THE BENEFITS OF TECHNICAL COMPUTING TO THE SOUTH AFRICAN GOLD MINING INDUSTRY

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DECLARATION

I declare that this project report is my own, unaided work. It is being submitted for the Degree of Master of Science in Engineering in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in any other University.

Robert M. Gilmour

19TH day of August 1988

To my wife Lyn

for her encouragement, patience and understanding.

ABSTRACT

The South African gold mining industry is currently faced with serious problems in terms of excessive cost escalation and declining grades. Various schemes such as economy of scale and mechanisation have been implemented in an attempt to improve profitability. The degree of success achieved to date by these measures has been insufficient to alleviate this situation.

Technical computing, which has enabled the implementation of techniques that were previously impossible by manual methods, is seen as an additional aid to improving profitability. This project report examines the benefits that can be gained by the use of one such technique, namely geostatistics.

A geostatistical analysis was conducted on a portion of the White Reef at West Rand Consolidated Mines Limited and the results compared against the mine estimation technique and total mining. The major benefit of geostatistics over the other methods was its ability to improve ore reserve estimation and consequently the extraction of payable ore. The case study showed that profits could be increased by R1.0 million per annum as a result of using geostatistics which would not be possible without a computer.

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1 INTRODUCTION

Though legend says that gold was first discovered in the Witwatersrand in 1852, the official discovery, that of the Main Reef on the farm Langlaagte, is attributed to George Harrison early in 1886. Since then the Witwatersrand has witnessed the opening of seven goldfields, the latest being the Evander field in the middle 1950's. From the commencement of production in 1887, up till the end of 1986, these fields have seen the establishment of approximately 125 mines, output of 40 851 784 kilograms of gold, at a realised value of 123 229 982 000 rand. Today 32 mines are still operational.

The majority of existing producers were established on reasonably high grade deposits and under relatively stable economic conditions. However, they now face a very different scenario as discussed below.

1.1 Problems Facing the Industry

Declining grades and escalating costs are the most serious problems currently facing the industry.

1.1.1 Grade

The reduction in grade currently being experienced initially originated from the policy of developing high grade reserves in the early days of a mine's life. This was done in order to improve its cash flow and internal rate of return. Consequently the stage has been reached where the majority of mines have fully developed and exploited their high grade reserves. This has led to mines being forced to extend operations into erratically mineralised areas of lower grade. The resulting drop in grade for the industry over the last 25 years is illustrated in Figure 1.1.

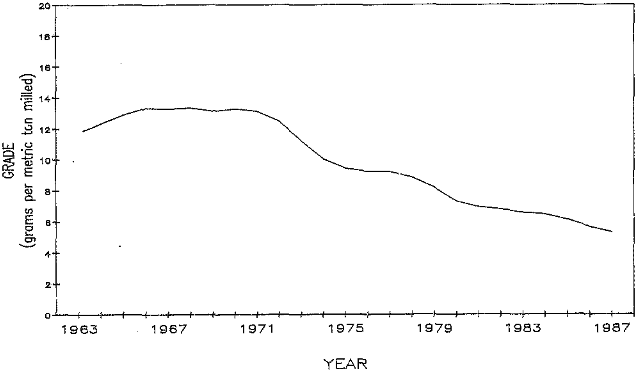
This trend has furthermore been exacerbated by the marked increase in mill throughput over the last ten years. This has been due mainly to mining companies taking advantage of the lower pay limits, resulting from the high rand gold price, in order to mill additional low grade tonnage and so extend the life of mining operations.

Occasional reversals to this trend due to the discovery of new goldfields may occur. The increase in grade in the early 1960's, for example, is attributable to the coming on stream of both the Klerksdorp and Orange Free State goldfields. However, the overall trend of decreasing grade cannot be disputed.

It is, however, not only existing producers who are faced with problems. New mines in existing goldfields are faced with the prospect of a lower average grade potential than their contemporaries. This is due to sophisticated large-scale exploration programmes, undertaken over the last few decades, having accounted for the the great majority of revenue generating projects under present economic conditions.

In the now almost defunct goldfields of the Central, East and West Rand the average grade can be seen to have declined over the

FIGURE 1.1 AVERAGE GRADE



SOURCE: *Chamber of Mines of South Africa*

years. For example, the average grade of mines that commenced production in 1888 was 9.47 g/t, compared to 7.1 g/t fifty years later in 1938. The correlation of grade and time is illustrated in Figure 1.2.

Comparison of the grade of new mines in the recently established goldfields further corroborates what was seen in worked out areas. In the context of this discussion new mines are those considered to have commenced production within the last ten years, a sufficiently long period for a mine to become fully established. Both the Far West Rand and Orange Free State goldfields can be considered in this context. The results are summarised in Table 1.1 below.

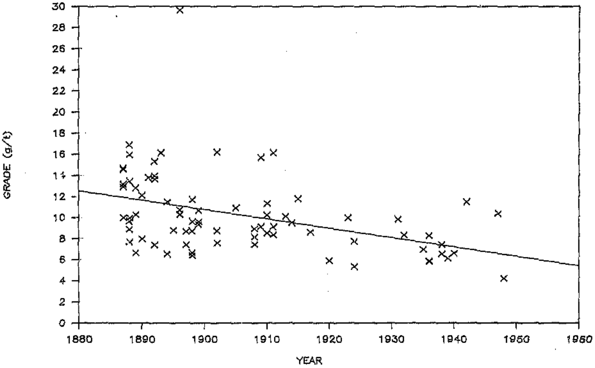
Table 1.1 Comparison of the grades of new mines with their goldfield average

	Grade to year end 1986 (g/t)	
	All mines	New mines
Orange Free State	10.69	6.40
Far West Rand	13.03	5.19
Evander *	8.04	4.50

* No new producers have been opened in this field. However, the published grade figure for the potential new Poplar mine is used to further substantiate this problem.

Results tabulated above are based on the principal Transvaal and

FIGURE 1.2 SCATTERGRAM OF GRADE AND TIME



Orange Free State producing mines that are members of the Chamber of ..nes of South Africa (C.O.M.).

1.1.2 Working costs

Prior to World War II costs remained virtually constant excepting for those years following the Boer War and the first World War. However, since the second World War costs have risen inexorably, a scenario which has accelerated from the middle 1970's and into the eighties, due to the high inflation rate. This is vividly illustrated in Figure 1.3.

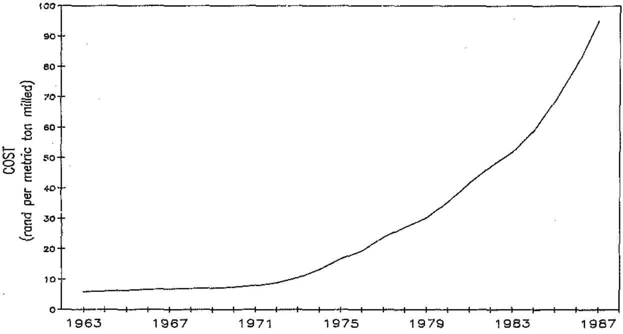
Though the inflation rate may well be reduced in the future, it is highly unlikely that the upward cost movement which has persisted for the last forty years will be reversed. In addition, added costs are being incurred as gold mining operations are pursued to greater depths and into increasingly hostile environments.

1.2 Effects on the Industry

The mining industry in recent years has yielded record rand profits comparable to those achieved at the beginning of the decade when the gold price reached its peak. This has been due mainly to the sympathetic link that started developing between the dollar gold price and the rand/dollar exchange rate from 1980 onwards.

This link, which began to significantly benefit the mines from 1984, can be clearly seen in Figure 1.4. It is due to the

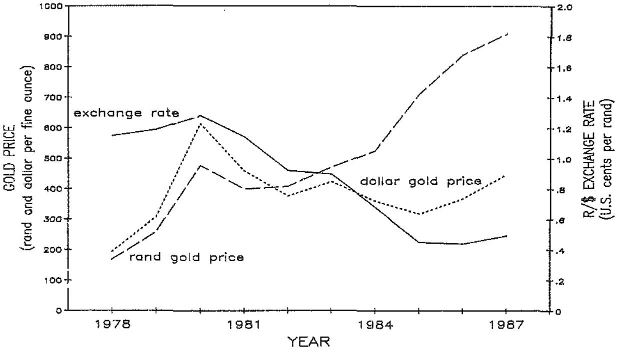
FIGURE 1.3 AVERAGE WORKING COSTS



YEAR

SOURCE: *Chamber of Mines of South Africa*

FIGURE 1.4 RELATIONSHIP BETWEEN THE GOLD PRICE AND THE R/$ EXCHANGE RATE



SOURCE: *Reserve Bank of South Africa*

depreciation of the rand against the dollar, the result of which has been to compensate for fluctuations in the dollar gold price.

However, profits in actual money terms disguise the effects of inflation, which in the context of the mining industry is primarily escalating working costs. When viewed in real 1980 money terms profits can be seen to have fallen from their high of R7335 million in 1980 to R2674 million in 1987 as shown in Figure 1.5.

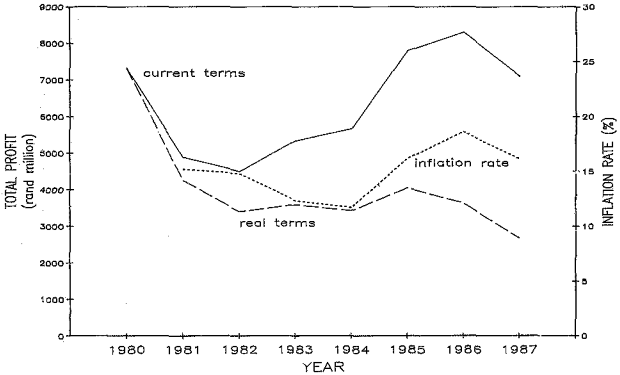
When combined with lower grades the effect of inflation is to decrease profit margins. This is resulting in cash flow problems similar to those which caused the closure of a number of marginal mines in the 1960's. New Kleinfontein and East Champ D'Or in the East and West Rand respectively, are examples of such closures. Likewise, a number of new prospects remain shelved until either one or both these factors can be alleviated. Gencor's Poplar project in the Evander area bears testimony to this.

Though the rand has strengthened against the dollar during 1986 and 1987, most economic scenarios predict it weakening in the long term. Should this not materialise and the gold price falls in rand terms, the effects on the industry, and specifically on marginal mines and new ventures, will be severe.

1.3 Industry Solutions

To alleviate this economic situation of declining grades and increasing costs, most mines have implemented various schemes,

FIGURE 1.5 EFFECT OF INFLATION ON TOTAL PROFIT



SOURCE: *Reserve Bank of South Africa*

some of which are discussed below.

1.3.1 Economy of scale

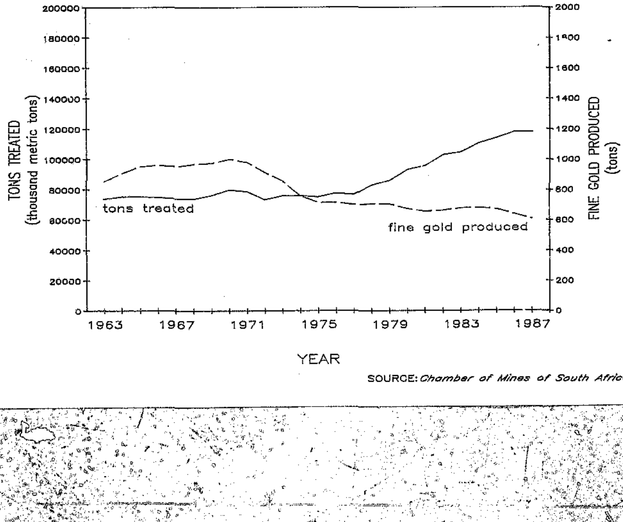
Many mines have capitalised on their ability to increase installed mill capacity, to further reduce already lower pay limits, thus enabling the inclusion of low grade ore which under normal circumstances would have been uneconomical. This mining practice generally leads to increased grade losses through excessive dilution, the net effect of which is that the savings in unit costs are partially countered by increased production costs. Thus the scope for improving profitability by mining at higher rates is limited.

Furthermore, it can be seen from Figure 1.6 that the additional tonnage milled has failed to compensate for its lower grade. Gold production has declined to 605 tons in 1987, a fall of nearly 400 tons since 1970 when the country's production peaked at 1 000 tons. The industry no longer has the proven high grade reserves to get production back to its former levels.

1.3.2 Mechanised mining methods

Mechanised mining in the context of the gold mining industry implies the introduction of trackless equipment in place of conventional labour-intensive mining methods. One of the many benefits cited for this type of mining and used for financial justification is the reduction in working costs. The major cost saving is derived from the significant reduction in waste development rock broken, as most of the development is now on

FIGURE 1.6 SOUTH AFRICAN GOLD OUTPUT



SOURCE: *Chamber of Mines of South Africa*

reef rather than in the footwall.

Two examples of the cost benefits derived from trackless mining operations are summarised below:

- WESTERN AREAS GOLD MINING COMPANY, LIMITED, NORTH DIVISION (NAUDE, 1986).

  Based on a production rate of 35 000 tons per month, total costs were reduced by 24 percent from R54.05 per ton for conventional mining to R40.93 per ton for trackless mining.

- RANDFONTEIN ESTATES GOLD MINING COMPANY, LIMITED, COOKE 2 SHAFT (RHODES, 1986).

  The comparison of costs for conventional as opposed to mechanised mining was based on the development and stoping component only. For a production rate of 40 000 tons per month , costs were reduced by 32 percent from R12.60 per ton for conventional mining to R8.57 per ton for trackless mining.

1.3.3 Mergers

The benefits of mergers involving neighbouring mines are well researched and documented. Rather than restate these, the results of a current operation are used to illustrate the advantages. A good example is provided by Free State Consolidated Gold Mines Limited (Freegold), which came into being as a result of merging the Orange Free State gold mines administered by Anglo American Corporation. This was brought about because of diminishing reserves, under utilisation of assets and high overhead costs. The new operation has combined assets which include 28 shafts, 8 metallurgical plants, 105 000 employees and a gold output of some 110 tons per annum.

The main advantages to Freegold were the creation of a stronger financial base and the maintenance of operating margins. A strong financial base has enabled the financing of various new major capital projects. These were previously impracticable due to the high capital cost and insufficient reserves in the area of the new venture.

In the maintenance of operating margins the improvement in asset utilisation such as shafts and plants was the most important factor. It contributed to an increased unit cost (Rand per ton milled) of 12.1% which was 5.9% below the inflation rate of 18% as measured by the consumer price index (CPI). Optimal extraction of ore reserves kept its unit cost increase (Rand per metre^2) 3.9% below the CPI, while the unit cost increase (Rand per ton treated) due to the rationalisation of the joint metallurgical scheme was 5.6% below the C.P.I.

Maximisation of the benefits achievable from the above, or any other, approach still requires the optimal extraction of ore through accurate and detailed mine planning. Inherent in this assumption is an accurate up-to-date geological model incorporating a detailed grade distribution.

1.4 Additional Solution - Computerised Mine Planning

Gold mining operations are becoming increasingly complex, as mining is pursued to greater depths and into geologically more complex areas. These complexities are making dynamic mine planning more difficult, with mining depths approaching 4 000 metres and geological displacements of up to 1 000 metres becoming common. In addition, mine management has to plan the operations of up to 20 000 men underground every day, base revenue forecasts on nearly 1 000 000 sample points, and conduct possibly 30 000 survey traverses and 100 000 trigonometric calculations every year for the new extensions to its workings and excavations.

In addition, the design of every excavation has to take into account geology, rock mechanics principles, environmental and mechanical engineering. Coupled with this is the fact that mine management is confronted with a continually moving target in the guise of changing gold price, escalating costs and a very dynamic industrial relations picture. Any change to one parameter has a ripple effect on others. For example, the result of a higher gold price is to lower the pay limit, the consequence of which is to increase the payable ore reserves, which in turn affects the development layout and mining sequence.

The environment described is far less stable than that which existed twenty or even ten years ago. In fact, the sheer volume of data and the range of interrelated factors, which must of essence be integrated in planning, have brought about a situation where mine planning can no longer be effectively carried out by manual methods which were suitable twenty years ago for a shallow, stable mining environment. Technical computing is seen as the vehicle to modernise mine planning and thereby improve cash flows.

2 TECHNICAL COMPUTING WITHIN THE MINING INDUSTRY

This report is devoted to evaluating the benefits that can be derived from technical computing. It is thus appropriate to begin by establishing both a brief historical perspective and current review of developments prior to examining individual application areas.

2.1 Historical Perspective

Computers as we know them today emerged five decades ago in the 1940's. The impetus and framework for their development was provided by the second World War, during which period the first full scale digital computer was completed. The main features of early computers were logic circuitry based on vacuum tubes, with punched card input and magnetic drum storage.

A quantum jump in computer technology was the replacement of the vacuum tube by the transistor in the 1950's. Transistors were smaller, generated less heat and thus proved more reliable. The middle 1960's saw the next innovation, the introduction of solid state logic technology and integrated circuits. This period also witnessed the introduction of the first minicomputer. Building on the micro-chip technology of the 1960's, the next decade saw the development of the microprocessor from which evolved the microcomputer.

The industry's rapid rate of technological progress from bulky and slow vacuum tube calculators, to compact, reliable and ultra-fast electronic machines has continued into the 1980's.

Silicon memory and logic circuits are being increasingly miniaturised, storage capacity is expanding and communications are being enhanced.

In brief, computer development can be said to have passed through four generations, each marked by a dramatic technological breakthrough. The first generation being the vacuum tube computer; the second the transistor computer; the third the integrated circuit computer, and the fourth the microcomputer. The four generations and their associated technological innovations are summarised in Table 2.1.

2.2 Current Developments

The 1980's, in common with the preceding decades, have witnessed the continuing trend of smaller size coupled with improved storage capacity and increased processing speed, all at lower costs. Innovations which have been developed in this decade include the workstation, networking, interactive graphics and parallel processing. Many of these concepts, though conceived in the 1970's, are products of this decade.

Traditionally the mining industry has been served by large mainframe computers. This is because mining software is very input/output intensive. The technological innovations of the 1980's, particularly the increase in memory and processing power, have allowed these tasks to be run on minicomputers or powerful microcomputers. These advances have conceived the concept of the workstation.

Table 2.1 Computer generations

	First	Second	Third	Fourth
Time frame	1944–1955	1956–1963	1964–1970	1971–Present
Technology	Vacuum tubes Magnetic drums Magnetic cores Punched cards	Transistors Magnetic discs	Integrated circuits Minicomputers Virtual machine Families of computers	Microprocessors Microcomputers
Programming	Stored program	Batch processing	Operating system	Virtual storage
Popular computers	UNIVAC I IBM 600/700	IBM 1401	IBM SYSTEM/360 HONEYWELL 200	IBM 303

The workstation is a stand-alone processor, ideally with high performance graphics, to which is attached a range of peripherals. In the context of the mining industry the peripherals would commonly be a digitising tablet, plotter and printer. The benefits of such a system are that it provides computing power where it is needed, that is on site. Furthermore, as the system is dedicated to a particular application area the user has immediate response.

Networking is the process of linking hardware and, as a result, software into a single environment. A central processor, workstations, plotters and other peripherals linked together is an example of such a system. These systems can be connected to others, or to foreign mainframe computers, in different locations. This concept is ideally suited to the mining industry where mines are widely and often remotely located. The benefit of such a system lies in the sharing of resources. Users have access to all the data, software and hardware in the network as if it were local to their system. This contributes to the maximum utilisation of system resources, improved communications and reduced system management functions.

Graphics in the form of plans, contours and sections, to name but a few, have long been used in the mining industry. The output media of printers and plotters are well suited to final production drafting. However, the repetitive nature of geological modelling or mine planning renders them inefficient for this type of work due to their relatively slow speed. In addition, the two-dimensional nature of the output is not conducive to error detection. The introduction of interactive graphics displays has revolutionised this area of computing. The user is now able to preview and interpret the results of his work in the minimum time. Furthermore, he can interface directly with the information, view it in any orientation and modify it quickly.

Conventional processing takes a systematic approach towards executing tasks, the job is done but not always in optimal time. Parallel processing divides the tasks into related segments and processes them simultaneously using multiple processors. Mathematical modelling and database management operations are good examples of applications which benefit from this architectural approach.

2.3 Application Areas

Computers and their first technical applications in the mining industry can be traced back to the late 1950's. These early uses were mainly confined to the fields of ore reserve estimation and statistical analysis.

Today, application areas cover a vast spectrum ranging from data management through orebody modelling and planning to process control. All these disciplines have a part to play in the overall mining operation. However, in the context of mine planning, only those of direct significance are considered in this report.

2.3.1 Database management

Mining is a multi-disciplinary operation involving the interaction and cooperation of the various disciplines involved in the overall operation. The effectiveness of these many disciplines relies heavily on the sharing of common data. With the volume of data becoming available daily in a mining operation, the database is fundamental. The database management system is the receptacle for all data storage and the means by which all other dependent systems access information. The provision of adequate data capture, verification and interrogation facilities to an acceptable and satisfactory degree must be dictated by the end user.

While the sharing of data is considered vital, security is of extreme importance in view of the highly sensitive nature of some of the content. The database must be structured for easy reference by multiple users, while at the same time allowing each discipline to retain control over the management and security of its own data.

Due to the fact that much of the data associated with mining is positional, graphical output is essential. It must include plans, contours, sections and isometric views. The user should be provided with the ability to rotate, pan and zoom these views, while also being able to control the picture content, format, size and colour. Flexibility in every detail of the output enables corporate layouts to be conformed with. Facilities must also be provided to link this graphic data to the non-graphical, for example, grade and unit costs to name two.

Data capture should cater for the many differing sources of data. Stereo-plotters, electronic theodolites and portable capture units such as the gold analyser (Stewart and Nami, 1986) are examples in this area. A provision should be made for the customisation of screen menus for the capture and modification of data. The ability for the user to export and import any data to or from an outside source, and the creation of macros to facilitate the automation of complicated or repetitive commands are further prerequisites.

Interrogation methods should be both graphic and non-graphic. Facilities should include a user definable data dictionary to enable the user to define entities and their characteristics in the database. A conventional query language must be provided to allow manipulation of the data in any desired fashion. Provision must also be made for the extraction and reporting of any adhoc user-specified information.

Many databases go beyond the preliminary verification and interrogation concepts and include facilities such as ore reserve estimation. These are considered specialist fields and as such should be treated appropriately.

2.3.2 Geological modelling

Detailed mine planning is a multi-disciplinary operation, incorporating the design and layout of development and stoping and the selection of associated mining techniques and equipment. It involves using standard mining methods to model irregular, uncertain and often changing interpretations of the geological structure of the orebody and its country rock. It is therefore a dynamic and constantly changing process, demanding ready access to the most recent data across many disciplines.

An accurate 3-dimensional representation of the geological structure is essential, as this forms the basis of mine planning. The importance of this cannot be over emphasised, as the orebody is the only revenue earning asset of the mine. The 3-dimensional model must use all available data, be able to represent all geological surfaces such as reef and fault planes, and generate and display lines such as reef/fault intersections.

The use of interactive graphics is essential in the modelling process. The user must have the ability to view any portion of the model in 3-dimensions, rotate it to any desired angle and interactively modify it to suit the human concept with immediate response. Interactive graphics purely represents an extension of the user's interpretive capabilities.

2.3.3 Ore reserve estimation

A wide variety of spatial data analysis techniques are in use, some with a theoretical basis, others justified on empirical grounds and some with no apparent vindication. Methods practiced are the simple polygonal method, weighted moving averages and trend surface analysis, to name but a few. There are a host of other methods in use, the most important of which are the geostatistical methods.

Geostatistics became possible with the advent of the computer. It has gained widespread acceptance and is employed for ore reserve estimation in all the major South African mining houses. However, the exact geostatistical methods used differ from one mining house to another depending on the prevailing school of thought. Irrespective of the technique, the successful application will depend on cognisance of the geological model and particularly on such features as structure and sedimentology.

2.3.4 Mine planning

This is the major area in which the cost benefit of effective mine planning systems will be seen. Planning and scheduling is in general an iterative process and if done by hand can never explore all the possibilities. The result is that the standard rule of thumb approach is traditionally adopted. With the ability to consider iteratively a range of options, the mining engineer is freed of the tedium of bookkeeping tasks and can make the judgmental decisions so important in production optimisation.

The deficiencies of manual methods need little elaboration. In the planning and scheduling environment the importance of graphics facilities is of paramount importance. Numerical output alone cannot equal a pictorial representation of the information.

At a glance the user is able to relate to the information, form a good understanding of the spatial relationships and quickly come to meaningful conclusions.

Via interactive graphics the user can define, design, enter or modify the information and obtain an immediate graphical response which can be viewed and evaluated quickly. These facilities must be flexible and function within the constraints imposed by geological complexity and mining methods. System limitations must not force the user to follow unfamiliar or impracticable design procedures, rather they must promote innovative problem solving.

Non-graphical information is also of importance to the user in the evaluation of his various alternative strategies. Data on costs and rates of advance for example, have a vital bearing on the ultimate option chosen.

2.3.5 Economic analysis

A detailed mine plan depends on the input of various disciplines, however, economic analysis is the cornerstone upon which the plan succeeds or fails. It is the yardstick by which the proposed plan and alternatives are measured.

As well as the standard financial criteria of rate of return and net present value, the facility to cater for uncertainty and risk is of extreme importance. Prior to the advent of the computer it was difficult if not impossible to consider more than a few alternatives. This is now catered for by sensitivity and risk analysis, tools which are essential in modern day planning where the margin between success or failure is often thin.

2.3.6 Ancillary facilities

The aforegoing areas are those considered essential in the improvement in *mine planning*. However, there are other disciplines that have a role to play. Rock mechanics and environmental engineering, to name but two, are fields of which cognisance must be taken. A mine plan, no matter how technically innovative or cost effective, cannot be pursued if for example the basic safety principles are not met.

2.4 *Summary*

In less than half a century computers have developed from bulky mistrusted devices to fast compact machines with universal application. However, it is only since the beginning of the 1980's that their use in the mining industry has started to become widespread. From initial data capture to final mine design, the various steps are being simulated in order to enhance our understanding of the orebody and its potential economic value.

3 BENEFITS OF TECHNICAL COMPUTING

The benefits of computer applications in the minerals industry are well known and documented. Weiss (1969, 1979) together with the proceedings of the APCOM symposia provide some of the best all-round references in this area of technology. The activities in exploration, mining and processing have become the outstanding applications where computer technology has shown the best benefits and returns.

The areas in which the major benefits accrue are:

- Quality, quantity and efficiency of work
- Timeliness of results
- Development of skills

These are discussed in general terms below and are followed by a technical computing case study as a practical example.

3.1 Quality, Quantity and Efficiency of Work

Expanded interpretation capabilities, such as the ability to consider interactively a range of options and not rely on a standard set of rules of thumb, enhances the user's efficiency and accuracy. Furthermore, it allows the user to apply techniques which hitherto were impossible due to the vast amounts of complex calculations required.

Two examples in this area are:-

- Risk Analysis

  Risk analysis represents an extension of sensitivity analysis, which is the process of showing the effect of individual economic parameter changes upon the measure of profitability. In the technique of risk analysis the sensitivity of the project is determined by varying some or all of the input parameters simultaneously.

The process of risk analysis involves determining a distribution function for each of the estimated economic parameters. Using these distributions a number of analyses, generally several hundred, are computed using sample values randomly drawn from each parameter population by the Monte Carlo simulation procedure. This procedure selects a parameter value based on its probability of occurrence. The outcome of such an analysis is a probability distribution of the profitability of the venture, from which a measure of risk can be determined.

Risk analysis, because it defines uncertainty, provides a more realistic basis for investment decisions than does an economic model based upon single estimates. The advantage of such a stochastic analysis is that both risk and profitability can be determined as probabilities. Examples of this would be the risk of not achieving a specified return or the risk of not having an investment paid back in full.

- Geostatistics

Various statistical procedures have been applied over the years in the valuation of ore reserves. The polygonal method and inverse distance weighing are two such techniques. In recent years there has been a tremendous surge of interest in the use of geostatistical techniques in the mining industry.

Geostatistics had its origins in the early work carried out by Sichel (1949) and Krige (1951) into the application of mathematical statistics to gold ore valuation. Since then the mathematical and statistical foundations of geostatistics have been developed and established in France by a research team under the leadership of G. Matheron.

The basic concept in geostatistics is that the variation in grade of an orebody is not usually random, but exhibits a spatial structure. This physical and statistical structure of the orebody is defined by a semi-variogram, which describes the degree of correlation between sample values at varying distances apart. Following the construction of the semi-variogram the deposit evaluation is performed using the estimation method of kriging. This technique involves assigning weighting factors to sample values based on the semi-variogram and the geometry of the sample points relative to the area being estimated.

The major benefit of geostatistics is that the technique of kriging provides the best unbiased linear estimator. That is, the computed value should be, on average, equal to the actual value and not systematically higher or lower. Furthermore this technique allows a determination of the reliability of the estimate by providing a measure of error (standard deviation) with every estimate.

Mine design work involving mine layouts, shaft design, mine planning and scheduling requires both innovative and routine thought processes, as well as the manipulation of large data sets and the use of both simple and complex mathematical techniques. Historically, these routine processes have been labour intensive, time consuming and prone to errors. Performance of these processes via technical computing reduces the design time. This is particularly important in the context of production planning where severe time constraints are imposed.

Major benefits in this area are:

- Increased accuracy in mineral grade prediction leading to better mining decisions and improved yields.
- The ability to consider more alternatives.
- Improved visual representation.
  The facility of interactive 3-D graphics allows more accurate modelling of the orebody which may result in less primary and secondary development being required to exploit the deposit.
- Improved mine design by reducing the possibility of geometrical and rock mechanics interference between one excavation, other excavations and geological disturbances.
- Improved utilisation of the resources necessary to exploit the orebody.
- Better utilisation of personnel required in mine planning and design.

3.2 Timeliness of Results

Mining is a dynamic operation, the efficiency of which depends on management's ability to react quickly to external influences. The key to rapid decision making is the availability of accurate up-to-date information.

Information on a mine becomes available on a daily basis from [illegible] sources such as surveyors, samplers and geologists. There is a marked time lag between the time that this data is captured until it has been transcribed onto plans and been made available to other mining disciplines. The nature of these new findings means that during this period key decision makers are working with obsolete data.

Major benefits in this area are:

- The ability to evaluate alternative mine design and planning options faster.
- The ability to design a mine with more accurate information.
- Up-to-date and accurate reporting of information.
- A common database which guarantees that data is complete and readily available for analysis by all disciplines concerned.
- Up-to-date drilling records and analyses, which will improve control of borehole drilling operations and efficiencies in future drilling operations, thus reducing exploration costs.

3.3 Development of Skills

This is probably one of the most underrated areas which is all to frequently overlooked. Mining companies spend millions of rands annually on education and training. In today's highly competitive environment the only way to stay in the forefront of technical developments, and benefit from their innovations, is to ensure that staff have access to the latest tools and state of the art technology.

In the field of education computers provide an ideal learning medium. With limited keyboard ability a user can quickly begin to reap the benefits of computer based education systems. A major area where this technology can be employed is in the training of unskilled labour on the mines. At the end of 1986 the unskilled labour force on South African gold mines was in excess of 400 000.

Today many education systems are commercially available. These systems replace the traditional method of instruction, through the use of modularly designed programs to guide the user in a structured manner through the various steps in the learning process. Courses ranging from the simple basics of education through to more specific work related areas such as mine safety can be taught in this environment.

Inadequate training is one of the main reasons why mining companies, who having invested heavily in the latest computer technology, have failed to see it realise its full potential. Training involving the correct choice of staff must not be viewed as a single operation. It must be seen as an ongoing development throughout the life of the system. New staff need to gain a comprehensive knowledge of the system. Existing staff require to learn additional skills, to enable them to fully utilise both their and the system's potential.

Training not only ensures the best utilisation of the software system, but encourages innovative solutions as a user's operating skills and confidence improve. Effective training is an investment which pays for itself.

The major benefits in this area are:

- a more highly educated, skilled and motivated workforce.
- a competitive business edge.
- the realisation of the benefits summarised under sections 3.1 and 3.2.

3.4 Summary

This chapter addressed the three main spheres of activity which can benefit most from the application of technical computing. Other areas not mentioned but worth noting are the increased flexibility and confidence in decision making and the ability to implement quickly the operational changes dictated by external influences.

The current discussion has purposely centered on a qualitative analysis of the benefits of technical computing. A quantitative assessment is given in the following chapter.

4 A CASE STUDY - GEOSTATISTICAL ANALYSIS OF THE WHITE REEF IN THE BATTERY SHAFT AREA AT WEST RAND CONSOLIDATED MINES LIMITED

This study focuses on a single application area selected from those outlined in chapter 2, section 2.3. The analysis involves a technical investigation coupled with an assessment of its economic and technical benefits. The latter form the basis of this project report.

4.1 Objective

The object of this case study is to illustrate and quantify the technical and financial benefits to be derived from the application of a computer based technique in the gold mining industry, namely geostatistics.

With the emphasis on the application of a technique rather than an investigative study, a detailed geostatistical treatise is not warranted. A straightforward geostatistical approach has been adopted for this investigation.

It is realised that more sophisticated geostatistical techniques exist today, each with its own school of thought. However, provided the basic underlying geostatistical assumptions are correct and the technique properly applied, the results irrespective of the procedure will be similar and superior to traditional methods. Indeed it has been shown (David, 1977) that many of the more complex methods show little or no advantage over the straightforward approach.

4.2 Study Outline

This study can be divided into two distinct parts. Sections 4.3 – 4.6 involve a brief review and statistical analysis of the data, followed by the construction of semi-variogram models representing the main features of the deposit. The remaining sections are primarily concerned with the estimation process, namely kriging, and an assessment of the technical and financial benefits to be derived from geostatistics.

4.3 Data for Analysis

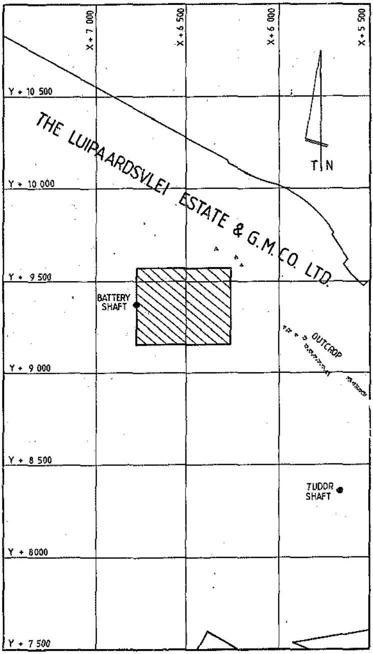
The data which was supplied in typed form by the mine consisted of 1013 underground raise and stope samples. The location of this data which covers an area of approximately 22 hectares is shown in Figure 4.1.

The data consisted of plane of reef co-ordinates which were scaled from 1:200 stope assay sheets together with two variables, gram per ton (g/t) gold grade and channel width. Prior to any manipulation of the data, the reef sample co-ordinates were plotted and superimposed on the stope assay sheets to verify their positional accuracy.

The data was then changed into the format required for use in the estimation procedures. This entailed the following steps:

- Rotation of the sample positions from plane of reef to the horizontal using a given dip and azimuth relative to a point of zero projection. Though there were slight variations in dip and strike between the three levels, it did not warrant

FIGURE 4.1 LOCALITY PLAN



the rotation of sheets on an individual basis.

- Calculation of centimetre gram per ton (cmg/t) values for each sample based on channel width and associated g/t gold value. This was done to satisfy the requirements of support as discussed in section 4.4 below.
- The identification of individual samples by numeric code to enable their correct usage in the estimation procedures.

This data, which is hereafter referred to as the Base Data, formed the basis for future geostatistical estimates.

4.4 Support

The basis for geostatistical analysis in at least the semi-variogram computation is one of equal support lengths. In the Witwatersrand geological environment one is faced with a series of grade intersections at variable thickness. This problem can be obviated by applying the simple technique of accumulation, whereby a new variable is created from the product of the grade times the thickness, namely cmg/t. Since both grade and thickness are regionalized variables, so also is their accumulation, which thus can be used in the semi-variogram analysis.

4.5 Statistical Analysis

The first step prior to an effective geostatistical analysis is a standard statistical analysis of the data. The statistical techniques of frequency distribution and correlation are considered sufficient for this part of the study. The analysis using a standard statistical package was performed on the three variables cmg/t, channel width and g/t.

4.5.1 Frequency distribution analysis

This analysis provides a clear overall picture of the data and its inherent relation[illegible]ps.

4.5.1.1 Centimetre gram per ton (cmg/t)

This analysis was carried out using both the untransformed and transformed (natural logarithm) data. The basic statistics are shown in Table 4.1 with [illegible]ency distributions being represented graphically b[illegible]s in Figures 4.2 and 4.3.

Table 4.1 Summary of statistics for centimetre gram per ton

	Untransformed	Transformed
Number of values	1013	1013
Mean	627.4545	6.0475
Variance	303750.6	0.9779
Standard deviation	551.1357	0.9889
Minimum	1.00	0.00
Maximum	4460.00	8.4029

Examination of the histograms reveal two important points:

FIGURE 4.2 HISTOGRAM OF UNTRANSFORMED CMG/T VALUES

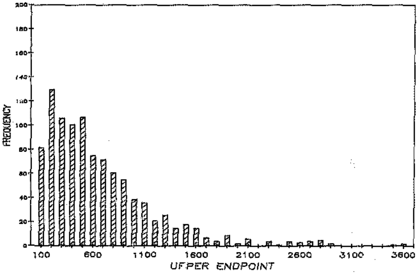
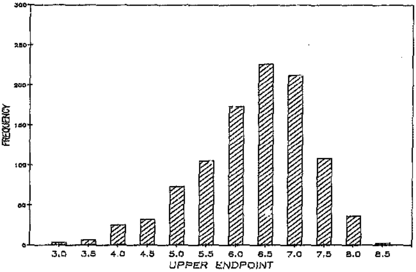


FIGURE 4.3 HISTOGRAM OF TRANSFORMED CMG/T VALUES



- The dis ibution is unimodal. This is a basic requirement in any ore estimation procedure.
- The occurrence of relatively high cmg/t values. These potential outliers having been rechecked were analysed and found to be synonymous with the lognormal pattern of ore value distribution common to Witwatersrand gold reefs.

While it is recognised that the distribution is probably three-parameter lognormal, estimation analysis was done on the untransformed data for the reasons stated in section 4.1

4.5.1.2 **Channel width**

The basic statistics are shown in Table 4.2 below.

Table 4.2 Summary of statistics for channel width

	Untransformed	Transformed
Number of values	1013	1013
Mean	86.5341	4.3666
Variance	1327.3740	0.2021
Standard deviation	36.4331	0.4496
Minimum	12.0000	2.4849
Maximum	220.0000	5.3936

From the above it can be seen that the reef varies considerably in thickness. This is consistent with geological observation which has shown that large *variations in reef* thickness occur, often over short distances. Whereas, cmg/t was clearly *lognormally distributed*, channel width can be seen to approximate a normal distribution as shown by the histograms in Figures 4.4 and 4.5. Unfortunately, one cannot accept this observation as a histogram's appearance is sensitive to the choice of class limits and intervals.

However, the statistical Chi-square test can be used to determine if the values are normally distributed. This classic test investigates the degree to which an empirically determined distribution fits a particular theoretical distribution such as the normal distribution. This is achieved by combining the data into groups and measuring the discrepancy between the observed and expected frequencies, the latter determined by the rules of probability.

If the discrepancy is zero both the observed and expected frequencies agree exactly, while the greater the value from zero *the* larger is the discrepancy between the two frequencies. This discrepancy is compared against a critical value. If the computed discrepancy exceeds the critical value it is concluded that the frequencies differ significantly and they do not belong to the same distribution.

Based on a grouping of eight classes and the generally accepted five percent level of significance as the critical value the

FIGURE 4.4 HISTOGRAM OF UNTRANSFORMED CHANNEL WIDTH VALUES

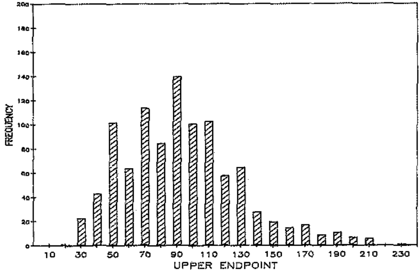
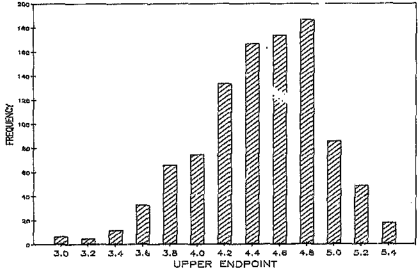


FIGURE 4.5 HISTOGRAM OF TRANSFORMED CHANNEL WIDTH VALUES



computed Chi-square value is 5.96. This is less than the expected value of 11.1 and it thus can be concluded that these values are normally distributed. Based on this the subsequent channel width analysis, which was limited to a semi-variogram study for the purpose of further showing the complementary nature of variography to geology, was done on untransformed values.

4.5.1.3 Gram per ton (g/t)

The basic statistics for gold grade are shown in Table 4.3 below.

Table 4.3 Summary of statistics for gram per ton

	Untransformed	Transformed
Number of values	1013	1013
Mean	7.7364	1.6804
Variance	48.5552	0.8471
Standard deviation	6.9682	0.9204
Minimum	0.0100	-4.6052
Maximum	58.4000	4.0673

As is to be expected the results conform closely to those of cmg/t as illustrated by the histograms in Figures 4.6 and 4.7. This is due to the highly skewed and variable nature of the g/t distribution as opposed to the symmetry displayed by the channel width values. The result of this is that when both g/t and

FIGURE 4.6 HISTOGRAM OF UNTRANSFORMED G/T VALUES

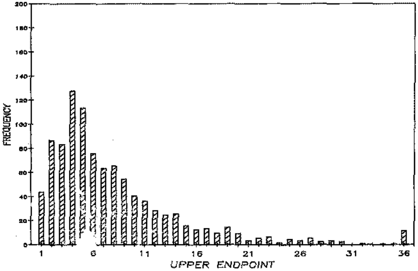
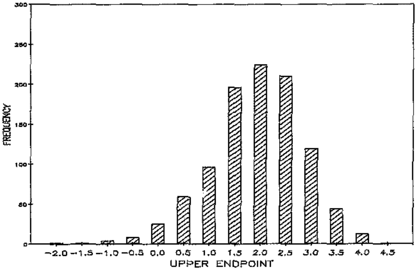


FIGURE 4.7 HISTOGRAM OF TRANSFORMED G/T VALUES



channel width are combined to form the new cmg/t distribution, it will reflect the characteristics of the dominant g/t variable.

4.5.2 Correlation analysis

This routine analysis was done to determine if any significant correlation existed between channel width and gold grade. A scattergram provides a visual qualitative measure of the degree of correlation. From the scattergram shown in Figure 4.8 it is clear that the two variables are uncorrelated.

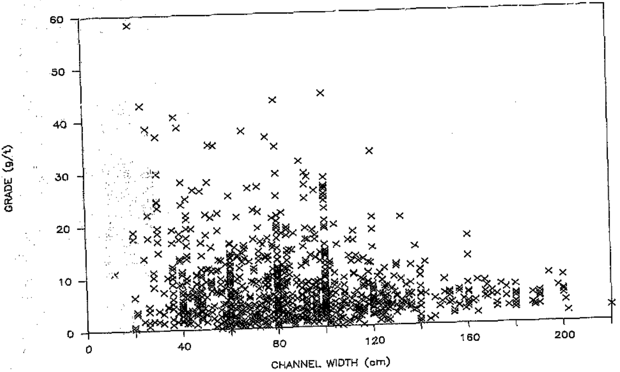
However, defensible conclusions cannot be drawn by merely examining a representation of the sample observations. A numerical quantity that measures the degree of relationship between the variables is required. The traditional correlation coefficient provides such a statistic.

The computed value for channel width and gold grade is -0.17. This figure, which implies very low correlation, substantiates the visual interpretation of no relationship existing between the two variables. This result serves to confirm the similarity of the White Reef with other Witwatersrand gold reefs in this respect.

4.6 Structural Analysis

In a geostatistical estimation, the fundamental tool of structural analysis is the semi-variogram. A semi-variogram is a graph of the variance of the difference in sample grades between points separ^ted by a given distance and calculated at multiples

FIGURE 4.8 SCATTERGRAM OF GRADE AND CHANNEL WIDTH



of that distance. It quantifies the spatial correlation that exists between samples within a continuous orebody. The semi-variogram provides three important geological concepts for use in future estimation procedures, namely:

- Measure of continuity (Nugget effect).

  The nugget effect characterises the magnitude of the random element of samples at distances significantly less than those available for observation. The presence of a nugget effect may occur for a number of reasons, such as sampling and assaying errors or the erratic nature of the mineralisation on a microscale. In general it is due to a combination of both the above factors. The nugget effect appears on the semi-variogram as an apparent discontinuity at the origin.
- Measure of influence (Sill and Range)

  The measure of independance between samples is quantified by the sill and the range. The sill is the value at which the semi-variogram reaches its plateau and the distance at which this occurs is the range. A useful characteristic of the sill is that it should equal the ordinary sample variance of the values within the deposit. The nugget effect, if present, is included in the estimate of the sill. If no correlation exists between sample values, the sill will equal the sample variance for all lag distances.
- Measure of trend (Anisotropy)

  Anisotropy in the data is indicated by the semi-variogram parameters changing with respect to direction. These changes can be broken down into two areas - geometric (where the range is a function of the direction) and zonal (where the sill is a function of the direction). Should the semi-variogram not change in different directions it is said to be isotropic.

4.6.1 Experimental semi-variogram analysis

The purpose of this analysis is to establish the behaviour of the mineralisation and structure of the deposit in the area under investigation.

4.6.1.1 Centimetre gram per ton (cmg/t)

Semi-variograms were computed at distances of multiples of 10 metres (10 metre lags) over a distance of 200 metres in the major directions of the compass to check for anisotropy. A tolerance angle of 22.5 degrees was used as the data was not located on a regular grid. The results are shown in Figure 4.9. They reveal no significant directional anisotropy and as such the deposit can be considered isotropic for this variable.

However, because of the skewness of the value distribution and the presence of some extremely high values, the semi-variograms were also computed using the natural logarithms of the values. The purpose of the logarithmic transformation was only to ensure that the variability of the values did not mask any anisotropy. While a more stable semi-variogram was obtained no genuinely major anisotropic structure could be identified as shown in Figure 4.10.

FIGURE 4.9 EXPERIMENTAL SEMI-VARIOGRAMS OF CMG/T IN THE FOUR MAJOR DIRECTIONS

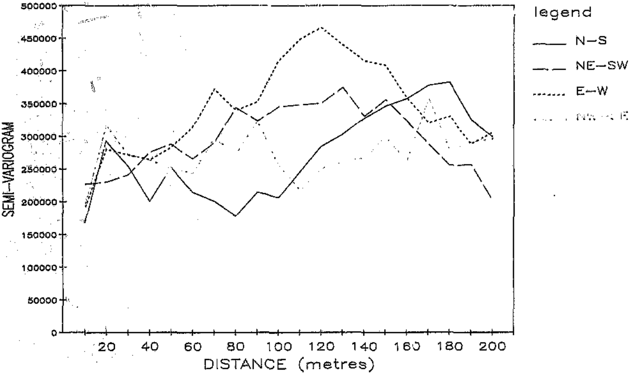
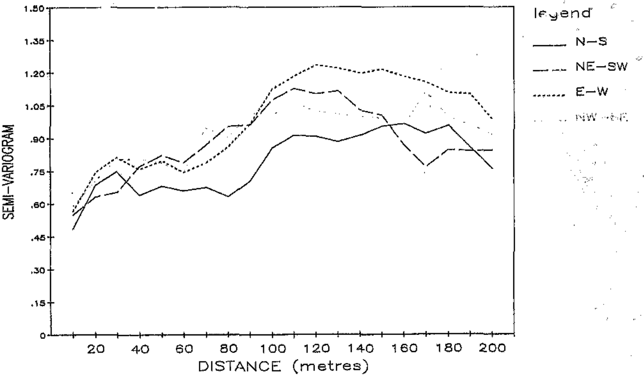


FIGURE 4.10 EXPERIMENTAL SEMI-VARIOGRAMS OF THE LOGARITHM OF CMG/T IN THE FOUR MAJOR DIRECTIONS



With no anisotropy having being established through the use of logarithms the area under investigation can be considered to be isotropic. The four *semi-variograms* based on the untransformed sample values can thus be averaged to form a single combined semi-variogram. This semi-variogram, which was constructed by using a 90 degree tolerance angle, is shown in Figure 4.11. It formed the basis of the geostatistical estimation process.

4.6.1.2 Channel width

The channel width semi-variograms were computed over the same lag and directions as those used for the cmg/t. However, the distance was increased to 300 metres to ensure the incorporation of multiple channels in the semi-variogram computation. The results are shown in Figure 4.12.

The North-South semi-variogram shows a zonal anisotropy (expected from prior geological knowledge) as indicated by its lower variance and clearly definable sill. A noticeable feature of the other semi-variograms particularly the East-West *semi-variogram* is the pronounced hole effect. Generally this can be explained by one of the following:-

- insufficient pairs of points for calculation purposes
- natural semi-variogram fluctuations
- physical depositional features

In this case study the first two options can be quickly discounted. When one considers the depositional nature of the

FIGURE 4.11 AVERAGE EXPERIMENTAL SEMI-VARIOGRAM
OF CMG/T

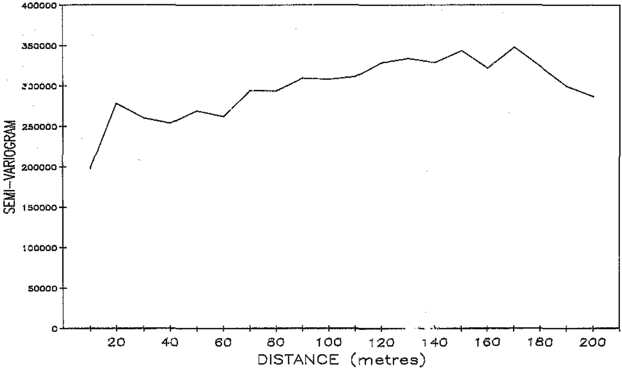
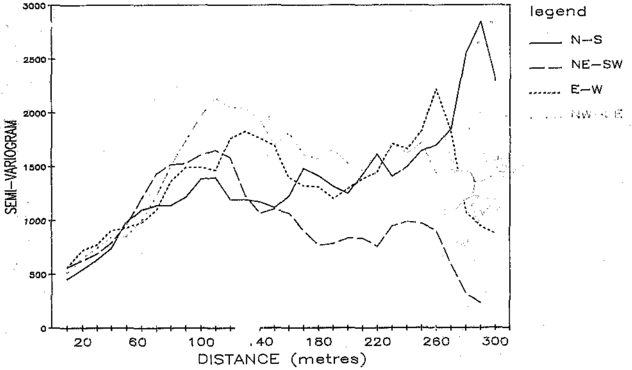


FIGURE 4.12 EXPERIMENTAL SEMI—VARIOGRAMS OF CHANNEL WIDTH IN THE FOUR MAJOR DIRECTIONS



Witwatersrand gold bearing reefs the hole effect can be readily explained. The structural continuitv shown by the North-South semi-variogram indicates this as the channel direction. Accepting this, the East-West semi-variogram and to a lesser extent the other semi-variograms would be expected to exhibit a *periodic profil as the semi-variogram computation passes* from channel to inter-channel facies.

This is because in the flow direction the semi-variogram *will in* general use homogenous data in its computation, either channel or inter-channel values. However, at right angles to this direction the computation will use, in addition to either the channel or inter-channel values, a combination of both data types. The result of this is a semi-variogram consisting of areas of low and high variance, the latter occurring when a combination of both data types are used in the computation. This hypothesis satisfactorily explains the theoretical semi-variogram behaviour.

This theory can only be accepted when substantiated by local geological evidence. The only investigation on the White Reef at West Rand Consolidated Mines was that reported by Steyn in 1975 and summarised by Briggs (1986). This was undertaken on a full time basis between 1963 and 1965 and intermittently thereafter until 1975 when a final report was produced. The objective of that study was to determine the controls of mineralisation in the White Reef.

Steyn classifies the White Reef as a braided fluvial fan deposit composed predominantly of longitudinal bars. Figure 4.13 adapted from Steyn shows the paleoflow directions in the vicinity of the area under investigation. From this plan it can be clearly seen that the paleoflow direction is estimated to be approximately South-South East. This is in good agreement with the results of the semi-variogram analysis, especially when it is remembered that a tolerance of 22.5 degrees was used in its computation.

The nature of this study did not warrant the future use of channel width in the estimation process. However, had it been required a more accurate definition of the anisotropy would have been attempted. This would have involved the re-computation of the semi-variogram at various directions around the anisotropy combined with a smaller tolerance on the search angle.

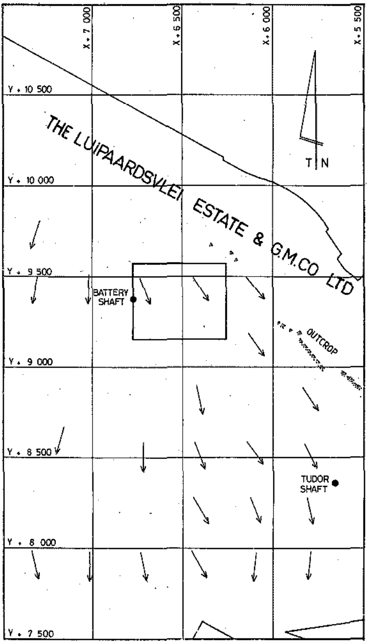
4.6.2 Semi-variogram modelling

An experimental semi-variogram is purely a data summary technique. It is used as an aid in determining the variability of the mineralisation and the structure of the deposit. Conclusions about the deposit must be made by a process of inference. In order to use the semi-variogram in the estimation procedure it must be related to a theoretical model. This process is analagous to that of constructing a histogram from a set of data values and inferring from it a theoretical distribution for the population.

4.6.3 Semi-variogram models

A number of models exist for semi-variogram analysis. They can

FIGURE 4.13 PALEOCURRENT PLAN



be divided into two groups, models with a sill and models without a sill. Brief descriptions of those most commonly used are given below:

4.6.3.1 **Models with a sill**

- Spherical model

  The spherical or Matheron model which was derived on theoretical grounds has been shown to satisfactorily represent the semi-variogram of many different ore deposits (David , 1977). This model is characterised by two parameters, the sill and the range. The spherical model rises rapidly from the origin, then gradually flattens off until at a certain distance it reaches its sill at which it remains. This distance at which samples become independent of one another is called the range of influence. The equation of this model is:

  $$\gamma(h) = C(3h/2a - h^3/2a^3) + Co \qquad \text{for } h \leq a$$

  $$\gamma(h) = C + Co \qquad \text{for } h > a$$

  where

  $\gamma(h)$ = semi-variogram
  C + Co = sill
  Co = nugget effect
  a = range of influence
  h = distance between samples

- Exponential model

  This model is also defined by the sill and the range. It likewise rises from the origin with increasing distance, but more slowly than the spherical model and asymptotically reaches its sill. This model is gaining acceptance in the South African gold mining industry where it has been demonstrated to be more reliable and robust under lognormal conditions. The equation of this model is:

$$\gamma(h) = C[1 - \exp(-h/a)] + Co$$

where

$\gamma(h)$ = semi-variogram

$C + Co$ = sill

Co = nugget effect

a = range of influence (one third practical range)

h = distance between samples

Figure 4.14 shows the general form of the two semi-variograms.

4.6.3.2 Models without a sill

- Linear model

  This is the simplest model and is encountered in cases where there is no range of influence. It is simply a straight line passing through the origin and is defined by its slope. The equation of this model is:

$$\gamma(h) = \alpha^2 h/2$$

where

$\gamma(h)$ = semi-variogram

α^2 = constant

h = distance between samples

- De Wijsian model

Also known as the logarithmic model, this semi-variogram also increases without bound. It was derived from the model of the variation in ore grades at different scales as developed by de Wijs (1951, 1953). This model is essentially a straight line relationship and is plotted against the logarithm of the distance. Because of its logarithmic nature it becomes inapplicable for distances less than one. It can thus only be applied for samples with support of finite dimensions and not point observations. This model is ideally suited to global estimation based on very limited data. Because of this it has been extensively used in the valuation of new deep Witwatersrand gold deposits where surface boreholes are widely spaced. The equation of this model is:

$$\gamma(h) = 3\alpha \ln h$$

where

$\gamma(h)$ = semi-variogram

α = constant, known as the absolute dispersion

h = distance between samples

Figure 4.15 shows the general form of these two semi-variograms.

4.6.4 Model selection

Models were selected for both cmg/t and channel width based on those described in section 4.6.3.

4.6.4.1 Centimetre gram per ton (cmg/t)

As was seen from Figure 4.11 the cmg/t semi-variogram has a clearly defined sill and nugget effect. Because the semi-variogram reaches a sill either a spherical or exponential model must be used. As mentioned earlier recent geostatistical research (Matheron, 1988) favours the use of the exponential model for skew distributions. In keeping with this trend an exponential model was used for this procedure.

The model was fitted iteratively using an IBM PC AT computer and software developed by the Mining Department of the University of the Witwatersrand under Professor D.G. Krige. The model represents a best visual fit of the data based on estimates for the nugget effect, sill and range. Figure 4.16 shows the fitted exponential model, the parameters of which are:

Nugget effect (Co) = 170 000

FIGURE 4.14 SEMI-VARIOGRAMS WITH SILL

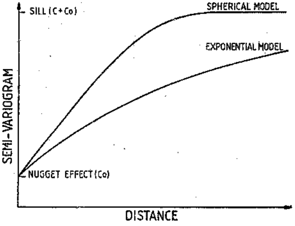


FIGURE 4.15 SEMI-VARIOGRAMS WITHOUT SILL

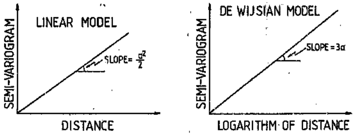


FIGURE 4.16 FITTED EXPONENTIAL MODEL TO AVERAGE EXPERIMENTAL SEMI-VARIOGRAM OF CMG/T

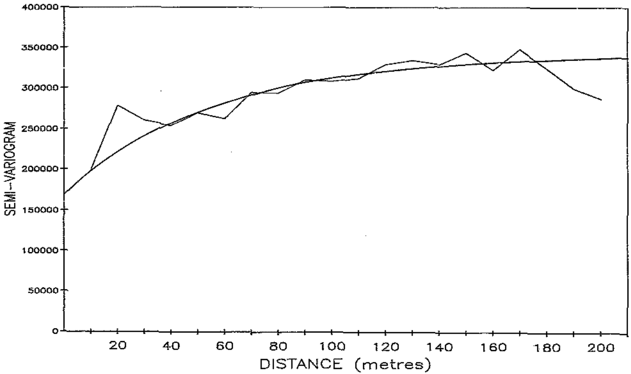
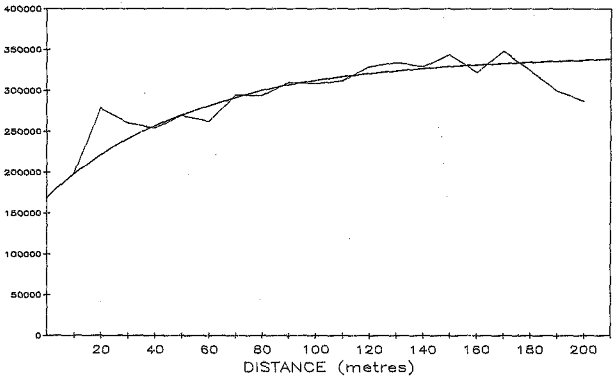


FIGURE 4.16 FITTED EXPONENTIAL MODEL TO AVERAGE EXPERIMENTAL SEMI-VARIOGRAM OF CMG/T



Sill (C + Co)	=	[illegible] 000
Range (metres)	=	55

Examination of both the model parameters and the fitted model reveal two areas of concern.

- The nugget effect
- The sill

Each of these are discussed separately below:-

- The nugget effect

  The behaviour at the origin, namely the nugget effect, plays a crucial role in the semi-variogram fitting as it has a marked effect on the kriging estimation. Figure 4.16 shows the nugget effect to be based on pairs showing a fairly scattered behaviour. Depending on the view taken in extrapolating the first few semi-variogram values back to the origin, different nugget effects can be obtained.

  The only way to improve the estimation of the nugget effect and reduce personal bias is to use additional data at closer distances. This was achieved by re-computing the semi-variogram based on a 2 metre lag distance. Though this reduced the number of pairs at the shorter distance, substantially more than the 30 considered sufficient by Journel and Huijbregts (1981) to enable a confident reassessment of the nugget effect were available for computational purposes. The results of this semi-variogram are shown on Figure 4.17 and tabulated in Table 4.4 below.

FIGURE 4.17 FITTED EXPONENTIAL MODEL TO AVERAGE EXPERIMENTAL SEMI-VARIOGRAM OF CMG/T AT 2 METRE LAG INTERVAL

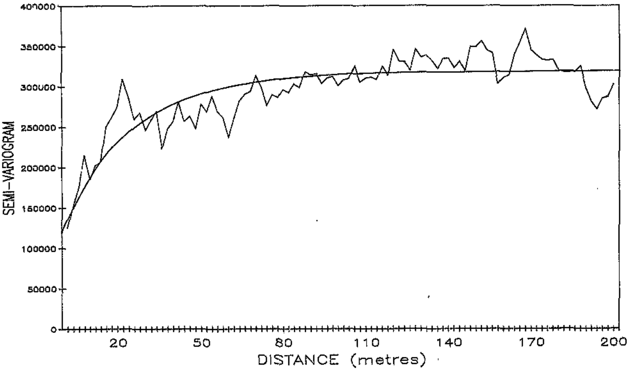


Table 4.4 Comparison of the nugget effect at different lag distances

Lag distance (metres)	Nugget effect (Co)
10	170 000
2	120 000

- The sill

The sill is set at the value where the semi-variogram stabilizes. In theory this should coincide with the overall variance of the data. In order to see if the sill could be brought into line with overall sample variance the experimental semi-variogram was re-computed over a distance of 300 metres.

The rationale behind increasing the distance was that the overall picture may be masked at shorter distances due to the variable nature of the sample pairs. Over a longer distance the variability should be minimised, thereby allowing a best estimate for the sill. *Figure 4.18* shows the new semi-variogram which does give credence to the idea in at least this case study. Table 4.5 below shows the new estimate for the sill compared with both the previous and theoretical estimates.

FIGURE 4.18 FITTED EXPONENTIAL MODEL TO AVERAGE EXPERIMENTAL SEMI-VARIOGRAM OF CMG/T OVER DISTANCE OF 300 METRES

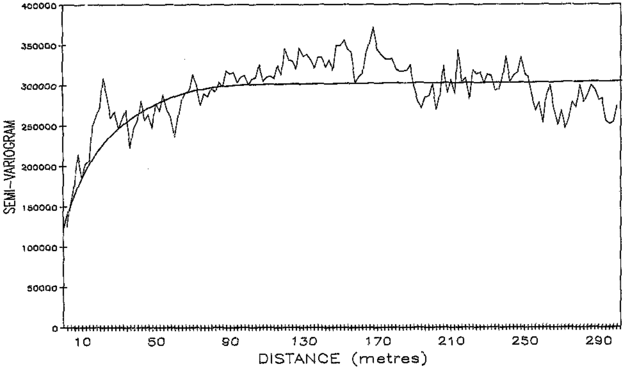


Table 4.5 Comparison of the estimated sill value at different distances

Distance (metres)	Sill (C + Co)	
	Theoretical	Estimated
200	303 750	340 000
300	303 750	305 000

Based on the above semi-variogram recomputations the following mathematical model was accepted as the basis for all further geostatistical estimation procedures. The model parameters are:

Nugget effect (Co) = 120 000
Sill (C + Co) = 305 000
Range (metres) = 25

4.6.4.2 Channel Width

Though, as previously stated, channel width was not used in the estimation procedure a model was fitted to further illustrate the technique of experimental semi-variogram modelling.

Based on the experience gained from fitting the cmg/t model, the North-South semi-variogram was re-computed at a lag interval of 2 metres. Both spherical and exponential models were fitted to the experimental semi-variogram. It was found that while both models adequately represented the sill, the former best accounted for the behaviour over the slope. The selected spherical model is shown in Figure 4.19. The model parameters are:-

Nugget effect (Co)	=	275
Sill (C + Co)	=	1330
Range (metres)	=	120

4.7 Comparison of the Actual Stope Panel Values with the Geostatistical and Mine Estimates

This comparison, based on cmg/t, allows both an illustration of the respective merits of the two estimation techniques and a quantification of the discrepancies which can occur between prediction and recovery when the two methods are used.

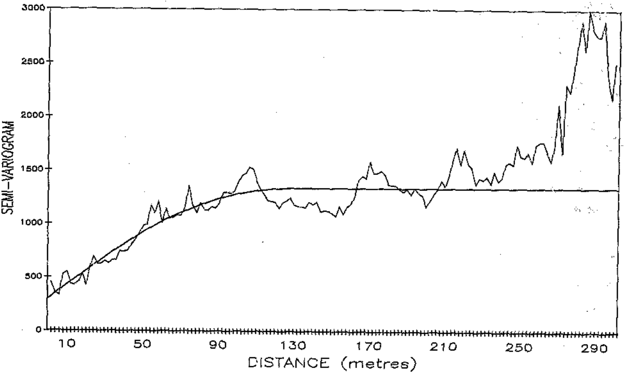
4.7.1 Definitions

- Mine Value: the value assigned to a stope panel by the mine sampling department prior to mining. A brief description of the sampling procedure and the methods used to determine the average value for a panel are given in Appendix 1.

- Geostatistical Value: the value assigned to a stope panel by kriging prior to mining. This is analogous to the mine value. A brief description of this technique is given in section 4.10 prior to its use in the estimation procedures.

- Actual Value: the average value assigned to a stope panel by kriging based on all sampling within the vicinity of the

FIGURE 4.19 FITTED SPHERICAL MODEL TO EXPERIMENTAL NORTH–SOUTH SEMI–VARIOGRAM OF CHANNEL WIDTH



panel. All sample values internal as well as external to the panel are used. It thus represents the best available estimate of the true value of the panel.

4.8 Additional Data

This data provided the basis for the comparison of the actual values with the estimated geostatistical and mine panel values.

4.8.1 Sample data

The additional data consisted of 562 raise and stope samples covering the period of the investigation from 12 November 1986 to 14 September 1987. During this period a total of 38 stope panels were mined. The mine supplied 427 of the additional data values in the same format as the Base Data, the balance of the values being captured from 1:200 stope assay sheets using a digitising tablet. The data was manipulated in the same manner as the Base Data with the exception of the gold content for the 135 sample values.

The digitally captured gold grades represented regressed g/t values. Calculation of the actual cmg/t values was achieved by application of the standard mine regression tables to the regressed values, in order to obtain unregressed bullion (gold/silver) values. The bullion values were then split to remove the silver content from which the cmg/t values were obtained.

4.8.2 Co-ordinate data

The co-ordinates of the 38 stope panels were digitised from the stope assay sheets and rotated to the horizontal on the same basis as the Base Data. In addition, the panels were assigned a numeric code to ensure their correct usage in the estimation procedures.

4.8.3 Data verification

The sample and panel co-ordinates were plotted in conjunction with the Base Data. This was done in order to confirm both the correctness of the additional data and its continuity with the Base Data. Figure 4.20 shows a plan of the data values and stope panels.

4.9 Statistical Analysis of Centimetre Gram per Ton

In the light of the previous statistical analysis the use of a frequency distribution was considered sufficient to ensure the consistency of the additional data with the Base Data.

4.9.1 Frequency distribution analysis

The additional data consisted primarily of stope samples based on the results of progressive face sampling. They would thus be expected to show a higher value than the Base Data which included the initial development results. Table 4.6 compares the results of the Base Data with the additional data and also shows their combined value.

FIGURE 4.20 PLAN MAP SHOWING THE LOCATION OF SAMPLE VALUES AND STOPE PANELS

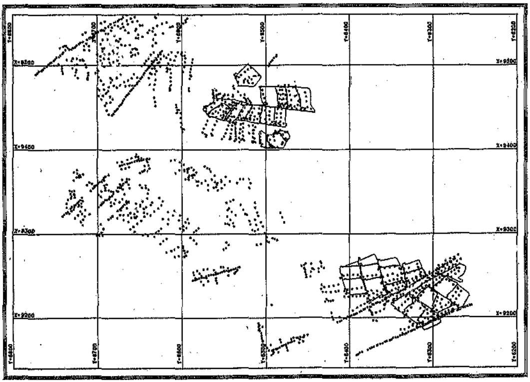


Table 4.6 Summary of statistics for centimetre gram per ton for base, additional and all data

	Base data	Additional data	All data
Number of values	1013	562	1575
Mean	627.4545	772.5632	679.2326
Variance	303750.60	380688.70	336036.7
Standard deviation	551.1357	616.9997	579.6867
Minimum	1.0	44.80	1.00
Maximum	4460.00	4370.80	4460.00

It can be seen from the above table that the results confirm what was geologically and statistically expected. The effectiveness of this tool in verifying the data was well demonstrated. The data was thus accepted for use in the estimation procedures.

4.10 Kriging

Kriging is a multiple regression procedure for arriving at the best linear unbiased estimator or best linear weighted moving average estimate of the ore grade for an ore block of any size. The technique was developed in France around 1960 by Matheron (Matheron, 1963) and named after D.G. Krige who is considered the first to make use of spatial correlation in the field of mineral resources valuation.

The method involves assigning an optimum set of weights to all the available and relevant data based on the semi-variogram. The main advantages of the technique are the avoidance of systematic bias errors and the minimisation of the error of estimation.

4.11 Computation of Geostatistical and Actual Values

The technique of Ordinary Kriging (kriging without the mean) was used to compute the kriged and actual values. This was due to the unavailability of a computer program for Simple Kriging (kriging with the local mean) at the time of the investigation. The exponential semi-variogram model determined in Section 4.6.4.1 formed the basis of the estimating procedure. The software used in the kriging estimation was the Geostokos System, which is the proprietary of Geostokos Limited, London.

Due to the small time frame and number of stope panel samples it was decided not to re-model the semi-variogram on a quarterly basis, but rather use the exponential semi-variogram model as fitted to the Base Data as the basis for the kriging procedure. The validity of this approach is discussed later.

4.11.1 Geostatistical values

This procedure was used to estimate the value of a stope panel prior to mining. Only Base Data values and additional values available prior to the mining of a panel were used. This was achieved by flagging each value and panel by a numeric code. Values within a panel were assigned a code of one greater than the panel. In the estimation procedure only values with a numeric code of less than or equal to the panel code were used.

This method assured that no values either *within* or ahead of the panel being estimated were used in the kriging procedure, thus simulating actual knowledge at the time of mining.

4.11.2 Actual values

In this procedure a stope panel was assigned a value with no restriction on selection criteria. The actual value assigned was thus based on all information available *prior to*, concurrent with, and after the mining of the panel.

4.12 Results

The results for both the actual and geostatistical panel values together with their equivalent mine estimates are collated on a panel by panel basis in Table 4.7.

4.13 Analysis of Results

This analysis involves a reconciliation of the differences between the actual values and the geostatistical and mine estimated values.

4.13.1 Reconciliation of actual and estimated values

The best comparative representation of actual and estimated values is a scattergram. This technique provides a good graphical method of summarising and illustrating the correlation between two variables. The general features of such a plot are shown schematically in *Figure 4.21* in which the scatter of points is replaced by its boundary line.

FIGURE 4.21 SCHEMATIC ILLUSTRATION OF THE EFFECT OF VALUE ESTIMATION ERRORS

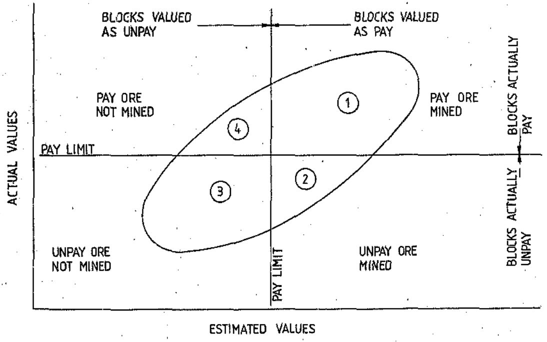


Table 4.7 Comparison of actual stope panel values with their kriged and mine estimates

Panel No	Mine estimate	Kriged estimate	Actual value
1	512.16	842.61	657.94
2	1346.15	928.66	491.34
3	929.39	648.49	487.72
4	1119.30	483.07	809.10
5	841.28	649.25	741.95
6	521.22	1081.40	1011.99
7	350.00	638.97	1043.72
8	265.98	379.33	582.85
9	288.97	288.93	216.84
10	707.44	305.01	328.24
11	724.96	667.23	910.07
12	865.84	579.49	554.23
13	555.74	473.83	203.65
14	323.00	237.63	146.02
15	342.86	564.95	266.71
16	1348.08	728.07	1027.04
17	1312.14	679.59	586.88
18	917.40	689.38	660.90
19	917.40	814.18	541.61
20	528.06	526.99	526.99
21	625.00	636.68	554.02
22	771.45	569.52	525.82
23	771.45	676.05	441.82
24	368.16	627.50	600.58
25	439.56	819.03	562.69
26	501.60	697.46	668.34
27	474.40	644.14	1008.06
28	529.20	630.42	1098.85
29	1270.86	1161.22	1451.89
30	997.92	1123.36	987.45
31	965.08	1236.96	981.84
32	1195.86	1562.69	991.25
33	950.95	1003.27	1032.16
34	845.00	1043.94	912.17
35	936.88	943.33	946.72
36	849.66	769.12	855.99
37	1123.36	871.01	783.79
38	487.20	1182.95	1146.30

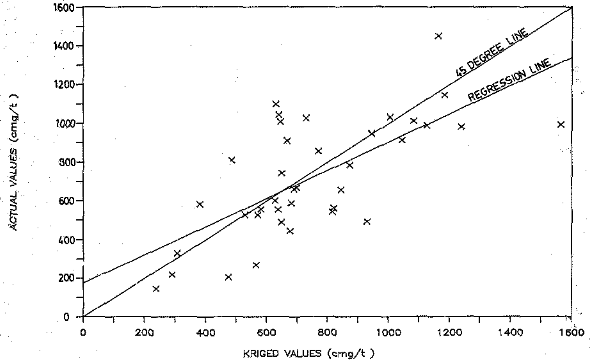
The horizontal and vertical lines represent the actual and estimated cut-off grade and divide the plot into four regions. Regions 1 and 3 represent actual values above and below the cut-off, respectively, and which are treated as such. Regions 2 and 4 contain observations which have been misclassified due to errors in the estimated values. In region 2, values estimated above the cut-off are in fact waste, whilst the reverse holds true for region 4.

The primary objective of any estimation technique is to minimise the number of points which plot in regions 2 and 4. In an ideal situation no points will lie within these two regions. However, in any selection process the estimated value is not obtained. Rather the true value, given the estimated value is above the pay limit or cut-off grade, is obtained. Thus discrepancies will occur between what is predicted and what is mined.

The intention of kriging is to minimise these discrepancies by providing an estimator which on average will produce estimated values lying along or close to the 45 degree line. The consequence of this on selection is to recover a value close to what was predicted.

Figures 4.22 and 4.23 show the scattergrams of the actual values against both the geostatistical and mine estimates respectively. It is clearly evident that the geostatistical method is the superior estimating technique. This is because both the scatter of points is minimised a the regression line is closer to the 45 degree line. The statistics for the two methods, which are given in Table 4.8 below, confirm this. The correlation coefficient implies a significant positive relationship between the actual values and the geostatistical estimates, whereas a low

FIGURE 4.22 SCATTERGRAM OF ACTUAL AND KRIGED VALUES



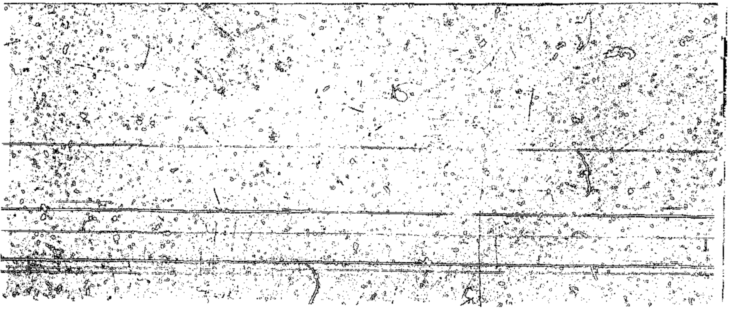
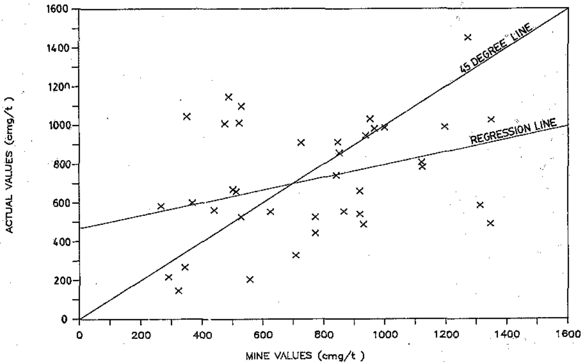


FIGURE 4.23 SCATTERGRAM OF ACTUAL AND MINE VALUES



correlation exists for the mine values.

Table 4.8 Correlation coefficient for centimetre gram per ton for kriged and mine estimation

	Kriged vs Actual	Mine vs Actual
Correlation coefficient	0.69	0.35

4.14 Reconciliation of the Semi-Variogram Model used for the Geostatistical Computations

In the early stages of the development of a project, where data is limited, it is unlikely that the experimental semi-variogram will be adequate for the definition of the underlying semi-variogram model. This was not the case in this investigation, as the exponential semi-variogram model used was based on 1013 sample values in the immediate proximity of the area to be mined. However, as mining progressed away from the worked-out areas, additional data would be incorporated into any new experimental semi-variogram calculation.

This could result in the semi-variogram model being inadequate at later stages of mining. To ensure the robustness of the exponential model it was fitted to the data available at the middle of the mining programme. Figure 4.24, shows the fitted model. The recalculated semi-variogram incorporating the additional data confirmed the validity of the original model parameters for the area of the investigation. Their use in the geostatistical estimates was thus accepted.

4.15 Benefits Derived from Geostatistics

The benefits derived from geostatistics can be divided into two categories, namely technical and financial. These are treated separately.

4.15.1 Technical benefits

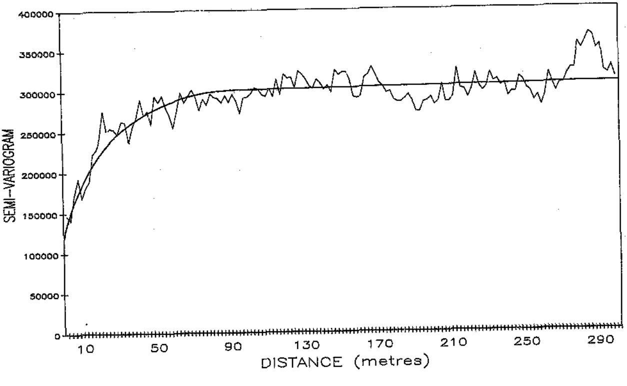
The principal benefits achieved by the successful implementation of geostatistics are outlined below.

4.15.1.1 Improved ore reserve estimation

When mining selectively a panel is mined provided the estimated value exceeded the pay limit. However, actual recovery is the true and not the estimated value. The accuracy of this estimate is of extreme importance in the case of mines which are faced with increasing selectivity as percentage payability decreases.

In any estimation procedure the overall value is a combination of undervaluation of the actual value in the lower grade categories and overvaluation in the higher grade categories (Krige, 1951). The effect of this is that individual panel estimates will vary from their actual value. However, the combined effect of under and overvaluation can be to produce an acceptable global estimate.

FIGURE 4.24 FITTED EXPONENTIAL MODEL TO AVERAGE EXPERIMENTAL SEMI-VARIOGRAM OF CMG/T AT MIDDLE OF MINING



For cut-off grades over the whole value range of the deposit comparisons have been made between the kriged and mine estimated values. These values which were derived from the respective regression equations are tabulated in Table 4.9 below.

Table 4.9 Comparison of kriged and mine estimated values at selected pay limits

Pay limit (cmg/t)	Kriged estimate (cmg/t)	Mine estimate (cmg/t)
200	35.81	-
300	172.77	-
400	309.74	-
500	446.71	92.22
600	583.68	395.58
700	720.65	698.93
*719.62	747.53	758.45
800	857.62	1002.29
900	994.59	1305.64
1000	1131.56	1609.00
1100	1268.53	1912.35
1200	1405.50	2215.71
1300	1542.47	2519.07
1400	1679.44	2822.42
1500	1816.41	3125.78

* Actual mean value

From the above table it can be seen that the kriged and mine estimates of 747.53 cmg/t and 758.45 cmg/t compare well with the actual mean value of 719.62 cmg/t. The estimates are within an acceptable tolerance for an ore reserve estimate, being 3.9 per cent and 5.4 per cent at variance with the actual value.

However, comparison of the individual stope panel values show the mine estimation method to be clearly unacceptable due to gross discrepancies at both the lower and upper value ranges. The result of this is that actual recovery will vary considerably from the predicted value if selection is based on this technique. Use of this method will result in a loss of revenue in the lower value ranges due to mineable panels being classified as unpay and not mined. In the upper value range all the mineable panels will in general be correctly classified, however, their revenue expectations will be unrealised.

The kriged values show a significantly smaller difference from the actual values and thereby predict more accurately what is mined. They thus provide a more accurate and confident framework from which to plan further mining activities and form a base for the prediction of future revenues.

Comparison of the errors (i.e., differences) between actual and estimated values further serve to illustrate the superiority of kriging over the mine method for local estimation. Histograms of errors between actual and kriged and actual and mine values are shown in Figures 4.25 and 4.26, respectively. It is clearly

FIGURE 4.25 HISTOGRAM OF ERRORS BETWEEN ACTUAL AND KRIGED VALUES

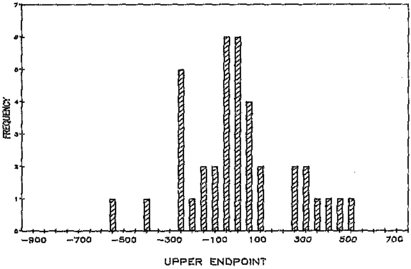
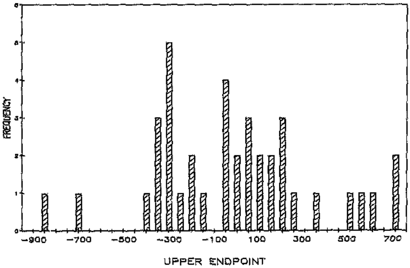


FIGURE 4.26 HISTOGRAM OF ERRORS BETWEEN ACTUAL AND MINE VALUES



seen that when mine estimates are used the histogram of errors is much wider than that between actual and kriged values. The results in Table 4.10 demonstrate this in statistical terms.

Table 4.10 Comparison of statistics for kriged and mine errors

	Kriged errors	Mine errors
Mean	-27.9047	-38.8271
Variance	51774.9700	121372.7000
Standard deviation	227.5411	348.3859
Minimum	-571.4400	-854.8100
Maximum	468.4300	693.7200

4.15.1.2 Improved block factors

A block factor is the ratio of the current sampled value of a block to its estimated value, expressed as a percentage. For the purpose of this study a panel is considered synonymous with a block.

The average block factors for the two estimating techniques were compared for different value categories over the whole value range of the deposit. The results are shown in Table 4.11 below. From this table it is seen that the utilisation of kriging gives a far more accurate estimate of the block factor than the mine method.

Table 4.11 Comparison of block factors for kriged and mine estimation

Value category (cmg/t)	Kriged block factor (percent)	Mine block factor (percent)
101 - 200	61.45	45.21
201 - 300	55.08	63.16
301 - 400	107.62	46.40
401 - 500	64.49	48.75
501 - 600	93.78	96.44
601 - 700	91.37	124.22
701 - 800	102.14	78.98
801 - 900	139.39	86.52
901 - 1000	92.48	103.02
1001 - 1100	138.61	182.87
1101 - 1200	96.90	235.28
1201 - 1300	-	-
1301 - 1400	-	-
1401 - 1500	125.03	114.24
	98.29	107.67

Comparison of the figures in Table 4.11 reveal an analagous situation to that seen in the previous section. While the overall estimated block factors are similar the estimates at different value categories can be seen to vary considerably, especially, in the case of the mine method. The statistics for the two methods which are shown in Table 4.12 below, illustrate the much reduced variance when using kriging.

Table 4.12 Comparison of statistics for kriged and mine block factors

	Kriged block factor	Mine block factor
Mean	98.2884	107.6731
Variance	1093.3480	3686.4590
Standard deviation	33.0658	60.7162
Minimum	42.9800	36.5000
Maximum	174.3000	298.2100

From the statistics it is evident that the average block factor should not be used alone, but in light of the variability of the individual block values.

4.15.1.3 Improved selective mining options

As mentioned earlier, manual estimation methods, while providing a reasonable overall value for an area, can prove dangerous as a guide to selective mining. When mining selectively the aim is to mine only those panels which have been estimated above the pay limit. It must be accepted however, that whatever estimation procedure is employed panels will be mined that are in fact below the pay limit, whilst others above the pay limit will be left unmined.

The implementation of geostatistics would lead to better mining as the effect of kriging, being a superior estimating technique, is to minimise the misclassification of panels. This is illustrated graphically for selected pay limits in Figures 4.27 and 4.28, where it can be seen that kriging minimises both the pay ore not mined and the unpay ore mined, whilst maximising the economically mineable ore.

In the context of this study the changes in ore classification at the mine pay limit of 400 cmg/t are the most relevant. The reclassification of panels at the pay limit by changing from the mine estimation method to the geostatistical technique of kriging is shown in Table 4.13 below.

Reference to Table 4.7 shows that panels 7, 10 and 24 were correctly reclassified and panel 15 wrongly reclassified by kriging. Thus the net effect of changing estimation methods was the ability to increase the number of pay panels mineable by two, namely numbers 7 and 24, from 30 to 32 out of a maximum of 33 actually available. This represents an increase of 6,06 per cent in pay mined.

FIGURE 4.27 CLASSIFICATION OF ORE AT SELECTED PAY LIMITS BY KRIGING ESTIMATION

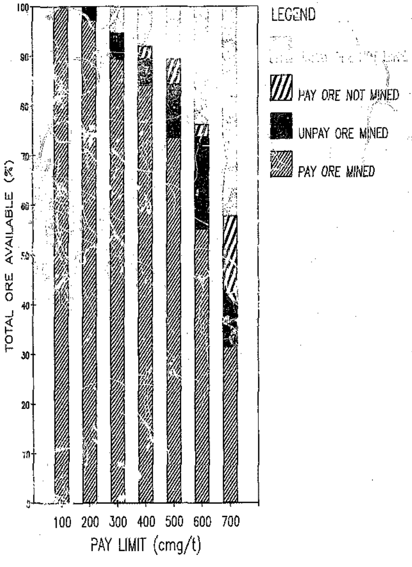


FIGURE 4.28 CLASSIFICATION OF ORE AT SELECTED PAY LIMITS BY MINE ESTIMATION

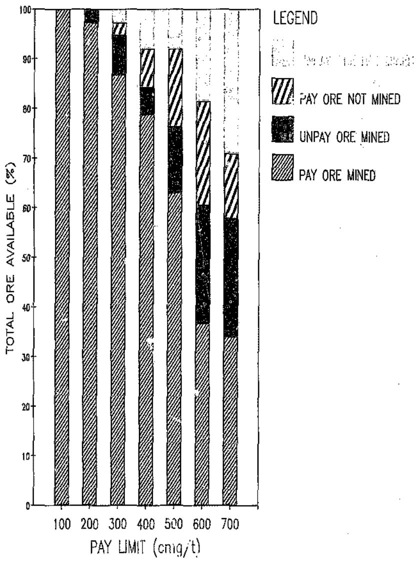


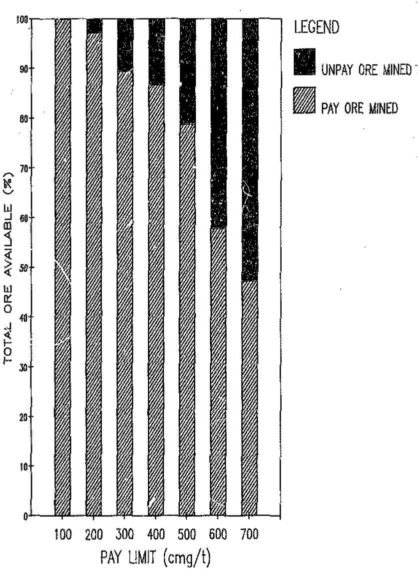
Table 4.13 Change in panel classification at the pay limit due to the use of kriging

Panel No	Mine estimation	Kriged estimation
7	Pay not mined	Pay mined
10	Unpay mined	Unpay not mined
15	Unpay not mined	Unpay mined
24	Pay not mined	Pay mined

Due to a lack of available face the mine did not adhere to planning derived from its estimation method, but pursued a policy of total mining in the area under investigation. While this was only a temporary measure it does allow a further comparison, namely the effect of non-selective mining. By practising no grade control all the pay ore is mined but at a price, namely the reduction of grade due to excessive dilution by the mining of all the unpay ore. Figure 4.29 allows the results of total mining to be viewed against the backdrop of what could have been obtained had selective mining been employed (Figures 4.27 and 4.28).

It is realised that the improvements which would be obtained by the use of geostatistics in a selective mining environment do not take into account the practical extent of stoping. These are addressed under financial benefits in Section 14.15.2.1 where a

FIGURE 4.29 CLASSIFICATION OF ORE AT SELECTED PAY LIMITS BY TOTAL MINING



simulated mining plan based on the results of the two estimation techniques is carried out.

4.15.1.4 Improved short term planning

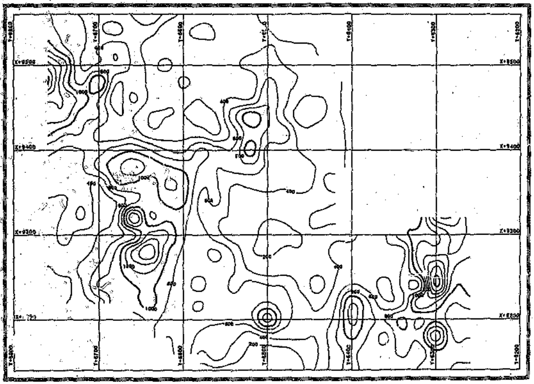
The decision on whether to advance a development end or raise is generally based on sampling and diamond drilling results. When values are below the pay limit the decision may be taken to stop development. This decision often takes no cognisance of possible pay trends that could be encountered.

Because kriging incorporates all available data and their spatial characteristics during interpolation, it will forecast the possibility of pay shoots. Furthermore as kriging can be used to produce a close grid of values, contour maps are easily generated. Such kriged contour maps provide a straightforward tool from which the prediction of possible trends can be made. Figure 4.30 illustrates an interpolated contour map kriged from all of the available data using a 20 metre by 20 metre grid spacing.

An additional advantage of kriging is that each estimate is accompanied by a corresponding estimation error, or standard error. This error can be used to provide a measure of the level of confidence that can be attached to the estimated value. The required confidence limits are obtained by transformation of the estimated errors into the desired intervals.

The lower confidence limits are the most important in ore reserve

FIGURE 4.30 KRIGED CONTOUR MAP OF CMG/T



estimation, as they provide a measure of the probability of the actual value occurring in practice. For example, the lower 90 percent confidence limit gives the value above which one can be 90 per cent sure of the actual mean occurring.

As was the case with the kriged estimates the errors can likewise be contoured. Thus each kriged contour map can be accompanied by a kriging error map, which can be used as a test of its reliability. Figure 4.31 is the accompanying error map for the value contour map.

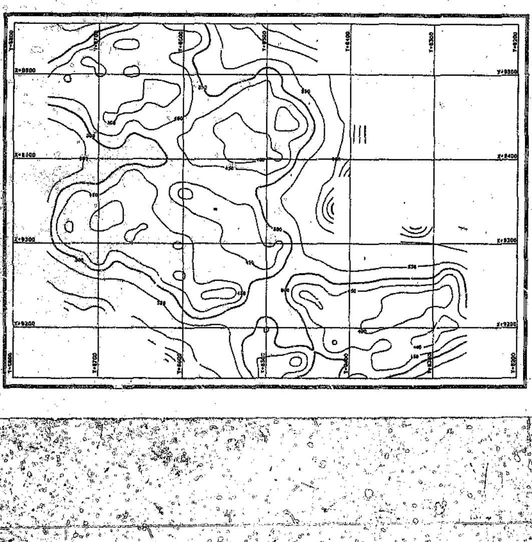
These maps c[illegible]ist in the control of short term planning as th[illegible]s will enable management to view the extent of planning in relation to the latest data. With this aid problem areas can be detected, thus enabling advance planning moves to be initiated with a minimum effect on production.

4.15.1.5 Improved sample spacing

Standard mine procedure is to sample at 3 metre intervals in development ends and raises and at 6 metre spacings in stopes. No provision is made for the variability of the mineralisation resulting from the alluvial nature of the deposition. This variability is of importance as it affects the level of confidence in grade prediction. In areas of low variability the level of confidence in grade prediction is high, whilst the reverse is true in areas of high variability.

The latter holds true for the area under investigation. This was

FIGURE 4.31 KRIGED ERROR MAP OF CMG/T



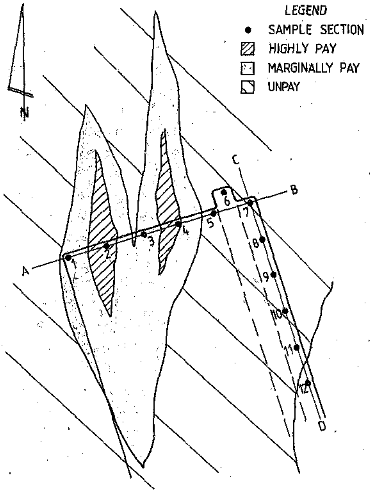
detected in the channel width semi-variogram by the anisotropic structure with lower variance in the North-South direction and subsequently confirmed by geological evidence. This structure has important implications for sample spacing. In order to achieve the highest sampling efficiency, sample spacing should be closer in the East-West direction to ensure that the best estimate of grade for the stope or development end is obtained.

This is illustrated in Figure 4.32 for stopes mining in an up-dip direction. It shows how samples taken at regular intervals vary according to the channel direction. Breast sampling has been included in the plan for illustration purposes.

It is clear that sample value variation is lower in the direction of the channel than normal to it. Consequently the stoping will have a greater variation of values across the face of the up-dip panels (Section A-B and samples 1 to 7) than those taken as breast samples (Section C-D and samples 7 to 12). Thus with a greater value variation as the face advances the sample distance should be reduced. An optimum sample spacing may be obtained by undertaking a detailed geostatistical study similar to that carried out by Rendu (1976).

Anisotropic semi-variograms facilitate the determination of an optimal sampling scheme. However, where sampling is still conducted on a regular interval basis the panel estimate is greatly improved by using kriging. This is because the weighting factors used in kriging are calculated taking into account the

FIGURE 4.32 SCHEMATIC ILLUSTRATION OF THE THEORETICAL EFFECT OF CHANNELING IN RELATION TO STOPE SAMPLING



directions of greater and lesser continuity as determined by the semi-variogram.

14.15.2 Financial benefits

From the viewpoint of economic value, the financial benefits of technical computing can often be tenuous and indirect. The application of geostatistics in this case study provides a clearly *quantifiable* monetary benefit. This is because it is aimed at maximising the extraction of payable ore, which is the only revenue earning asset of the mine. This is discussed in more detail below.

14.15.2.1 Improved profit potential

The progressive improvements gained by changing from a mining method of total extraction to a selective *mining environment*, based firstly on mine estimates and then on kriged estimates, have been demonstrated. However, unless these improvements have a significant effect on profitability the advantages to be gained would be purely of academic interest.

Selective mining based on the panel estimates derived from the two estimation techniques were compared with total extraction. The operating and marginal pay *limits during the period* of the investigation are summarised in Table 4.14 below.

The operating pay limit defines whether ore can be mined at a profit. It takes into account *the changed or changing* conditions such as mining rate, working costs and gold price.

Table 4.14 Pay limits applicable during the period of the investigation

Period	Pay limit (cmg/t)	
	Operating	Marginal
Oct - Dec 1986	400	310
Jan - Mar 1987	380	310
April - June 1987	370	310
July - Sept 1987	420	350

Total working expenses incurred in mining can be split into variable and fixed. The marginal pay limit is based on the variable cost component. This includes the cost of stoping, tramming, hoisting and milling. Development and pumping costs are excluded. Fixed costs (overheads) which are not included must be carried regardless of the volume of production.

The marginal pay limit thus comes into effect when the mill is running under capacity and a mining face has reached an unpayable block of ground that can be left as a pillar. This pay limit defines whether the unpayable ore can be mined and make a contribution to overheads. As any ore mined at the marginal pay limit makes no contribution to offset overheads the practice must only be followed on a limited scale.

The following criteria, as supplied by the mine, were used as guidelines for selective mining in this exercise:

- all panels with estimated values above the operating pay limit are to be mined.
- the mining of panels with estimated values above the marginal pay limit should be restricted to 15 per cent (six panels in the case of this investigation).
- the mining of panels with estimated values below the marginal pay limit should be confined to 5 per cent (two panels in the case of this investigation).

These conditions are subjective and depend on the technical and ...ancial conditions existing at the time of mining.

The area of the study can be divided into two distinct mining environments in terms of payability, namely a northern and a southern portion. These areas were 67 per cent and 100 per cent pay, respectively. All panels in the southern area were estimated as pay by kriging and would have been mined. In the case of the mine estimation technique, only panel 24 was estimated as unpay. However, its estimated value (368 cmg/t) was only marginally below the operating pay limit of 380 cmg/t. This, coupled with the fact that all the adjacent mined panels were pay would have ensured that it was mined. In summary all panels would have been stoped regardless of the estimation technique used.

A different situation existed in the northern area due to its lower payability. The panels available for mining, together with those already stoped and showing their actual value, are indicated in Figure 4.33. The selection criteria for the three different mining scenarios for this area, together with any salient features, are outlined below.

- Total extraction
  All panels are mined. The mine adopted this policy of non-selective mining due to the lack of available face.

- Selective mining using mine estimates
  The bases for the mining decisions are outlined in Table 4.15 below. The outcome of the selection procedure was that three panels above and one below the marginal pay limit were mined. Of the panels mined only three, namely, 8,10 and 14 warrant discussion.

  Panel 8 was mined though the estimated value was below the marginal pay limit. This decision was based on the fact that the adjacent mined panels indicated improving value ahead which should be encountered by this panel. In addition the raise to mine this stope panel had been developed. The alternative to leaving this panel unmined would be to re-raise and re-establish the stope face further ahead in payable ground. The additional cost [illegible] redevelopment was rejected due to the proximity of better values. Accordingly the panel was mined to expose the pay ore ahead.

FIGURE 4.33 PLAN MAP SHOWING THE LOCATION OF STOPED PANELS AND THOSE AVAILABLE FOR MINING

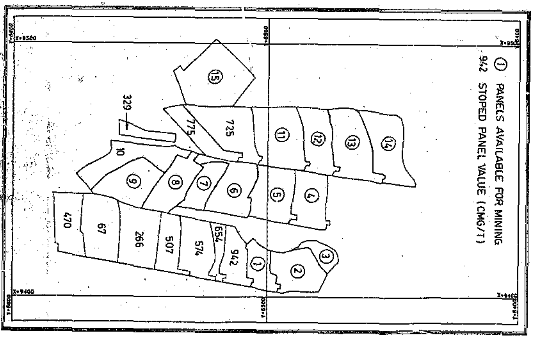


Table 4.15 Basis for panel selection using mine estimation

Month	Panels available	Basis for mining decision	Mine
December	1	Panel estimate (512 cmg/t) pay. Preceeding four panels had pay values of 507,574, 654 and 942 cmg/t respectively. See Figure 4.33. Improving pay value indicated.	Yes
	11	Panel estimate (725 cmg/t) pay. Preceeding two mined panels had pay values of 775 and 725 cmg/t, respectively. See Figure 4.33. Improving pay value indicated.	Yes
	8	Panel estimate (266 cmg/t) unpay and below marginal pay limit. Adjacent mined panels on either side and in direction of mining pay. See Figure 4.33. Development in position to commence mining.	Yes
January	2	Panel estimate (1346 cmg/t) pay. Preceeding panel (1) pay (658 cmg/t).	Yes
	12	Panel estimate (866 cmg/t) pay. Preceeding panel (11) pay (910 cmg/t).	Yes
	7	Panel estimate (350 cmg/t) unpay, but above the marginal pay limit. Preceeding panel (8) pay (583 cmg/t). Adjacent panels on either side and ahead pay. See Figure 4.33.	Yes
February	3	Panel estimate (929 cmg/t) pay. Preceeding panel (2) pay (491 cmg/t).	Yes
	13	Panel estimate (556 cmg/t) pay. Preceeding panel (12) pay (554 cmg/t).	Yes
	6	Panel estimate (521 cmg/t) pay. Preceeding panel (7) pay (1044 cmg/t).	Yes
March	5	Panel estimate (841 cmg/t) pay. Preceeding panel (6) pay (1012 cmg/t).	Yes
	9	Panel estimate (289 cmg/t) unpay and below the marginal pay limit. Adjacent mined panels (67 cmg/t and 266 cmg/t) unpay. See Figure 4.33.	No
	14	Panel estimate (323 cmg/t) unpay but above the marginal pay limit. Preceeding panel (13) unpay (204 cmg/t). Unpay value indicated.	Yes
April	4	Panel estimate (1119 cmg/t) pay. Preceeding panel (5) pay (742 cmg/t).	Yes
	10	Panel estimate (707 cmg/t) pay. Preceeding panel (8) pay (583 cmg/t). Ajacent unmined panel (9) and mined panel had estimated and actual unpay values of 289 cmg/t and 329 cmg/t, respectively. See Figure 4.33.	Partly
	15	Panel estimate (343 cmg/t) unpay but above the marginal pay limit. Adjacent mined panels had pay values of 725 cmg/t and 775cmg/t. See Figure 4.33	Yes

The estimated value of panel 10 (707 cmg/t) was based on the sample values in panel 8 contiguous with it, where the average value was 717 cmg/t. The estimated value is, however, not consistent with either of the adjacent values in the direction of mining, which were unpay. The approach adopted here was that the panel would be mined in conjunction with strict sampling control. Once sampling indicated the face had become unpay mining would stop. Approximately 60 per cent of the panel was mined at an average value of 412 cmg/t before sampling indicated the face unpay.

The estimated value for panel 14 was above the marginal pay limit, whilst the actual value for panel 13 was below this pay limit. On the absence of any additional evidence panel 14 would be mined to prove if mining had entered an unpay area or whether the previous panel constituted an erratic value.

- Selective mining based on kriged estimates

  The bases for the mining decisions are outlined in Table 4.16 below. The outcome of the selection procedure was that one panel above and no panels below the marginal pay limit were mined. In terms of the guidelines provided for selective mining this represented almost perfect mining. The basis for the mining decision for each panel is clear and does not warrant any further discussion.

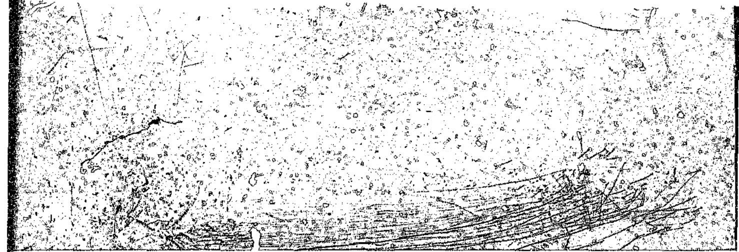
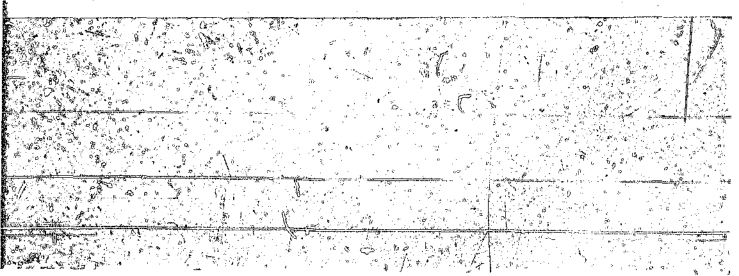


Table 4.16 Basis for panel selection using kriging estimation

Month	Panels available	Basis for mining decision	Mine
December	1	Panel estimate (843 cmg/t) pay. Preceeding four panels had pay values of 507, 574, *654 and 942 cmg/t respectively.* See Figure 4.33. Improving pay value indicated.	Yes
	11	Panel estimate (667 cmg/t) pay. Preceeding two mined panels had pay values of 775 and 725 cmg/t, respectively. See Figure 4.33. Improving pay value indicated.	Yes
	8	Panel estimate (379 cmg/t) marginally unpay. Adjacent mined panels on either side and in direction of mining pay. See Figure 4.33. Development in position to commence mining.	Yes
January	2	Panel estimate (929 cmg/t) pay. Preceeding panel (1) pay (658 cmg/t).	Yes
	12	Panel estimate (579 cmg/t) pay. Preceeding panel (11) pay (910 cmg/t).	Yes
	7	Panel estimate (639 cmg/t) pay. Preceeding panel (8) pay (583 cmg/t).	Yes
February	3	Panel estimate (648 cmg/t) pay. Preceeding panel (2) pay (491 cmg/t).	Yes
	13	Panel estimate (474 cmg/t) pay. Preceeding panel (12) pay (554 cmg/t.	Yes
	6	Panel estimate (1081 cmg/t) pay. Preceeding panel (7) pay (1044 cmg/t).	Yes
March	5	Panel estimate (649 cmg/t) pay. Preceeding panel (6) pay (1012 cmg/t).	Yes
	9	Panel estimate (289 cmg/t) unpay and below the marginal pay limit. Adjacent mined *panels (67 cmg/t and 266 cmg/t) unpay. See Figure 4.33.*	No
	14	Panel estimate (238 cmg/t) unpay and below the marginal pay limit. Preceeding panel (13) unpay (204 cmg/t).	No
April	4	Panel estimate (483 cmg/t) pay. Preceeding panel (5) pay (742 cmg/t).	Yes
	10	Panel estimate (305 cmg/t) unpay and below the marginal pay limit. Adjacent unmined panel (9) and mined panel had estimated and actual unpay values of 289 cmg/t and 329 cmg/t, respectively. See Figure 4.33.	No
	15	Panel estimate (565 cmg/t) pay. *Adjacent mined panels* had actual pay values of 725 cmg/t and 775 cmg/t, respectively. See Figure 4.33.	Yes



Based on the results of mining, cashflow statements were generated for each mining scenario. The key elements of the cashflows are outlined below:

- Square metres mined

  The square metres mined were obtained by planimetering each panel using an automatic digital planimeter.

- Tons milled

  The mill tonnage was calculated on a panel by panel basis using their respective stope widths and a reef density of $2.7t/m^3$.

- Yield

  Individual panel yields were obtained from the actual values by application of the mine call factor and the percentage metallurgical recovery.

- Gold price

  The gold price used in the revenue calculation was the price realised by the mine for each quarter.

- Cost per metre squared

  The costs used were those for the Battery Shaft area. They were supplied by the mine on the same quarterly basis as the gold price. Development costs were excluded as the main developement was well in advance of mining.

The detailed cashflows for total extraction, selective mining based on mine estimates and selective mining based on kriged estimates are set out in Appendices 2,3 and 4, respectively. The results are summarised in Table 4.17 below.

Table 4.17 Summary of results of cashflow analysis for the different mining scenarios

Valuation technique	Total extraction	Selective mining	
		mine estimation	kriging estimation
Tons milled	34620.6	33628.5	32260.7
Yield (g/t)	5.18	5.26	5.40
Revenue ('000)	5288	5223	5139
Costs ('000)	2711	2602	2468
Working profit ('000)	2577	2621	2671
Percentage gain over total extraction	-	1.71	3.65

It can be seen from Table 4.17 that if selective mining based on kriging as opposed to mine estimation was practiced, working profit would have been improved by approximately 2 per cent. When considered in the context of the total volume of production and associated revenue, a 2 per cent increase in working profit represents a substantial monetary benefit.

The tonnage mined in the area of the investigation represented 5 per cent of the total White Reef stope tonnage milled on an annual basis. Application of geostatistics to this reef over the balance of the lease area with the same success, could improve total working profit by R1.0 million per annum. There is also no reason why a similar improvement could not be obtained from the other source of mill tonnage, namely the Ventersdorp Contact Reef.

It should be borne in mind that the area of the investigation was one of high payability (87 per cent). The scope for improvement offered by selective mining is thus limited. In areas of lower payability the advantages of selective mining based on geostatistics should prove greater.

It is evident from the above that there are financial benefits to be gained by the application of geostatistics in conjunction with selective mining strategies. However, to be accepted it must be confirmed in economic terms as the only justification to shareholders for capital investment is the prospect of improved profits.

The capital outlay required in terms of hardware, software and training to implement a geostatistical analysis system is presently estimated at R200 000 for this mine. Hardware is the major cost component, comprising well over half the total outlay.

The high hardware cost is due to the adoption of a decentralised approach whereby all the computing power resides with the user at the mine. Implicit in this is the concept of the microcomputer workstation comprising central processing unit, slave terminals, digitiser tablet, plotter, printer and associated peripherals. A breakdown of the total costs are detailed in Appendix 5.

For the purpose of measuring profitability the total capital outlay is assumed to have been made at the outset of the project. While this approach may not be strictly correct the effects on the cashflow are minimal. No profits are considered to be generated during the first three months of the project. This period is considered necessary to ensure that both training and data capture have reached a position whereby the system can be operated to its full potential. Thereafter profits are considered to be generated mo[illegible]ly with the optimal benefits being achieved one yea[illegible]nstallation.

In view of the relative magnitude of the returns, compared with the investment, inflation and escalation have been disregarded. This is equivalent to assuming a zero per cent real growth rate [illegible] the rand gold price, a scenario which is in line with current [illegible]ket perception.

Profitability has been measured by the Net Present Value (N.P.V.), Internal Rate of Return (I.R.R.) and Payback Period. While the latter is a measure of liquidity or exposure to risk rather than profitability, it does provide in conditions of scarce capital a yardstick by which investments can be compared.

A two year time frame has been used to illustrate these measures. The results of the cashflow which is set out in Annexure 6 are given below.

Net Present Value (6%) = R1 043 140
Internal Rate of Return = 251%
Payback Period = 10 months

The above measures demonstrate very clearly the financial benefits to be gained by the implementation of geostatistics in a selective mining environment. In the case of this mine these results would have contributed to offsetting the current working loss.

5 CONCLUSION

The objective of this project report was to demonstrate how the implementation of technical computing could benefit the South African gold mining industry. This was shown by a case study involving the application of geostatistics to a portion of the White Reef at West Rand Consolidated Mines Limited. The major conclusion to be drawn from this investigation is that geostatistical estimation is superior to the method currently employed on the mine.

The areas of improvement highlighted in this study were:

- improved ore reserve estimation
- improved block factors
- improved selective mining options
- improved short term planning
- improved sample spacing

In financial terms the benefit of superior estimation in a selective mining environment was to improve working profit by 2 per cent. In the context of the total mining operation on the White Reef this could amount to R1.0 million per annum.

The computations necessary to derive these benefits were processed on an IBM PC AT microcomputer and required the minimum of computer expertise. Likewise, the geostatistical techniques employed were relatively straightforward and should be easily applied by those with a sound statistical background.

Geostatistics was used in this case study to demonstrate the benefits of technical computing. However, it should not be viewed in isolation but rather in conjunction with the many other technical applications available to the mining industry. These cover the broad spectrum of a mining operation ranging from the peripheral areas of exploration through to smelting and refining. The potential benefit in both technical and financial terms of the combined applications available is considerable.

Productivity is also improved by technical computing. It is obvious that the ease of access to and availability of up-to-date information, computational power and speed, increased interpretation capabilities and better utilisation of resources, must yield a direct benefit in terms of productivity through savings of time and manpower.

In summary, failure to take advantage of the benefits offered by technical computing will, in today's highly volatile and competitive markets, result in a loss of profits and operational efficiency.

APPENDIX 1

UNDERGROUND SAMPLING AND ESTIMATION PROCEDURES

1 Sampling Procedure

Samples are taken at 6 metres centres in stopes and at 3 metre centres in raises and other development ends. Stope and development faces are in general sampled once per month. However, where high sample variability is encounterd in stopes, sampling frequency is increased to two and in some cases three times per month. This occurred in the area under investigation.

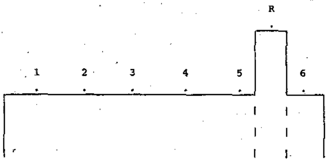
2 Calculation of Average Panel Value

The following steps are carried out in panel estimation.

- Reduction of all values in excess of 5225 cmg/t to this value. This is done to avoid excessive overvaluation of panels.
- Once the highest values have been reduced to the maximum permissible value a second method of reduction is used. A regression factor is applied to the g/t stope width values. The effect of regression is to reduce the calculated value to a more representative average. The lowest values have a small regression whilst the high but erratic values have a much larger regression which can reduce the true value by over half. It is likely that this factor originated from the fact that the majority of samples have a low value.
- Once the reduction of the values has been completed a weighted arithmetic average is calculated, using the

sections in the panel (plus the one from the raise heading when present) to provide a mean cmg/t value for the panel.

This is illustrated below:



$$\text{Average Value} = \frac{V1*SW1+V2*SW2+V3*SW3+V4*SW4+V5*SW5+VR*SWR+V6*SW6}{SW1+SW2+SW3+SW4+SW5+SWR+SW6}$$

$$\text{Average Width} = \frac{SW1+SW2+SW3+SW4+SW5+SWR+SW6}{7}$$

where:-

V = cmg/t value of section

SW = stoping width of section

This method for calculating average panel values does not cater for samples taken at irregular intervals. Furthermore, despite sampling having taken place in primary and secondary raises prior to ledging and stoping no use is made of these values when evaluating stopes.

APPENDIX 2

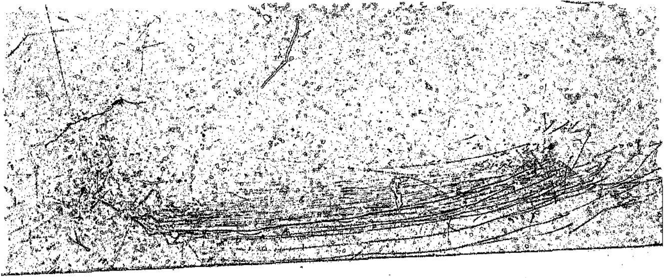
CASHFLOW FOR NON-SELECTIVE MINING

	QUARTER ENDED				
	DECEMBER 1986	MARCH 1987	JUNE 1987	SEPTEMBER 1987	TOTAL
TECHNICAL					
Square metres mined	1042.2	4876.9	3241.2	3928.3	13088.6
Tons milled	3374.9	11492.8	7287.9	12465.0	34620.6
Yield (g/t)	5.02	4.99	3.91	6.13	5.18
Gold recovered (kgs)	16.94	57.35	28.50	76.41	179.20
FINANCIAL					
Gold price received (R/kg)	28829	28179	29946	30506	29514
Revenue ('000)	488	1616	853	2331	5288
Cost per metre squared (R)	212	189	216	221	207
Working cost ('000)	221	922	700	868	2711
Working profit ('000)	267	694	153	1463	2577

APPENDIX 3

CASHFLOW FOR SELECTIVE MINING BASED ON MINE ESTIMATION

	QUARTER ENDED				TOTAL
	DECEMBER 1986	MARCH 1987	JUNE 1987	SEPTEMBER 1987	
TECHNICAL					
Square metres mined	1042.2	4876.9	2737.2	3928.3	12584.6
Tons milled	3374.9	11492.8	6295.8	12465.0	33628.5
Yield (g/t)	5.02	4.99	4.18	6.13	5.26
Gold recovered (kgs)	16.94	57.35	26.32	76.41	177.02
FINANCIAL					
Gold price received (R/kg)	28829	28179	29946	30506	29508
Revenue ('000)	488	1616	788	2331	5223
Cost per metre squared (R)	212	189	216	221	207
Working cost ('000)	221	922	591	868	2602
Working profit ('000)	267	694	197	1463	2621



APPENDIX 4

CASHFLOW FOR SELECTIVE MINING BASED ON KRIGING ESTIMATION

	QUARTER ENDED				
	DECEMBER 1986	MARCH 1987	JUNE 1987	SEPTEMBER 1987	TOTAL
TECHNICAL					
Square metres mined	1042.2	4876.9	2117.2	3928.3	11964.6
Tons milled	3374.9	11492.8	4928.0	12465.0	32260.7
Yield (g/t)	5.02	4.99	4.77	6.13	5.40
Gold recovered (kgs)	16.94	57.35	23.51	76.41	174.21
FINANCIAL					
Gold price received (R/kg)	28829	28179	29946	30506	29501
Revenue ('000)	488	1616	704	2331	5139
Cost per metre squared (R)	212	189	216	221	206
Working cost ('000)	221	922	457	868	2468
Working profit ('000)	267	694	247	1463	2671

APPENDIX 5

TOTAL WORKSTATION COSTS

HARDWARE	COST (R)
1 X Micro (386 Processor, 300Mb)	32 000
3 x Slave terminals	7 500
1 x Digitiser tablet (A0)	16 000
1 x Plotter (A0)	38 000
1 x Printer (A4 Laser printer)	9 000
Peripherals (Tape streamer, RAM)	11 000
GST on above	13 620
TOTAL HARDWARE	127 120
SOFTWARE (including 12% GST)	56 000
TRAINING (GST excluded)	15 000
<u>TOTAL COST</u>	<u>198 120</u>

APPENDIX 6

WORKSTATION CASHFLOW

	YEAR								
	1				2				
	QUARTER				QUARTER				
	1	2	3	4	1	2	3	4	TOTAL
Investment (R)									200 000
Working Profit (R)	0	55 555	138 888	222 222	250 000	250 000	250 000	250 000	1 416 665
Cashflow (R)	-200 000	55 555	138 888	222 222	250 000	250 000	250 000	250 000	1 216 665

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