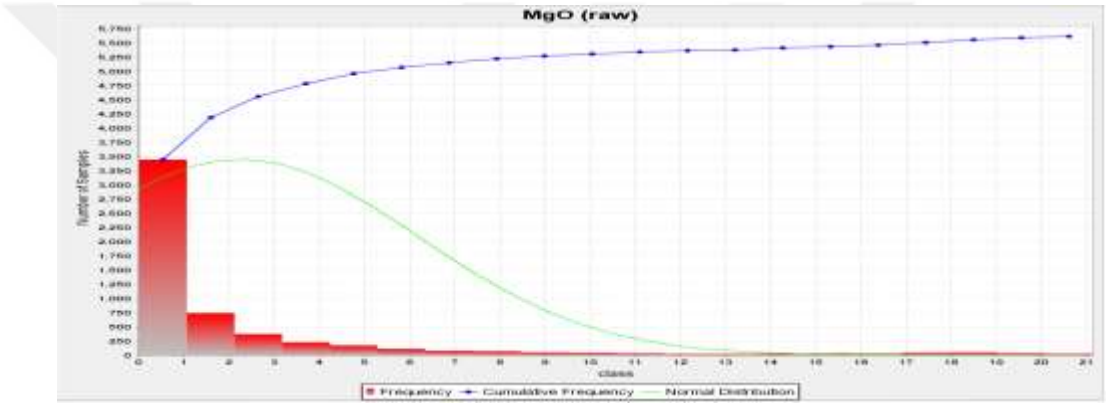


Figure 4.3 Distribution graph of MgO content of the original data

As a result of the evaluations of all drillings, the general quality values of the Cankurtaran Field are given in Table 4.1.

Table 4.1 The original quality values of the Cankurtaran limestone field

Parameter	Average	Standard Deviation	Min.	Max.
%MgO	2.113	4.189	0.00	22.25

When the average and standard deviation values of these data are examined, it is possible that the whole area is usable. For this reason, in order to be able to get a general idea, all data are classified by different intervals and it is aimed to have an idea about the percentage of limestone that can be used in the original data. The figures related to the classified quality obtained are given in table 4.2. The sample percentages

in this chart, which consist of the original data, have been given and it is aimed to provide a general idea of the class ranges within the total. Cumulative averages provide the overall average with the previous class. (Onur, 2014).

Table 4.2 Original classified quality values for the Cankurtaran limestone field MgO

%MgO Content class	**0.0-0.5**	**0.5-1.0**	**1.0-1.5**	**1.5-2.0**	**2.0-3.0**	**3.0-4.0**	**4.0-5.0**	**5.0-6.0**	**6.0>**
Number of samples	1886	452	189.00	134	156	96	81	50	356
Sample (%)	55.2	13.4	5.5	4.0	4.6	2.8	2.5	1.5	10.5
Average	0.229	0.711	1.242	1.748	2.452	3.515	4.497	5.532	12.84
Cumulative average	0.229	0.324	0.393	0.462	0.572	0.67	0.774	0.852	2.113

Statistical analyzes using the original data are given in the above charts. Evaluation of the obtained raw drilling data by direct statistical analysis in this way may suggest erroneous results. Because the drillings are sampled at different intervals in different core sizes, the weighted average of each drilling is necessary for a more precise and accurate result. For this very reason, it is necessary to divide the data into equal intervals in order to create a block model to be used in the reserve account and quality evaluation. In this method known as integration (compositing), the drilling data is arranged in the equivalent thickness range of 5 meters to provide a better representation of limestone raw materials. By this means, it will be possible to represent the limestone, which will be produced with 10 meters height steps, more appropriately. In the statistical analyzes, the thickness effect has been removed, and the weighted averages of the 905 data obtained have been included in the modeling process by dividing 3375 pieces of drilling data into 5-meter intervals. Statistical analyzes have been performed for the new integrated data set obtained by dividing the original data into 5 meter intervals using existing computer software and the obtained results are given in Table 4.3.

Table 4.3 Integrated quality values for 5-m intervals for the Cankurtaran limestone field MgO content

%MgO Content class	0.0-0.5	0.5-1.0	1.0-1.5	1.5-2.0	2.0-3.0	3.0-4.0	4.0-5.0	5.0-6.0	6.0>
Number of samples	425	115	76.00	44	61	35	26	21	102
Sample (%)	47	12.7	8.4	4.9	6.7	3.9	2.9	2.3	11.2
Average	0.240	0.705	1.241	1.714	2.393	3.496	4.523	5.351	12.624
Cumulative average	0.240	0.339	0.450	0.535	0.692	0.822	0.945	1.060	2.262

When the statistical analyzes of original data and integrated data are examined, it can be observed that there are differences between them. In the analyzes obtained from the original core data, the percentage of data with percent MgO content value is 55.2, the average value is 0.229; while the data percentage for data grouped at the same interval has decreased to 47.0 and the average percent MgO content value has increased to 0.240. This is an important indication that the cores are not equal in size and that the assessments should be done according to the weighted average values. The last value in the cumulative average column of the grouped data gives the general average values obtained before the modeling of the region.

The variogram is the key function in geostatistics as it will be used to fit a model of the temporal/spatial correlation of the observed phenomenon. One is thus making a distinction between the experimental variogram that is a visualisation of a possible spatial/temporal correlation and the variogram model that is further used to define the weights of the kriging function. Note that the experimental variogram is an empirical estimate of the covariance of a Gaussian process. As such, it may not be positive definite and hence not directly usable in kriging, without constraints or further processing. This explains why only a limited number of variogram models are used: most commonly, the linear, the spherical, the Gaussian and the exponential models.

Modeling was done using spherical function. Correctness in the literature is the most widely accepted and statistically reasonable approach.

The modeling of the Cankurtaran limestone field in terms of percent MgO contents is important for determining the amount of reserve distribution according to quality. For the modeling process, variogram analyzes are performed by using drilling data in different orientations, directions and slopes according to percent MgO contents obtained from drillings. After making sure that the models obtained are within the confidence interval by applying cross-testing, the variogram models are used to predict (estimate) the midpoints of the blocks in the Block Model prepared in certain dimensions. In the light of the general evaluations given above, a data set integrated at 5 meters intervals has been used to implement the block model. In order to reveal the distribution structure in different directions, variogram model analyzes have been performed in different directions. (Figure 4.4 and Figure 4.5) In the model developed in the perpendicular direction, a meaningful impact area is revealed towards depth while in the models in the horizontal direction, the effect distance is determined. Both variograms have been prepared using percent MgO contents.

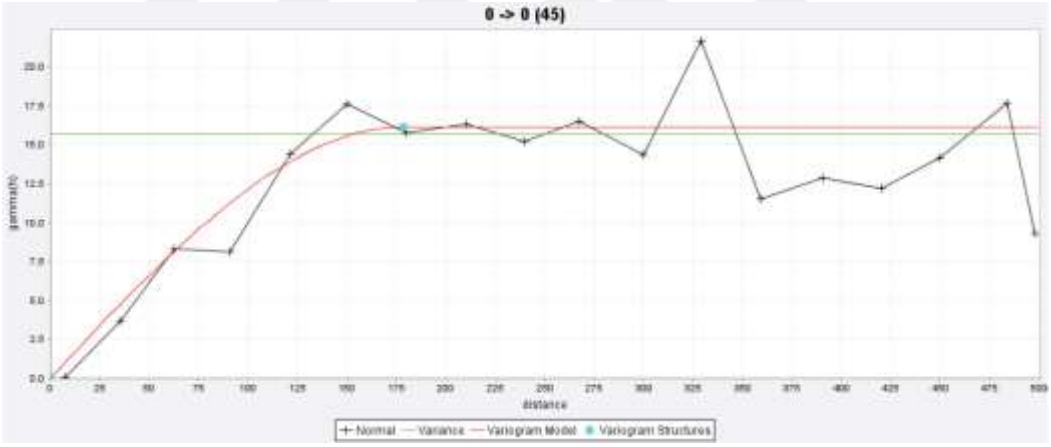


Figure 4.4 Horizontal variogram model

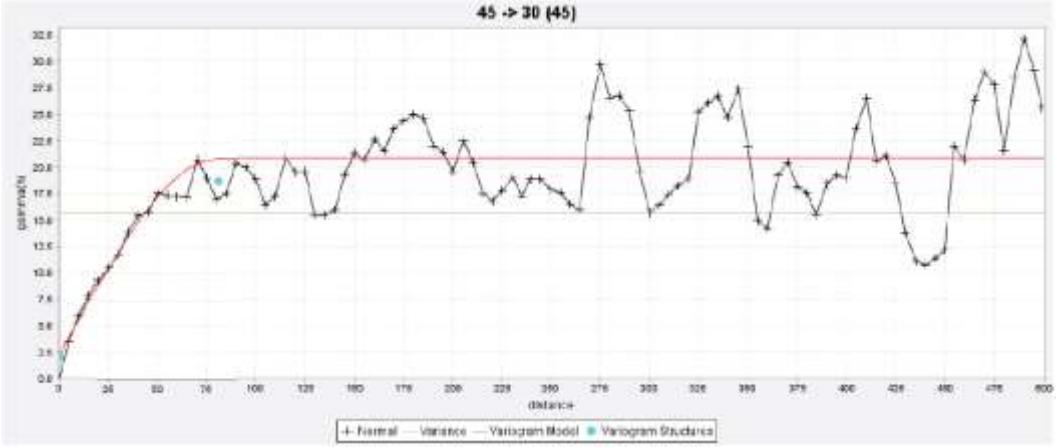


Figure 4.5 Perpendicular variogram model

By means of variogram model analysis, variance changes related to direction have been examined and information about the direction, slope and slope direction of high MgO content zone in limestone formation has been obtained and model parameters have been established.

In order to determine the usability of the modeling parameters obtained from the graphs in the block modeling process, a model conformity test called cross-testing needs to be performed. During this test, all the data in the data file are extracted one by one using the obtained model values and the value is assigned by performing the estimation with the kriging method. The difference between the actual data value and the predicted values assigned gives the modeling error. The value of the correlation coefficient obtained from the result of cross-testing performed is given in Figure 4.6.

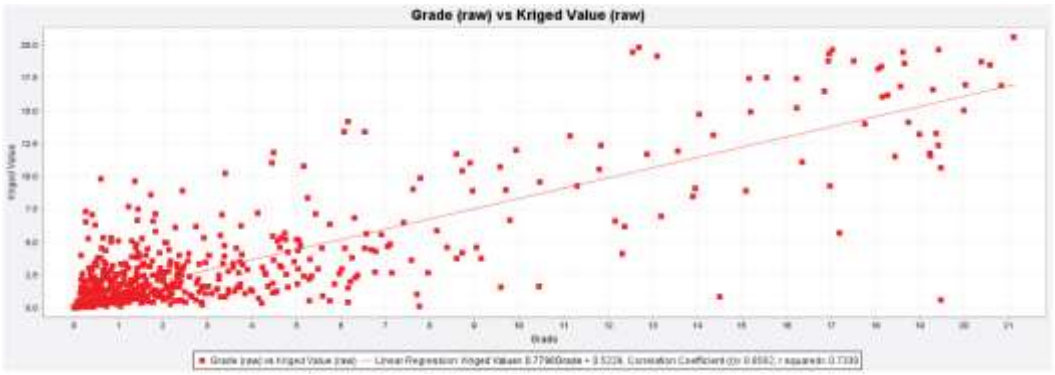


Figure 4.6 Correlation coefficient obtained from cross test.

The model obtained as a result of the cross test has been obtained from the integrated 905 data and estimated with an accuracy (correlation coefficient) of 0.856. This value is within the confidence limits and will be used in block modeling.

The block model has been created considering the maximum limestone benches that can be produced within the license limits of the Cankurtaran limestone field. In the field, the western part which has not been not drilled due to the known quality high-slope bench, and the parts that do not have adequate drillings in the north-eastern part, are excluded from the block model. The block model has been developed for the percent MgO content in the section between the maximum production limit and the existing topographic surface and is given in Figure 4.7.

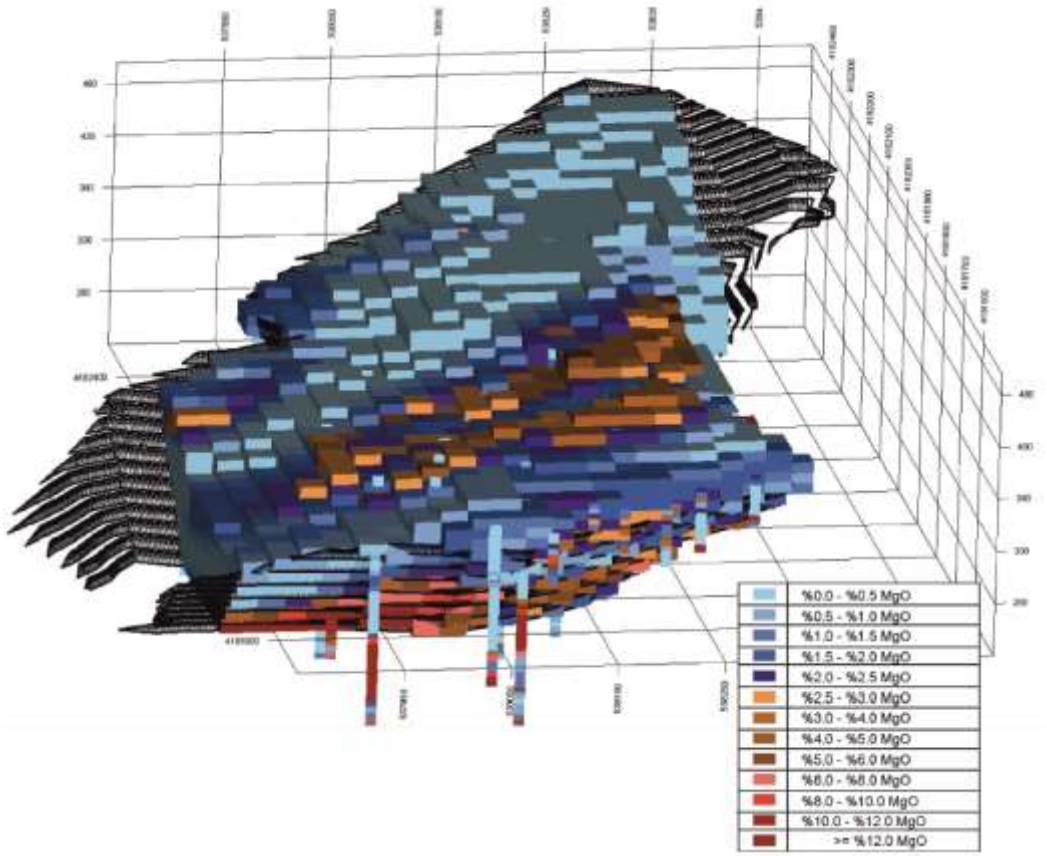


Figure 4.7 3-D view of the block model for MgO content

Blocks modeled with 0.5 percent quality ranges for each level starting from the base elevation of 230 m are colored and the areas to be excluded from production are determined in terms of reserve quality and MgO content in order to determine the zone

with low quality value (percent MgO> 5). An image of a 230m elevation as an example level is given in Figure 4.8.

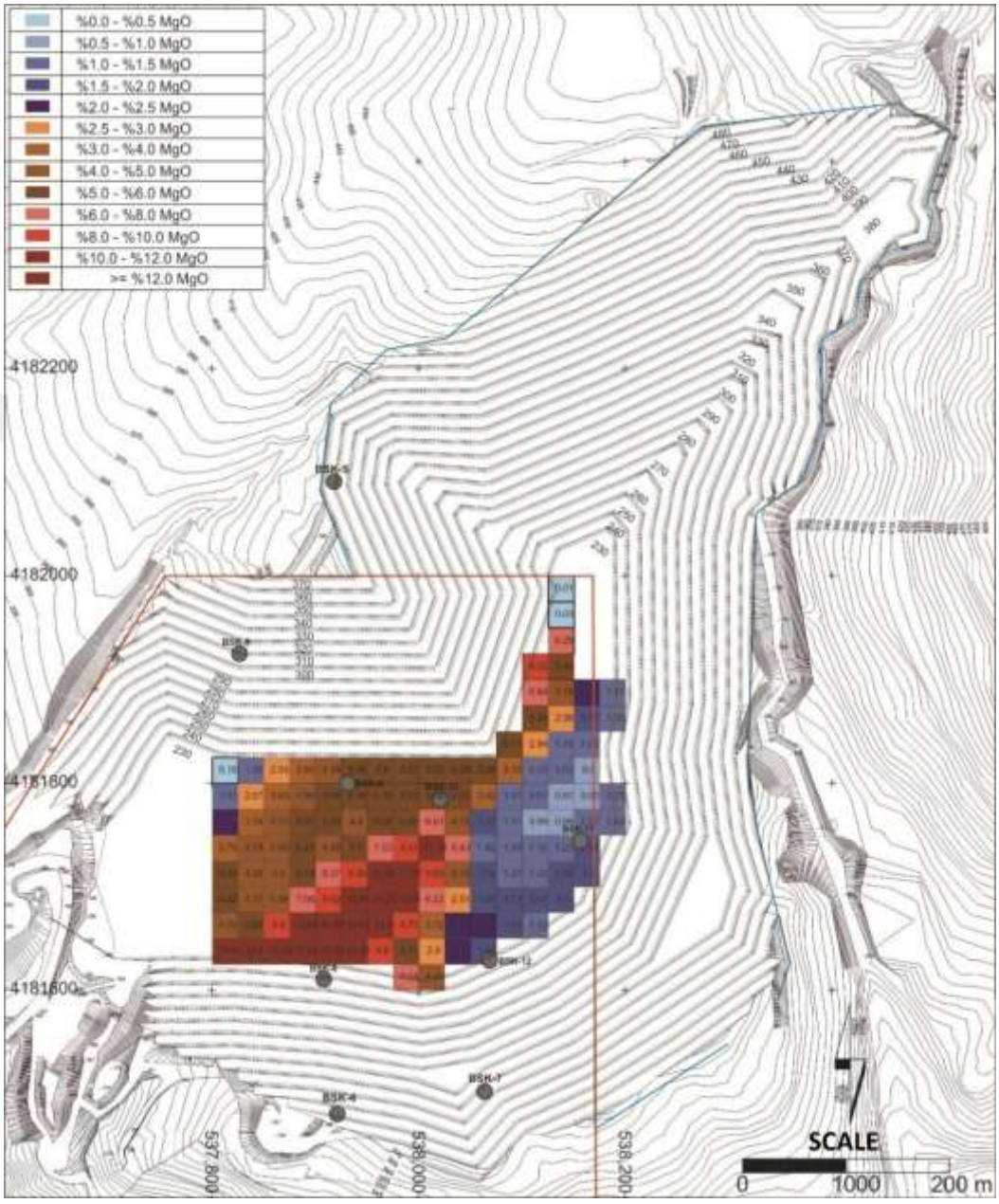


Figure 4.8 The 2-D view of the block model constructed for MgO for 230m thick and the final excavation plan within the license limit

4.6 Modelling of the Quarry

Limestone reserves and content have been determined by modeling the raw material over 360 m elevation within the scope of project in the Cankurtaran limestone field.

Production planning of 51 829 265 ton limestone reserves determined over 300 m elevation has been carried out by using data obtained from field drillings.

In the northern and northwest parts of the Cankurtaran limestone field, there is a raw material that can be used in the cement factory, while in the south and southwest parts of the site there is an inhomogeneous raw material with a very variable content of percent MgO. Especially in the southeast part of the site is a dolomitic structure with a content of 10-20 percent MgO reaching a thickness of 60-70 m in a certain area. It is not possible to produce this material. In this study, projecting work has been carried out to determine the amount of raw materials that could be produced from the field, so as to exclude high MgO content from production within the existing license boundaries. Both the modeling, the production plan and the determination of the amount of raw material that can be excavated have been prepared according to the present condition of the quarry. In other words, calculations have been made according to the amount of material that can be produced from the quarry instead of the amount of raw materials remaining below 300 m.

4.6.1 Bench Dimensions

The bench dimensions have been determined by taking into account the production technique and environmental factors along with the safety of slope projected between 490 m. And 230 m in the Cankurtaran limestone field. Accordingly, the slope geometry and bench dimensions to be applied in the field are given in Figure 4.9

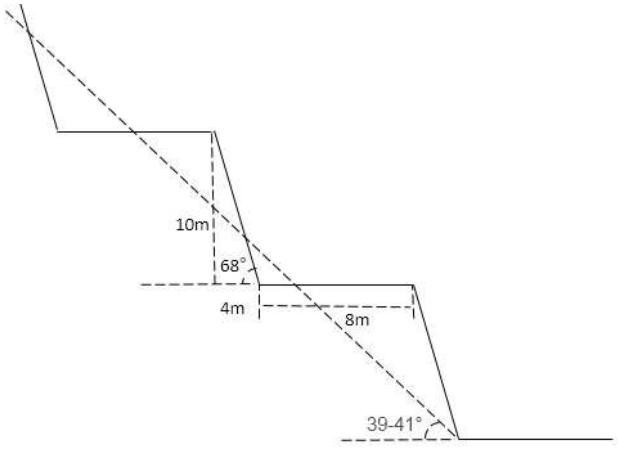


Figure 4.9 The bench geometry to be implemented in the field

As seen in the figure, 10 m of bench height has been selected. There are two reasons for this. The first is to ensure that visual pollution is perceived at least in a visual manner by means of 10-meter benches in the final slope. The second is that 10-12 m of bench height is ideal in terms of production technology (hole drilling machines, etc.). The final bench widths have been projected to be 8 m, step slope angles to be 68°, and general slope angle to be 40°.

4.6.2 Cankurtaran Limestone Field Production Planning

The production plan for the Cankurtaran limestone field has been performed in two stages. Initially, a project has been designed to determine the amount of material that could be produced in the South and North zones without regard to quality within the operating permission limits (Figure 4.10). The total amount of material that can be excavated from the field has been calculated as 115 916 404 tons. In the raw material found in the field, there is a mass which can not be used in the cement plant and in the gravel plant, especially in terms of percent MgO content.

Considering the quality distribution of the Cankurtaran limestone field and the working conditions related to the structure of the land, the production planning has been prepared in order to determine the operable reserves. In this context, a planning study has been carried out with the condition of leaving the poor quality raw materials concentrated in a certain area of the field unexcavated. In this study, the massive parts with MgO content higher than 5 percent have been excluded from the excavation plan, and the geometric shape that the quarry will take has been formed (Figure 4.11). In this way, a production plan has been created to keep the stripping excavation at the lowest level, leaving the poor quality material in the southern and southeastern part of the quarry to remain in place.

The amount of limestone to be produced in 10 m bench slices between 490 m and 230 m elevations is calculated to be 90 350 000 tonnes. In Table 4.4, the excavation amounts to be performed on the basis of the benches in the case of complete excavation

of the site and implementation of the recommended optional excavation plan are given. As can be seen from the chart, 25 566 404 tonnes of low quality material with mostly high MgO content of 5 percent have been left unexcavated. In most of this material MgO content is higher than 6 percent, but due to the inhomogeneous structure, some low MgO content limestone has been excluded from excavation. Especially in the northern part of the site where production benches can not be established due to the narrowing of the excavation boundaries, there is a quality raw material with a low MgO content below 290 m elevation (Figure 4.15). Raw materials with a MgO content higher than 5 percent will be produced in a small amount within the production plan framework to be applied in a similar manner.

As seen in Table 4.5 there are 6 808 750 tonnes of raw materials which can not be quality assessed and which will be excavated according to the production plan. Due to the limited number of drillings on the field, there has been no quality evaluation for the regions in the western and northeastern parts of the modeling work outside the area of drillings. The limestone amounts of these parts are given in Table 4.5 on a benc-by-bench basis. In addition, the quality distribution plans according to the percent MgO content of the generated production plan are given in Figures 4.13 and 4.16 respectively for each bench. When the excavation plans prepared on the basis of the benches are examined, the average MgO content of the raw material to be produced up to 380 elevation is below 1.5 percent. In the four benches between 370 m and 340 m elevations, averages vary between 1.54 percent and 1.72 percent. The low-quality material coming from this area during production should be sent to the gravel plant or mixed with low MgO content and fed to the plant. A similar situation exists in the last four benches that form the basis of the operation between 260 m and 230 m elevations. In these benches, the content of MgO is higher, but the amount of material to be excavated is low, so the usage planning can be made more comfortably.

Table 4.4 Proposed excavation plans and quantities of materials remaining within the license boundaries

Step level (m)	Amount of final excavation (tons)	Amount of excavation plan production (tons)	Amount of material left out of the excavation plan (tons)
480-490	48750	48750	0
470-480	243750	243750	0
460-470	406250	406250	0
450-460	682500	682500	0
440-450	1218750	1218750	0
430-440	2242500	2242500	0
420-430	2811250	2811250	0
410-420	3851250	3851250	0
400-410	4777500	4777500	0
390-400	5768750	5768750	0
380-390	6922500	6922500	0
370-380	7702500	7702500	0
360-370	7816250	7816250	0
350-360	7913750	7913750	0
340-350	7800049	6776250	1023799
330-340	7377923	6061250	1316673
320-330	7052825	5070000	1982825
310-320	6597955	4387500	2210455
300-310	6029319	3835000	2194319
290-300	5525504	3461250	2064254
280-290	5037988	1820000	3217988
270-280	4518020	1755000	2763020
260-270	4079189	1430000	2649189
250-260	3575504	1332500	2243004
240-250	3217858	1072500	2145358
230-240	2698020	942500	1755520
Total	**115 916 404**	**90 350 000**	**25 566 404**

Table 4.5 Cankurtaran limestone zone production quantities planned at steps according to MgO content

Level	Amount of excavation (tons)											Limestone reserve without quality assessment	Total limestone excavation
	%MgO 0.0-0.5	%MgO 0.5-1.0	%MgO 1.0-1.5	%MgO 1.5-2.0	%MgO 2.0-3.0	%MgO 3.0-4.0	%MgO 4.0-6.0	%MgO 6.0-8.0	%MgO 8.0-10.0	%MgO 10.0-12.0	Step average %Mgo		
480-490	16250										0.16	32500	48750
470-480	162500										0.15	81250	243750
460-470	292500										0.19	113750	406250
450-460	520000	16250									0.24	146250	682500
440-450	845000	146250	32500								0.33	195000	1218750
430-440	1625000	325000				48750					0.37	243750	2242500
420-430	1673750	390000	130000	81250	211250	48750	16250				0.66	260000	2811250
410-420	1722500	585000	162500	292500	503750	227500	48750				1.05	308750	3851250
400-410	1511250	1040000	568750	406250	568750	308750	48750				1.17	325000	4777500
390-400	1348750	1641250	926250	455000	471250	422500	178750				1.27	325000	5768750
380-390	1300000	2128750	763750	893750	796250	471250	260000	16250			1.40	292500	6922500
370-380	1446250	2015000	893750	1088750	877500	682500	455000	16250			1.54	227500	7702500
360-370	1511250	1998750	698750	926250	1235000	520000	633750	81250	16250		1.67	195000	7816250
350-360	2015000	1771250	731250	682500	1072500	682500	617500	146250	65000		1.72	130000	7913750
340-350	2307500	1218750	747500	503750	796250	455000	438750	130000	48750	16250	1.56	113750	6776250
330-340	2210000	1413750	780000	471250	373750	178750	276250	146250	16250	32500	1.32	162500	6061250
320-330	2096250	1722500	471250	292500	81250	32500	97500	65000	16250	16250	0.89	178750	5070000
310-320	2193750	1105000	373750	325000	16250	16250	32500	32500		16250	0.73	276250	4387500
300-310	2080000	698750	438750	195000	48750	32500		16250			0.67	325000	3835000
290-300	1690000	845000	341250	97500	48750	16250	16250				0.64	406250	3461250
280-290	617500	568750	97500	32500	81250						0.68	422500	1820000
270-280	503750	406250	97500	81250	81250	65000	65000	16250			1.20	438750	1755000
260-270	325000	276250	81250	97500	32500	81250	113750	32500			1.64	390000	1430000
250-260	113750	195000	32500	65000	65000	97500	162500	113750	32500	16250	3.19	438750	1332500
240-250	81250	81250		81250	97500		243750	81250	16250	16250	3.74	373750	1072500
230-240	16250		32500		48750	65000	308750		16250	48750	5.06	406250	942500
Total	**30 225 000**	**20 588 750**	**8 401 250**	**7 068 750**	**7 507 500**	**4 452 500**	**4 013 750**	**893 750**	**227 500**	**162 500**	**1.29**	**6 808 750**	**90 350 000**

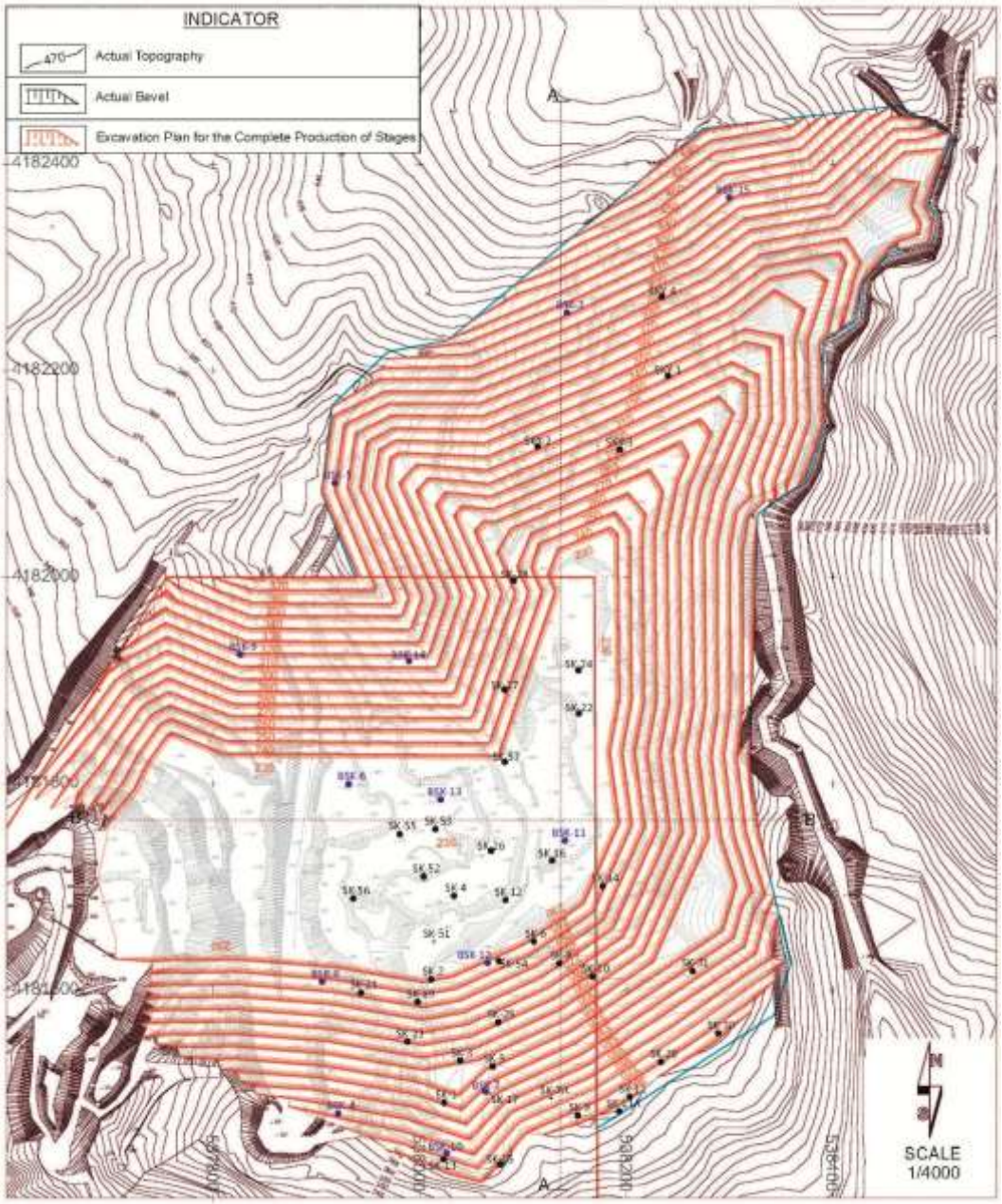


Figure 4.10 Excavation plan for the complete production of north and south fields

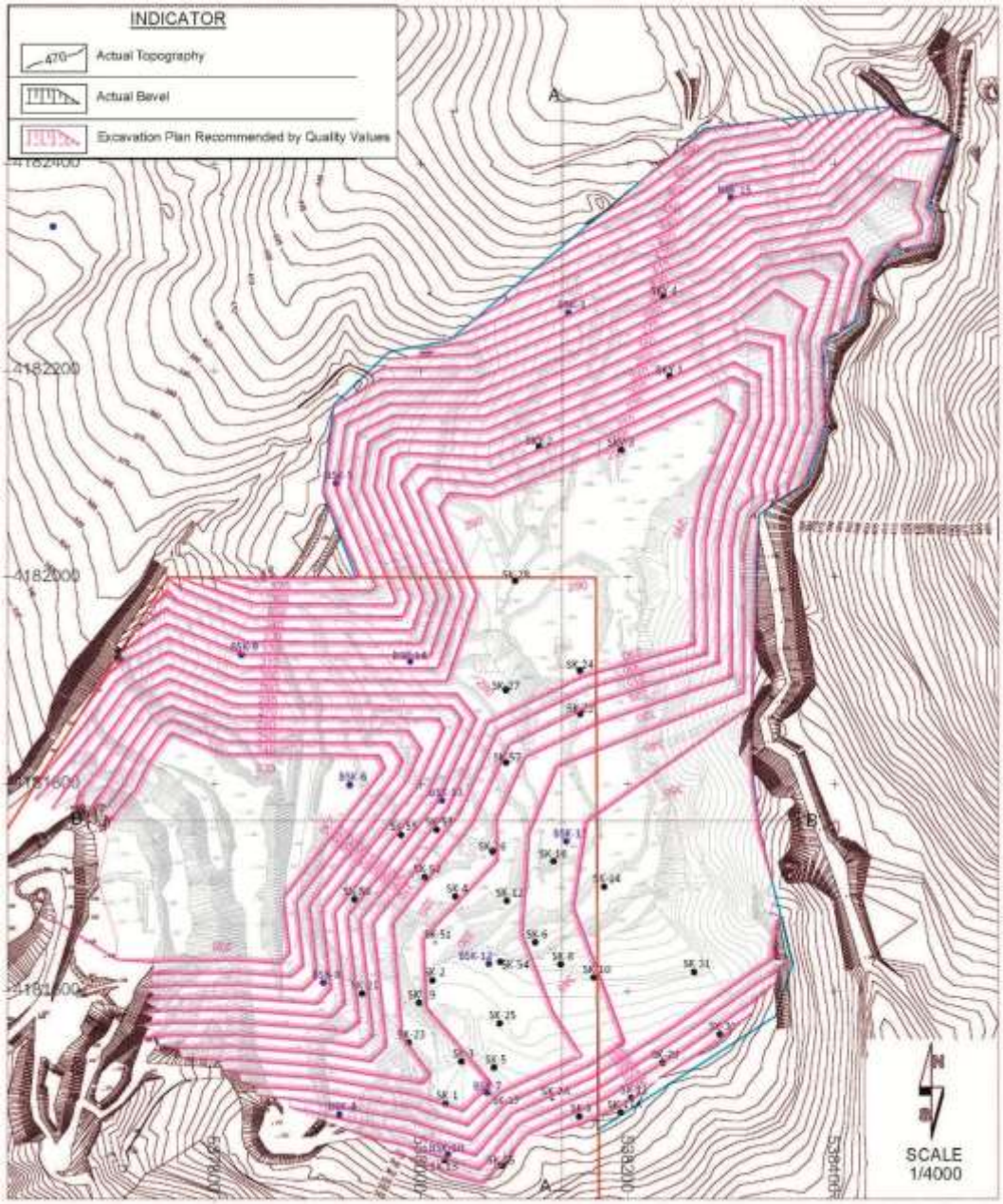


Figure 4.11 The proposed production plan according to the quality values on the south and north sides

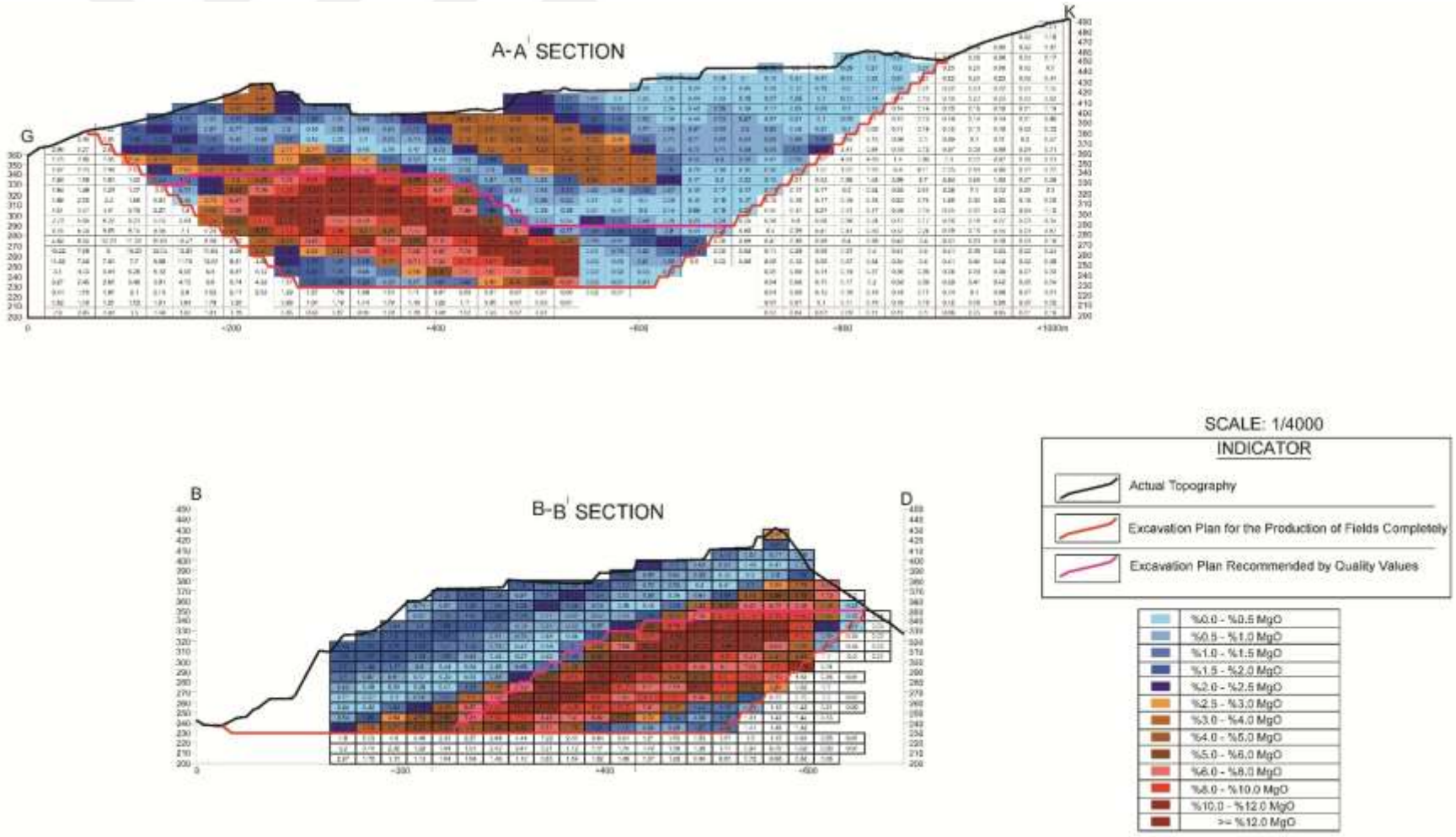


Figure 4.12 Production planning sections according to quality values in north and south areas

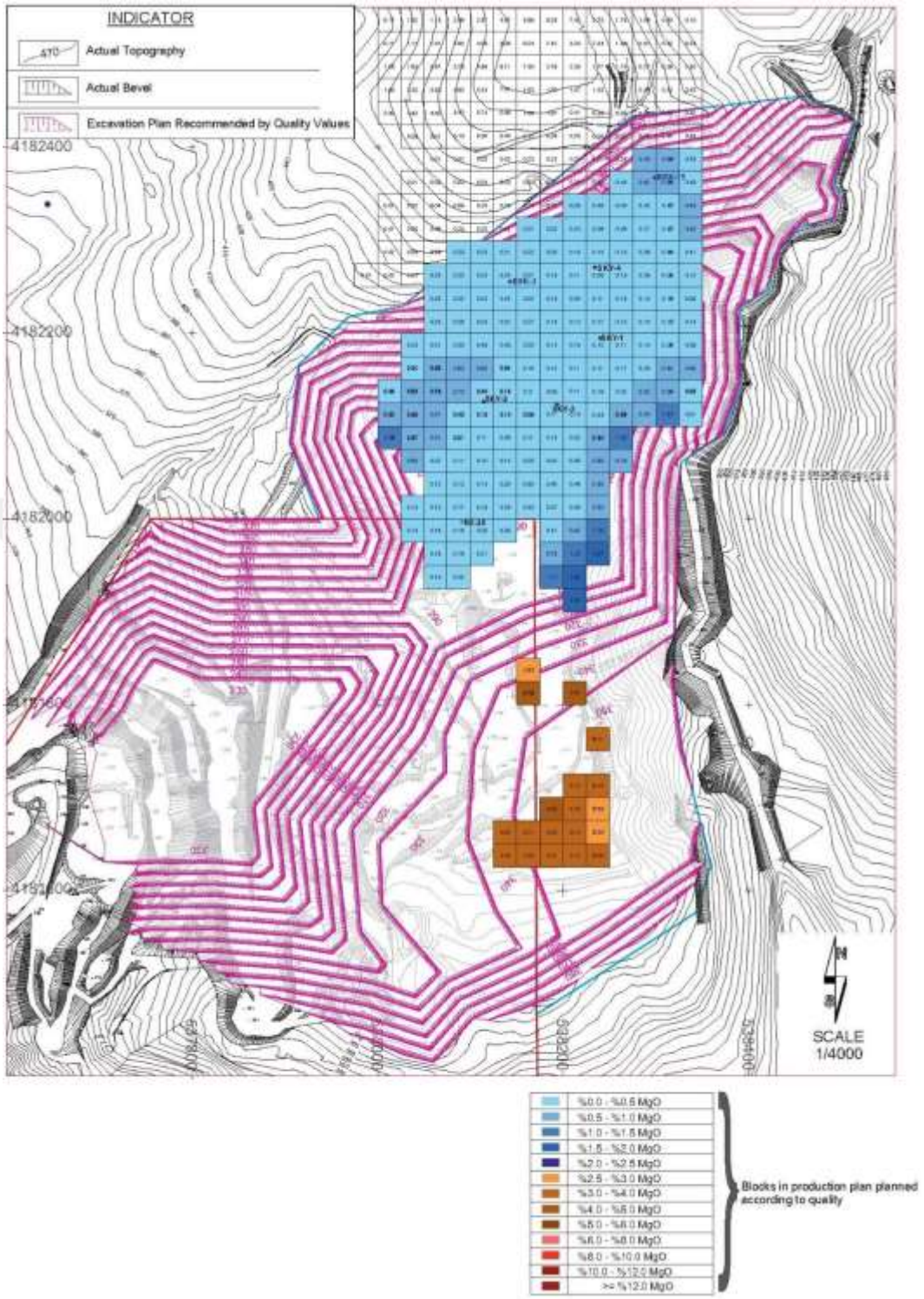


Figure 4.13 430 - 440 m stepwise quality distribution map based on MgO content

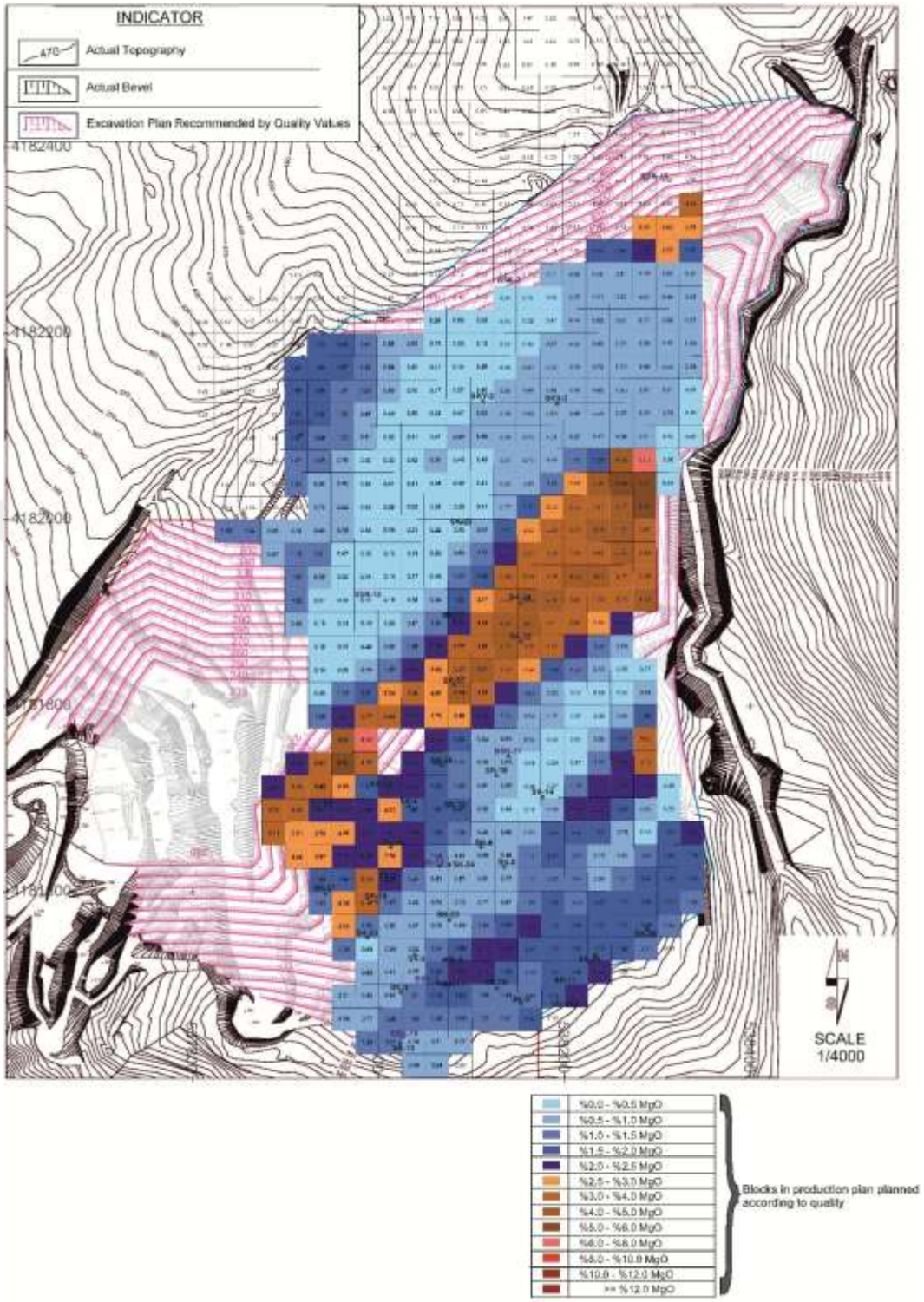


Figure 4.14 380 - 390 m stepwise quality distribution map based on MgO content

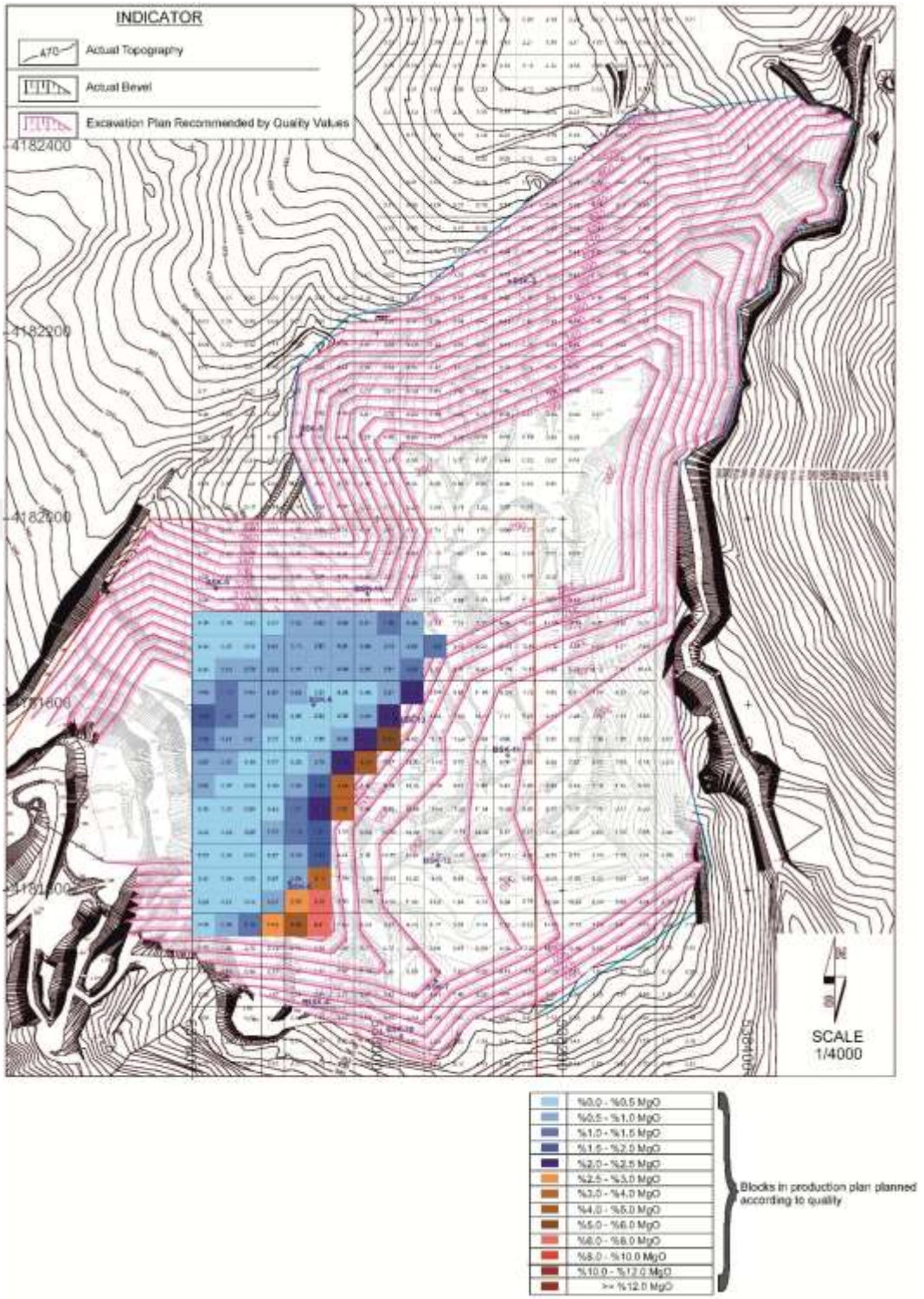


Figure 4.15 280 - 290 m stepwise quality distribution map based on MgO content

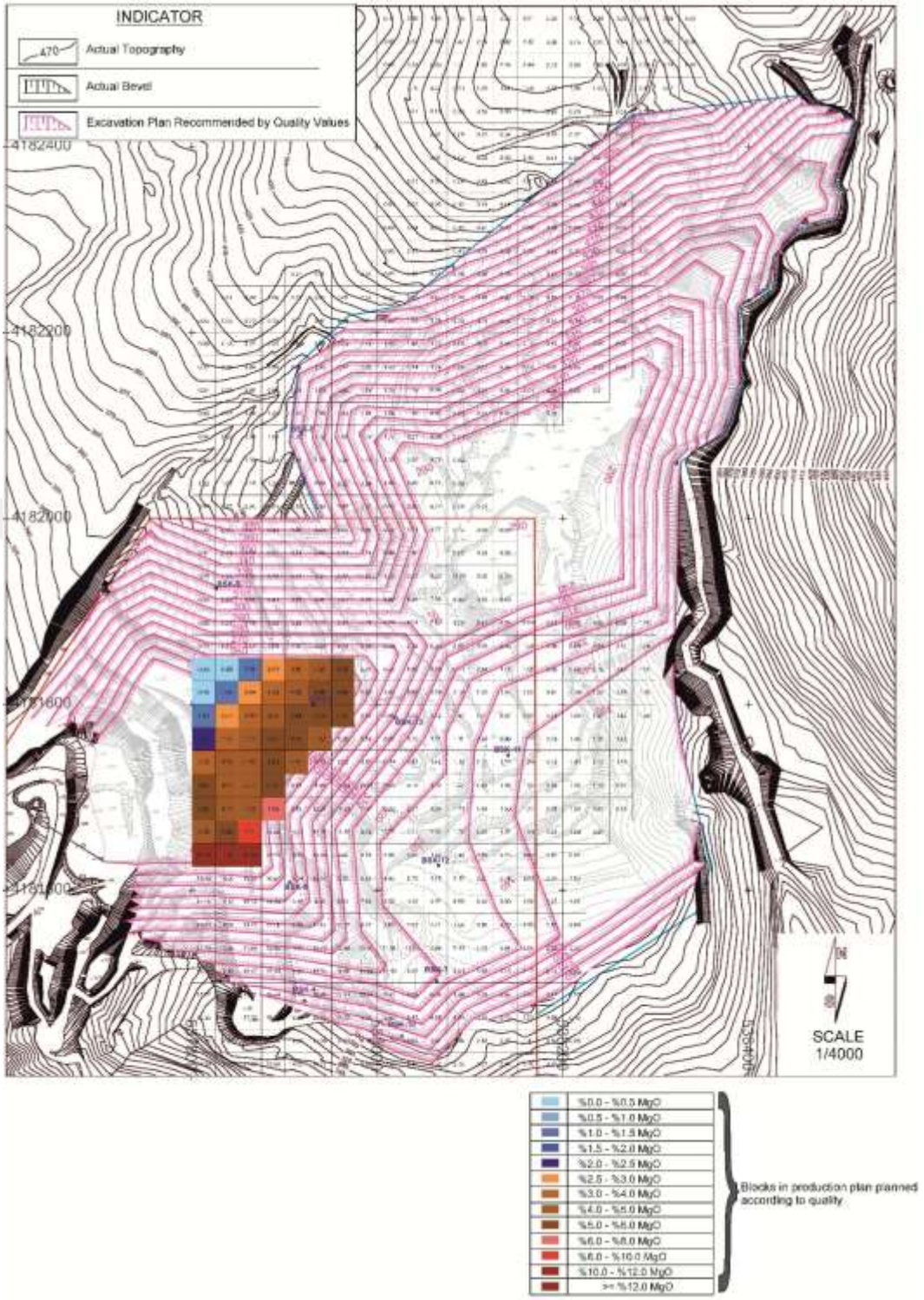


Figure 4.16 230 - 240 m stepwise quality distribution map based on MgO content

CHAPTER FIVE
CONCLUSIONS

Such mining computer aided software packages will be beneficial in manufacturing if used by businesses. By doing that it will be possible to avoid unnecessary overburden, monitor overburdens status by computers and taking precautions where needed will also be financially beneficial and efficient for the orginisation. By doing that it will be possible to avoid unnecessary overburden, monitor overburdens status by computers and taking precautions where needed will also be financially beneficial and efficient for the orginisation. By doing that it will be possible to avoid unnecessary overburden, monitor overburdens status by computers and taking precautions where needed will also be financially beneficial and efficient for the orginisation.

By doing that it will be possible to avoid unnecessary overburden, monitor overburdens status by computers and taking precautions where needed will also be financially beneficial and efficient for the orginisation. Mining softwares will increase competence so the production quality. Hence continuos development and advancing can be succeeded. Especially mining is an industry which requires major amount of investments and this necessitates predictability. Thereby all projects better be tested and then implied to the projects which also brings easiness on monitoring as well as reducing the time spent on planning.

By the aid of the international mining software SURPAC, a geologic database for a metal mine was established, 3D models of the surface, geologic of the mine based on the 3D model was calculated. In order to compute the reserves, a grade block model was built and MgO element grade was estimated using Ordinary Kriging. Then, the reserve of MgO element and every sublevel of the mine was worked out. Finally, the calculated result of percent MgO reserve to its actual prospecting reserve was compared, and the results show that they are all almost equal to each other. Obviously, the built models are reliable and the calculated results of reserves are correct. They can be used to assist the geologic and mining engineers of the mine to do research work of reserves estimation, mining design, plan making and so on.

Of course, it is necessary to examine it economically to form a production plan. However, in the thesis, the quantities of excavation and quality were taken into consideration and an economic evaluation was not made.

The quality of the raw material found in the Cankurtaran limestone field is determined according to percent MgO contents and the operational reserves have been calculated and production plans have been prepared. A total of 3375 drilling data have been evaluated statistically using all drilling data. In current situation, a production plan of 10 m benches has been created by using the mathematical model data of the raw material between the highest elevation of 490 m in the northern part of the field and the lowest elevation of 230 m in the southern part of the field.

Firstly, excavation planning of the site has been carried out up to a height of 230 m within the limits of the operation permit with the aim of determining the amount of raw materials found in the area without considering the quality values. However, when the data obtained from the modeling studies are evaluated, there exists a mass which can not be used in the cement plant and the gravel plant in the raw material contained in the field, especially in terms of percent MgO content. The amount of raw materials that can be excavated from the area has been calculated as 115.916.404 tons according to the created plan. Especially in the southern and southeastern parts of the site there is an increasing pollution in terms ofpercent MgO content. Considering these conditions, instead of producing the entire limestone, it is envisaged to implement an operation plan that will exclude poor quality materials from the excavation plan. A plan has been made to exclude the low quality raw material excavation plan in the southeastern part of the field. In the plan, three-dimensional mathematical model data have been used to exclude massive parts with MgO content higher than 5 percent from the excavation plan. According to the excavation plan, the amount of raw material that can be produced between 490 m and 230 m elevations has been calculated as 90 350 000 tons. The average MgO content of the raw material to be produced has been determined to be 1.29 percent.

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