COMPUTER–AIDED MINE DESIGN AND PLANNING AT PANASQUEIRA, PORTUGAL

by

Daniel MIREKU-GYIMAH
MSc, CEng, MIMM

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Computer Aided Design Research Group
Department of Mineral Resources Engineering
Imperial College of Science, Technology and Medicine
London SW7 2BP

ABSTRACT

At the Panasqueira mine in Portugal, the principal ore mineral (wolframite) occurs erratically as individual crystals or clusters within a series of thin, near-horizontal, hydrothermal quartz veins, making conventional channel or chip sampling methods impractical. Measurements of areas of the wolframite crystals exposed in the side walls of mining faces or development ends are used to estimate ore grade and reserves. The quartz veins must hence be exposed before sampling and mining can be carried out effectively.

This complex vein system is emplaced in a schistose country rock which shows no apparent variations by which zones of common but distinct characteristics can be delineated. It has not been possible, therefore, to date, to geologically connect, correlate or interpolate individual vein intersections observed in diamond drill cores. The complexity of the vein system and the concomitant difficulty of establishing ore grade and reserves have led, in the past, to much development in unprofitable ground.

This research analyses the widths of the quartz veins, observed in diamond drill cores from a representative part of the mine, with statistical and Markovian probability methods, and uses a geostatistical technique to estimate the quartz content on a block by block basis. A step-by-step approach is suggested to identify areas which contain maximum quartz and can be mined conveniently with a mechanised room and pillar mining method currently employed at the mine. A development schedule is worked out to expose the quartz for subsequent sampling and mining.

Six possible mineable horizons are identified within the study area. The elevations of these horizons are found to be similar to those at the mine, but plan areas occupied by selected blocks in each of the horizons are generally larger, and estimates of quartz tonnages obtainable from the horizons are found to be significantly higher, than those at the mine.

A model for the initial design of a room and pillar layout is developed on the basis of the Tributary Area and the Plate theories. This model is used to examine the current layout design at the mine which is found to be unfavourable; a modification in the layout is suggested to reduce the risk of pillar failure.

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CHAPTER 1

GENERAL INTRODUCTION

1.1 Background

The general approach used in designing and planning a mine is to establish the location, quantity and economic value of mineral or minerals in a given deposit from grade values obtained by assaying rock samples from boreholes. The location, quantity and economic value of the mineral are then used as a basis to design an appropriate mining system, to select suitable equipment and to work out a production schedule. This approach is particularly suitable for massive orebodies, especially where the location of the orebody is such that surface mining technology is applicable.

Vein type deposits, that would normally be exploited with underground mining techniques, present two problems to the mine designer and planner. Firstly, their complex nature makes it difficult to determine the spatial and geometric arrangement, as well as the continuity or extent of the individual veins, using information obtained from diamond drill holes. The vein pattern may not necessarily fall into the standard classifications found in literature (McKinstry (1948)) and may be either unique or a combination of various patterns which cannot be easily determined, even with good geological information. Secondly, their economic value can only be properly established after the veins have been exposed through extensive and costly development. The reason for this is that samples from boreholes are not adequately representative, even when the drilling programme is extensive.

The location and extent of veins are usually established by extrapolating the vein intersections observed in diamond drill cores, making use of geological information of the country rocks. Once the veins are exposed there are standard estimation procedures to establish the grade value for subsequent design and planning of mining operations. These procedures have been outlined, for example, by Clark (1978; 1984), David (1977), Readdy *et al.* (1982) and Tapp (1982). Channel or chip samples are taken from the veins exposed in the drives or cross-cuts which border the piece of ground to be valued. The assays of these samples are then analysed with classical or geostatistical methods to establish the grade value of the ground. The process is clearly illustrated by Koch and Link (1983) in their work at the Frisco mine:

> " The assays investigated in detail were taken from the brown vein exposed in a 756 m long drift on level 9 and a 930 m long drift on level 10 ... "

Any effective design and planning strategy will therefore depend firstly, on the correct prediction of the location of the initial development, using drill hole information, to expose the veins for subsequent sampling and secondly, on the ability to establish the ore grade. In these aspects, there are unique problems at the Panasqueira mine in Portugal which demand an alternative approach.

1.1.1 Mine Planning Problems at Panasqueira

The Panasqueira mine in Portugal, the major producer of tungsten in Europe, is a classic example of a vein type deposit where the problems discussed in the previous section are aggravated by two additional factors:

(i) The principal ore mineral (wolframite) occurs erratically as individual crystals or clusters within a series of near horizontal, hydrothermal quartz veins in a schistose country rock. This has made conventional channel or chip sampling methods impractical. Measurements of areas of the wolframite crystals exposed in the side walls of mining faces or development ends have been utilised to estimate ore reserves, using an empirical factor known as the Mineral Evaluation Factor (MEF). The method of estimating the grade value and the MEF are explained in Chapter 2 and in Appendix 2.

(ii) To date, it has not been possible to use geological information to connect, interpolate or correlate the individual vein intersections observed in diamond drill cores. The meticulous, detailed recordings of the inter-burden schist rock show no apparent variations by which zones of common characteristics can be delineated or the extent of individual veins can be established. Since, in addition to this problem, no core sampling or assays are present the only meaningful information obtainable from the diamond drill logs, recorded for over half a century, are the vein widths and the x-, y-, z-co-ordinates at which they were intersected. A typical section of the mine showing this information is discussed in Section 3.3.1. The fundamental problem of deciding

> which vein intersection from one borehole connects or correlates with that of an adjacent borehole, and therefore which horizons are worthy of development for subsequent sampling, still remains a matter of intuition and experience. In practice, a single promising vein is developed and followed, its direction being inferred from past experience and information at the face.

The problems associated with the multi–veins of Panasqueira were best expressed by Allen *et al.* (1947):

> " This irregular habit of wolfram mineralisation makes it impossible in practice to determine the value of any vein by normal sampling methods. It is difficult to establish ore reserves in any accepted meaning of the word. When some of the stopes were opened, it was found that many had to be abandoned owing to apparent weakness in vein width or mineralisation, yet a few metres ahead good ore was again encountered. "

Due to the increasing cost of labour and timber, the Panasqueira mine has recently replaced its long-wall mining method with a mechanised room and pillar mining system. To support 5 m wide rooms, 15 m by 15 m pillars were initially left and then eventually reduced to 5 m by 5m square pillars. Over the years, the pillar size has been reduced to 3 m by 3 m to increase the ore extraction ratio. To facilitate mechanisation, the stoping room height has been increased from 1.8 m to 2 m. Sometimes, either to be able to mine more than one vein or to further facilitate mechanisation, the stoping height is increased to as much as 2.5 m. Two factors militate against this action: first, dilution is increased — the average width of the veins mined is only about 0.30 m; and second, the factor of safety of the pillars can be reduced to a point where the pillars will fail. Although no *in situ* tests have been undertaken to permit the use of finite element or boundary element methods to study the state of total stress and induced displacements around rooms, it was considered worthwhile as part of this project to examine the present room and pillar mining layout based on the available geomechanical properties of the rocks.

1.2 Research Aims

The complexity of the Panasqueira vein system and the difficulty of estimating the grade and local ore reserves have led, in the past, to much development in unprofitable ground.

The primary aim of this research is to develop a computational methodology to identify areas of optimal concentration of quartz veins for subsequent planning of development work that will expose the veins for effective sampling and mining.

In the latter part of the research, the current room and pillar layout design is examined to verify the stability of the pillars and to find the implications of increasing the stoping room dimensions, especially the height.

The research is based on the diamond drill logs supplied by the Geology Department at Panasqueira via Charter Consolidated Plc. in London, as well as the available geomechanical properties of the rocks. The logs consist essentially of vein widths and the x-, y-, z-co-ordinates at which they were intersected.

The aims which were initially set out and finalised during the author's visit to the mine are:

(i) to establish a measure of predictability of the vein occurrences;

(ii) to evolve a decision process by which mineable horizons can be established;

(iii) to develop planning tools that will be practically useful at Panasqueira;

(iv) to examine and so justify, or otherwise, the current room and pillar layout design; and

(v) to computerise any methods that will be employed.

The ultimate aim is to find an appropriate technique by which the mine can achieve optimal extraction of the complex, multiple quartz veins.

1.3 Previous Work

Williams (1985) analysed vein widths within Level 2 of the mine and arrived at global estimates of "vein density" which he defined as the quantity of veins per 60 m height of ground within the mine limits. His work demonstrated the applicability of geostatistical methods to the Panasqueira veins *but the global estimates failed to indicate the location of veins or the ground that could be mined.*

In the second part of his work, Williams analysed the 'mineral measurements' of a mined out area and showed that there is no correlation between the vein widths and the amount of wolframite contained in them.

However, other investigations carried out by Hebblethwaite, who was the chief geologist at Panasqueira from 1981 to 1984, and the mining staff indicate that the ratio of wolfram (WO_3) to quartz (SiO_2), in the veins, is constant within 90% confidence limits (Hebblethwaite (1981)). Consequently, the more quartz mined, the more wolfram obtainable. This observation, which was further supported by the current technical staff at Panasqueira during the author's visit to the mine in 1987, is in fact one of the bases upon which this research was outlined.

1.4 Thesis Outline

This thesis is divided into seven main chapters. The subject matter of each chapter is as follows:

(i) The first chapter serves as the introduction which highlights the technical problems at the Panasqueira mine and defines the aims of this research.

(ii) In the second chapter, relevant information about the Panasqueira mine is presented.

(iii) In the third chapter, an attempt is made to qualify the subjective decisions employed by the mine geologists to connect vein intersections and select horizons for development, so as to be able to computerise the process. The third chapter also analyses the vein widths of a representative section of the mine using classical statistical methods and Markovian probability concepts. A derived variable termed as *vein concentration*, in this thesis, and defined as the quartz abundance or vein width accumulation per 2 m height of ground, is also statistically analysed.

(iv) The fourth chapter uses semi-variography to study the variability of the vein width and the vein concentration within the study area of the mine. Vein concentration of the pre-ordained development blocks at the mine is then estimated using a linear kriging technique. This results in the creation of a vein concentration inventory and the establishment of a vein concentration-tonnage relation.

(v) Chapter five features the selection of possible mineable blocks from the vein-concentration inventory. It also presents a development planning strategy to expose the veins for sampling and mining.

(vi) In chapter six, the current room and pillar mining system is analysed using the available geomechanical properties of the rock. Based on the Tributary Area and Plate theories, a computer model for designing a room and pillar layout is developed. This model is then used to investigate the mining system at Panasqueira, examining the sensitivity of pillar strength factor of safety and ore extraction ratio to variable pillar and room dimensions.

(vii) Chapter seven summarises the work carried out, the observations made and the conclusions drawn during the course of this project. Recommendations for further research work at Panasqueira are also presented.

CHAPTER 2

THE PANASQUEIRA MINE

2.1 Introduction

Panasqueira is the second largest mine in Portugal and the major wolfram producer in Europe. It employs a room and pillar mining system. The average monthly production is 42000 tonnes run-of-mine, from which 170 tonnes of wolframite concentrate (75.5% WO_3) are processed. In addition, 13 tonnes of tin concentrate (73% Sn) and 80 tonnes of copper concentrate containing some silver (25% Cu with 650gm/t Ag) are produced each month. The tin and copper concentrates are sold in Portugal, and the wolframite concentrate is exported mainly to England, Holland, France, Japan and the USA.

Panasquiera is situated in the Beira Beixa province of central Portugal, on the southern flank of the Serra da Estrela mountains which cross the country in a NE–SW belt (0.3°E; 40.02°N). The mine is about 34 km west of Fundão, which, by road, is about 300 km north-east of Lisbon (see Figure 2.1). The mine concession, shown in Figure 2.2, covers an area of approximately 34 km^2 extending over an 8 km long NW–SE strip of land, from the Cebola river in the north to the Zezere river in the south. The elevation of this fairly rugged concession ranges from 1083 m at Chiqueira to about 300 m in the Zezere valley. Mine operations are now centred at the town of Barroca Grande with only minor access and facilities remaining at the old village of Panasqueira. The historical developments in the organisation of the mine are recounted in Appendix 1.

2.2 Geology

Panasqueira is located in the Beiras Belt of metamorphic rocks. The belt is mostly composed of an undifferentiated Precambrian–Palaeozoic phyllistic-schist complex intruded by a great number of granitic and basic-doleritic rocks (Gaines and Thadeu (1971)). The region is at the southern edge of the Hercynian Granitic Complex of Iberia. The granites do not crop out at Panasqueira. The surface rocks are the tightly folded pellitic schists of the more ancient rocks. Figure 2.2 shows the surface geology of the mine's concession area.

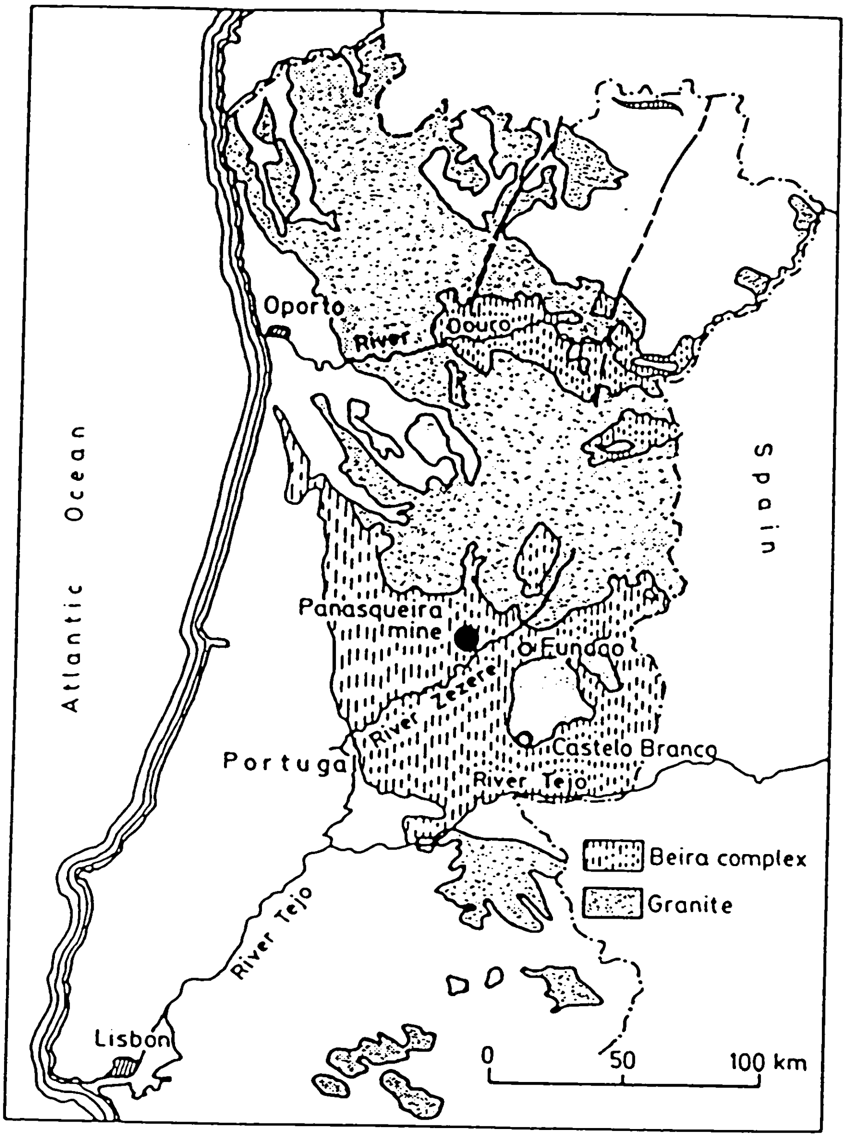


Figure 2.1. Location of the Panasqueira mine (after Smith, 1977).

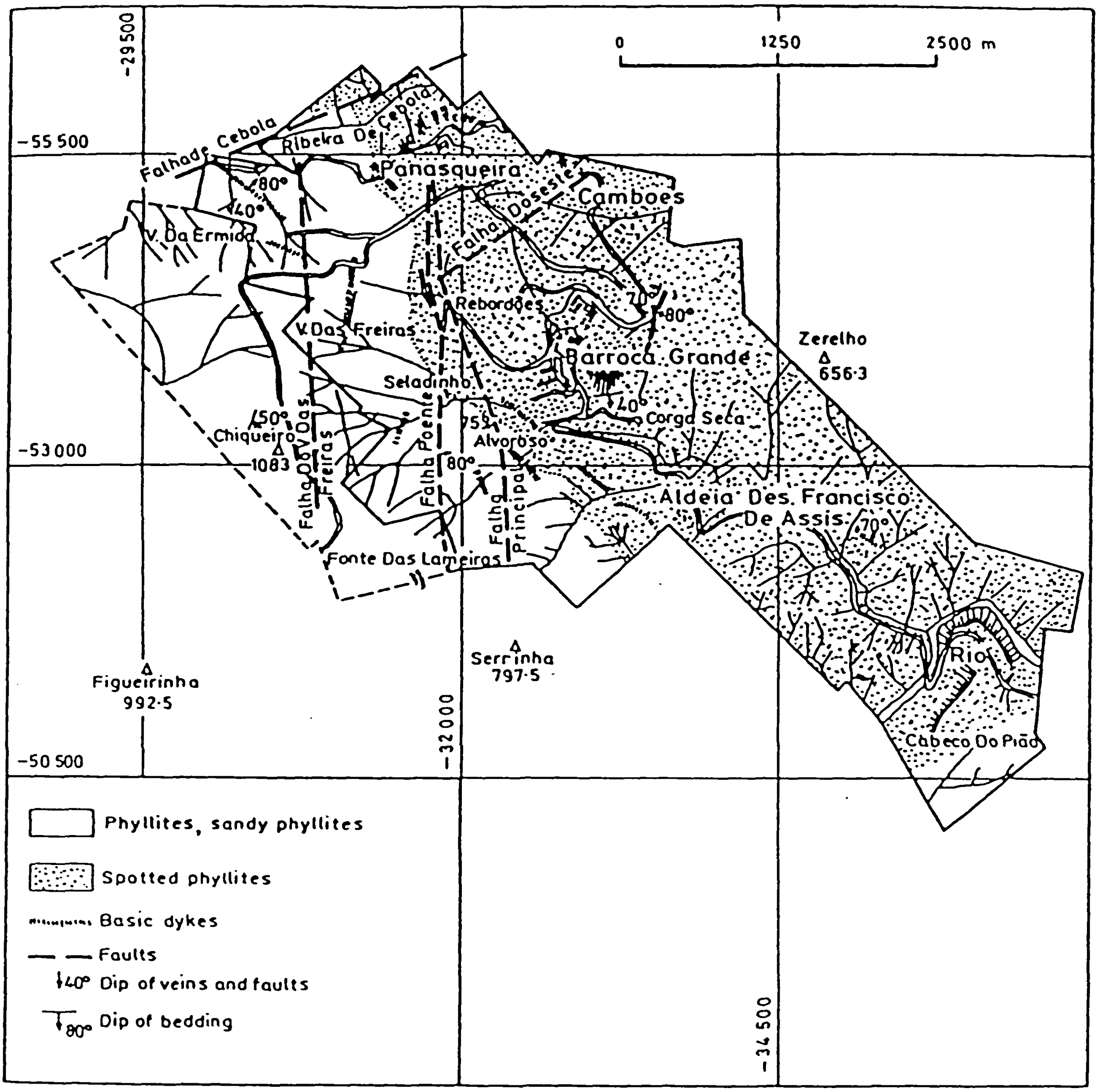


Figure 2.2. Limits of the mining concession, and surface geology (after Smith, 1977).

The region has been subjected to three major tectonic events that have contributed to the formation of the deposit at Panasqueira (McNeil (1982)): Caledonian, Hercynian and Alpine orogenies. Figure 2.3 shows a summary of the geological evolution of the Panasqueira orebody.

The orebody is a highly complex series of near horizontal, hydrothermal quartz veins associated with pre-existing joint planes caused by local stress conditions and subsequent Hercynian tectonic activity (personal communication with Hebblethwaite, 1987). The country rock is schist. Within the individual veins, mineralisation is irregular. According to Kelly (1974), Kelly and Rye (1979), Schemerdon (1981) and Hebblethwaite (1983), these quartz veins bear a mineral assemblage which includes wolframite, cassiterite and chalcopyrite in currently economic quantities. The associated sulphides are essentially arsenopyrite and pyrite, but sphalerite and pyrrhotite are also present. Other minerals include siderite, fluorite, mica, triplite and marcasite.

The veins are believed to have been emplaced by vertical dilation along openings created on the near horizontal, pre-existing joint sets. There are varying opinions on the origin of the flat lying joint sets: the joints could have been formed as "sub-horizontal fractures resulting from vertical pressure release caused by slight sagging of the underlying mass when it contracted during consolidation" (Schemerdon (1981)); or "by tectonic unloading" (Hebblethwaite (1982)).

Normally, the veins have an average inclination of about 7° to 15° and a dip to the SE and NW. The average width is reported in most of the literature as being 30–40 cm (Thadeu (1980) and McNeil (1982)). Williams (1985) pointed out that this reported average vein width may refer to veins that were mined in older parts of the mine.

The deposit is cut by a series of steeply dipping faults. These faults are thought to have been initiated by strike-slip movements during the Hercynian episode, and to have been reactivated during the Alpine orogeny. The wolfram mineralisation does not extend into these faults, which are sometimes filled with carbonate cemented gouge (Thadeu (1980)). Polya (1987) suggests that these faults are the original mineralising fluid "conduits" from which the fluid passed into the horizontal fractures.

Geological Evolution of the Panasqueira Orebody

The sequence of major events in the evolution of the Panasqueira orebody may be summarised as follows:

60 my	**Cenozoic.** Erosion and weathering.
75 my	**Late Cretaceous.** Re-heating and uplift.
152 my	**Late Jurassic.** Rotation, re-heating and uplift.
200 my	**Late Triassic.** Alpine Orogeny. Faults reactivated with secondary mineralisation.
290 my	**Late Carboniferous - Early Permian.** Hercynian Orogeny comprising the following stages:- 1. Folding - compressional phase, giving rise to isoclinal folds with strong foliation. 2. Regional Metamorphism - greenschist facies. 3. "Seixo Bravo" or barren quartz formation. 4. Faulting - minor N-ENE. 5. Jointing - 3 sets formed by tectonic unloading. 6. Dyke intrusion - dolerite and aplite. 7. Granite intrusion - porphyry 2-mica (S-type) granite batholith with one or more cupolae. 8. Contact metamorphism - cordierite-hornfels facies. 9. Joint dilation - tectonic unloading. 10. Vein mineralisation - comprising:- a. Oxide-silicate stage - (quartz, wolframite, cassiterite) b. Main sulphide stage - (pyrite, arsenopyrite, pyrrhotite, sphalerite, chalcopyrite) c. Pyrrhotite alteration stage - (marcasite, siderite) d. Late carbonate stage - (dolomite, calcite) 11. Hydrothermal alteration - tourmalinisation.
500 - 410 my	**Early Ordovian-Silurian.** Caledonian Orogeny. Gentle warping of the sediments in a series of NNE - ENE trending fold axes.
600 - 500 my	**Late Precambriam-Cambrian (pre-Ordovian).** Sedimentary accumulation. Argillaceous-arsenaceous facies.

Figure 2.3. Geological evolution of the Panasqueira orebody (after Hebblethwaite, 1983).

2.3 Mining System & Mine Planning

2.3.1 Mine Layout

The mine is developed from an orthogonal system of haulages (drives and cross-cuts) laid on a 100 m grid. The cross-cuts are orientated at about 30° east of the national grid north. Main levels are connected by 60 m vertical raises at every 50 m along the drives to serve as ore passes and service-ways for compressed air and electricity, and to act as part of the ventilation network. Figure 2.4 shows a three-dimensional sketch of the main haulages and raises. There are also spiral ramps to provide access for equipment to individual veins between the main levels. From the raises, drives known as inclines, are developed to open the veins for subsequent "mineral measurements", and to provide access when stoping commences.

Access to the mine was, until 1982, by means of an adit at Level 0 (at an elevation of 680 m). An inclined shaft was started in 1977, commissioned in 1982 and now serves as the main access to the mine. The shaft is 1203 m long and has a vertical drop of 200 m. It is equipped with a 0.9 m wide conveyor belt over its entire length. The conveyor system has a designed capacity of 250 t/hr. at a belt speed of 2.13 ms^{-1}. Ore and waste from Level 1 (between 620 and 680 m) and Level 2 (between 560 and 620 m) pass to a 250 t/hr. capacity crusher that is located 30 m beneath Level 2. The inclined shaft has three sections: 575 m, of cross-section 3 m by 4 m and inclined at 13°, accommodates the main conveyor and a track beside it for vehicles and materials; a short section of 46 m, on the surface, is enclosed as it crosses Reboedoes valley; a final section, 600 m long and inclined at 8°, terminates 43 m above Level 0. Access to the mine workings for trackless equipment is through the exposed section of the shaft in Rebordoes valley. Storage bins are situated at the portal end of the shaft for both ore and waste before a final transfer conveyor transports ore to the crushing and screening plant, and waste to waste dumps. The conveyor is also available for transport of men — transfer points are provided at each level. The system is capable of transporting the entire underground labour force of 500 men in 30 minutes.

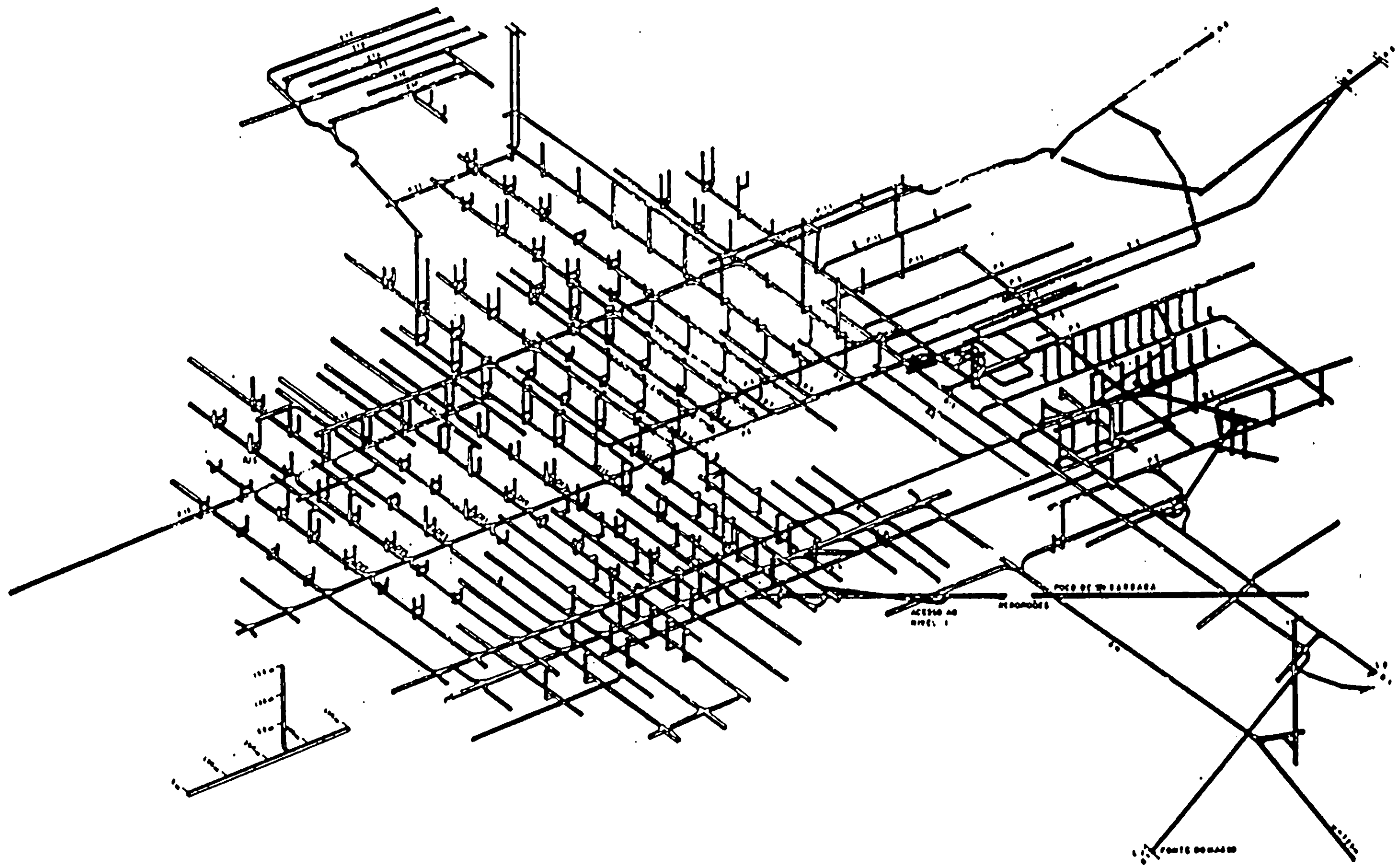


Figure 2.4. Three dimensional sketch of main haulages and raises
(Source: Minas da Panasqueira, 1987).

2.3.2 Mine Planning

The mine is currently planned on the basis of information obtained from surface and underground boreholes, most of which are drilled vertically. The underground boreholes are drilled from the pre-ordained, orthogonal network of drives and cross-cuts. They are now generally drilled to the entire depth of the level, 60 m, with Diamec 250 rigs. Holes up to 120 m are also being drilled to investigate the prospective Level 3. The drilling spacing was originally 100 m, but has been progressively reduced to 25 m.

The drill logs are plotted by the Geology Department onto vertical sections along the drives and cross-cuts. The correlation between vein intersections shown in the cross-sections is still a matter of experience and subjective judgements based on the known behaviour of veins in that part of the orebody, particularly the likely dip and continuity.

Decisions as to which veins to develop are made by the mining staff with the assistance of the geology personnel. There are apparently no hard and fast rules as to what constitutes a payable proposition at this stage (since there are no assays). Veins with widths of 0.18 m or more are included in the initial "geological reserves", but 20–30 cm is often quoted as the minimum vein width mined (Smith (1979) and McNeil (1982)).

Once an area with a promising vein has been delineated, drives (called inclines) are developed on the selected vein and ideally connect all the raises on this vein. Very occasionally, the potential stope incorporates more than one vein, either due to the original one thinning off to be replaced with another, or through the stope stepping up or down to an adjacent vein.

When the vein is exposed it can be sampled using the Mineral Measurement Method. An average grade in kg/m^2 of wolframite concentrate is calculated by the Survey Department and mineable blocks are established. The block can now be entered in the "virgin" reserve inventory. Generally, 13 kg/m^2 is regarded as the cut-off grade below which a block is considered to be unpayable. The development ground may be trammed as ore and can make a significant contribution to the mine production.

Mining is planned to progress in two main phases:

> *Phase 1.* 5 m wide rooms are mined within a block which will normally lie on an orthogonal grid parallel to the mine grid. The stoping height

is 2 m. The rooms are separated by 11 m by 11 m pillars (Mendes *et al.* (1987)). The pillars are then sampled using the mineral measurement method and assigned grade values in kg/m^2. The aggregated average grade of all the pillars in a stoping area is also computed and used in the planning process (Williams (1985)). If the grade values are satisfactory and tonnages are acceptable, mining proceeds to the second phase.

Phase 2. The 11 m by 11 m pillars are split with 5 m wide rooms so as to obtain, from each pillar, 4 smaller pillars of 3 m by 3 m. The overall ore recovery is 86%. There is no planning programme to recover the remaining pillars, however where the veins are very rich some pillars are recovered.

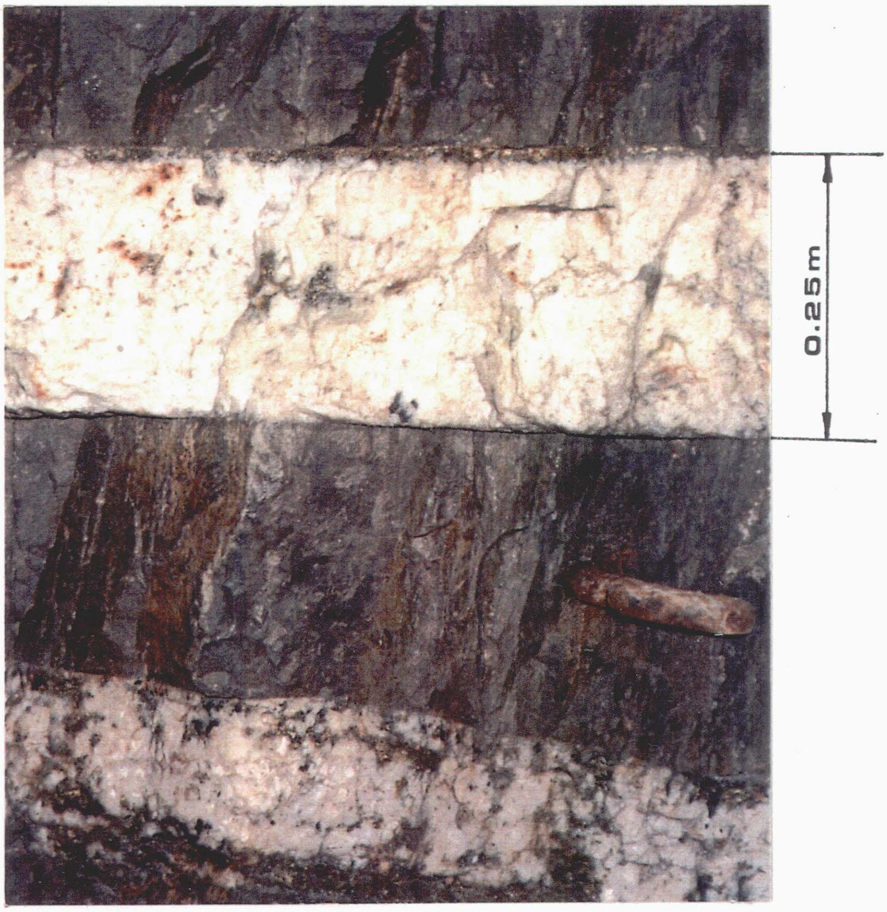
With this room and pillar mining system, Panasqueira targets to produce 2000 t/day run-of-mine, working for 21 days per month throughout the year.

Short-term planning is based on the most recent mineral measurement results and tonnages obtained from active stopes and incline development, together with the actual concentrate production at the processing plant. The mineral measurement readings are taken approximately every two weeks in advancing stopes. An average grade and tonnage of the whole mined out area is then calculated. The monthly plan is contained in the *Planeamento do Desmonto*. This is a detailed table showing the original estimated grade of each stope when it was first opened, and its currently estimated grade from the additional measurements. The area of ground planned to be mined in each stope in the coming month is also indicated. In this way, the overall run-of-mine grade for the coming month is estimated and the allocation of men, machinery and materials can be made to achieve production targets.

2.3.3 Sampling & Mineral Measurement

At Panasqueira, the main ore mineral (wolframite) occurs erratically as individual crystals or clusters of crystals. Figures 2.5a and b are photographs of quartz veins exposed in a stope face on Level 2 showing black wolframite crystals, with schistose interburden rocks. The thickness of the vein in Figure 2.5b is about 20 cm. With such crystals chemical assay based on any conventional chip or channel sampling will produce many nil values and few high ones. The alternative method to sampling any vein *in situ* by channel or chip sampling (as can be carried out in vein deposits of less erratic mineralisation) is bulk sampling.

(a)



(b)

Figure 2.5. Photographs of exposed veins in stope faces.

This could be achieved by having a separate sampling plant underground. It is reported by Williams (1985) that Panasqueira has ruled this out on the basis that the high capital and operating costs of such a system are not justified, and that the system may not, in the end, provide better results than the current method in use i.e. the 'mineral measurement method'.

The mineral measurement method employed at Panasqueira was originally formulated by Mendes (1958) and has been explained by other authors e.g. Linzell (1966), Reis (1971) and Hebblethwaite (1981). In this method, areas of the wolframite crystals exposed in a stope face or the side walls of a development end are measured and converted into kilograms of concentrate per square metre using an empirical factor known as the Mineral Evaluation Factor. The description, given below, of the method is based on the author's observations during his visit to the mine in 1987:-

(i) divide the face into standard measurement lengths (usually 2 m);

(ii) measure the dimensions of each wolframite crystal exposed in the face;

(iii) calculate the total area of the wolframite crystals (in cm^2);

(iv) divide the total area by the overall length of the face in metres;

(v) divide the result by the Mineral Evaluation Factor (MEF) of which the current value is 1.8; and, thus,

(vi) obtain the face grade in kilograms of concentrate per metre square.

The method can be expressed by the relation:

$$V = \frac{S}{\text{MEF} \times b}$$

where

$V =$ face grade in $[kg/m^2]$;

$S =$ sum of areas of wolframite at the face in $[cm^2]$;

$b =$ overall length of face in $[m]$; and

$\text{MEF} = 1.8$

The outline of the theory behind this formula is presented in Appendix 2. Despite the apparent simplicity of this method and its dependence on the empirical factor (MEF), it has proven to be so reliable that it is unlikely to be replaced in the near future (personal communications with Panasqueira mining staff, 1987).

The subject of concern at the Panasqueira mine is how to interpolate the vein intersections and, hence, how to delineate the horizons which will contain more quartz veins. Unless the veins are exposed the mineral measurement cannot be carried out. It is this problem which the research project addresses primarily.

2.3.4 Mining Methods

Before the seventies, long-wall stoping was the main mining method employed at the Panasqueira mine. Allen *et al.* (1947), and Reis (1971) have given good descriptions of this method and variations which were employed at the mine. Faces were generally advanced parallel to the 'incline' development between raises, using two scraper units for mucking. Alternatively, a fan-shaped pattern was employed, centred around an ore pass so that a single scraper unit could be used. Timber packs provided support for the back and a system of pack-walls, parallel to the face, served to minimise loss of fines during blasting. Hand-held drills on air-leg supports were utilised, and the stoping height was maintained at 1.4 m, although this increased gradually to 1.8 m during the seventies.

In the late seventies, due to increasing cost of labour and timber, the traditional long-wall stoping was replaced by a mechanised room and pillar mining system. Low profile trackless tramming equipment and electric-hydraulic jumbos were introduced. There are two Secoma single boom, four Secoma twin boom and one Tamroc single boom jumbos. The LHDs include Eimco 911 (0.76 m^3), Wagner EHST1 (0.76 m^3) and Wagner EHST2 (1.52 m^3).

The room and pillar mining system started on a test basis with 5 m wide rooms and large pillars measuring 15 m by 15 m. The stoping height was 1.8 m. The pillar size was later reduced to 5 m by 5 m, corresponding to an extraction ratio of 75%. Since 1983, the pillar size has been further reduced to 3 m by 3 m in order to increase the ore extraction ratio. To facilitate equipment movement, the stoping height was increased to 2 m. To further enhance mechanisation or to be able to mine more than one vein, the stoping height is often increased to as much 2.5 m. Two factors militate against this action:-

(i) Dilution increases — the average thickness of the quartz veins actually mined is about 30 cm which constitutes only about 15 % of the run-of-mine when the stoping height is 2 m.

(ii) The factor of safety of the pillars is reduced and pillar stability may be sacrificed — at the 2 m stoping height, some pillar failures have been observed (personal observation in the stopes) when the pillars measure 3 m by 3 m. This issue will be discussed in Chapter 6.

2.4 Mineral Processing

Ore from the mine is processed in two stages: first, the ore is crushed at Barroca Grande where the coarse fraction is upgraded by heavy media separation using hydrocyclones, and the fine (-1 mm) fraction by tabling; second, the bulk concentrate containing 4% WO_3, 1.5% Cu and 0.3% Sn is transported by aerial ropeway to the processing plant, in Rio (see Figure 2.2), where it is further upgraded by gravity, flotation and magnetic separation. It is eventually split into three concentrates: wolfram (75% WO_3), tin (73% Sn) and copper (25% Cu with 650 gm/t Ag).

CHAPTER 3

VEIN SYSTEM ANALYSIS

3.1 Introduction

The following points, noted in Chapter 2, are the characteristics of the deposit which make geological and geometrical interpretation, as well as grade estimation, impossible, using borehole information:-

(i) The vein system is very complex and is best described as a lattice-work of interconnecting and bifurcating veins comprising predominantly hydrothermal quartz that occupy selectively dilated, pre-existing tectonically induced sub-horizontal joint fractures.

(ii) The detailed lithological recordings show no apparent variations, in the predominantly schistose country rocks, by which zones of common but distinguishing characteristics can be delineated.

(iii) Because of the peculiarly erratic nature of mineralisation, assays are absent.

However, from previous works (Hebblethwaite (1981) and Williams (1985)), and from personal conversations with the current mine geologists and technical staff, there is evidence which indicates that the vein width is a useful variable that can be analysed with simple statistical methods in order to predict the likely volumes of veinage. The analysis which follows is based therefore on the vein widths.

The objectives of this chapter are:

(i) to examine the mine's interpolation procedures with a view to computerising it;

(ii) to statistically analyse the vein width in order to know the nature of its distribution and to establish the probability of the vein occurrences;

(iii) to find out whether the *vein concentration** can be analysed with simple statistical methods and hence examine the nature of its distribution; and

(iv) to derive the mean vertical separation between successive veins.

3.2 The Mine Database

The information on Panasqueira veinage, obtained from detailed computer-based drill logs, was provided by the Geology Department of the mine via Charter Consolidated Plc. in London. This information was contained in three main files: BORHO, LITHO and VEINAGE recorded in fixed format on labelled 9-track magnetic tapes at a tape density of 6250 BPI. These tapes were loaded and read onto a permanent-user file base with the READ9T facility available at the Imperial College Computer Centre. Three respective short programs —LISTB**, LISTL and LISTV— were used to list the files in more readable formats for subsequent study. A description of the information contained in the files together with brief explanations are given in Appendix 3.

3.3 Creation of Working Database

The data contained in the three files described above (called the Mine Data Base in this report) include information about veins in old areas of the mine, dating as far back as 1940 and well beyond the limits of current mining operations. These very early data have some inconsistencies which are likely to introduce errors in the analysis. In addition, the files occupy 5000 computer sectors of the Cyber 855 which is equivalent to about 3.2 megabytes of memory. Moving the whole data through even simple programs can be costly. It was considered prudent therefore to extract only the most relevant information from the files, and to

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* *Vein concentration* in this thesis means abundance of quartz or aggregate vein width per 2 m height of ground; it is measured in metres. It is this attribute of the deposit that will finally be used to identify the potential ground to be developed in order to expose the maximum quantity of quartz. The vein concentration is explained in Section 3.3.1.

** All computer programs mentioned in this thesis were written in FORTRAN77. Because their combined volume is enormous the programs are not listed, but they are deposited with the Mineral Resources Engineering Department, Imperial College, for reference.

concentrate on an area which covers the ground of current mining operations. This area is enclosed within the eastern co-ordinates: 30800.00 to 31700.00 m (i.e. from drive D15 to D25), northern co-ordinates: 53200.00 to 54000.00 m (i.e. from panel P01W to P06W) and elevation: 620 to 560 m — these elevations define Level 2 of the mine. The reasons for selecting this area are twofold:-

(i) The area has a concentration of boreholes and therefore is more likely to reveal any characteristics of the quartz veins.

(ii) Mining activities are concentrated in this area and so the results of the work carried out in this project can be cross-checked.

All the three data files contain 'key' numbers which identify a given borehole. By using this key number as a flag it is possible to extract all the necessary information belonging to that borehole from the files. The three main variables extracted are:

(i) the vein width,

(ii) the vein concentration, and

(iii) the vertical separation between successive vein intersections.

A simplified flowchart illustrating the procedure of creating the Working Database is shown in Figure 3.1. To extract the vein width, a program SORTVLX was developed to sort out all the veins intersected within a given level, and enclosed in the geological region of the study area, i.e. within the eastern co-ordinates: 30300.00 m to 32000.00 m and northern co-ordinates: 52800.00 m to 54900.00 m.

To extract information about the veins in Level 2, the program reads all the three files, picks the key number and co-ordinates of a borehole or raise within Level 2 (from 620 m to 560 m) from the BORHO file and selects the matching recordings from the VEINAGE and LITHO files. The program also computes the elevations of the mid-points of the vein widths. The output is kept on a permanent direct access file. VEINS2L, for example, is a file containing the co-ordinates of the veins intersected in Level 2, their widths, the elevations of their mid-points and any descriptions as contained in the original files. The vein concentration is the aggregate vein width per 2 m height of ground, and is calculated by program SORTVLX and kept on a separate file, VCON2L.

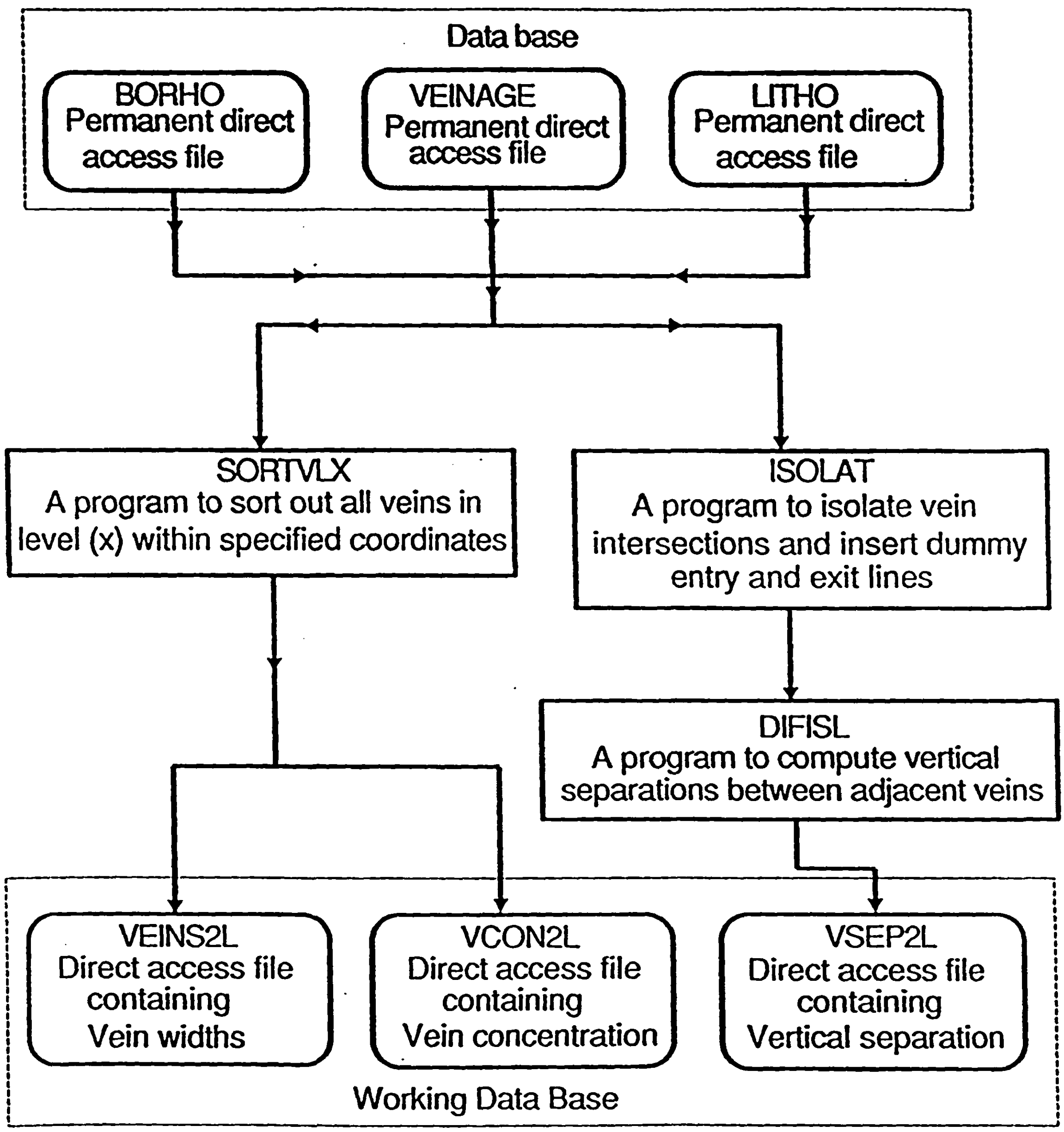

Figure 3.1. Schematic diagram to explain the creation of the Working Data Base.

In order to extract the vertical separations between successive veins, a program ISOLAT was used to isolate all the vein intersections within the supplied eastern and northern co-ordinates, and specified elevations as in the case of SORTVLX, but here dummy entry and exit lines are entered by the program to identify the top and base of each log in the specified level. These dummy veins are flagged with the following numbers:

1 — to indicate the top of the chosen level when the log continues upward beyond the top of the level.

2 — to indicate the top of the log, but inside the level i.e. the collar of a downhole or the top of a raise or an uphole.

8 — to indicate the base of the log inside the level i.e. the start of an uphole, the start of a raise or the end of a downhole.

9 — to indicate the base of the level when the log continues beneath the level.

Figure 3.2 illustrates the eight cases which can occur with the vertical logs in the vertically sided slices (levels). The total length of the drill holes is recorded in the LITHO file, making it possible to locate the end of a log.

The output from ISOLAT is used by DIFISL to compute the vertical separation between successive veins, the number of veins, and the vein width accumulation for the entire height of Level 2. The results are kept on a permanent indirect access file VSEP2L.

By changing the elevations in the programs, information about the veins in Level 1 was extracted. This level is within the elevations 680 m and 620 m. The extracted vein width, vein concentration and vertical separation were kept in separate files, VEINS1L, VCON1L and VSEP1L, respectively.

Both programs SORTVLX and ISOLAT have mechanisms to trap errors, such as wrongly recorded zero widths, and to calculate the true widths of the veins in case any of the holes are inclined (there are only a few inclined holes).

A plot of the locations of all the boreholes and raises within Level 2 of the mine was generated, as shown in Figure 3.3. Figure 3.4 shows the representative part of the mine (marked ABCD) selected for detailed analysis. It can be observed that although smaller in area compared to the whole of Level 2, this selected area has the greater proportion of drill holes and is therefore likely to

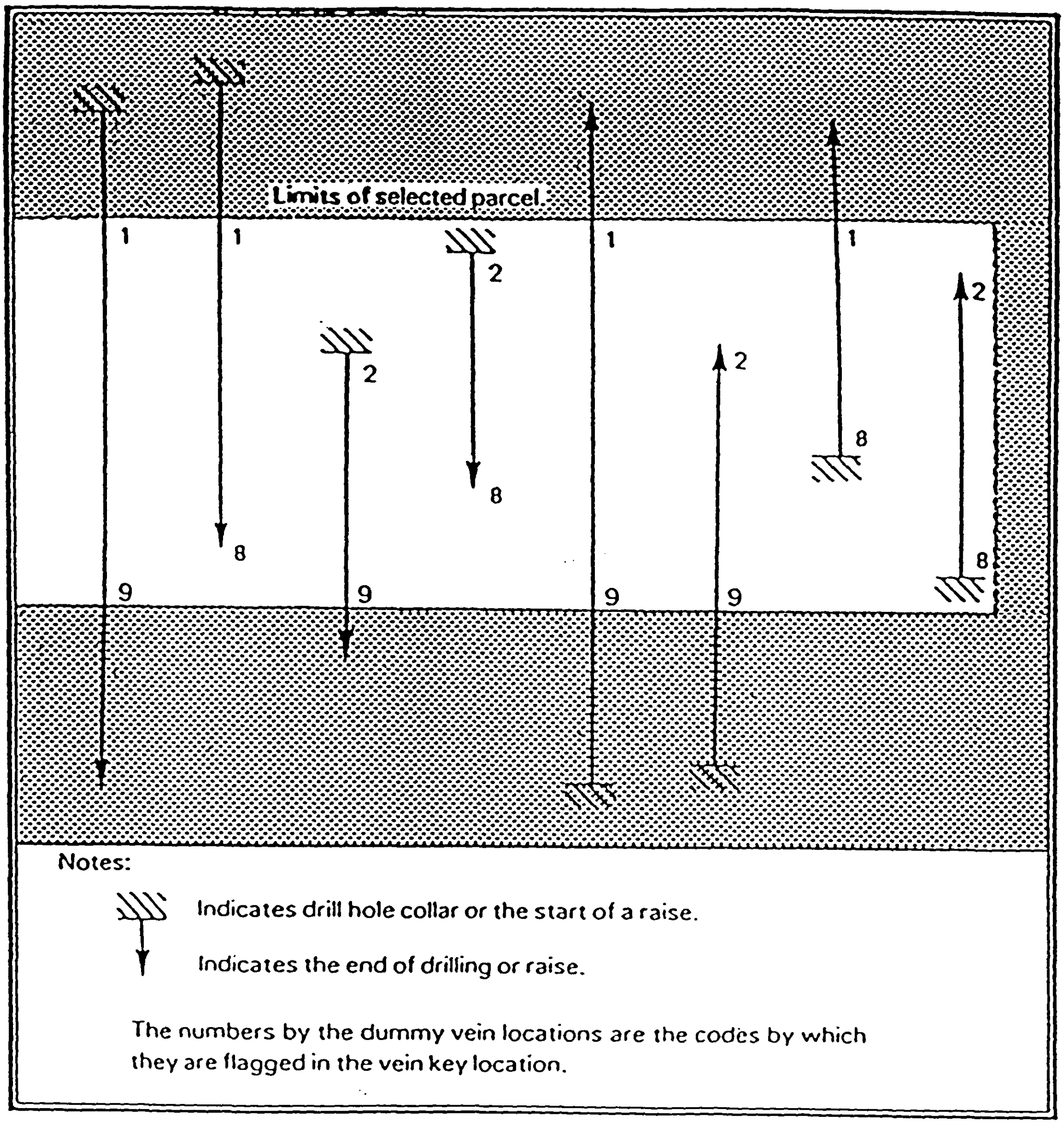

Figure 3.2. Diagram illustrating the insertion of dummy vein locations by program ISOLAT to indicate the beginning and end of logging (after Williams, 1985).

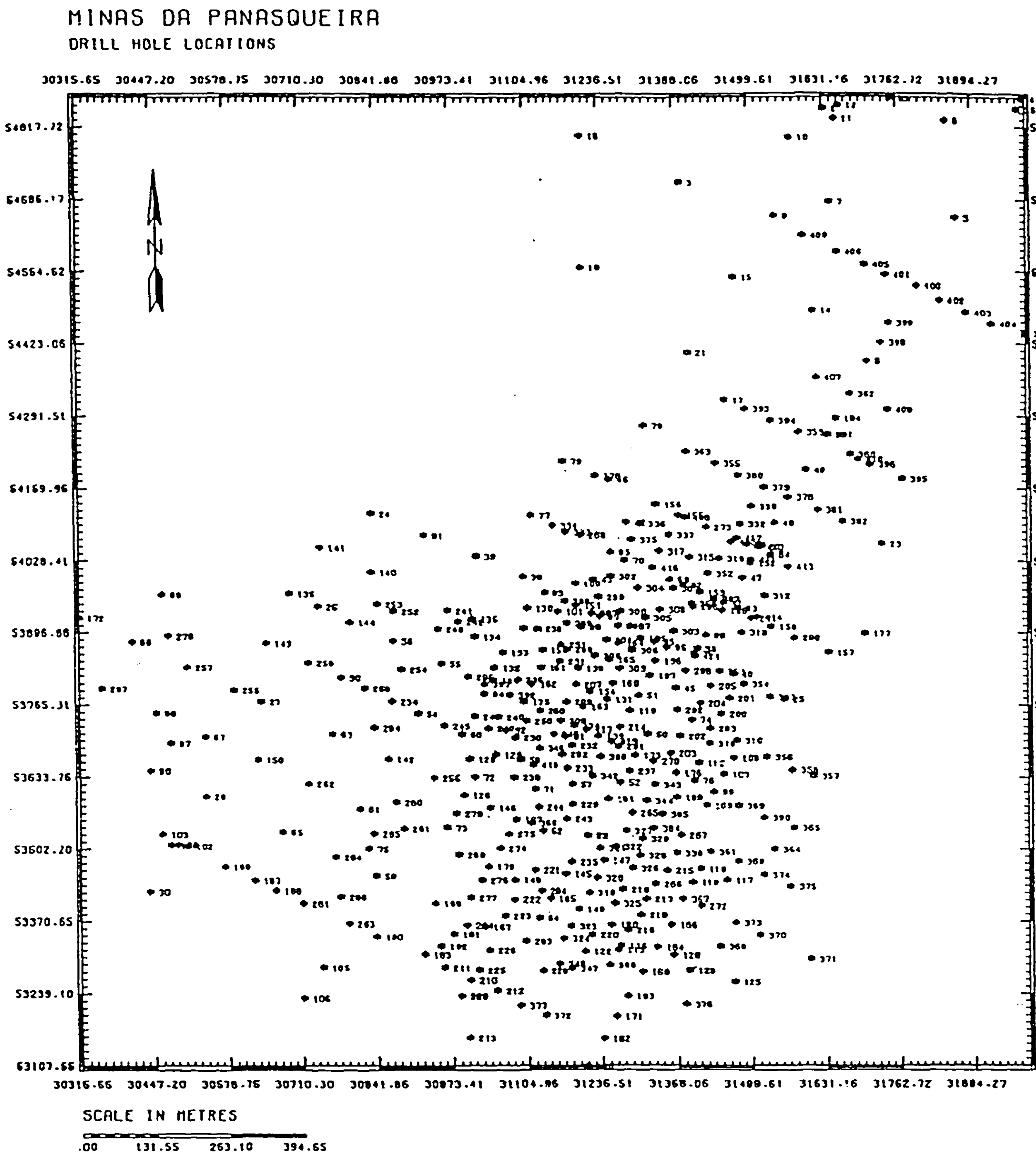

Figure 3.3. Plan of Level 2 showing the locations of boreholes and raises.

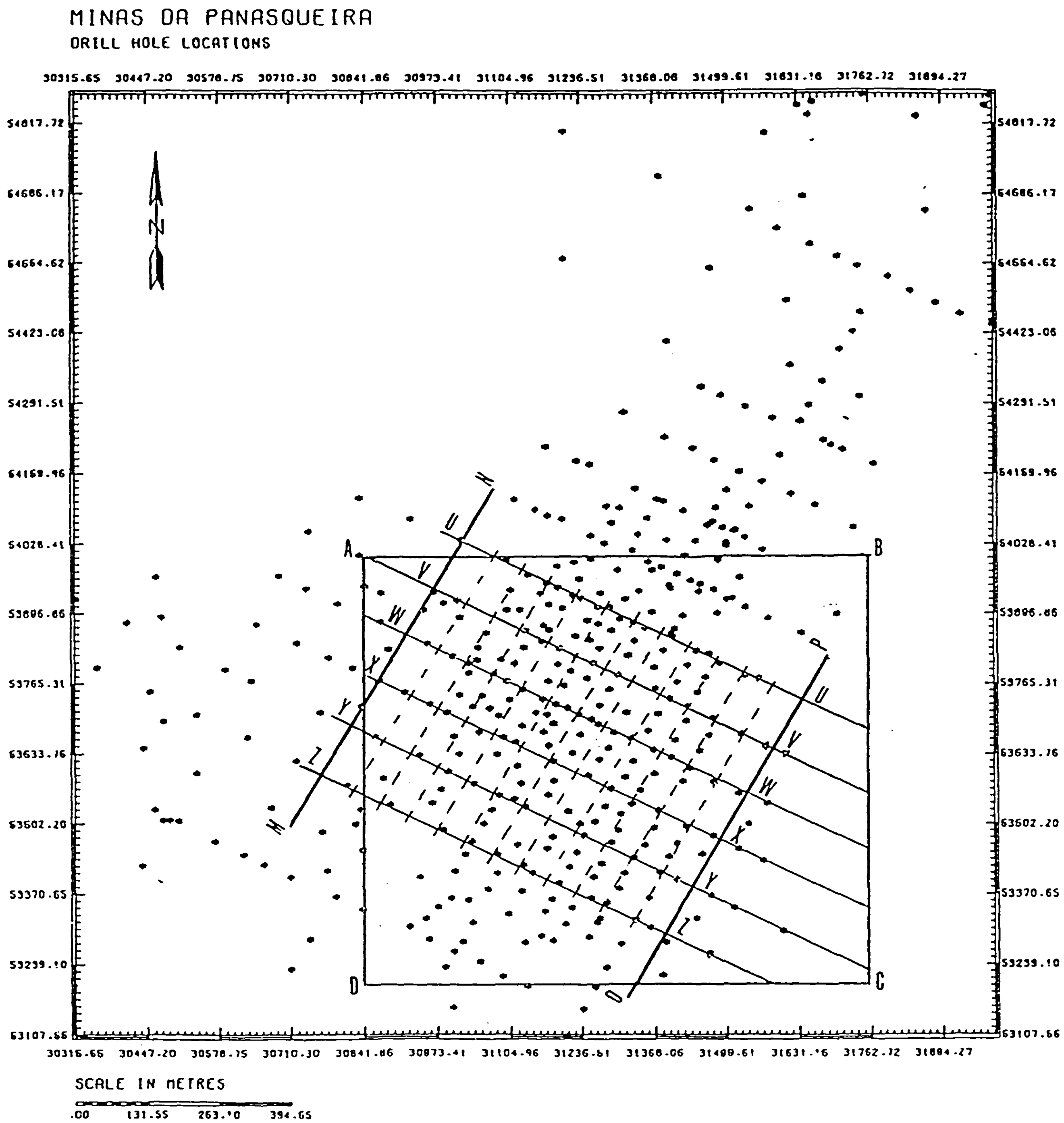

Figure 3.4. Plan of Level 2 showing the representative area selected for detailed analysis (marked ABCD) and sectional lines.

have more information on the veinage. It is also the area of current mining operations, thus making comparison of results obtained in the course of this project with those of the mine possible.

Since drilling is a dynamic process, the mine updates the data set described previously, i.e. BORHO, LITHO and VEINAGE, on a monthly basis. The initial data set provided for this research was from the records of January, 1984. In October, 1986 an updated set of data was made available. Again in January, 1987, a still more recent data set was provided. The analysis carried out in this research has hence had to be repeated several times to improve the results. The results shown in this report are mainly based on the data set of January, 1987.

3.3.1 Contour Map and Cross-sections

Within the selected area, the vein width accumulation in the entire 60 m height of Level 2 is presented in the form of a contour map in Figure 3.5 to give an overview of the veinage distribution. This map was generated with the Tile, Solid and Conicon package produced at the Mathematics Department of Bath University (Sibson (1981)). The package uses the Natural Neighbour Interpolation procedure described by Sibson (1980).

The contour map shows the distribution of all the quartz contained in the study area. The more darkly shaded areas have greater quartz content and can be observed to be surrounded by areas of less quartz content. It can be seen that apart from some few small areas with no quartz, the whole of the study area is mineralised. This overview of quartz distribution, however, is not useful to Panasqueira because the material in the entire 60 m height of Level 2 is not mined — the total quartz mined is less than 6% of the total volume of material in the entire level. *The strategy that is taken in this project is therefore to consider the allowable stoping height and to estimate the aggregate quartz within it.*

Cross-sections were generated along sectional lines UU, VV, WW, YY, XX and ZZ, as shown in Figure 3.4. The orientation of these lines corresponds to that of the drives at the mine. Figure 3.6 shows the cross-section along WW which is typical of all the cross-sections. Actually, the cross-section shows the vein widths projected onto the nearest 50 m regularly spaced planes. The cross-section does not portray any apparent pattern in the occurrences of the veins. Moreover, there is no correlation between the vein widths and their elevation;

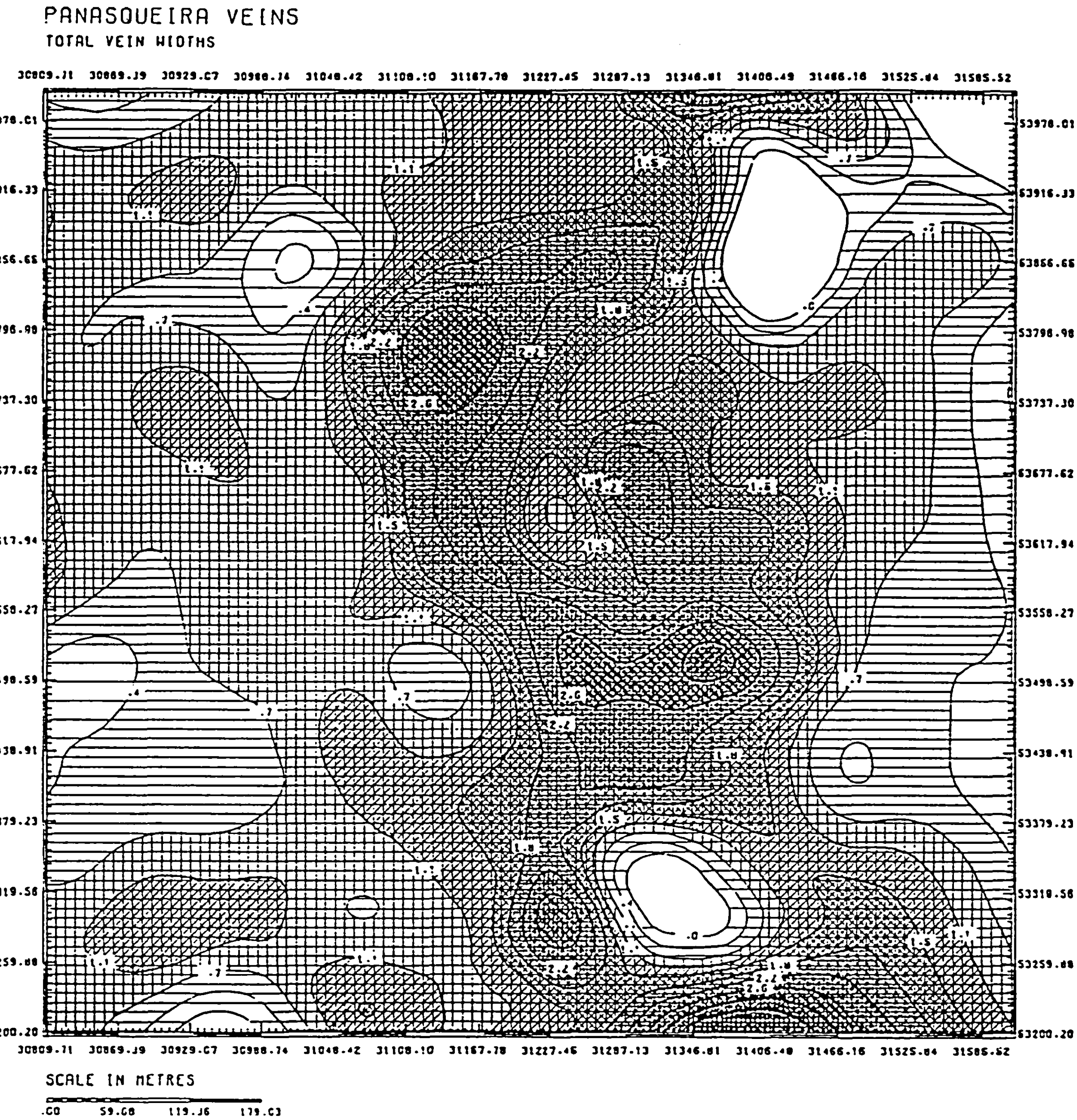

Figure 3.5. Contour map of vein width accumulation.

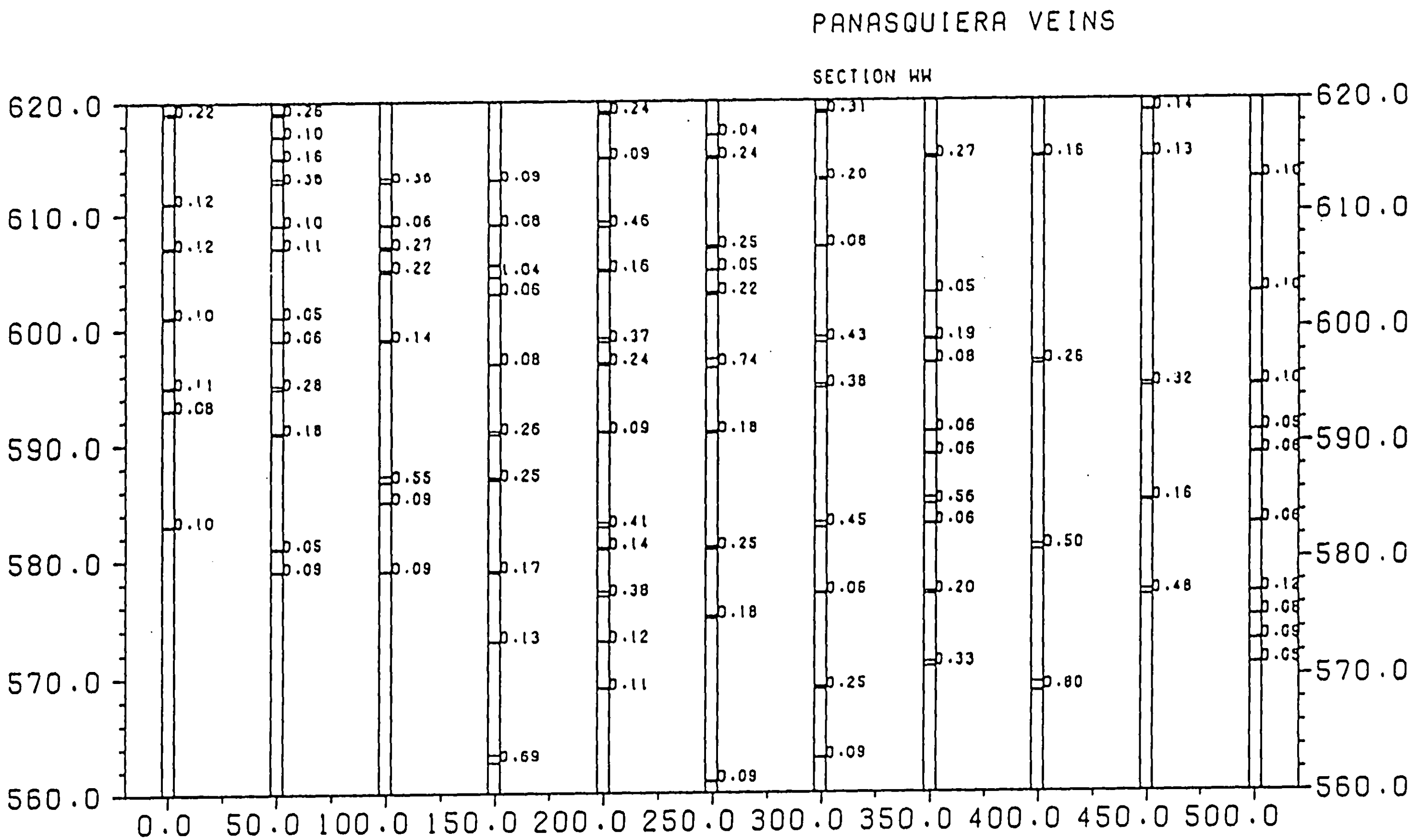

Figure 3.6. Section W-W showing intersected vein widths.

in fact, for all the veins in the mining concession, the coefficient of correlation is only -0.064 for Level 1 and -0.003 for Level 2. The logic of serial correlation, for instance, outlined by Davis (1986), based on constant spatial sequence, is not applicable to the Panasqueira vein system because in practice the veins form a complex network which are predominantly sub-horizontal with interwoven 'fingerlets'.

Figure 3.6 highlights the problem of vein width interpolation, and thus the delineation of potential mineable horizons, discussed earlier. There can be several variations in the joining up of the intersections. An artistic impression of three hypothetical variants derived by three independent geologists is shown in Figure 3.7, which is based on section W-W. Each colour represents the work of one of the geologists. The work in green is based on the principle that the veins are strictly horizontal and cannot extend any further than the intersections shown — the work in blue presupposes that the veins do not terminate but rather they form a complex network, the nodes of which are intersected by the boreholes.

The work in red suggests that the veins are fairly extensive and can rise and fall in their extent. The author is inclined to support this view as a result of the outcome of the analysis carried out in Section 3.4. From the mining point of view, this work is more interesting since it shows possible development horizons which will expose the veins.

3.4 Observations from Mined out Areas

Mined out areas of the deposit were studied with two aims in mind:

(i) to acquire more knowledge of the behaviour of the veins; and

(ii) to translate into rules the judgements used by the mine geologists to connect the vein intersections observed in diamond drill cores – the rules can then be used to reproduce mathematically the interpolation process at the mine.

For this exercise, available mine sections of mined out areas in Level 1 were used. The original logs of identifiable boreholes and raises were isolated from the Mine Data Base. Then the widths of the intersected veins were presented in the form of bar graphs along the length of each borehole or raise. These graphs were cut up and glued to their corresponding positions on the sections. Figure

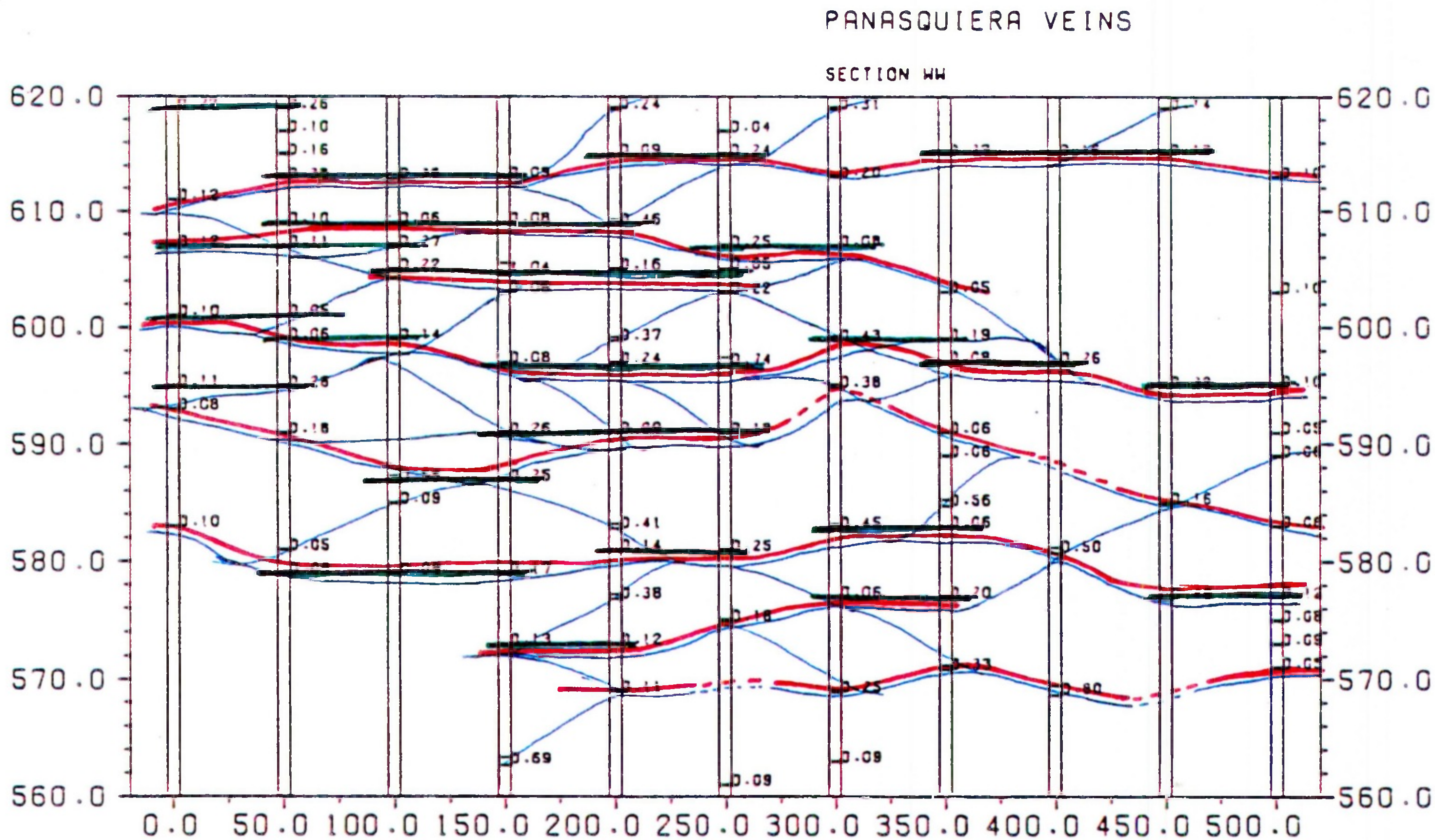

Figure 3.7. Possible variants of joining the vein intersections.

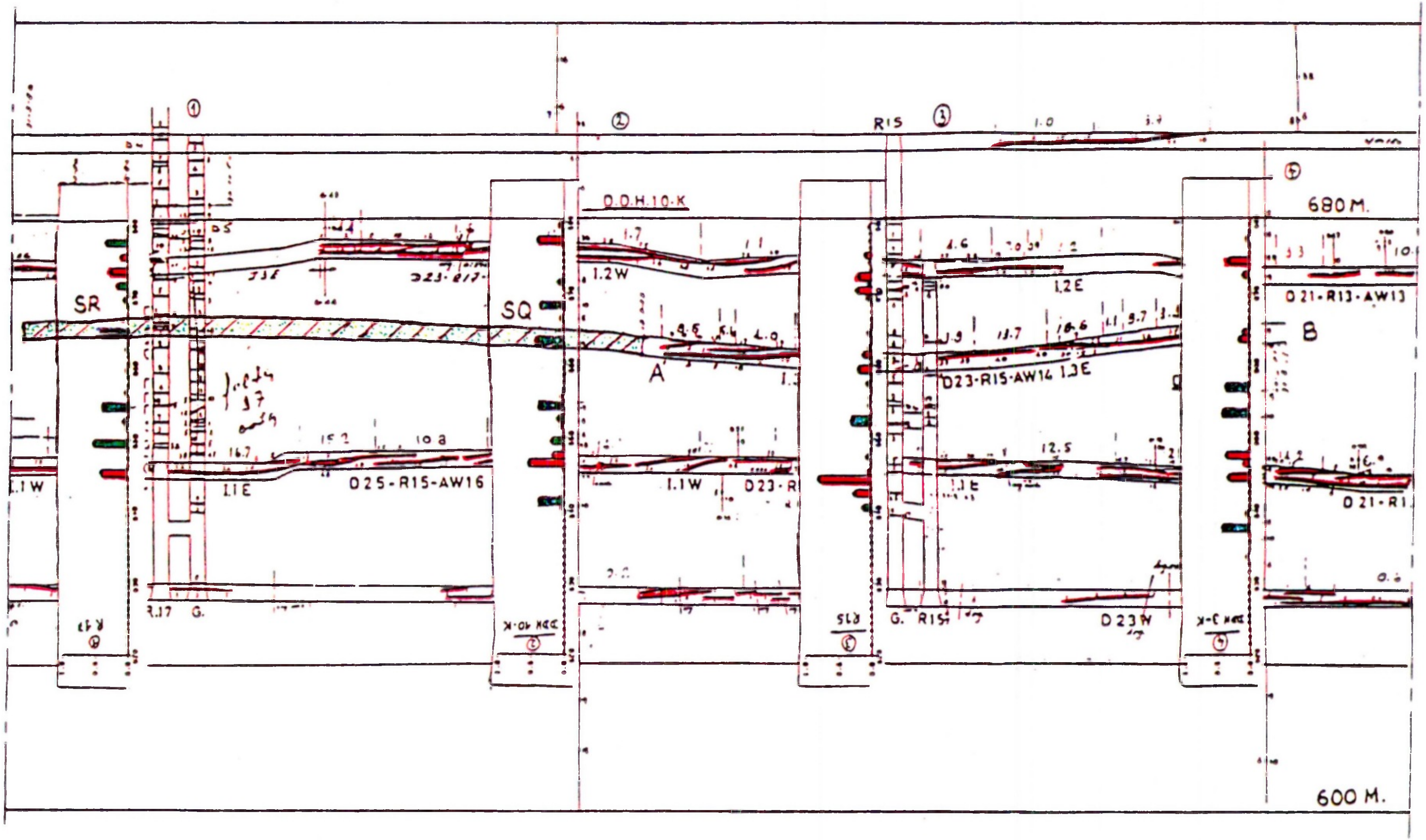

Figure 3.8. A section of a mined out area in Level 1 with bar-graphs of vein widths intersected by initial boreholes or raises superimposed on it. The veins picked up for development are shown in red colour. The veins in green colour were left out.

3.8 is an example of the results obtained. In this figure, the veins picked up for development are shown in red. The veins in green might have been left out, or mined later — there is no evidence as to what happened to them. Based on the study, and through personal communications with the mining staff, the following conclusions were drawn:-

(i) The quartz veins are near horizontal and predominantly parallel.

(ii) The angle of inclination to the horizontal is 0-10°.

(iii) Veins may continue to be horizontal in their extent, or fall, or rise, or fall and rise, within the general angle of inclination.

(iv) Veins may extend in some direction for up to 50 m.

(v) The veins are almost tabular.

(vi) Mineralisation is present in these more or less horizontal veins.

The following facts are also known:-

(i) Geostatistical structural analysis, carried out in Chapter 4, shows that the vein widths have predictability. Closeness in distance implies closeness in value.

(ii) Only single veins with width values of 0.20 m or more are mined; several intersected veins are left out, either due to their narrow widths or due to the fact that they cannot be joined up with other veins, or less commonly that their vertical separation from other veins is not enough to provide the necessary support for mining.

3.4.1 Planning Cut-off Criteria

Panasqueira sets a cut-off width of 0.18-0.2 m. This figure applies to individual veins. Veins with widths of at least 0.18 m are included in the geological reserves, but only veins with a width of at least 0.2 m are currently mined. This is based on the general belief, and supported by regressional analysis carried out at the mine, that the more quartz mined the more wolfram obtained and that a grade of approximately 13-15 kg/m^2 corresponds to 0.18-0.2 m of quartz mined.

Based on this *a priori axiom*, and in the absence of assays, the strategy taken in this project is to consider the quantity of quartz contained in the mined ground. It is logical to state that a number of narrow veins with an aggregate width of 0.18-0.2 m must equally yield approximately 13-15 kg/m^2 since mining them will give the same quantity of quartz. It is recommended to maintain the current stoping height of 2 m (see Chapter 6) and therefore one must be interested in the aggregate width of quartz veins in this 2 m height of ground. *In this thesis, the aggregate width of quartz veins in every 2 m height of ground is termed as vein concentration.** The cut-off criterion then becomes a vein concentration of 0.2 m, or indeed some other value. It is a planning cut-off, at this stage of mining when assays are absent, and is in line with the operational cut-off defined by Taylor (1972; 1985). It is, in fact, also a measure of dilution and relates directly to mining and processing costs. A selected value of the vein concentration as the cut-off is, therefore, also a measure of the degree of dilution that the mine is prepared to accept.

3.5 Computerised Delineation of Development Horizons

Taking the observations made in Section 3.4 into account, an interactive graphics program, MBANDS, was developed to delineate possible development horizons across a given cross-section of the mine. The required input data for the program are the co-ordinates of drill holes along a given sectional line. The program isolates the drill holes from the working database. For example, from file VEINS2L, it generates on a display screen a section showing the vein intersections and a 2-metre segment to which each intersection belongs. The user is prompted to select two adjacent segments which are then checked against the vein limiting inclination and cut-off vein width. They are also checked against the allowable interpolation distance as portrayed by the semivariogram model discussed in Chapter 4 of this report. If all the criteria are satisfied, the intersections are joined and boundaries of 2 m height are drawn to form a 'band'. Otherwise, the user is informed of a shortcoming and asked to select two other adjacent segments. In this way, several permissible bands can be linked to form a possible mineable horizon.

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* The term *vein concentration* will be often used. The reader is hereby notified that it is not meant to suggest the number of veins, grade or any unit of measure of mineral content, i.e. saleable product content. When encountered in this thesis, it is nothing other than as it is defined, i.e. the abundance of quartz represented by the aggregate vein width per 2 m height of ground, and it is measured in metres.

The program makes use of graphic routines from the Common Plotting Library available at the Imperial College Computer Centre. It is fast and so can be used to quickly delineate the potential ground for development. The limiting values of the operational control variables, i.e. the cut-off and the limiting inclination of the horizons, can always be changed if desired. Despite its simplicity, the program emulates the interpolation process at the mine. Figure 3.9 is a typical output of MBANDS showing the possible development horizons in the section shown in Figure 3.6. However, because of the capricious nature of the veins, different users of the program can delineate the potential ground for development in different ways. Outputs of program MBANDS showing possible development horizons along the remaining sectional lines are presented in Appendix 3.

3.6 Markovian Analysis of the Vein Pattern

This section aims at analysing the frequency of vein occurrences in a given section and to investigate the possibility of generating a simulated section showing possible vein occurrences. The concept employed is an extension of the Markov phenomenon.

The concept of the Markov model is that random natural events may exhibit a short memory, i.e. an event is influenced by the immediately preceding events. The Markov process is one in which the probability of being in a given state at a given time or place is based on the preceding state or states. When the frequencies or probabilities attached to the transitions from one state to the next are established, it is possible to predict the times or places of occurrences of other states.

A common natural process that has Markovian properties is the weather. The changes or transitions of the different states of the weather (sunny, cloudy, raining, snowing) form a chain of events in which the state of today's weather has some influence on tomorrow's weather. In management science, the Markov analysis is used to predict the likely future demand of commodities because they are greatly influenced by current supply and demand (Thierauf *et al.* (1985)). These examples are characterised by chains or sequences of discrete states in time or place that are essentially 'one-dimensional'.

The Markov concept has also been applied in geology. The successive occurrences of different minerals (states) in a column of rock represent a 'one-dimensional' sequence of events that exhibit strong Markovian properties. Davis

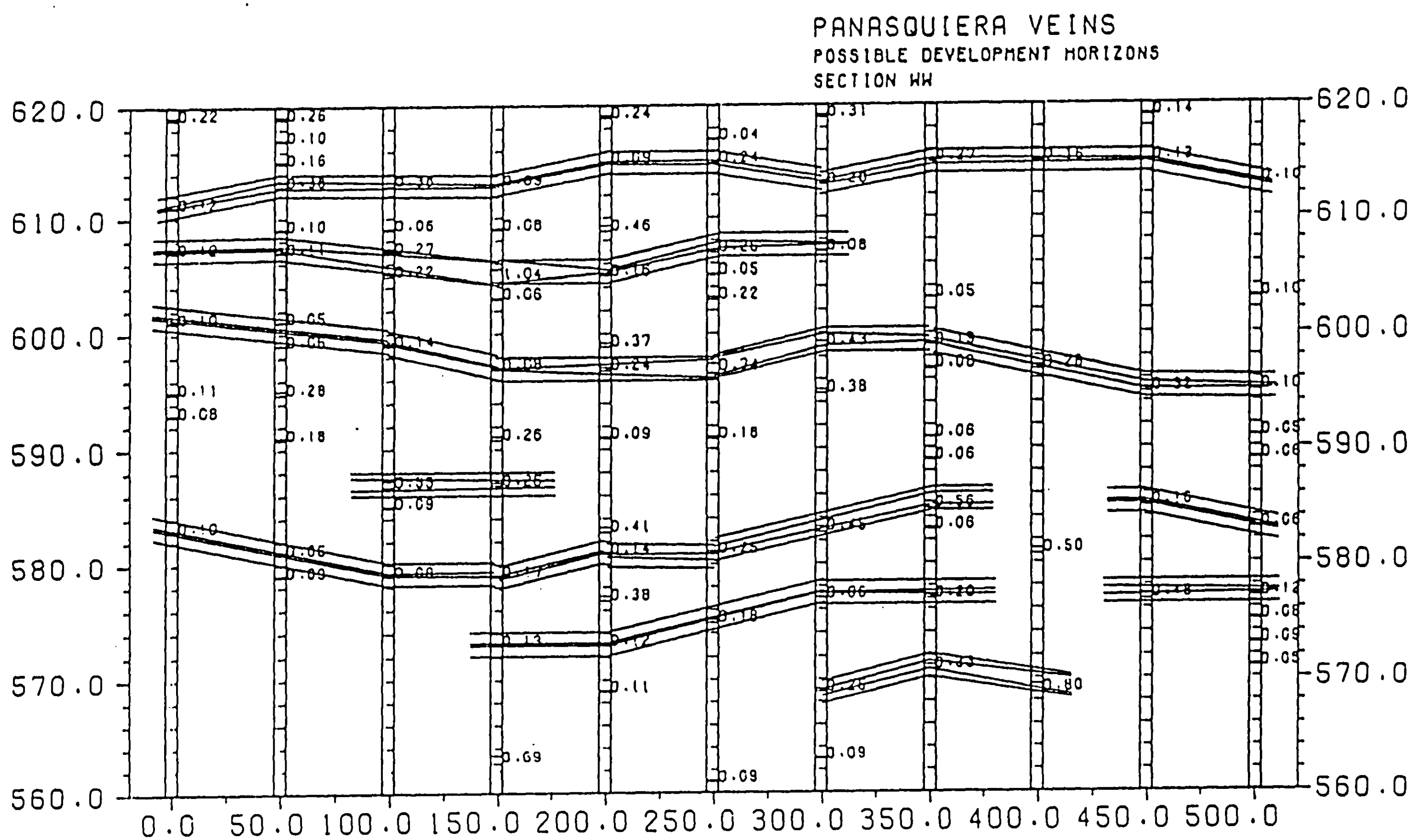


Figure 3.9. Output of program MBANDS showing possible development horizons with respect to Section W-W.

(1986) demonstrates the use of Markov concept in studying natural stratigraphic sequences.

Switzer (1965) extended the application of the Markov concept in one-dimensional sequences to that of the plane (i.e. two-dimensions), making it possible to study planar patterns. His theorem states that *a finite state random process can exist in the plane with such properties that its alternations between states along any straight line are Markovian.* Switzer's main concepts in extending the Markovian process to isotropically random planar patterns are as follows:

(i) A random planar pattern can be constructed by assigning two or more independent states into random polygons that are defined and bounded by intersections of a set of random lines which could be described by polar co-ordinates (p, θ). When p and θ are distributed uniformly in a circle of radius R such that $-R \leq p \leq R$ and $-\frac{\pi}{2} \leq \theta \leq \frac{\pi}{2}$, respectively, random lines will be generated in the circle. The number of lines is assumed to be a random variable that follows the Poisson distribution with a mean N given by:

$$N = 2\pi\lambda R \tag{3.1}$$

where λ is a constant governing the distribution. The steps in the conceptual construction of a random planar pattern are shown in Figures 3.10a and b.

(ii) For any set of n collinear ordered points within the circle, a random sequence of states is assigned to them according to the state of each of the random polygons through which the collinear points pass, yielding

$$A_1, A_2, \ldots, A_n$$

where A_i denotes one of the m states. If A_1 and A_2 are located in different random polygons, then the following conditional probability relations exist:

$$P(A_1|A_2,\ldots,A_n) = P(A_1)[1 - exp(-2\lambda L)] \quad \forall\ A_1 \neq A_2 \tag{3.2}$$

$$P(A_1|A_2,\ldots,A_n) = P(A_1)[1 - exp(-2\lambda L)] + exp(-2\lambda L) \quad \forall\ A_1 = A_2 \tag{3.3}$$

where $P(A_1)$ is the marginal probability of state A_1 in the plane, and L is the distance between the two points with states A_1 and A_2. The

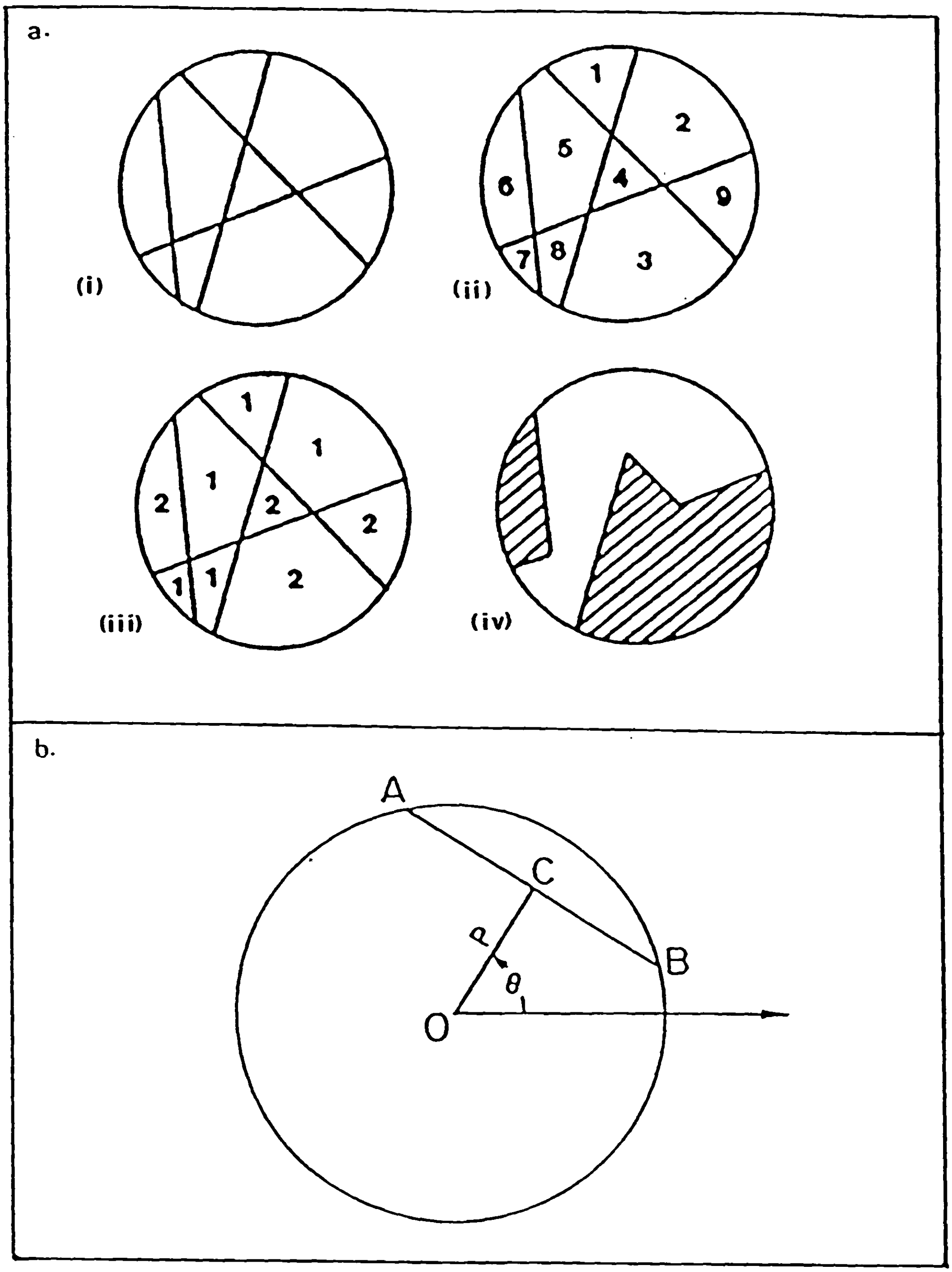


Figure 3.10a. Steps in the construction of a random pattern:
(i) generate a set of random lines in a circle,
(ii) label each resulting random polygon with an identity number,
(iii) independently assign states (e.g. 1 or 2) to the polygons at random, and
(iv) erase the the lines between polygons assigned with the same state. (after Lin and Harbaugh, 1984)

Figure 3.10b. Diagram to illustrate that a random line AB can be specified by polar co-ordinates (p, θ) of a radial line OC to which the line AB is erected as a perpendicular. Angular origin is erected by a vector emanating from the centre of the circle.

quantity $2\lambda L$ indicates the number of random lines that cut through any line segment of length L in the circle.

Because there are only two states, there is no information pertaining to states $A_3, \ldots, A_n$ specified on the right sides of Equations (3.2) and (3.3). The general form of the relation may be written as:

$$P(A_1|A_2, \ldots, A_n) = P(A_1|A_2) \tag{3.4}$$

which indicates that the probability of state A_1 existing, given a sequence of preceding states $A_2, \ldots, A_n$ is equal to the probability of state A_1, given the immediately preceding state A_2.

In employing Switzer's theorem to study a two-dimensional Markov process e.g. planar patterns, there are two main objectives:

(i) to analyse the actual planar pattern to provide essential statistics, namely the transition and marginal probabilities of the states, and

(ii) to construct a simulated pattern that utilises the statistics derived from the actual pattern.

In this process, the initial problem is to estimate λ, which governs the number of random lines N, where $N = 2\pi\lambda R$. The value of λ can be estimated from both Equations (3.2) and (3.3). From Equation (3.3), the value of λ is:

$$\lambda = -\frac{1}{2L} ln\left[\frac{P(A_1|A_1) - P(A_1)}{1 - P(A_1)}\right] \tag{3.5}$$

Equation (3.5) refers to the case where a state makes a transition to itself. If there are m states, A_1 can be anyone of them. Hence there are m estimates of λ among which the differences are statistically insignificant. It follows from Equation (3.5) that the determination of λ for a specific pattern is related to the transition probabilities and the marginal probabilities for that pattern. Only the constant λ and the marginal probabilities are required for the simulation of random patterns, inasmuch as λ, the transition probabilities, and the marginal probabilities are mutually interdependent.

Anisotropic Planar Pattern:

The equations described above refer to isotropic planar patterns where the parameter θ of random lines is distributed uniformly between $-\frac{\pi}{2}$ and $\frac{\pi}{2}$. In this case, the random lines and their intersections yield polygons that are distributed randomly irrespective of direction.

An anisotropic pattern is one in which the pattern is elongated in one direction. In this case, the random lines are defined as in the isotropic case, but the parameter θ is constrained. The degree of elongation of the polygons produced by intersecting lines within a circle of radius R is characterised by the range of θ. If the desired range of the azimuth of the random lines lies between two angles $\theta_1 + \frac{\pi}{2}$ and $\theta_2 + \frac{\pi}{2}$, such that $-\frac{\pi}{2} \leq \theta_1 < \theta_2 \leq \frac{\pi}{2}$, then the two parameters $(p,\ \theta)$ are distributed as $p \in [-R,\ R]$; $\theta \in [\theta_1,\ \theta_2]$. The number of random lines in the circle are assumed to follow the Poisson distribution with a mean N given by:

$$N = \int_{-R}^{R} \int_{\theta_1}^{\theta_2} \lambda dp\ d\theta = 2\lambda R(\theta_2 - \theta_1) \tag{3.6}$$

When $\theta_2 - \theta_1 = \pi$, the random lines within the circle are not constrained; $N = 2\pi\lambda R$, and the pattern is isotropic.

In practice, it is difficult to create sets of random polygons that accord well with a given pattern. Moreover, the counting process can be laborious. Therefore the approach adopted here is to superimpose a grid with square cells, assumed to be random, over the pattern and record the state encountered at each cell.

To calculate the value of λ, it is necessary to obtain the transition frequencies by counting transitions at grid points, both in the horizontal and vertical directions. The two sets of counts can be combined in an isotropic case to arrive at a single λ for each state.

In an anisotropic case, the transition frequencies in the horizontal and vertical directions must be treated separately to arrive at a pair of λ for each state. Lin and Harbaugh (1984) have derived the factors that must be incorporated in Equation (3.5) to take care of the anisotropic case — to calculate λ, the distance L in Equation (3.5) can be set to the value of 1 in the isotropic case for simplicity, and later replaced by one of the following values:

(i) to obtain λ in the horizontal direction,

$$L = sin\left(\frac{\theta_2 - \theta_1}{2}\right) cos\left(\frac{\theta_2 + \theta_1}{2}\right) \quad \forall\, \theta_1,\ \theta_2 \tag{3.7}$$

(ii) to obtain λ in the vertical direction,

$$L = -\sin\left(\frac{\theta_2 - \theta_1}{2}\right) \sin\left(\frac{\theta_2 + \theta_1}{2}\right) \quad \forall\, \theta_1 \leq 0,\ \theta_2 \leq 0 \qquad (3.8)$$

$$L = \sin\left(\frac{\theta_2 - \theta_1}{2}\right) \cos\left(\frac{\theta_2 + \theta_1}{2}\right) \quad \forall\, \theta_1 \geq 0,\ \theta_2 \geq 0 \qquad (3.9)$$

$$L = 1 - \cos\left(\frac{\theta_2 - \theta_1}{2}\right) \cos\left(\frac{\theta_2 + \theta_1}{2}\right) \quad \forall\, \theta_1 \leq 0,\ \theta_2 \geq 0 \qquad (3.10)$$

3.6.1 The Computer Program for the Analysis

The computer program used in this research to carry out the Markov analysis of the vein widths, MAK2, is a modified version of the program MARKOVII originally written by Lin and Harbaugh (1984). The program uses graphical routines from the Common Plotting Library as well as a random number generation routine all of which are available at the Imperial College Computer Centre. There are six main steps involved in the preparation of the input data file and the execution of the program. These are as follows:

(i) A data file pertaining to the original planar pattern (section showing vein intersections in our case) must be prepared. The file must contain the radius R, the number of states m, the elongated direction angles D_1 and D_2 of the pattern, and the locations of grid points describing the alternation of states i.e. the grid number in which the states are encountered. Only the row and column numbers of the grid have to be supplied. One of the states can be omitted – the omitted state must be included in the data file.

(ii) The rows and columns of the grid must be equal otherwise the program creates a square array, and then forms a circle of radius R' within the array by setting elements of the array that lie outside the circle to zero. It then supplies the omitted state at all array elements within the circle. Parts of the elements within the circle are replaced by other states recorded in the data file.

(iii) Based on the digitised data, transition frequencies are counted separately along both horizontal and vertical directions (i.e., along rows and columns in the array), and marginal frequencies are obtained. If the pattern is isotropic, the counts obtained from the two directions are merged, but if the pattern is anisotropic the counts remain separate. The transition and marginal frequencies are tabulated and transformed into estimates of transitional and marginal probabilities. The program also calculates λ values for each state, using the diagonal elements of the transition probability matrix. The results are recorded and displayed on the computer screen.

(iv) The program requests input from the user as to whether the original digitised pattern is to be plotted. The user must respond with yes or no. A negative answer implies a simulated pattern is to be generated and the program produces a synthesised pattern for which three parameters must be supplied – the constant λ, radius R and a seed number for initialising the pseudorandom number generator. The value of λ in an anisotropic case may be taken as the mean of the λ values obtained in step (iii)

(v) The program creates a circle of radius R and then generates $2\pi\lambda R$ random lines within the circle. This results in a group of polygons formed by intersection of the random lines. Based on the marginal probabilities of states obtained from the original pattern, the program assigns m states to the polygons. Transition frequencies are counted automatically in the simulated pattern, and the transition and marginal probabilities are estimated as well as λ values for the states. The statistics obtained from the simulated pattern can be checked against those derived from the actual pattern.

(vi) The program has a facility that allows the user to plot an actual or a simulated pattern.

3.6.2 Application to Vein Cross-sections

Any planar pattern, isotropic or anisotropic, can be simulated by superimposing on it a square grid, assumed to be random, and counting the state encountered at each grid to obtain the transition frequencies. The derived statistics can then be used to regenerate the actual pattern or similar patterns. Two conditions must be satisfied. The pattern must be random and must have Markovian properties.

The vein width intersections as portrayed by the cross-sections are random. If it is assumed that the pattern of vein occurrences have Markovian properties, then, by assigning one state to the vein widths and another to the schistose country rock, it is possible to analyse the pattern and deduce:

(i) the probabilities of encountering veins in the horizontal and vertical directions, and

(ii) a possible pattern of vein occurrences at some other place using the statistics from a known pattern.

In this process, a square grid, assumed to be random, was superimposed on a cross-section showing the vein intersections. The grid distance was set at 12 m and each vein intersection was assumed to extend for 6 m near-horizontally to either side of its location, thus making the pattern anisotropic with the elongated direction of the veins which are predominantly horizontal, between -10° and 10°.

Table 3-1 is a summary of the statistics of the vein patterns as portrayed in sections along the the sectional lines U-U, V-V, W-W and X-X shown in Figure 3.4 as well as those of a simulated section generated with the statistics derived from the pattern in section V-V and the mean value of λ from the actual patterns in sections V-V and W-W. It can be seen that in the horizontal direction, the transition probability from vein to vein is approximately 0.5. *The inference is that if a vein is intersected then there is a 0.5 probability that it will extend horizontally for at least 12 m on either side of its location along the cross-sectional line.* In the vertical direction, the probability is only about 0.07, but this has no practical significance, knowing that the veins are sub-horizontal and thin.

The characteristics of the simulated pattern are similar to those of the actual patterns, meaning its picture could well represent possible vein occurrences in some section of the mine. Figures 3.11a, 3.11b and 3.11c show respectively the actual pattern in section V-V, the simulated pattern and the actual pattern in section W-W. There is some similarity between the actual and simulated patterns, the main difference being that some of the vein occurrences in the simulated pattern are singular, i.e. they do not occupy two grids all the time. This arises from the fact that each intersected vein in the actual patterns was assumed to extend for 12 m, thus occupying two grids. There is however some similarity in that in both cases the vein occurrences are distributed throughout the sectional plane without any particularly discernible consistency. The Markov concept is therefore applicable to the vein occurrences. It follows that once the vein occurrences in a given section are known, e.g. after the veins have

Table3-1. Statistical characteristics of actual and simulated patterns of vein intersections.

Section		U-U		V-V		Simulated		W-W		X-X		
		To		To		To		To		To		
	From	Vein	Schist	Vein	Schist	Vein	Schist	Vein	Schist	Vein	Schist	
Transition freq. matrix	Vein	96	85	88	88	86	90	92	92	89	89	Horizontal direction
	Schist	85	2284	88	2286	88	2280	92	2274	89	2283	
Transition prob. matrix	Vein	.530	.470	.500	.500	.489	.511	.500	.500	.500	.500	
	Schist	.036	.964	.037	.936	.037	.963	.039	.961	.038	.962	
Marginal freq.		181	2369	176	2374	174	2370	184	2366	178	2372	
Marginal prob.		.071	.929	.069	.931	.068	.931	.072	.928	.070	.930	
Lambda		.352	.354	.385	.384	.397	.379	.387	.390	.387	.391	
		To		To		To		To		To		
	From	Vein	Schist	Vein	Schist	Vein	Schist	Vein	Schist	Vein	Schist	
Transition freq. matrix	Vein	28	153	30	146	29	147	28	156	38	140	Vertical direction
	Schist	153	2216	146	2228	145	2223	156	2210	140	2283	
Transition prob. matrix	Vein	.155	.845	.170	..830	.165	.835	.152	.848	.213	.787	
	Schist	.065	.935	.061	.939	.061	.939	.066	.934	.052	.941	
Marginal freq.		181	2369	176	2374	174	2370	184	2366	178	2372	
Marginal prob.		.071	.929	.069	.931	.068	.931	.072	.928	.070	.930	
Lambda		1.201	1.235	1.111	1.077	1.131	1.070	1.225	1.242	1.226	1.242	

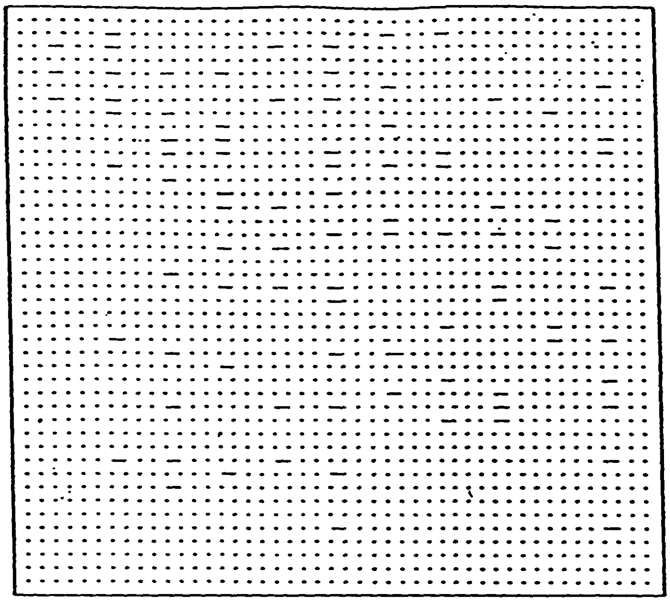
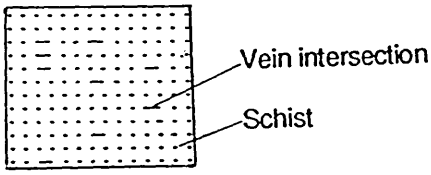


Figure 3.11a Output of program MAK2 showing pattern of vein occurrences in Section V-V.

Key



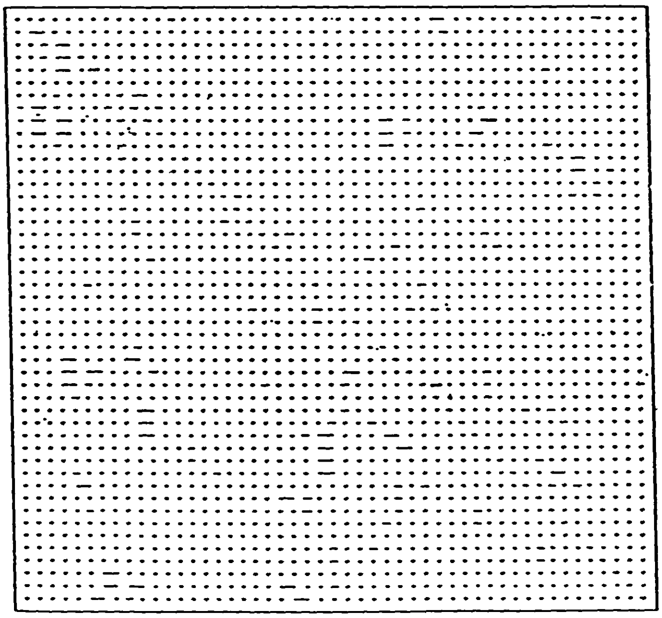


Figure 3.11b Output of program MAK2 showing simulated pattern of vein occurrences generated with statistics from Section V-V and mean value of lambdas from Sections V-V and W-W.

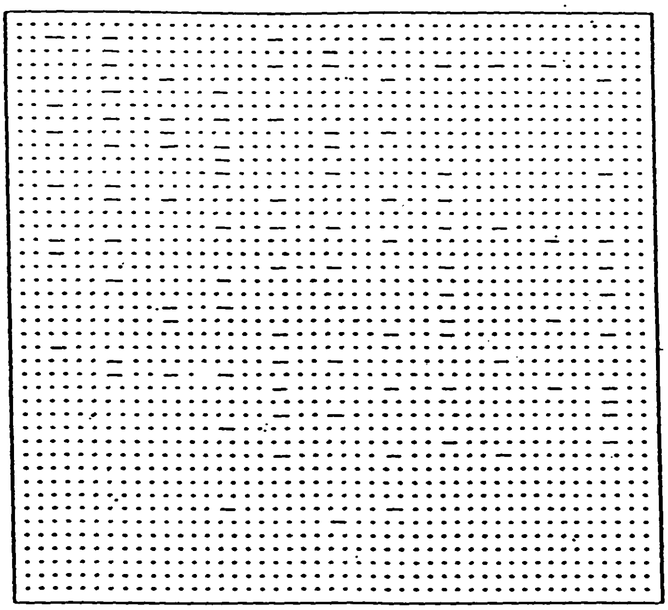


Figure 3.11c Output of program MAK2 showing pattern of vein occurrences in Section W-W.

been developed and exposed, the statistics derived from the pattern of the vein in the section can be used to generate possible patterns in the vicinity of the section.

3.7 Classical Statistical Analysis

The purpose of this section is to study the three attributes of the deposit, i.e. the vein width, the vein concentration and the vertical separation between successive veins, using classical statistical methods.

The theory of classical statistics is based on the concept of a parent population, infinite in size, from which independent samples can be drawn or measured. The aim is to study the sample values from a given population in order to be able to make qualified statements about the population, since it is virtually impossible to obtain the infinitely large number of values of the population.

In the geological environment, the population can refer to the infinitely large number of values of an attribute, such as thickness or grade, assumed to be randomly distributed in a mineral deposit. Statistics aims to make statements about the mineral deposit with respect to its size and economic viability, using measured values (samples) of the attribute. The sample values are variables such as the thickness or the grade measured at a number of locations which are conveniently placed within the deposit. The samples are themselves quite independent of each other and the results of their measurement are totally chance dependent. Thus the population parameters can be estimated without bias by means of statistics calculated from the sampling data.

It must be mentioned that the results of statistical analysis have a representativity (i.e. there is a limit to which they describe the actual natural phenomenon in the deposit) that is largely dependent on the quality of the data. Consistency and uniformity in the sampling of the population are the key to achieving reliable data. Haphazardly sampled or recorded data are a source of misleading results, no matter how sophisticated the analysis may be. Some methods for controlling the quality of the data have been discussed by Gy (1968). In general, these methods are based on sampling and sub-sampling, and the study of theoretical mixtures with different compositions.

As stated earlier, the purpose of this section is to analyse the vein width, the vein concentration and the vertical separation between successive veins with the most common univariate methods used in the analysis of sampling data obtained from exploratory programmes. The details of these methods are well presented in a number of text books and publications e.g. Krige (1962a), David (1977), Readdy *et al.* (1982), Yeoman (1982), and Davis(1986). Where appropriate, the theoretical points which are necessary and relevant in order to appreciate the approach adopted in this project are highlighted.

3.7.1 Computer Tools for the Analysis

The analysis was carried out using a program MINGEOS which was specially designed as part of this project. The program makes use of the MINPAK library of statistical and geostatistical routines developed by Forkes and O'Leary (1981) which are available at the Imperial College Computer Centre. The various routines and their functions are explained in a manual (O'Leary (1981)) available at the Mineral Resources Engineering Department, Imperial College. The routines in this library were written to suit the needs of the mineral industry with respect to ore-reserve estimation and therefore incorporates all the fundamental theories of classical statistics and geostatistics that are applicable to various mineral deposits.

3.7.2 Distributions and their Parameters

Histograms and cumulative probability curves were generated to examine and fit models to the distributions of the vein width, vein concentration and vertical separation. The program (MINGEOS) computes the following statistical parameters:

(i) the mean,

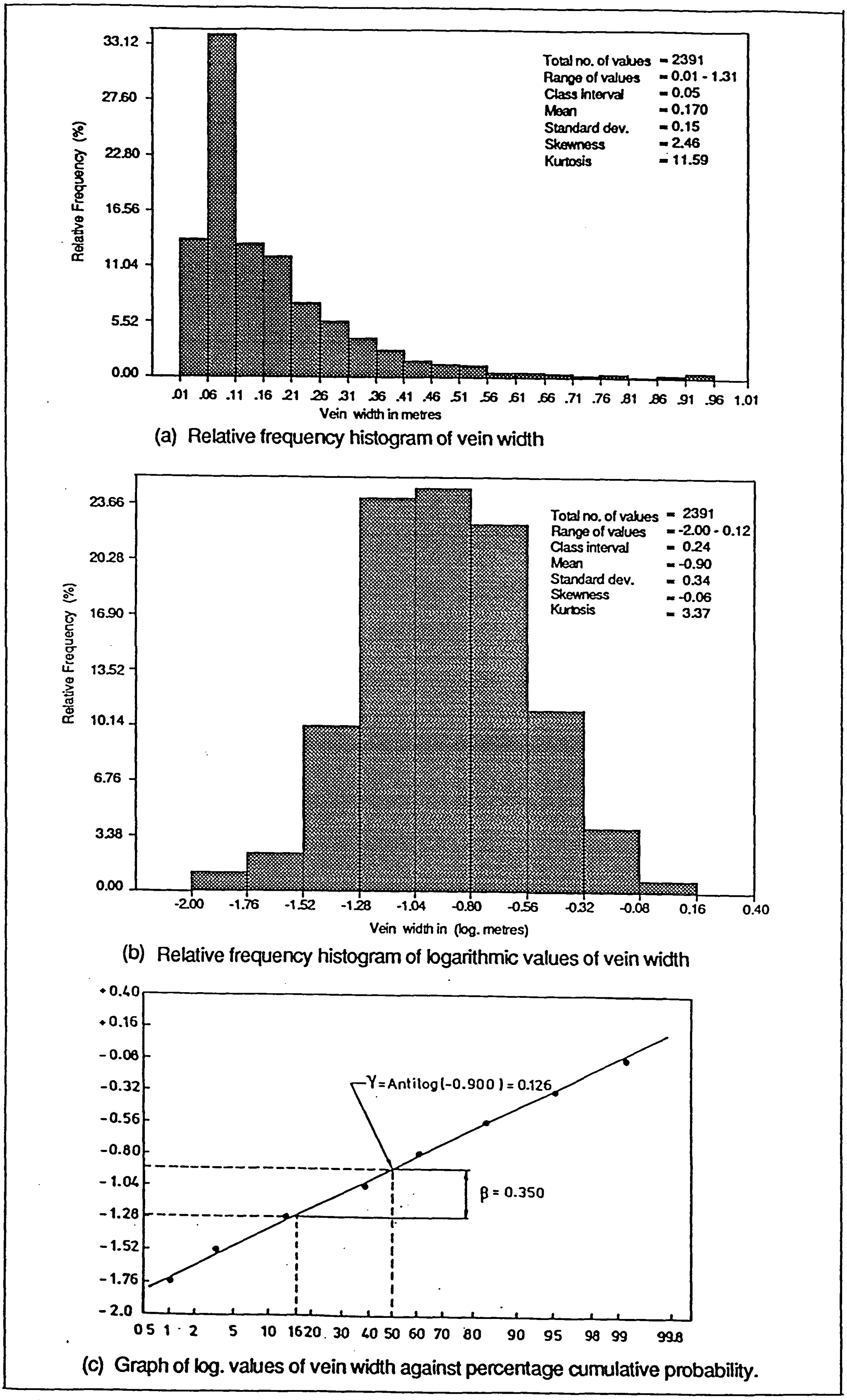
(ii) the variance,

(iii) the standard deviation,

(iii) the skewness, and

(iv) the kurtosis.

Figures 3.12a, 3.12b and 3.12c show the vein width distribution and its parameters. It can be observed in Figure 3.12a that the vein width has a positively skewed distribution. The histogram of the logarithmic values of the vein widths,



(a) Relative frequency histogram of vein width

(b) Relative frequency histogram of logarithmic values of vein width

(c) Graph of log. values of vein width against percentage cumulative probability.

Figure 3.12. Vein width distribution.

as shown in Figure 3.12b, is considered to be approximately normal on the basis of its skewness and kurtosis, the chi-square test and the curve obtained by plotting the values against their cumulative frequencies on the arithmetic probability scale.

For a perfect normal distribution, Spiegel (1972) and Yeoman (1982) show that skewness (SK), which is the third moment of the sample values about their mean, has a value of zero and the kurtosis (K), which is the fourth moment of the sample values about their mean, has a value of 3. The following mathematical expressions define skewness and kurtosis:

$$SK = \frac{\sum_{i=1}^{n} (x_i - \overline{x})^3}{nS^3} = 0 \tag{3.11}$$

and

$$K = \frac{\sum_{i=1}^{n} (x_i - \overline{x})^4}{nS^4} = 3 \tag{3.12}$$

where $\overline{x}$ is the mean, S is the standard deviation and n is the the number of samples. As can be observed, the skewness and kurtosis of the distribution of the logarithmic values of the vein width approximate to these figures.

The chi-square test is based on the sum value χ^2 which is found from the relation:

$$\chi^2 = \sum_{i=1}^{nc} \frac{(O_i - E_i)^2}{E_i} \tag{3.13}$$

where

nc = is the number of class intervals,

O = observed frequencies, and

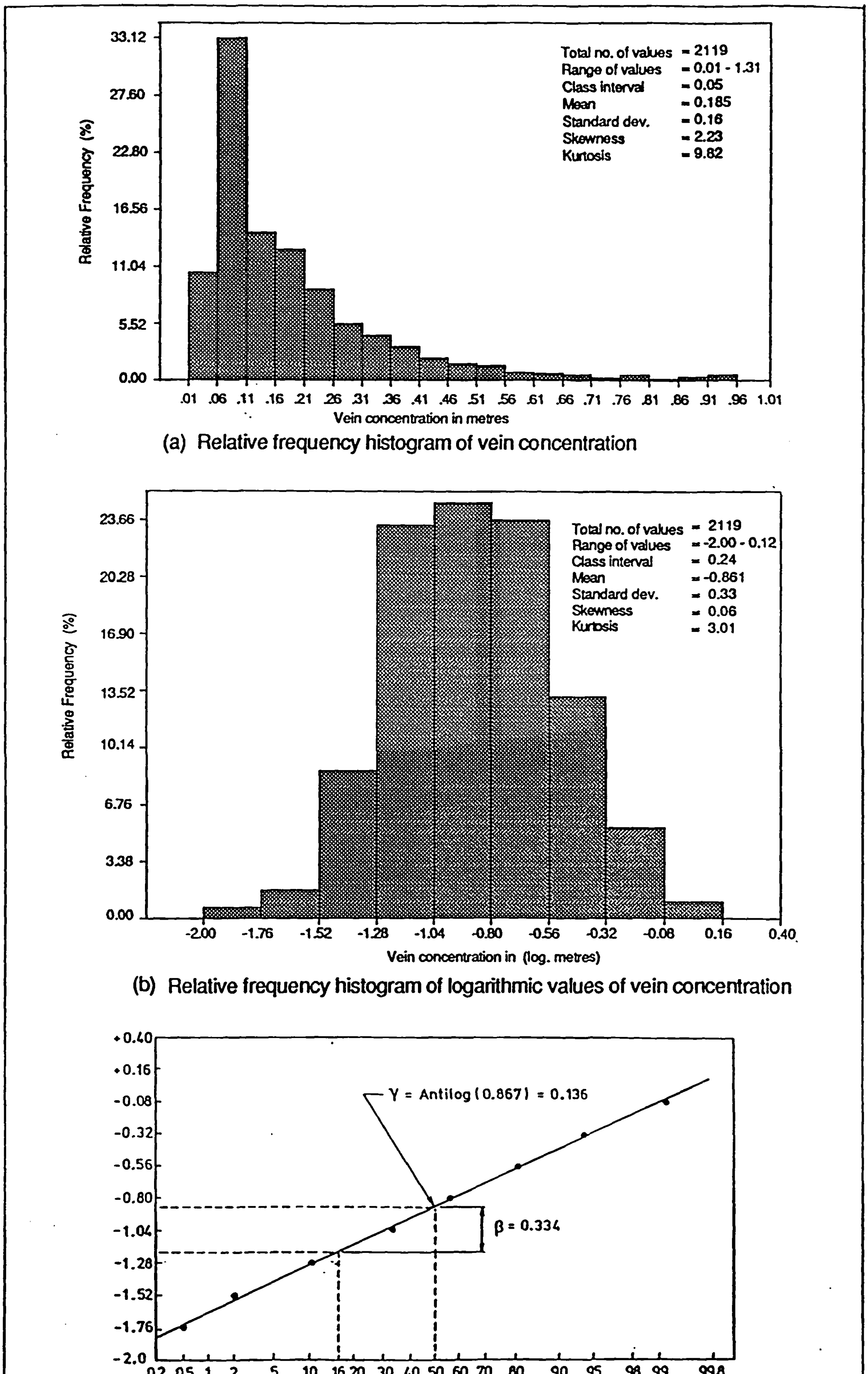
E = expected frequencies.

The chi-square value calculated for the distribution of the logarithmic values of the vein width, at 6 degrees of freedom, is 16.5 which is greater than the table value of 12.5 at 5% significance level. Therefore the null hypothesis that the distribution is normal cannot be accepted, at 5% significance level. However, this calculated value of chi-squared is less than the table value of 16.8 at 1% significance level and therefore, at this significance level, the distribution can be considered to be approximately normal. Moreover, the graph of the logarithmic values of the vein widths against their percent cumulative probabilities on the arithmetic probability scale show the characteristic straight line of normal distributions, as can be seen in Figure 3.12c. The parameters, i.e. the median γ and the standard deviation β, estimated graphically and shown in the figure are in fact close to the calculated values, giving evidence of the closeness of the graph to that of a normal distribution. It is suggested that the failure of the chi-square test, at 5% significance level, may arise from inconsistencies in the sampling practices, for example disregarding vein widths below 5 cm in some cases whilst recording them in other cases.

On the basis of all the factors discussed above, it was considered fair to fit the log-normal model to the vein width distribution. Although log-normal distributions have been observed mainly for assay values of certain mineral deposits, e.g. precious metals (Krige (1962b)) and stream sediments, it is fair to analyse the distribution of the vein width along these lines if one considers that the emplacement of the veins are related to their mineral characteristics.

The distribution and its characteristics of the vein concentration are shown in Figures 3.13a, 3.13b and 3.13c, and can be observed to be similar to those of the vein width. In Figure 3.13a, the distribution of the vein concentration can be seen also to be positively skewed. The percentage frequency of the first class (.01 - .06 m) is lower, compared to that of the vein width. Some of the values were obviously added to those of other classes during the calculation of the vein concentration. The distribution of the logarithmic values, shown in Figure 3.13b, has a skewness of 0.06 and a kurtosis value of 3.01, which are close to those of a normal distribution. The chi-square test also fails to justify this distribution to be normal, at 5% significance level. The calculated value of chi-square for this distribution, at 6 degrees of freedom and 5% significance level, is 15.8 which is higher than the table value of 12.59 at 5% significance level, but less than the table value of 16.81 at 1% significance level. Therefore at 1% significance level, the distribution can be considered to be normal. Again, the graph of the values against the percentage cumulative probability on the arithmetic probability scale shows the characteristic straight line of normal distributions (see Figure 3.13c). The vein concentration can, therefore, also be said to be approximately log-normally distributed.

Figures 3.14a, 3.14b and 3.14c feature the distribution and its characteristics of the vertical separation between successive veins and is found also to be positively skewed as can be seen in Figure 3.14a. The distribution of the logarithmic values, shown in Figure 3.14b, has, in fact, an additive constant of 2.5 and is observed to be also approximately normal. In this case, the calculated chi-square value at 6 degrees of freedom is 1.70 which is well below the

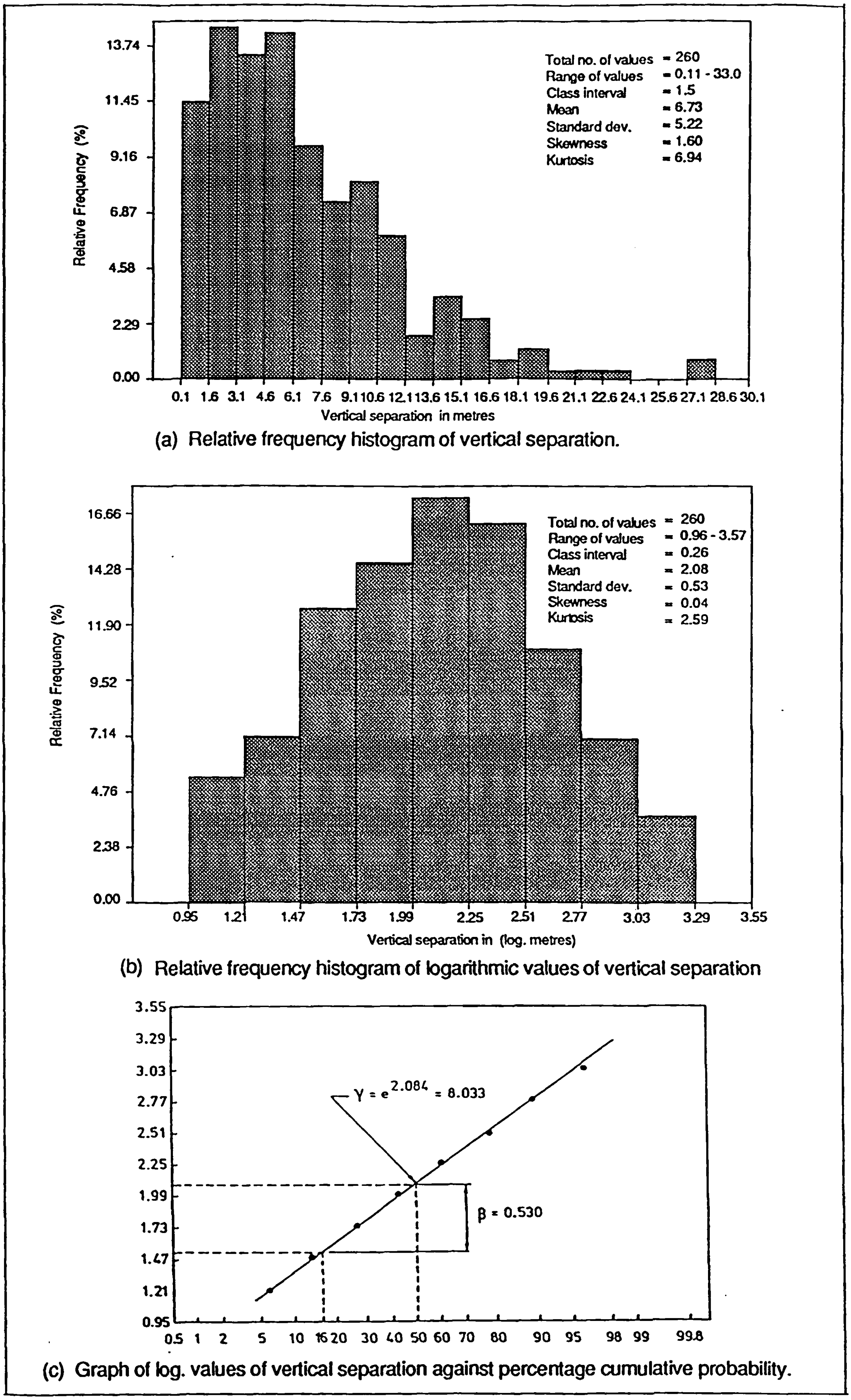


(a) Relative frequency histogram of vein concentration

(b) Relative frequency histogram of logarithmic values of vein concentration

(c) Graph of log. values of vein concentration against percentage cumulative probabilitiy.

Figure 3.13. Distribution of vein concentration.



(a) Relative frequency histogram of vertical separation.

(b) Relative frequency histogram of logarithmic values of vertical separation

(c) Graph of log. values of vertical separation against percentage cumulative probability.

Figure 3.14. Distribution of vertical separation between successive veins.

table value of 12.5 at 6 degrees of freedom and 5 % significance level. Therefore the null hypothesis that the vertical separation is log-normally distributed is acceptable by the chi-square test. Moreover, the graph of the logarithmic values against their percent cumulative probabilities is a straight line as shown in Figure 3.14c. Hence, it can be concluded that the distribution of the vertical separation is also log-normal.

Thus, all the three attributes of the deposit, i.e. the vein width, the vein concentration and the vertical separation between successive veins have distributions which can be considered to be log-normal. The inherent disadvantage of such distributions, compared to Gaussian or normal distributions, lies in the fact that their arithmetic mean tends to overestimate the actual mean. They are called log-normal distributions because the logarithmic values of the samples tend to be normally distributed, as discussed above. The equation that describes log-normal distributions is:

$$f(x) = \frac{1}{x\beta\sqrt{2\pi}}\, exp\left[-\frac{1}{2}\left(\frac{ln\,\gamma - ln\,x}{\beta}\right)^2\right] \tag{3.14}$$

According to David (1977), the parameters that describe log-normal distributions are:

(i) The median of the distribution, which is $\gamma = e^{\alpha}$, where α is the average of the logarithmic sample values.

(ii) The standard deviation of the logarithmic values which is β. The value of β can also be evaluated from the arithmetic probability scale as the difference of the log-values associated with the interval 16% to 50%, or 50% to 84%.

Theoretically the mean, m, can be estimated using the following relation:

$$m = \gamma e^{\beta^2/2} = e^{\alpha+\beta^2/2} = \gamma\, exp\left(\frac{\beta^2}{2}\right) \tag{3.15}$$

Equation (3.15), however, is a valid relation between real values and not between the estimated parameters (David (1977)). Probably a better way to estimate the mean of log-normal distributions is by using the Sichel's t-estimator technique (Sichel (1966)). This method requires the knowledge of the median, γ. A factor is then read from Sichel's Table A, which is available in some books on Geostatistics, e.g. in *Geostatistical Ore Reserve Estimation* (David (1977, pp. 20-21)). This factor is a function of the number of samples, n, and the natural logarithmic variance V. The t-estimator gives the mean as the product of the median and the factor. Sichel's Table B (David (1977, pp. 36-37)) provides the lower and upper limit factors which must be multiplied by the mean to obtain the lower and upper limits of the mean, at 90% confidence level. It should be noted that Sichel's tables feature natural logarithmic variances and therefore caution must be taken to convert any log_{10} parameters into those of ln. The natural logarithmic variance V and the log_{10} variance $(\beta')^2$ have the relation: $V = 5.3019(\beta')^2$. The median $\gamma = antilog_{10}(\alpha')$ if α' is the average of the base 10 logarithmic sample values.

Table 3-2 shows the means of the distributions estimated using Sichel's t-estimator. The mean of the vein concentration is found to be only about 0.01 m higher than that of the vein width, but this difference can obviously mean a significant increase in mineable potential.

Table 3-2. Estimated means of vein width,vein concentration and vertical separation between successive veins.

Distribution	Lower mean	Estimated mean	Upper mean
Vein widths in the Study Area	0.157 m	0.165 m	0.175 m
Vein concentration	0.170 m	0.178 m	0.186 m
Vertical Separations	6.367 m	6.607 m	6.87 m

3.8 Summary

In the absence of distinguishing lithological or mineral characteristics by which the vein intersections can be extrapolated, or by which zones of common characteristics can be delineated, the procedures and some of the the subjective judgements used by the mine geologists to identify areas to be developed have been translated into simple rules and coded into a program MBANDS. This is an interactive program that generates a section showing the vein intersections. The user is allowed to select vein intersections which are then checked against a set of rules that may or may not permit their connection to enable possible mineable areas to be delineated. Thus, it emulates the interpolation and delineation process at the mine. Its advantage lies in the fact that it is fast and can be used to quickly define possible mineable ground and hence assist in mine planning. However, because of the capricious nature of the veins, different users can arrive at different ways of delineating the potential mineable ground. Obviously, a method that provides an objective and reproducible result would be more useful; the suggested approach to achieve this is presented in Chapters 4 and 5.

It has been demonstrated that the vein occurrences have Markovian properties and hence by superimposing a grid on the sections showing the vein intersections it is possible to identify data matrices representing the different geological states, i.e. the quartz veins and the schistose country rock. The resulting matrix is diagnostic of the pattern of vein occurrences in the plane of the section and can be analysed to establish the probabilities of vein occurrences in a given direction at distances corresponding to the resolution of the grid. The program, MAK2, used for this analysis is based on Switzer's concepts of extending the Markov phenomenon to planar patterns. The program also has the facility to allow simulated patterns to be generated.

The results of such an analysis carried out during the research show that once a vein is intersected there is a 0.5 probability of it continuing to be present through a distance of 12 m, or that another vein will be intersected through that distance, in the near horizontal direction, but only a 0.07 probability in the vertical direction. This observation is useful in that it provides a good guide as to when to abandon the development of a vein which suddenly disappears — an event that is not uncommon at Panasqueira. Upon this evidence, it is suggested that if, for example, during the development of a reef drive the quartz vein should suddenly disappear, the development work should continue for at least 12 m before being abandoned.

The vertical separation between successive veins has been analysed statistically and found to be log-normally distributed. Using Sichel's t-estimator, the mean value has been estimated to be 6.6 m with a lower limit of 6.4 m and an upper limit of 6.9 m. This information is valuable at Panasqueira where a room and pillar mining method is employed in extracting the veins. Veins that are very close can be mined together. On the other hand, the vertical separation between veins that will be mined separately must be enough to provide the necessary support in the form of crown and sill pillars. A mean vertical separation of 6.6 m suggests, for example, that room and pillar mining, as pertains at Panasqueira, will be successful if and only if crown and sill pillars measuring 6.6 m or less are stable.

The vein widths intersected within the study area have also been statistically analysed and found to be approximately log-normally distributed. Again, using Sichel's t-estimator, the mean vein width has been estimated to be 0.165 m with a lower limit of 0.157 m and an upper limit of 0.175 m. If the veins are to be mined individually, as is the case at present, then an economically important aspect of mineralisation is the average quantity of the quartz vein which is available at an expected grade. Since at Panasqueira the grade is unknown until the veins are exposed, and the average quantity of veinage is directly proportional to the vein width, the mean vein width becomes a dominant factor.

It has been noted in Chapter 2 that only veins with widths of 0.20 m or more are currently mined at Panasqueira. Moreover, the average width of veins mined is reported to be 0.30 m. These two facts clearly indicate that the veins selected for mining are picked from the tail of the positively skewed vein width distribution. Thus, it can be concluded that the majority of veins are left unmined simply because they are thin. The fact still remains, however, that if these veins can be mined together, then their aggregate width, and hence the quantity of quartz obtainable, can be equal to, or even greater than, that obtained by mining single veins.

A more understandable variable that has been derived and considered as constituting a measure of mineable potential at Panasqueira is the *vein concentration*. It has been defined as the vein width accumulation per 2 m height of ground. The word concentration is, therefore, not meant to suggest grade or any unit measure of the mineral content, i.e. saleable product content of the ground. However, an *a priori axiom*, at the mine, holds that the more quartz mined the more wolframite obtainable. Consequently, the aggregate quartz content of the ground mined is of prime importance, at least, until the grade is known.

The vein concentration has been statistically analysed and also found to be approximately log-normally distributed. Its mean value has been estimated to be 0.178 m $\approx$ 0.18 m with a lower limit of 0.170 and an upper limit of 0.186 m. Since the minimum width of veins included in the mine's geological reserves is 0.18 m, there is no reason why the ground containing a vein concentration of at least 0.18 m cannot also be included in the geological reserves. Thus, more ground is expected to become mineable if the vein concentration is considered as the measure of mineability rather than the vein width.

CHAPTER 4

GEOSTATISTICAL MODELLING

4.1 Introduction

It was mentioned in Chapters 2 and 3 that the amount of mineral (wolfram) obtainable from the deposit is directly proportional to the quantity of quartz mined, and that the quartz must be exposed before the actual mineral potential can be measured.

The objective of this chapter is to estimate the likely volumes or tonnages of quartz in a given mining unit, i.e. a block, in the study area using geostatistical methods. Since the tonnages of quartz are also directly related to the vein width, or more precisely to the aggregate vein width, the primary aim is to estimate the aggregate vein width in the block.

In order that the final results of this project can be cross-checked with actual stope production records, it is proposed to use the exact blocks predetermined by the mine. It will be recalled from Chapter 2 that these blocks are oriented at 30.06° to the national grid north and measure 50 m by 100 m in the horizontal plane. The block height is taken to be 2 m which is the recommended stoping height (see Chapter 6). The selected block size therefore represents the level of resolution that is vital for effective mine planning.

As explained in Section 3.4.1, ***the aggregate vein width per 2 m height of ground is, by definition, the vein concentration.*** Thus, we are talking about estimating the vein concentration in the mine's pre-ordained blocks. Geostatistics is the method selected as the estimation technique because it is generally accepted in the mineral industry to be 'superior' to other traditional methods since it provides a basis for quantifying the geological concepts of:

(i) the area of influence around a sample,

(ii) the continuity of mineralisation within an orebody, and

(iii) changes in mineralisation according to the trend of an orebody.

The basis of Geostatistics is the theory of 'regionalised variables' developed by Matheron (1971). A variable is said to be regionalised when its value depends on neighbouring values that are distributed in two- or three-dimensional space (Barnes (1980)). There is an underlying assumption that the difference between values at two sample points is related to the distance between them and their orientation.

In the particular case of this project, it is assumed that if there are two vein intersections some distance apart, there is some form of physical continuity between them. This assumption does not imply that individual veins are continuous between intersections, but:

(i) that the general nature of the deposit is physically continuous;

(ii) that faulting, for example, is unlikely to have removed significant volumes of mineralised ground from between boreholes; and

(iii) that the general physical nature of the vein swarm does not change radically.

These assumptions are justified on the grounds of previous observation of the mineralisation exposed in underground workings. It is necessary, however, to justify the underlying assumption, i.e. that the value of a vein concentration depends on the neighbouring values, before applying geostatistical methods.

The semi-variogram is a plot of the semi-variance between sample values at increasing distances apart, and is used to establish the interrelation among samples. It is necessary to verify whether the difference between any two values of the vein concentration is related to their distance apart and their orientation. For completeness and to satisfy curiosity, the vein width will also be examined to establish whether it is a regionalised variable. Thus, the work carried out in this chapter has two main parts:

(i) semi-variogram analysis of the vein width and the vein concentration, and

(ii) a block by block estimation of the vein concentration, using linear kriging.

A full account of geostatistical theories can be found in David (1977), Journel and Huijbregts (1978), Rendu (1981) and Clark (1984). A few essential definitions and equations that are relevant to this work are given where appropriate.

4.2 Semi-variogram Analysis

The semi-variogram is the function which numerically represents the spatial variability among sample values from a deposit. Theoretically, it is defined as one-half of the variance of the difference between sample values at points separated by distance h. The variance calculation is simplified by the assumption that the trend in the deposit, or at least in the area of interest, is negligible (Clark (1984, p. 10)). The semi-variance is calculated as one-half of the average square difference between the sample values, that is:

$$\gamma(h) = \frac{1}{2N} \sum_{i=1}^{N} \left[X_i - X_{(i+h)}\right]^2 \tag{4.1}$$

where

$\gamma(h)$ = semi-variance between sample pairs,
X_i and $X_{(i+h)}$ = the sample pairs
N = the number of pairs, and.
h = the distance between sample pairs in a specified direction.

The three main steps involved in the process are:

(i) construction of an experimental semi-variogram by plotting the semi-variance $\gamma(h)$ against distance (h) in a given direction within the deposit (the semi-variance is calculated using Equation (4.1));

(ii) fitting of a model semi-variogram (Table 4-1 summaries the various models that can be fitted to the experimental semi-variogram); and

(iii) cross-validation of the model semi-variogram, using point kriging.

Table 4-1. Semi-variogram models.

Model type	Equation	Comment
Spherical	$\gamma(h) = C_0 + C\left(\frac{3}{2}\frac{h}{a} - \frac{1}{2}\frac{h^3}{a^3}\right) \quad \forall\, h < a$ $= C_0 + C \quad \forall\, h \geq a$	Found to give a satisfactory representation of the semivariogram of many different deposits.
Linear	$\gamma(h) = Ah + B$	The simplest model without a range.
de Wijsian	$\gamma(h) = A\,ln(h) + B$	An extension of the linear model and found in some hydrothermal deposits.
ah^{λ} model	$\gamma(h) = ah^{\lambda}$	Observed in elevation semi-variograms or in the study of mill feed variability.
Exponential	$\gamma(h) = C_0 + C\left[1 - exp(-\frac{h}{a})\right]$	The slope at the origin is C/a. This model is rare in mineral deposits.
Gaussian	$\gamma(h) = C\left[1 - exp(-\frac{h^2}{a^2})\right]$	The curve is parabolic near the origin and the tangent is horizontal at the origin.
Parabolic	$\gamma = \frac{1}{2}a^2h^2$	Observed when there is a linear drift.
Random	$\gamma(h) = S^2$	This model has no continuity, indicating there is a high degree of randomness in the variable distribution. $\gamma(h)$ is then equal to the statistical variance.
Hole-effect	$\gamma(h) = C\left[1 - \left(\frac{sin(ah)}{ah}\right)\right]$	This model has a periodic behaviour and is observed when there is a succession of rich and poor zones.

The symbols stand for the following :

C_0 = nugget variance, C = transition variance, h = distance between sample pairs, a = range, A, and B = constants, λ= slope, and S^2 = statistical variance of sample population.

(Sources: Blais and Carlier (1968); David (1977); Rendu (1981))

4.2.1 Experimental and Model Semi-variograms

In this project, semi-variances were computed by making calls within the program MINGEOS to two subroutines, VARDIR and VARISO, from the MINPAK library of geostatistical routines developed by Forkes and O'Leary (1981). VARDIR was used to compute experimental semi-variance at constantly increasing distances, known as the lag, in a specified direction. VARISO was used to compute isotropic semi-variances at a given lag distance.

The results of the computation, which are normally written in a standard format to a file to provide a permanent record, were plotted at a graphics terminal with the AUTOGRAPH package (Raby (1984)) available at the Imperial College Computer Centre.

Semi-variograms were constructed for both the vein width and the vein concentration within the study area, at bearings of 90°, 135°, 180° and 225° and a lag interval of 10 m. For each of the directions, a 23° tolerance angle on either side of the directional line, known as the *angle of regularisation*, was used.

Figure 4.1a shows each of the experimental semi-variograms, in the various directions, for the vein widths in the study area. It can be seen that none of the curves start from the origin, indicating the presence of a nugget variance. Each curve can be continued to intercept the semi-variance axis, at a lag distance $h = 0$, between 0.012 and 0.016 m^2 corresponding to the nugget variance. From this intercept value, each curve rises to a value between 0.020 and 0.024 m^2 at a lag distance of about 50 m and then appears to stabilise. There is thus a clear indication of the existence of a geostatistical structure, in the deposit, which is similar in all directions, reflecting an isotropic behaviour. The behaviour of these experimental semi-variograms corresponds to that of a spherical model with a nugget variance. The equation describing this model is given in Table 4-1.

Figure 4.1b shows each of the directional and experimental semi-variograms constructed for the vein concentration in the study area. It can be seen that each of the curves can be continued to intercept the semi-variance axis at some value above zero, between 0.008 and 0.020 m^2 corresponding to the nugget variance. From this value, each curve rises steadily to a value between 0.024 and 0.030 m^2 corresponding to a lag distance of approximately 50 m and then appears to stabilise. Evidently, therefore, there exists a geostatistical structure in the deposit which is similar in all directions, and reflects an isotropic behaviour. In

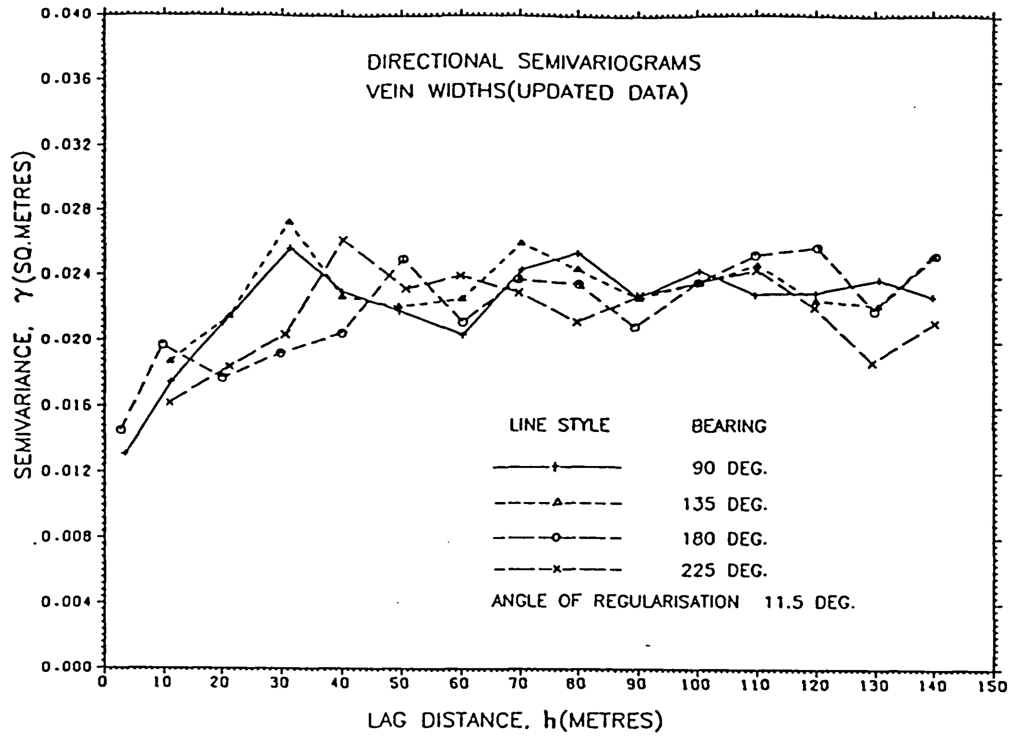


Figure 4.1a. Directional Semi-variograms of Vein Width.

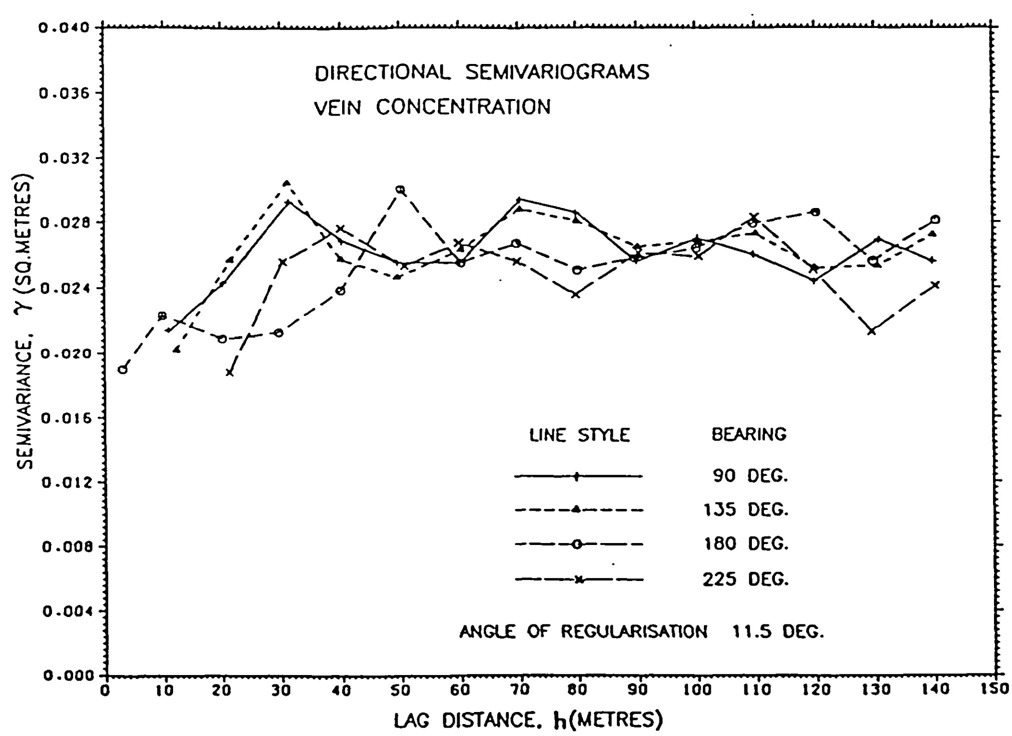


Figure 4.1b. Directional Semi-variograms of Vein Concentration.

this case also, the behaviour of the semi-variograms corresponds to that of the spherical model.

Isotropic semi-variograms were constructed at the same lag interval of 10 m for both the vein width and the vein concentration. In this case, the angle of regularisation was set at 90° and, thus, sample pairs at all bearings were used. The isotropic semi-variogram therefore summaries the observed behaviour of all the directional semi-variograms; it is used here as the basis for fitting a model to the deposit.

It is clearly not appropriate to consider a vein deposit of this type to be truly isotropic other than in the horizontal plane, as it consists of alternations of schistose country rock and quartz veins. The implication of the isotropic behaviour is that the character of veinage, as a whole, does not change with direction. The isotropy exists both within the individual veins and for the orebody as a whole. Since mineralisation is present in the near-horizontal swarm of veins and the deposit consists of alternations of schistose country rock and quartz veins, the observed isotropic behaviour of the semi-variograms must be taken to apply in the horizontal plane.

Figure 4.2a shows the isotropic semi-variogram constructed for the vein width as well as the spherical model with a nugget variance fitted to it. Initially, the following values were read from the graph and assigned to the model:

$$\begin{aligned} \text{a nugget variance } C_0 &= 0.0015\ m^2; \\ \text{a transition variance } C &= 0.007\ m^2; \\ \text{and a range } h &= 47\ m. \end{aligned}$$

Figure 4.2b features the isotropic semi-variogram constructed for the vein concentration as well as the spherical model with a nugget variance fitted to it. In this case, the following initial values were read from the graph and assigned to the model:

$$\begin{aligned} \text{a nugget variance } C_0 &= 0.017\ m^2; \\ \text{a transition variance } C &= 0.007\ m^2; \\ \text{and a range } a &= 48\ m. \end{aligned}$$

David (1977) attributes the nugget variance to poor analytical precision or, more often, highly erratic mineralisation at low scale. *The fact that the Panasqueira veins can thin out, or split up abruptly, giving rise to erratic vein widths, may explain the presence of the nugget effect.* This erratic behaviour

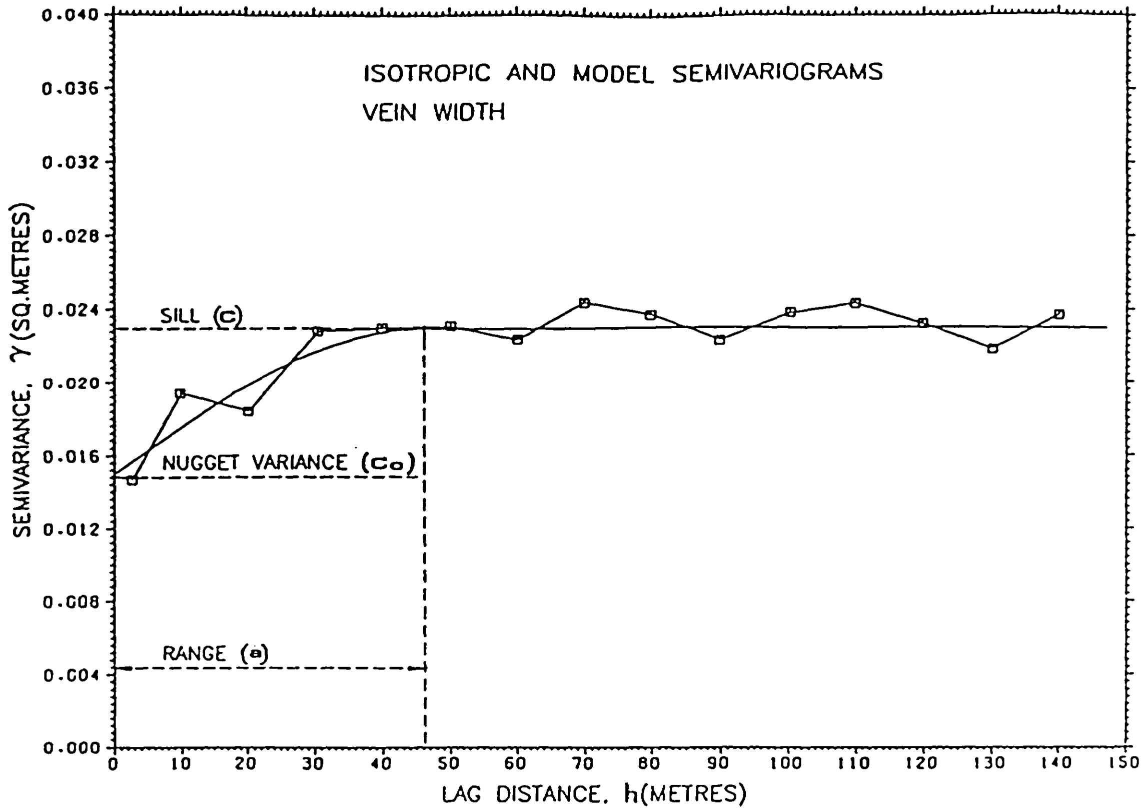


Figure 4.2a. Isotropic Semi-variogram of Vein Width.

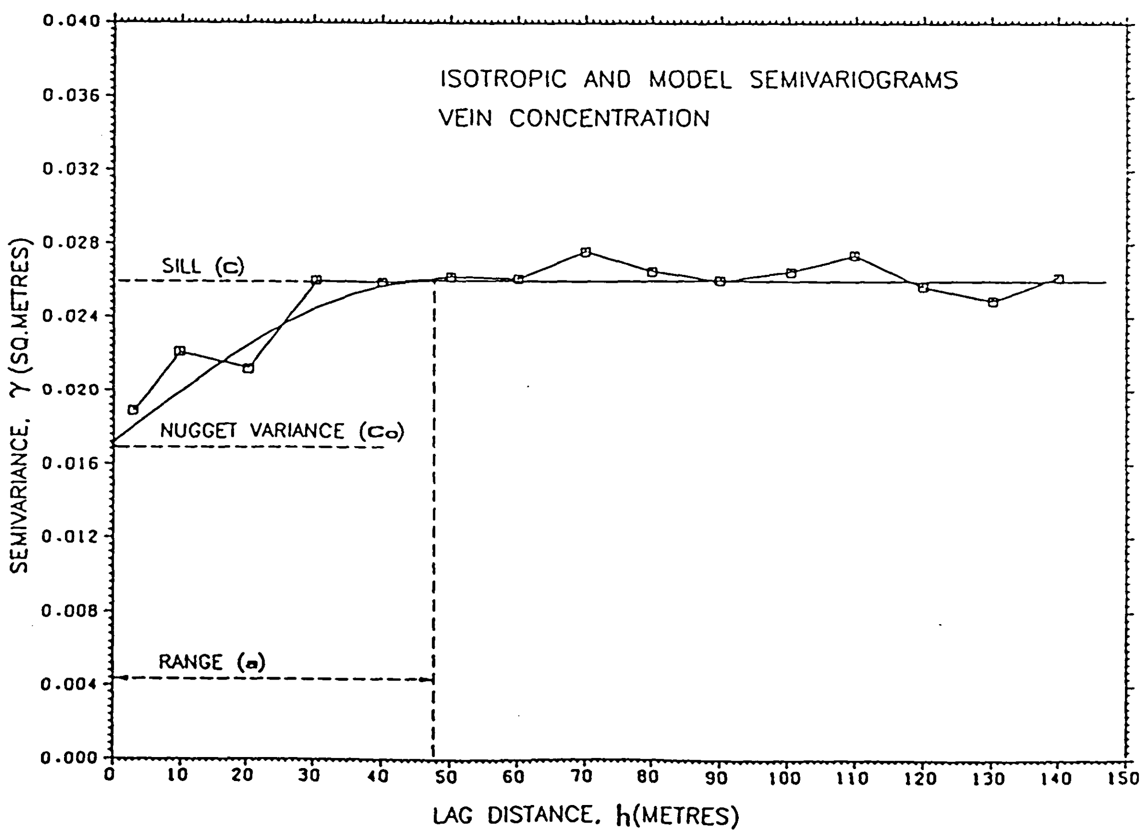


Figure 4.2b. Isotropic Semi-variogram of Vein Concentration.

of the veins has genetic origins, i.e. the random dilation of pre-existing joint fractures caused by tectonic and hydraulic forces.

4.2.2 Model Cross-validation

The preliminary selected models were cross-validated, using point kriging. In this process, each sample is removed in turn from the data set and its value is estimated with the neighbouring sample values. The agreement between the estimated values and the actual values is used in validating the model. According to O'Leary (1981), the requirements for a good fit are:

(i) the mean of kriged estimates must equal the mean of actual values,

(ii) the ratio of estimation variance (E_v) to kriging variance (K_v) must be equal to one, and

(iii) the mean algebraic error of estimation must be close to zero.

In practice, the $E_v : K_v$ ratio is acceptable if its value lies between 0.95 and 1.05 (O'Leary, personal communication). In addition to these criteria, the model must show a reasonable graphical fit to the experimental semi-variogram. By varying the initially selected values of the nugget variance, transition variance and range, with the objective of obtaining a unitary ratio of $E_v : K_v$, the best model can be fitted.

The smooth curve in Figure 4.2a shows the final model fitted to the experimental semi-variograms with respect to the vein width. It has the following parameters:

$$\begin{aligned} \text{a nugget variance } C_0 &= 0.0159\ m^2; \\ \text{a transition variance } C &= 0.0064\ m^2; \\ \text{and a range } a &= 46.5\ m. \end{aligned}$$

The equation, therefore, which describes the spatial variability among the vein widths in the study area is:

$$\begin{aligned} \gamma(h) &= 0.0159 + 0.0064\left(\frac{1.5h}{46.5} - \frac{0.5h^3}{46.5^3}\right) \qquad && \forall\ h < 46.5 \\ &= 0.0223 && \forall\ h \geq 46.5 \end{aligned} \tag{4.2}$$

In Figure 4.2b, the final model fitted to the experimental semi-variogram with respect to the vein concentration is also shown as a smooth curve with the following parameters:

$$\begin{aligned} \text{nugget variance } C_0 &= 0.0172\ m^2; \\ \text{transition variance } C &= 0.0088\ m^2; \\ \text{and a range } a &= 47.5\ m. \end{aligned}$$

Similarly, the equation which describes the spatial variability among the vein concentrations is:

$$\begin{aligned} \gamma(h) &= 0.0172 + 0.0088\left(\frac{1.5h}{47.5} - \frac{0.5h^3}{47.5}\right) \qquad && \forall\ h < 47.5 \\ &= 0.026 && \forall\ h \geq 47.5 \end{aligned} \tag{4.3}$$

Table 4-2 shows the resulting cross-validation indices which illustrate the goodness of fit of the semi-variogram models described previously. In this table, $Z - Z^*$ is the difference between sample value and the corresponding estimated value. For both attributes of the deposit, vein width and vein concentration, Table 4-2 shows acceptable indices for the fitted models. ***It is therefore established that there is recognisable geostatistical structure in the study area with respect to both the vein width and the vein concentration.***

Table 4-2. Indices of cross-validation.

Index	Vein width	Vein concentration
Number of points available for kriging	2391	2119
Number of points kriged	2330	2079
Points not kriged due to too few samples	61	40
Points not kriged due to too many samples	0	0
Mean of points used for kriging	0.1702	0.1854
Mean of kriged estimates	0.1707	0.1859
Variance of points used for kriging	0.0233	0.0254
Mean kriging variance (K_v)	0.0201	0.0243
% Error due to parameters	-0.37	-0.27
Mean of ($Z - Z^*$)	-0.0006	-0.0005
Mean estimation variance (E_v)	0.0210	0.0250
Ratio of $E_v : K_v$	1.04	1.03

Krige (1978) suggests that if the distribution of observed samples used in calculating block estimates within a deposit is log-normal, as is the case with both the vein width and vein concentration, then construction of the semi-variogram as well as block kriging should be based on the log-transformed values of the samples. The resulting block estimates can then be back-transformed using the relation:

$$Z_v^* = exp\left(Z'_v + \frac{(\sigma'_k)^2}{2}\right) \tag{4.4}$$

where

Z_v^* = expected block estimate,

Z'_v = kriged block estimate using log-transformed sample values, and

$(\sigma'_k)^2$ = kriging variance.

Rendu (1981, p. 21) explains that this procedure is necessary only when the sill value of the semi-variogram constructed with samples in different parts of the deposit increases proportionally with the square of the local mean. This is known in geostatistics as the proportional effect. The usual practice to overcome the proportional effect is to construct the semi-variogram using the log-normalised variable. Another suggested approach is to use the average of 'relative semi-variograms', i.e. semi-variograms divided by the square of the local mean, however Clark (1984, pp. 34–37) has shown that these methods may yield erroneous results.

In the particular case of this project, log-transformation had not been considered because semi-variograms constructed for subsets of the data conferred the absence of a proportional effect. Moreover, semi-variograms based on log-transformed values of both the vein width and the vein concentration showed no geostatistical structure in the deposit. These semi-variograms are shown in Appendix 4. In contrast, the semi-variograms of the observed, untransformed values of both variables show good geostatistical structure. Cross-validating the fitted models is a way of ensuring that adequate estimates can be made. After all, "the construction of the experimental semi-variogram and the estimation procedures used in geostatistics do not depend on the type of sample distribution" (Clark (1984, p. 35)).

The two semi-variogram models presented in this section are therefore considered adequate for block kriging. Both models may be used to estimate the likely volumes of quartz, but from the explanations given in Chapter 3 and earlier in this chapter, the vein concentration is considered to give a better representation of what is mined than the individual vein widths. Hence the estimation which follows in Section 4.4 is based on the vein concentration model.

4.3 Block Kriging Process

Kriging is the name given to the geostatistical interpolation technique which calculates the best estimates for *in situ* block values by developing a set of linear weighting coefficients that are applied to sample values in the vicinity of the block. The position of the samples with respect to the block and the continuity of mineralisation are taken into account.

The theory of the kriging equation is described in detail by David (1977), Journel and Huijbregts (1978), Rendu (1981) and Clark (1984). The kriging estimator has the general form:

$$Z_v^* = \sum_{i=1}^{n} a_i X_i \tag{4.5}$$

where Z_v^* is the best estimated value of the true but unknown value Z_v of the *in situ* block of volume V; X_i is the value of a sample in the vicinity of the block; a_i is the weighting coefficient assigned to X_i; and n is the selected number of samples in the vicinity of the block to be used in the estimation. For Z_v^* to be the best estimated value, two conditions must be satisfied:

(i) the estimation variance must be a minimum, i.e. $\sigma_E^2 = E\left(Z_v^* - Z_v\right) =$ a minimum; and

(ii) the estimation must be unbiased, i.e. $\sum_{i=1}^{n} a_i = 1$.

The estimation variance can be expressed in terms of the semi-variance (Rendu (1981)):

$$\sigma_E^2 = 2\sum_{i=1}^{n} a_i \overline{\gamma}\left(X_i, V\right) - \sum_{i=1}^{n}\sum_{j=1}^{n} a_i a_j \overline{\gamma}\left(X_i, X_j\right) - \overline{\gamma}\left(V, V\right) \tag{4.6}$$

where

$\overline{\gamma}\left(X_i, V\right)$ = the average semi-variance between sample X_i and every point in block V,

$\overline{\gamma}\left(X_i, X_j\right)$ = the average semi-variance between all pairs of points in the sample set, and

$\overline{\gamma}(V,V)$ = the average semi-variance between all pairs of points in block V.

To minimise the error of estimation, under unbiased constraint, a Lagrange multiplier μ is introduced and a function G is defined as follows (Rendu (1981)):

$$G(a_1, a_2, \ldots, a_n, \mu) = \sigma_e^2 - 2\mu\left(\sum_{i=1}^{n} a_i - 1\right) \tag{4.7}$$

The kriging weighting coefficients a_i are solutions to the following equations:

$$\begin{cases} \dfrac{\partial G}{\partial a_i} = 0 \quad \text{for } i = 1, 2, \ldots, n \\ \dfrac{\partial G}{\partial \mu} = 0 \end{cases} \tag{4.8}$$

This is a system of $n+1$ equations, where n is the number of samples and can be written as:

$$\begin{cases} \sum_{j=1}^{n} a_i \overline{\gamma}(X_i, X_j) + \mu = \overline{\gamma}(X_i, V) \quad \text{for } i = 1, 2, \ldots, n \\ \sum_{j=1}^{n} a_j = 1 \end{cases} \tag{4.9}$$

This system of equations is the general kriging system for an unknown mean. Multiplying the first n equations by a_i ($i = 1, 2, \ldots, n$ respectively) and by summing the results so obtained, the following is arrived at:

$$\sum_{i=1}^{n}\sum_{j=i}^{n} a_i a_j (X_i, X_j) + \mu = \sum_{i=1}^{n} a_i \overline{\gamma}(X_i, V) \tag{4.10}$$

From Equations (4.6) and (4.10) the minimum estimation variance which is known as the kriging variance σ_K^2 can be expressed as:

$$\sigma_K^2 = \sigma_E^2 = \sum_{i=1}^{n} a_i \overline{\gamma}(X_i, V) + \mu - \overline{\gamma}(V, V) \tag{4.11}$$

This equation is the general expression of the kriging error for an unknown mean. The kriging system and the kriging error can be written in a matrix form as:

$$[S][A] = [D] \tag{4.12}$$

with the solution

$$[A] = [S]^{-1}[D]$$

where

$$S = \begin{bmatrix} \overline{\gamma}(X_1,X_1) & \overline{\gamma}(X_1,X_2) & \dots & \overline{\gamma}(X_1,X_n) & 1 \\ \overline{\gamma}(X_2,X_1) & \overline{\gamma}(X_2,X_2) & \dots & \overline{\gamma}(X_2,X_n) & 1 \\ \vdots & \vdots & \ddots & \vdots & \vdots \\ \overline{\gamma}(X_n,X_1) & \overline{\gamma}(X_2,X_n) & \dots & \overline{\gamma}(X_n,X_n) & 1 \\ 1 & 1 & \dots & 1 & 0 \end{bmatrix}.$$

$$A = \begin{bmatrix} a_1 \\ a_2 \\ \vdots \\ a_n \\ \mu \end{bmatrix} \qquad \text{and} \qquad D = \begin{bmatrix} \overline{\gamma}(X_1,V) \\ \overline{\gamma}(X_2,V) \\ \vdots \\ \overline{\gamma}(X_n,V) \\ 1 \end{bmatrix}$$

It is now possible to solve the problem since all terms from $\overline{\gamma}$ are derived from the semi-variogram.

It follows from Equation (4.11) that the following factors are taken into account in estimation (Brooker (1979)):

(i) the variability of mineralisation, that is the characteristics of the semi-variogram: all terms with $\overline{\gamma}$,

(ii) the geometrical relation between samples and a block: $\sum_{i=1}^{n} a_i(X_i,V)$,

(iii) the geometry of the block to be estimated: $\overline{\gamma}(V,V)$, and

(iv) the configuration of the samples: $\sum_{i=1}^{n}\sum_{j=1}^{n} a_i a_j \overline{\gamma}(X_i,X_j)$.

The adjustment of these factors will change the magnitude of the estimation variance. *The following will decrease the variance:-*

(i) An increase in the block size — geological and mining constraints put a limit to this increase. Royle (1977) suggests that the ideal size of a block should be that of a future stope and that blocks should possibly be of the same size so that they are estimated with the same magnitude of error.

(ii) An increase in the number of samples in or around the block. This can be achieved when drilling density is high so that samples can be taken more often, but the cost of drilling limits the number of holes that can be drilled. In estimation, the minimum number of samples required to krige a block is recommended to be at least 4 (O'Leary, personal communication).

(iii) A reduction in the average distance of samples from the block's centre. David (1977, p. 283) gives a 'rule of thumb': *the minimum size of a block should not be less than $\frac{1}{4}$ of the drilling distance.*

A selected block size of 100 m by 50 m by 2 m satisfies the above conditions because the blocks are the eventual stoping blocks. It can be recalled from Chapter 2 that the boreholes are drilled from an orthogonal network of drives and cross-cuts. The drives are 100 m apart and the cross-cuts (called panels) are 50 m apart. The current drilling spacing is 25 m. Therefore, the selected block size ensures that each block is, in fact, surrounded by a number of drilled holes and hence has vein widths measured on all sides.

4.4 Estimation Procedure and Results

In this project, the block estimates of vein concentration, i.e. the aggregate vein width per 2 m height of ground, were calculated using linear kriging by making calls within the program MINGOES to kriging routines in the MINPAK library of geostatistical routines. The following are the main subroutines employed and their functions:-

(i) EXTV3D2 : calculates the directional 3-D variance of a block.

(ii) SLECT3D : selects the nearest maximum number of points to the block centre and within specified boundaries or search radii.

(iii) PNKM3D1 : establishes a 3-D block kriging matrix.

(iv) F04AAF : inverts the semi-variogram type kriging matrix and calculates the weighting coefficients.

(v) SOLVE2: calculates the kriged estimates and kriging variance.

The area of the Panasqueira mine delineated for study was divided into a series of contiguous blocks measuring 100m by 50 m by 2 m to form a model comprising a total of 2160 blocks. As has been mentioned earlier in this Chapter, the selected block size corresponds to that used at the mine and is considered to be at a level of resolution suitable for effective mine planning.

The estimation was carried out using the exact blocks predetermined by the mine; in fact, the centre co-ordinates as well as the corner co-ordinates of each block within the study area were obtained from the mine. This enables results to be cross-checked with actual stoping production records. These blocks are oriented at 30.06° to the national grid north. Figure 4.3 shows the plan of these blocks.

Semi-variogram analysis of both the vein width and vein concentration has shown that values measured at particular points bear some relation to values around those points. Since "mineralisation is present almost entirely in the more or less horizontal veins" (Thadeu (1980)), and the deposit consists of alternations of schistose country rock and quartz veins, the isotropic continuity demonstrated by the semi-variograms is taken to apply locally within the near-horizontal plane. Consequently the following measures were taken to ensure that a block is estimated with samples selcted from within a near-horizontal plane and that a block's value is estimated only when there are sufficient number of samples in and/or around it:-

(i) To ensure that samples used to estimate a block's value are selected within the limits of the general rise or fall of the veins and are therefore more likely to belong to the same vein, an angular search criterion was incorporated to define around a block a bi-conical volume, from which samples could be selected (See Figure 4.4). In this figure, ABCD represents the sectional view of a block. MNOPQRST is the sectional view of the volume from which samples can be selected to estimate the value of the block. Angle θ was restricted to 10° corresponding to the general rise or fall of the veins. The radius of search in the horizontal plane R_s was kept to the value of the semi-variogram range, i.e. 47.5 m. H_r is the radius of search in the vertical direction and was restricted to 1.5 m, preventing the selection of samples likely to belong to different veins in vertically adjacent blocks. The algorithm of this selection criterion was built into the subroutine SLECT3D which is, in fact, a modified version of an original subroutine SLEC3D from the MINPAK library.

(ii) The minimum number of samples required to krige a block was set at 4 (O'Leary, personal communication). This ensured that only blocks with a high probability of containing a vein were kriged.

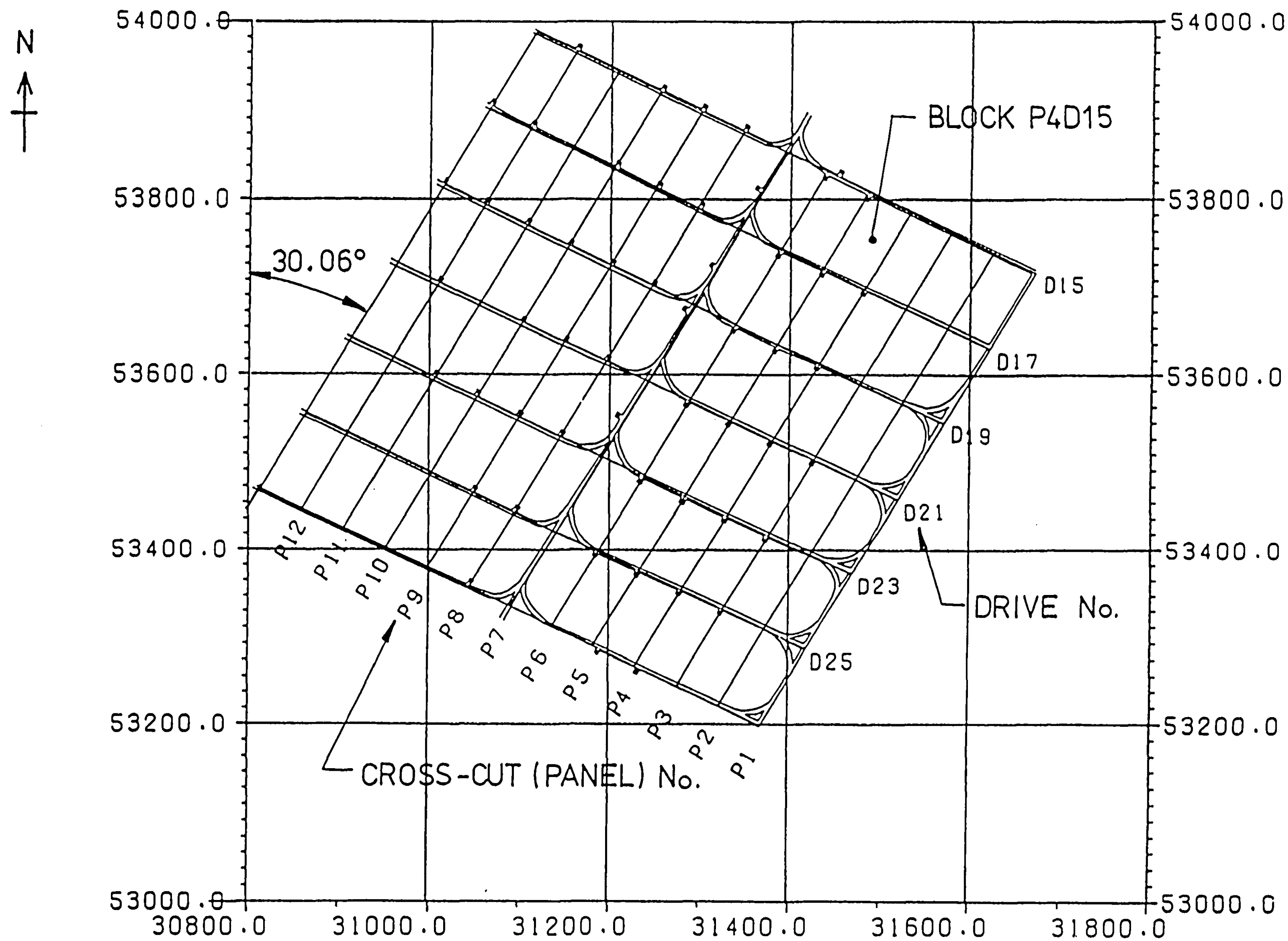


Figure 4.3. Plan of Blocks showing Panel and Drive Numbers.

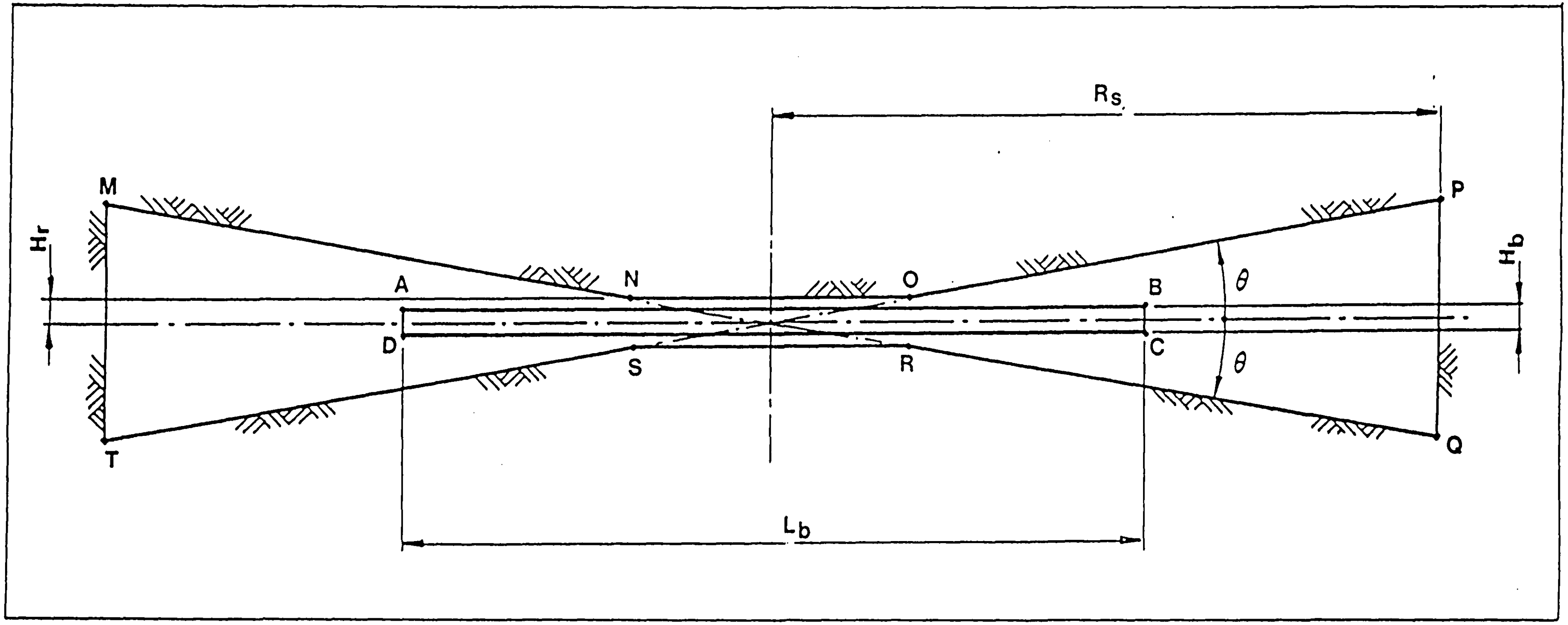


Figure 4.4. Diagram showing the sectional view of a block and the boundary of the ground from which samples can be selected to estimate the value of the block.

ABCD is a block with length L_b and height H_b. MNOPQRST is the ground boundary. R_s is the radius of search in the horizontal plane. H_r is the radius of search in the vertical direction, near to the block's centre. θ is the general angle of fall or rise of the veins and used to define the boundary.

A sample of the resulting block by block kriging estimates of vein concentration, which represents the metalogenic model within the study area, is shown in Table 4-3. In this table, XC1, YC1 and ZC1 are the centre co-ordinates of a block, Z^* is the estimated vein concentration of the block, σ_k^2 is the kriging variance and n is the number of samples used to estimate the block. The kriging variance is a measure of confidence of the estimated value. The number of samples indicates the sample density in and around the block.

Table 4-3. A Sample List of Block Kriging Results.

XC1	YC1	ZC1	Z_v^*	σ_k^2	n
31520.97	53511.92	583.00	.1454	.0136	4
31420.11	53338.73	583.00	.1394	.0133	4
31112.67	53933.43	585.00	.1488	.0137	4
31011.81	53760.25	585.00	.1625	.0127	5
30910.94	53587.06	585.00	.2347	.0140	4
30860.51	53500.47	585.00	.1805	.0113	4
31158.96	53910.85	585.00	.1567	.0127	4
31108.53	53824.26	585.00	.1517	.0132	5
31058.10	53737.67	585.00	.1694	.0098	7
31007.67	53651.07	585.00	.3712	.0125	5
30957.23	53564.48	585.00	.3155	.0125	4
30906.80	53477.89	585.00	.1803	.0123	4
31205.25	53888.27	585.00	.1486	.0115	6
31154.82	53801.68	585.00	.1515	.0109	6
31104.39	53715.09	585.00	.1851	.0091	11
31053.95	53628.50	585.00	.2941	.0106	8
31003.52	53541.91	585.00	.3375	.0115	6
30953.09	53455.32	585.00	.2510	.0103	4
31251.54	53865.70	585.00	.1612	.0100	8

The vein concentration estimates represent the entire quartz content of the blocks. When specific mining or economic constraints are applied, the blocks which satisfy the constraints form the potential mineable ground. However, the total tonnage of *ore** available at various cut-off values of vein concentration can be calculated to give the basis for any initial assessment. There are basically two methods for doing this calculation:

(i) a numerical method and

(ii) an analytical method based on a model of statistical distribution.

In this project, the calculation was carried out by calling a subroutine GTCURV in the MINPAK library within the program MINGEOS. This subroutine uses a numerical method which is considered appropriate because it gives reliable results independent of the distribution of the block estimates and when the data is fairly large (Anon (1981)). The estimates under consideration are numerous. Moreover, their distribution is only approximately log-normal. Figures 4.5a and b show respectively the vein concentration-tonnage curve and the curve of the average vein concentration above various cut-off values, the initial value being set at 0.06 m. From the curves, it can be seen that below the cut-off value of 0.1 m tonnages of ore and quartz material remain almost constant with a change in cut-off; the average vein concentration above cut-off also remains constant. In this zone, the practical implication is that increasing or reducing the cut-off value will not have any significant effect on the quality of ore, i.e. the change in the percentage of quartz in the ore will be irrelevant.

Between 0.1 m and 0.2 m cut-off, the tonnages of ore material fall more significantly with small changes in the cut-off than do the tonnages of quartz; the average vein concentration above cut-off increases exponentially with an increase in cut-off. In this region, the inference is that the cut-off can be raised to improve the quality of ore without a significant loss in the tonnages of quartz obtainable.

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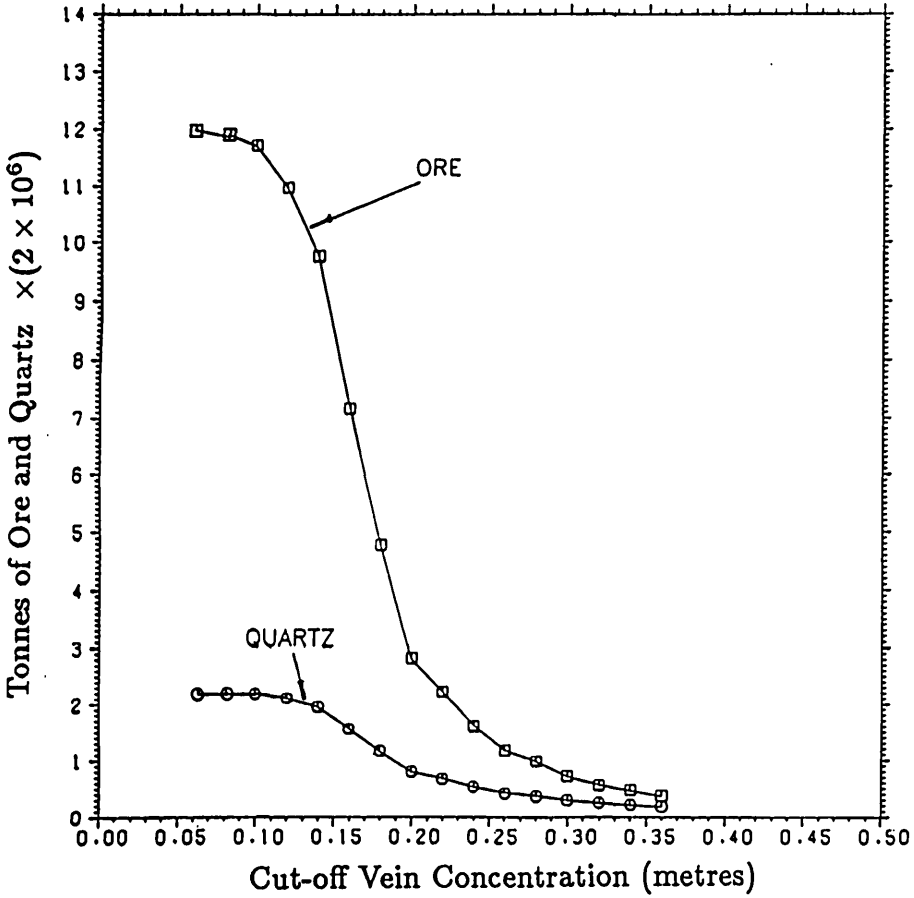
* This refers to the ground, comprising schist and quartz, that must be mined. Strictly speaking, it is not yet ore until mineral measurement has established its grade to be at least equal to the economic cut-off value of 13 kg/m^2 of wolfram. It is however the potential ore material since the wolfram obtainable from it is directly related to its quartz content.

Graph of Cut-off Vein Concentration against Tonnes of Ore and Quartz.



Graph of Cut-off Vein Concentration against Average Vein Concentration above Cut-off.

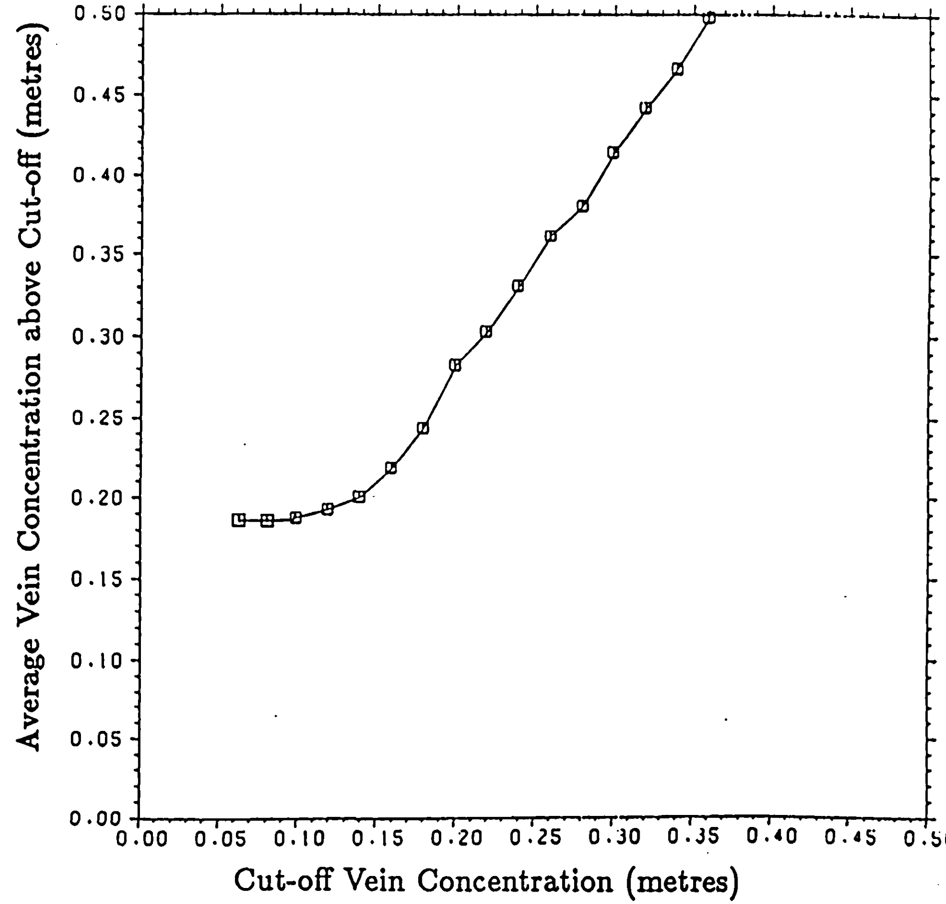


Figure 4.5. Graphs showing Overall Tonnages of Ore and Quartz at various Cut-off Values, and Average Vein Concentration above Cut-off.

Above 0.2 m, the difference between tonnages of ore and quartz reduces gradually as the the cut-off increases; the average vein concentration above cut-off increases linearly with an increase in cut-off. Thus, increasing the cut-off beyond 0.2 m improves the ore quality, but it also means losing quartz.

It can also be seen, from the curves, that if all the ore could be mined (this very unlikely) then, using the mine cut-off value of 0.2 m, 5.66 Mt of ore could be extracted from the study area, at an average vein concentration of 0.28 m. The quartz content will be 1.60 Mt.

4.5 Summary

In this Chapter, it has been demonstrated that within the study area, geostatistical structure clearly exists for both the vein width and the vein concentration. The features revealed by the experimental semi-variograms computed for each of these two main variables conform to those of a spherical model with nugget variance which is explained by the tendency of the veins to split up or thin out abruptly and so give rise to erratic vein widths.

The semi-variograms based on log-transformed values of the vein width and the vein concentration show no apparent structure in the deposit. Therefore, in the estimation process, log-transformation was considered unnecessary.

Using the natural values of either of the two attributes, for estimation of quartz, is considered acceptable since the semi-variogram for each of them shows a well-defined structure with no indication of 'proportional effect', which log-transformation would have eliminated.

The vein concentration model is used to compute estimates of quartz tonnages because it is considered to give a better representation of the material to be mined; the semi-variogram in this case has a range of 47.5 m, a transition variance of 0.0088 m^2, a sill value of 0.0260 m^2 and a nugget variance of 0.0172 m^2. A model block size of 50 m by 100 m by 2 m is used because it represents the level of resolution considered suitable for effective mine planning. Each block estimate is calculated using linear kriging on samples previously identified as falling within a 'bi-conical search volume' and therefore likely to belong to veins occurring in the block, rather than in a vertically adjacent block.

The resulting vein concentration model represents the total quartz available at specific locations within the study area. Since, at this stage, no economic or mining constraints have been applied, the quartz reserves cannot be considered mineable. However, the vein concentration-tonnage curve indicates that at a cut-off value of 0.2 m, there are 5.66 Mt of ore available in the study area. This ore material contains 1.60 Mt of quartz. The average vein concentration above this cut-off value is 0.28 m. In practice, this is only hypothetical, even if bulk mining methods were employed, since the imposition of technical and economic constraints would restrict the mineability of a sizeable proportion of this 'global' ore reserve.

With the current room and pillar mining system, it is obviously desirable to select only areas with high quartz content. Moreover, the blocks in such areas should be contiguous in such a way that mining can be continuous. These blocks should ideally be at the same elevation so that mining stopes can be horizontal.

The vein concentration model will serve as the basis for the selection of possible development horizons. This is described Chapter 5.

CHAPTER 5

PLANNING STRATEGY

5.1 Introduction

The complexity of Panasqueira's vein system and the absence of assay values require a step by step approach to solve the specific problem of where to develop in order to expose the maximum quantity of quartz for future mining.

The work carried out in this Chapter is based on the metalogenic model, i.e. the *vein concentration inventory,** created in Chapter 4 and has two basic objectives:-

(i) To delineate the ground, within the study area, which has the highest quantity of quartz and which can be extracted conveniently with the mechanised room and pillar mining system currently in use at the mine.

(ii) To plan the secondary or 'in-vein' development work necessary to expose the quartz veins in good time for subsequent sampling and mining to meet future production targets.

5.2 Block Selection Criteria

The metalogenic model takes no account of any techno-economic constraints. When such constraints are imposed, the blocks which satisfy the constraints will form the 'ore reserves'.

* The reader is reminded that the term *vein concentration* is the aggregate width of quartz veins per 2 m height of ground measured in metres. The database containing all the block estimates of this variable is hence the *vein concentration inventory* and forms a metalogenic model of the deposit.

5.2.1 Economic Constraints

Economic constraints constitute a defined economic cut-off value at and above which mining will be profitable. Definitions and classical methods of determining the economic cut-off value are all related to the economic value of the mineral content of declared reserves. The underlying principle is to compare the costs of mining and processing with revenue accrued from sales of the product (mineral). The method has varied from obtaining simple Break Even Grade or BEG (Mortimer (1950); Krige (1962b)), through finding the grade which gives the maximum Net Present Value or NPV, assuming constant prices and mine costs (Douglass (1971); Lane (1964; 1979); Halls *et al.* (1969)) to a more sophisticated determination of NPV using dynamic programming where the changing nature of markets and other constraints are taken into account (Dowd (1976)).

In the particular case of this project, a normal economic cut-off cannot be determined at Panasqueira because assays are absent. The quartz veins must be developed and exposed before mineral measurement can be carried out to establish the wolframite content (See Chapter 2). Consequently, the economic value of ore material in the vein concentration inventory is unknown.

Therefore, the approach used in this project is to set an economic aggregate quartz width in the mining unit of 0.18 m (this being the vein concentration) deduced from historical ore value. As explained in Chapter 3 (Section 3.4.1), this figure corresponds to an ore grade of 13-15 kg/m^2 which is considered to be the economic cut-off value.

5.2.2 Technological Constraints

Apart from the economic cut-off, blocks selected from the vein concentration inventory must satisfy technological constraints which are mainly dictated by the mining method and equipment to be employed. In underground mining, methods with high selectivity, e.g. cut and fill, and room and pillar, permit orebody boundaries to be defined so as to include only those blocks which satisfy the cut-off criteria because any waste blocks enclosed between ore blocks can be mined separately and dumped as waste, or alternatively, can be left unmined to serve as support. In contrast, if bulk mining methods, e.g. block caving and sub-level caving, are to be employed then the orebody boundary may have to be defined to include waste blocks or exclude ore blocks.

Because the shape of an orebody can be complex, it is useful to develop algorithms which can, for example, objectively generate at a graphics terminal the desired boundaries, given a cut-off value and mining method (Betty and Arcamone (1988)).

One such approach, suggested in this project, is to generate at a graphics terminal sections showing blocks satisfying the 0.18 m cut-off criterion and then interactively delineate the areas that can be developed to expose the quartz. An interactive approach is desirable because engineering judgements can be inserted to achieve the best results.

5.3 Delineation Process and Results

Delineation of the possible development horizons was carried out in two stages:-

(i) The selected cut-off value of 0.18 m was used to identify blocks from the vein concentration inventory. Sections were then generated to show the spatial arrangement of the selected blocks.

(ii) From these sections possible development horizons were identified. The resulting arrangement in space of the selected blocks as well as the expected tonnages of quartz in each of the horizons were calculated and compared with those of the Mine.

For this exercise, an interactive graphics program, BKPLOT, was developed to generate sections showing the selected blocks and their estimates. The program can also generate plans of the blocks at any specified elevation. The input data required by the program are:

(i) a file which contains the vein concentration inventory;

(ii) the co-ordinates of the centres of all blocks lying along a named sectional line in the case of sections, or the elevation in case of plans; and

(iii) initial and increment values of cut-off, and the number of increments.

Using program BKPLOT, sections were generated along drives D15, D17, D19, D21, D23 and D25, as well as along panels P1 to P12. The locations and names of these drives and panels are shown in Figure 4.3. Figure 5.1 shows the

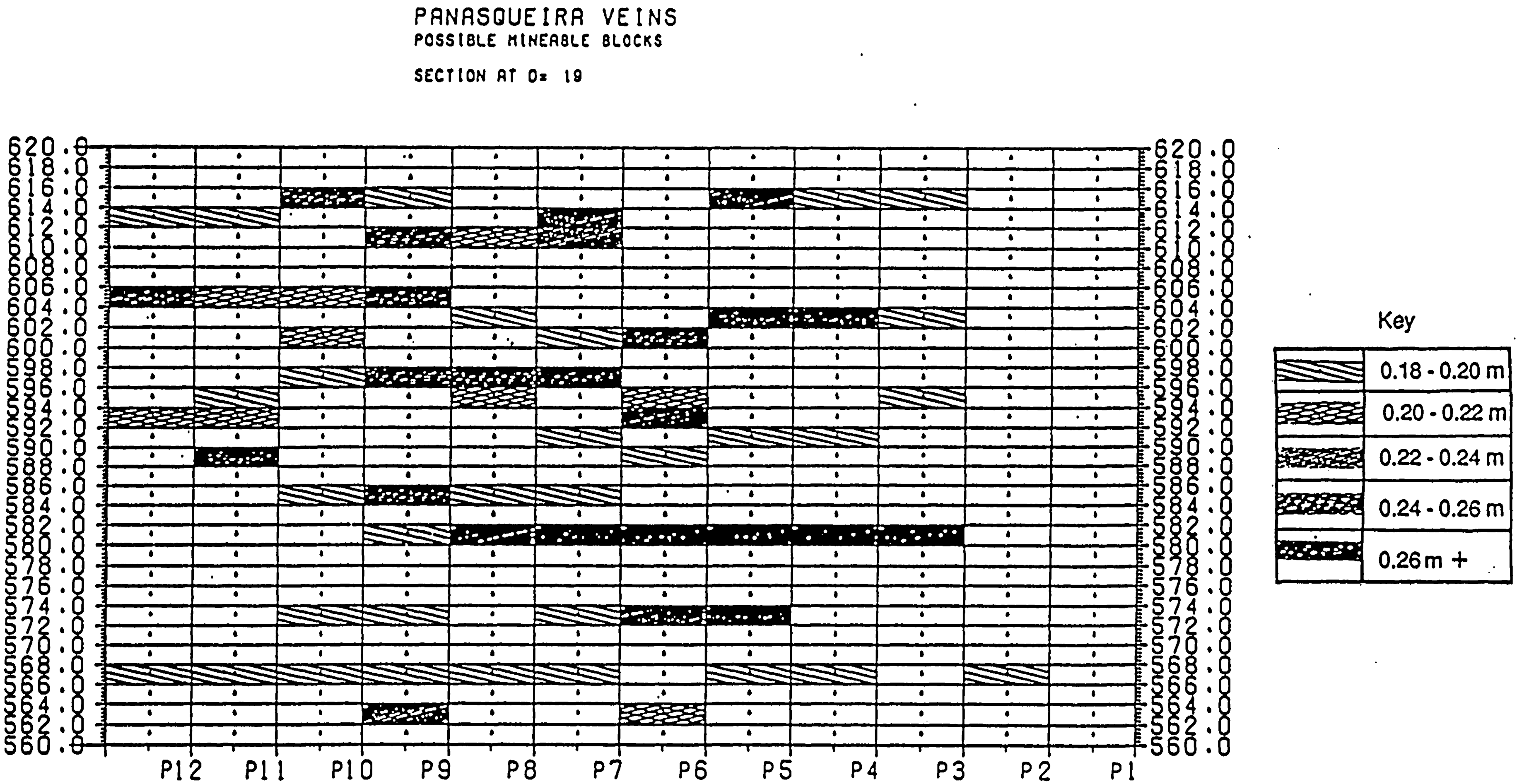


Figure 5.1. Section along Drive D19 showing blocks at 0.18 m cut-off.

section generated along drive D19 which is typical of all the sections obtained. The remaining sections are presented in Appendix 5.

In Figure 5.1, the darker the shade of a block the higher the value of vein concentration. *A closer look at this and other sections reveals that the selected blocks form clusters which define 6 main horizons of obvious development potential. The horizons are not perfectly horizontal; they rise and fall within 10 m elevations across the section. However, it is clear that within the study area, from the top at an elevation of 620 m down to the bottom at 560 m, the mineable veins occur at approximately 10 m intervals.* The horizons and the elevations within which they lie are summarised in Table 5-1. From the section, it is evident that Horizons 3 and 4 contain more and richer blocks followed by Horizons 1 and 2. Horizons 5 and 6 contain less and poorer blocks. This observation is confirmed by the overall estimated tonnages of ore and quartz which are also shown in Table 5-1. The implication of this observation is that at the same rate of mining, more quartz can be expected from Horizons 3 and 4 than from Horizons 1 and 2. At the same rate of mining, Horizons 5 and 6 are expected to last for shorter times unless further drilling or development reveals more veins, or unless the grade is so high as to warrant a low run-of-mine throughput.

Table 5-1. Block Clusters and Tonnages at 0.18 m Cut-off.

Cluster	Horizon	Elevation [m]	Ore [Mt]	Quartz [Mt]
1	1	620 — 610	1.764	0.398
2	2	610 — 600	1.736	0.432
3	3	600 — 590	1.964	0.532
4	4	590 — 580	2.320	0.652
5	5	580 — 570	0.812	0.190
6	6	570 — 560	0.952	0.182

In each of the horizons, individual blocks can be linked together to form continuous ground and so indicate where reef drives might be developed to expose the quartz veins. The suggested approach to perform this exercise will be discussed later in this Chapter.

The overall distribution of the blocks in each horizon can be seen better in a plan. For example, from Figure 5.2 which shows the plan of blocks in Horizon 1, it can be observed that waste blocks are mostly distributed around the SW and NE corners of the study area. The richer blocks run from the NW to the SE corners. The plan of each horizon shows the locations of richer and poorer blocks. Such plans are useful in mine planning in that they provide a basis upon which production can be scheduled to obtain ore with the same quartz content, or to achieve greater quartz output when needed.

The distribution of blocks varies from horizon to horizon. The clusters of blocks in some horizons are denser and richer in quartz content than in others. Thus, different tonnages of ore and quartz can be expected from each horizon. The vein concentration-tonnage curve constructed for each horizon using program GTCURV, described in Chapter 4, is similar to the overall curve presented in Figure 4.5, although the actual tonnages obtained at the various cut-offs vary from horizon to horizon.

For example, Figures 5.3a and 5.3b are the vein concentration-tonnage curve and the graph of average vein concentration above cut-off against cut-off, respectively, for Horizon 3. From these figures, it is evident that, below the cut-off value of 0.1 m, tonnages of *ore* and quartz material remain almost constant with a change in cut-off; the average concentration above cut-off also remains constant. In this zone, the practical implication is that increasing or reducing the cut-off grade will not have any significant effect on the quality of ore, i.e. the change in the percentage of quartz in the ore will be insignificant.

Between 0.1 m and 0.2 m cut-off, the tonnages of ore material fall more significantly with small changes in the cut-off than do the tonnages of quartz; the average concentration above cut-off increases exponentially with an increase in cut-off. In this region, the inference is that the cut-off can be raised to improve the quality of ore without a significant loss in the tonnages of quartz obtainable.

Above 0.2 m, the difference between tonnages of ore and quartz reduces gradually as the cut-off increases; the average concentration above cut-off increases linearly with an increase in cut-off. Thus, increasing the cut-off beyond 0.2 m improves the ore quality, but it is accompanied by a loss of quartz.

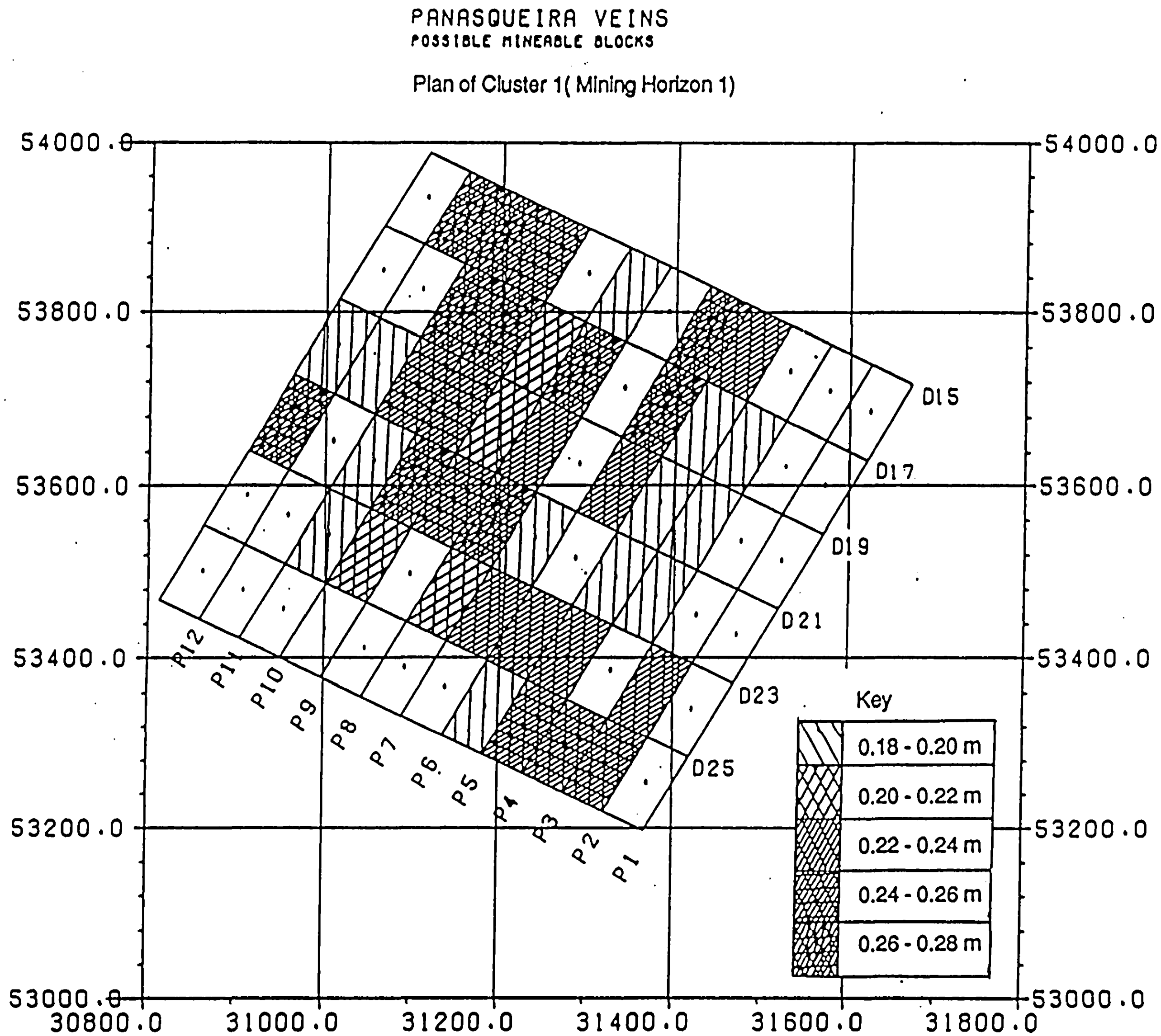


Figure 5.2. Diagram showing the plan of blocks in Mining Horizon 1.

(a.) Graph of Cut-off Vein Concentration against Tonnes of Ore and Quartz.

(b.) Graph of Cut-off Vein Concentration against Average Vein Concentration above Cut-off.

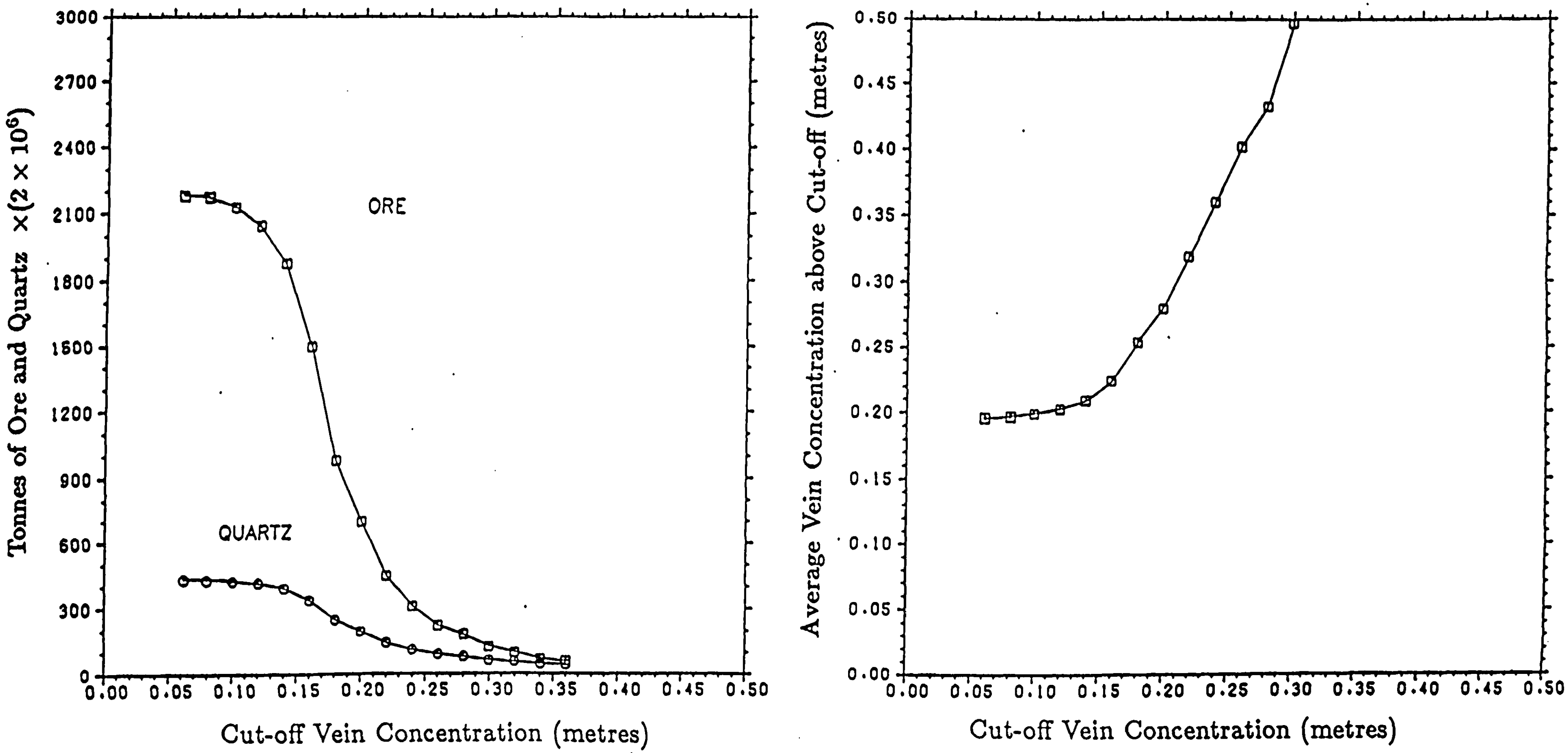


Figure 5.3. Graphs showing tonnages of ore and quartz at various cut-off values, and average vein concentration above cut-off in Horizon 3.

A summary of the tonnages of ore and quartz obtainable in each of the horizons, at the selected cut-off value of 0.18 m, is shown in Table 5-1. It may be observed that the quartz content increases generally from Horizon 1 to Horizon 4 and then diminishes. Horizons 3 and 4 have the highest quartz content, meaning that, at the same rate of mining, quartz production is expected to increase significantly in these two horizons. The pronounced fall in quartz content in Horizons 5 and 6 suggests that more drilling must be carried out to explore the possibility of further vein occurrences.

Given the current room and pillar mining system, the blocks selected with the cut-off criterion must also satisfy two other conditions:-

(i) Ideally, the selected blocks should be contiguous such that mining can be more or less continuous. Although this requirement may be circumvented with the room and pillar mining method, it would be impractical to mine a number of isolated ore blocks surrounded by numerous waste blocks.

(ii) Also, the contiguous mineable blocks must be approximately at the same elevation so that mining levels can be horizontal to facilitate equipment movement and to simplify drilling and mucking operations.

With these conditions in mind, the computer program BKPLOT was designed to draw a continuous boundary around selected blocks on a display screen. Two special subroutines, VCHK and CLCK, were incorporated in the program to perform the following tasks as an aid to the planning engineer in making decisions on selected blocks:

VCHK:- To check the vein concentration estimate ZV of a selected block against the cut-off, and on the angle of elevation VANG, between a selected block and the previously selected one, against a specified maximum angle of 10°, i.e. within the general rise or fall of the veins. The program reports if ZV is below the cut-off, or if VANG is above the maximum angle of elevation. One could choose to ignore the report in which case the boundary will enclose the block in question. The report can be ignored, for example, in a situation where the block in question is enclosed between two ore blocks and the block estimate is low. When an adverse report cannot be discarded, one could try selecting other blocks in the vicinity, or simply stop and start tracing a new boundary. In short, the planning engineer's judgement plays a significant role in the use of the program.

CLCK:- To check the estimates of the closest 8 blocks that surround the selected one in the horizontal plane and to return the number of blocks found to be ore as a percentage. The returned value may be used in discriminating between two blocks which may be of equal advantage in the mine schedule. Again, one may chose to ignore this value, or use it as a selection criterion.

These facilities make it possible to draw boundaries interactively around all blocks which satisfy the cut-off and which, according to the planner's judgement, will be sufficiently continuous and near-horizontal to satisfy room and pillar mining conditions.

The drawing of such boundaries automatically delineates the areas which must be developed. Figure 5.4b shows the blocks delineated for development in the section along drive D19, using the procedure described above. The 6 separate mining horizons can be identified. In addition, the outlines of the delineated areas have been traced on to a transparency overlay to show the 6 separate mining horizons (see Figure 5.4a). One cannot expect this section to be representative of the entire mine. The sections are 100 m apart and therefore the areas delineated can be viewed as extending 50 m into the deposit on either side of the sectional line. Figures showing the blocks satisfying the selected cut-off as well as the delineated areas along all the other drives, i.e. Drives D15, D17, D21, D23 and D25 are shown in Appendix 5.

In order to view all the sections simultaneously and hence to obtain a 3-D effect, each section along the drives was photocopied onto a transparent plastic sheet and then glued on to a specially prepared rectangular piece of clear glass. The four corners of each glass were drilled so that all the panes could be screwed together to form a single physical model of the deposit showing all the selected blocks. Figure 5.5a is a photograph of this model. The physical model itself is only a rudimentary form of what could be achieved with holography, but nevertheless can assist in understanding how the delineated areas change from place to place through the deposit.

The transparency overlay copies showing the delineated areas in the sections along all the drives are shown in Figures A5a to A5f in Appendix 5 so the reader may superimpose one on another to see the correspondence between the delineated mining horizons. When only two or three successive sections are superimposed on each other the correspondence between the delineated areas becomes clearer. For example, Figures 5.5b and 5.5c are the transparency

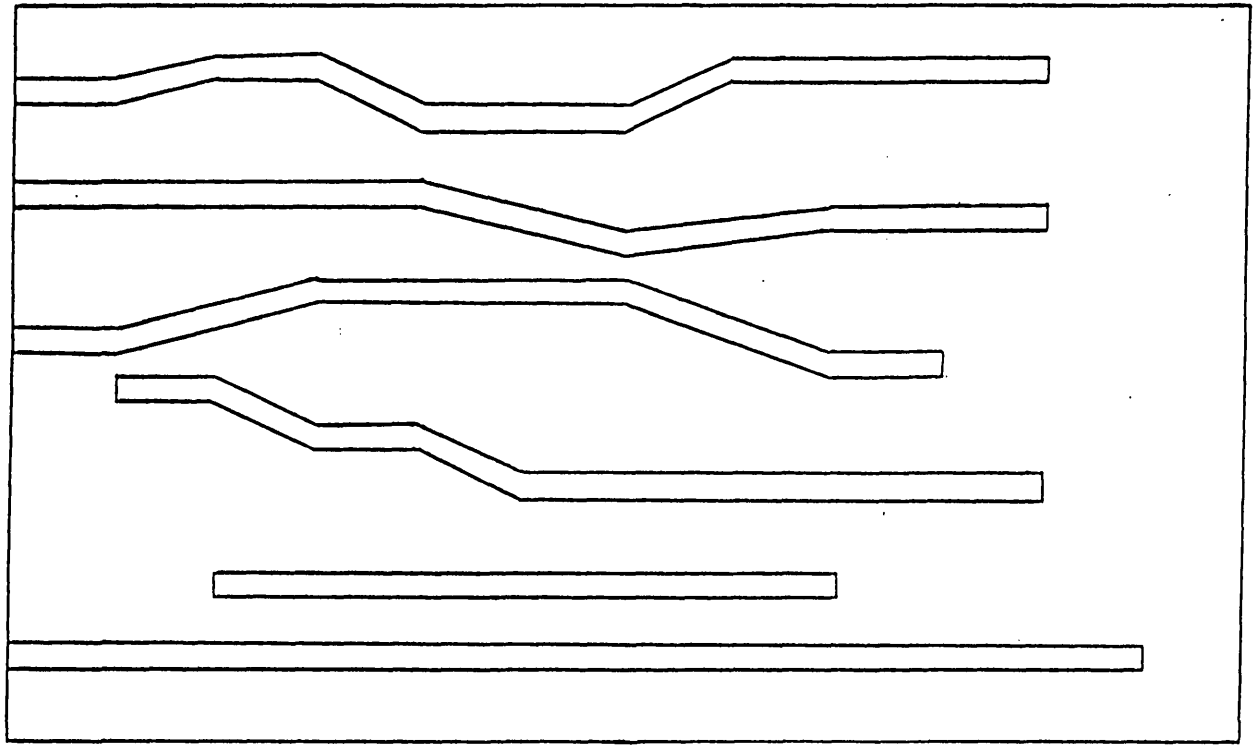


Figure 5.4a. Section along Drive D19 showing areas delineated for development.

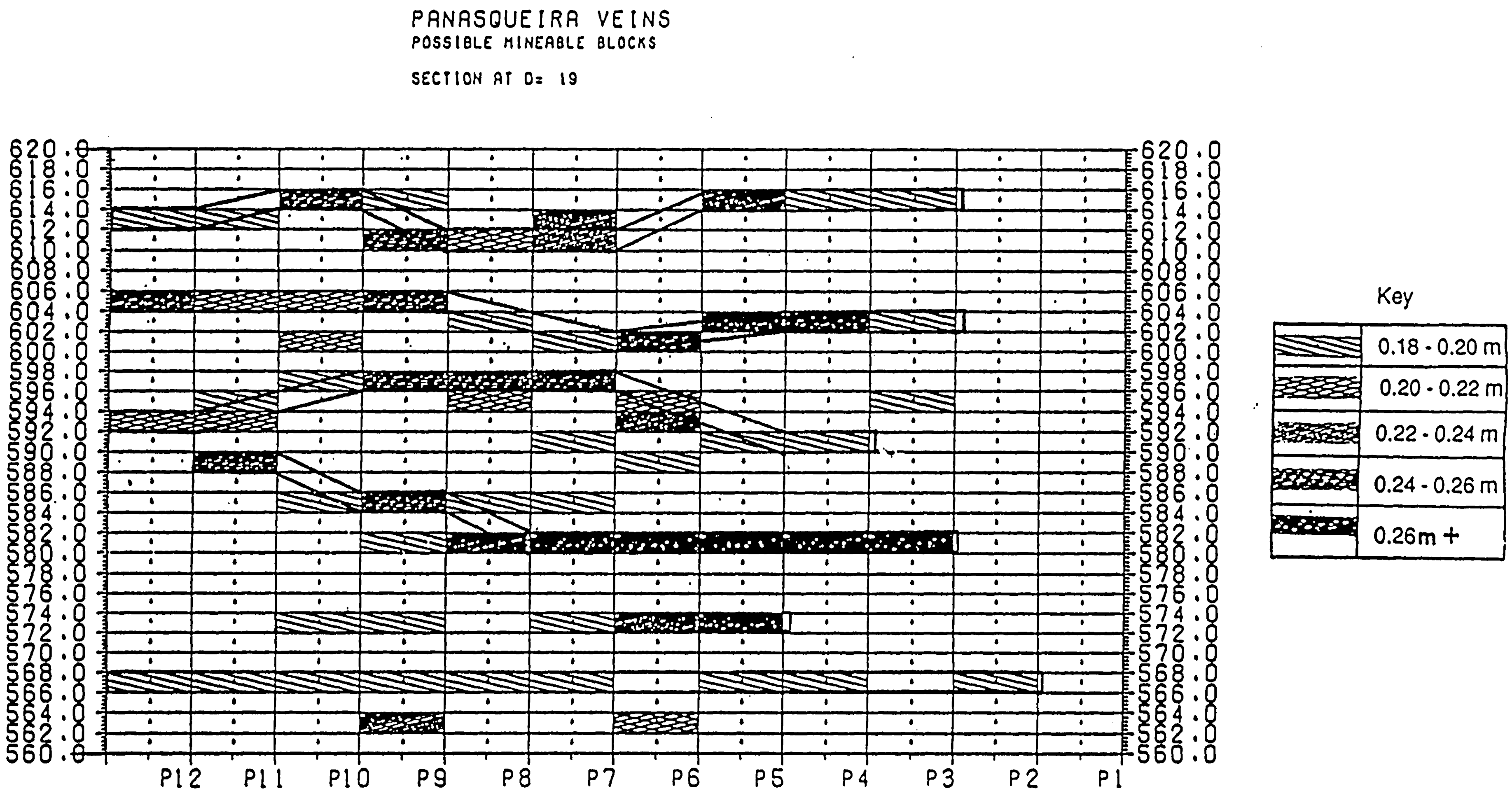


Figure 5.4b. Section along Drive D19 showing areas delineated for development in the 6 Mining Horizons.

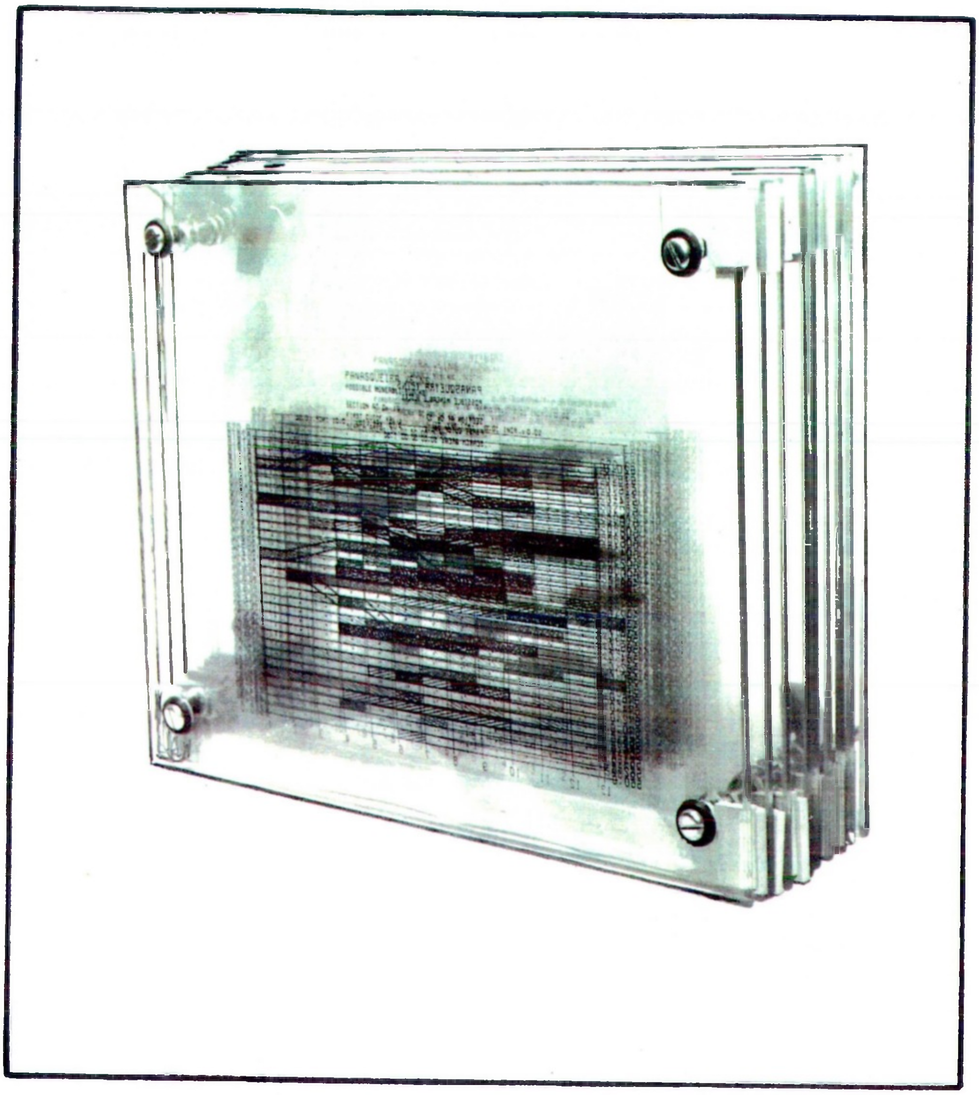


Figure 5.5a. A picture of the model built with the sections on clear glass.

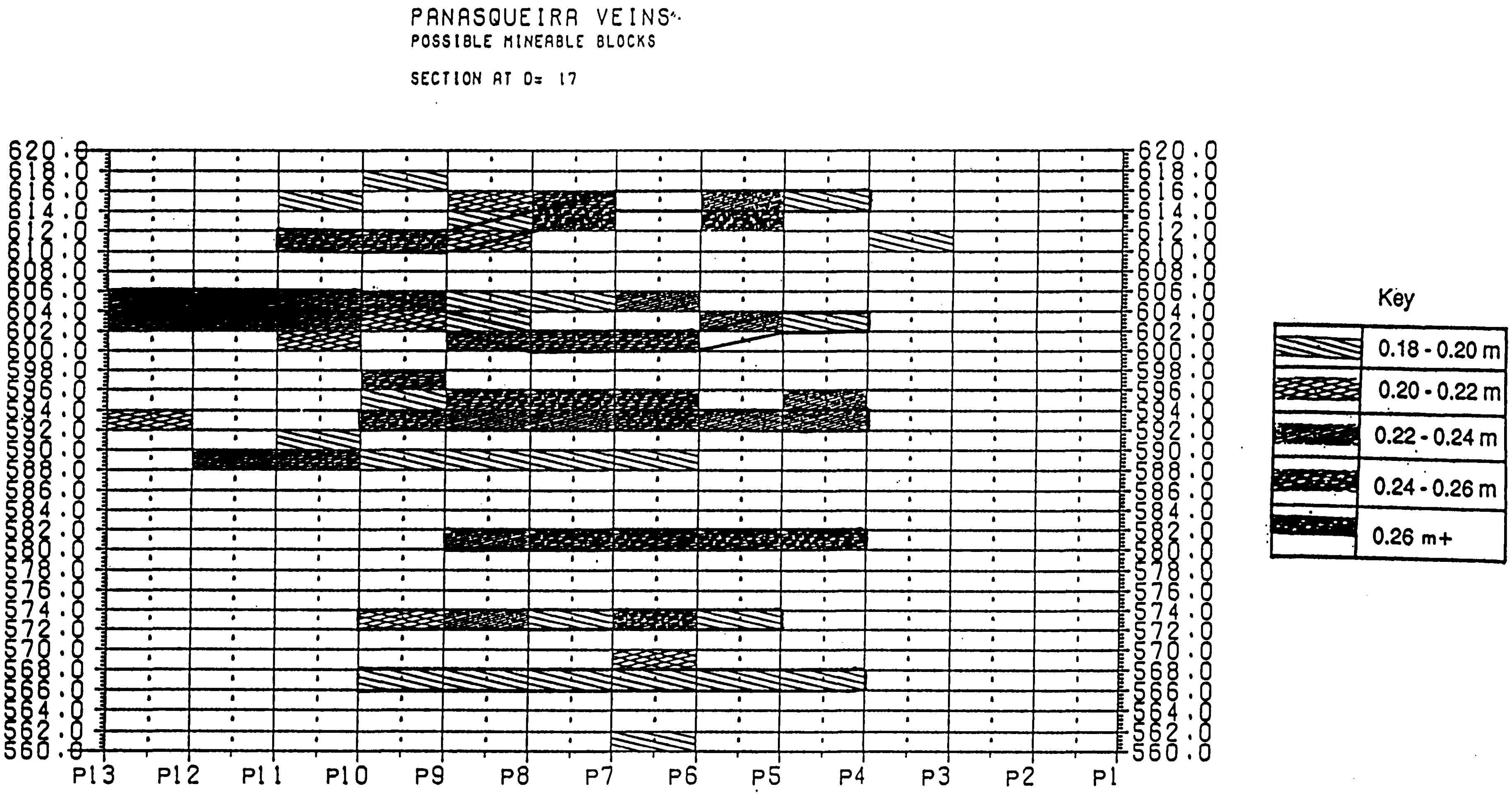


Figure 5.5b Section along Drive D17 showing delineated areas for development.

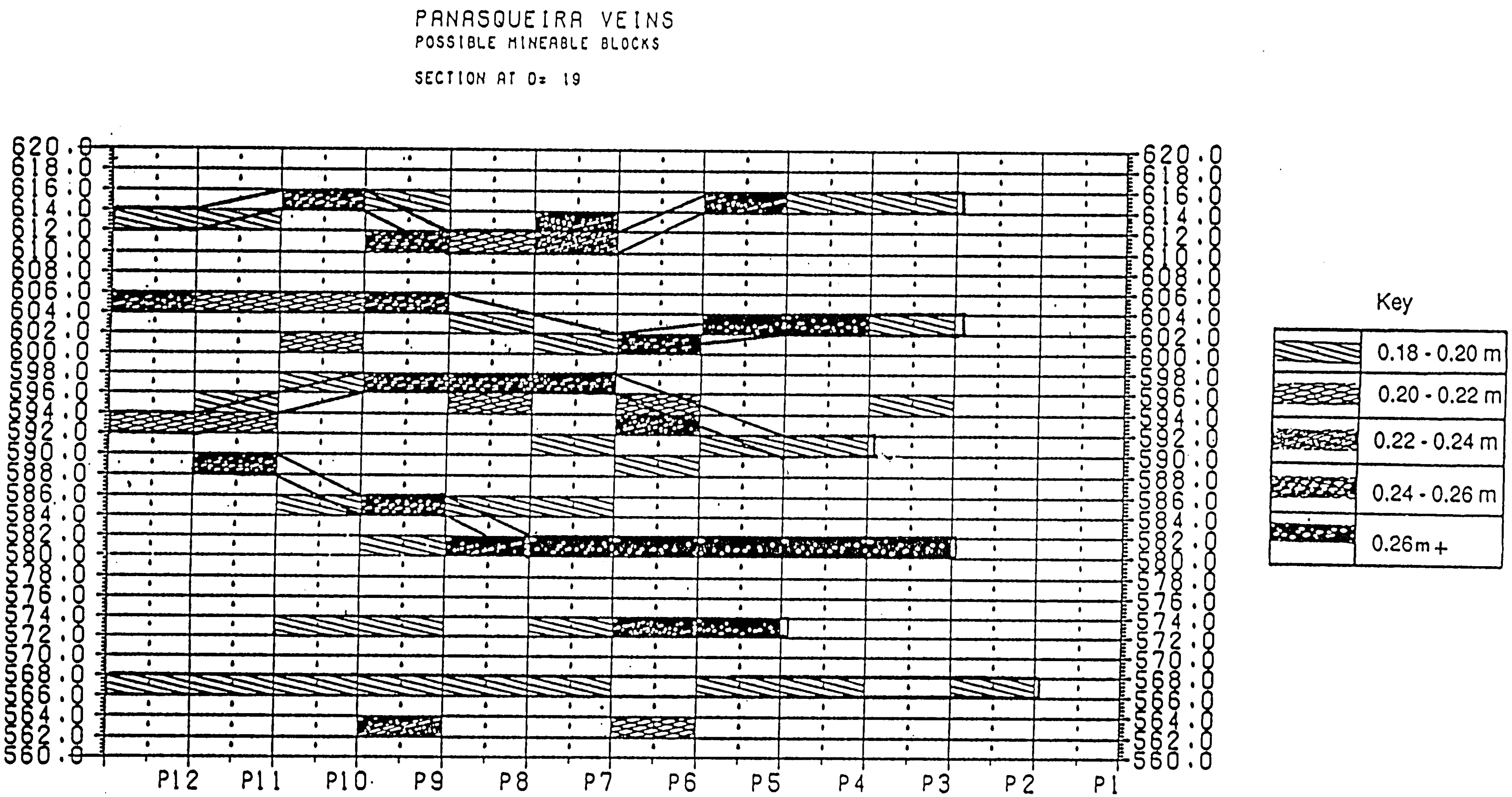


Figure 5.5c Section along Drive D19 showing delineated areas for development.

overlay copies of the sections along drives D17 and D19. These two adjacent sections can be viewed individually, and can be superimposed on each other to show that there is some correspondence between the delineated areas. This correspondence diminishes as the distance between the sections increases. This must be expected since the semi-variogram range is only $\approx$ 50 m, as seen in Chapter 4. As mentioned earlier, the sections are 100 m apart so the areas delineated in each section can be extrapolated up to 50 m into the deposit on either side of the sectional line.

Out of the seemingly uninterpolatable intersections of veins presented at the beginning of this project, we have arrived at a stage where we can conveniently indicate those areas which must be developed so as to expose the greatest quantity of quartz for subsequent mineral measurements.

The techniques used in delineating these areas provide a creditable alternative to the Mine's procedures and have the following advantages:-

(i) They are scientifically based and therefore give objective results.

(ii) They are pragmatic in that they allow engineering judgement to be exercised.

(iii) In the final delineation process several variants cannot be constructed and, thus, different engineers delineating the areas of high vein concentration will eventually produce similar results.

Before taking the final suggested step of using these estimates to plan the medium-term 'in-vein' development, it is considered prudent to recount some of the current developments at the Mine which have led to the method used in cross-checking the results obtained so far with those determined at the Mine. The effect of lowering the cut-off value on the identified mining horizons is also analysed.

5.4 Cross-checking of Results and Observations

The following current information supplied by Panasqueira provided a basis for cross-checking the results of the work carried out in Chapter 4 and in this Chapter, i.e. the observed mineable horizons and the tonnages of quartz expected from them.

By the end of 1987, it had been concluded that the veins within Level 2 could be subdivided into 7 successive horizons and that each horizon contained a *"unit of veins"*. Starting from the topmost horizon, the units were identified as follows:

Unit 1	'Light Green'
Unit 2	'Dark Green'
Unit 3	'Orange'
Unit 4	'Red'
Unit 5	'Light Brown'
Unit 6	'Black'
Unit 7	'Dark Brown'

These 'colour' names were apparently chosen arbitrarily and do not suggest any physical description of the veins; they represent the initial colours used by the mine geologists to codify the different units. For each unit, areas with a vein width of 0.18 m or more were delineated, using the Mine's interpolation process, and considered to form the geological reserves.

By May, 1988, the Light Green Unit in the study area had almost been mined out. Veins in all the remaining units had been developed, initial mineral measurements had been carried out and mineable areas had been delineated. The plans of these areas as well as the tonnages expected from them were made available to provide a basis for cross-checking the results of this project.

The initial cross-checking exercise was carried out at the mine during the author's visit in December, 1987. With the help of the staff in the Geology Department, a one-to-one comparison was made between the *clusters* identified in the course of this project and the *vein units* determined at the mine, paying particular attention to elevation, and number of horizons. The general elevations were found to correspond, but there were some disparities in the actual areas delineated in each of the horizons for development. The main reason for these disparities was found to be that, at the mine, only single veins with widths of at least 0.18 m had been considered as being potentially mineable and consequently areas where the vein widths are below 0.18 m had not been included. On the other hand, the delineation methodology outlined in the preceding sections are based on the vein concentration estimates and therefore the areas delineated contain a number of thin veins, yet the sum of their widths satisfies the 0.18 m cut-off. Unit 7 of the Mine occurs mostly below the 560 m elevation and hence outside the study area. The following correspondence was established:-

Cluster 1	corresponds to		Unit 1
Cluster 2	"	"	Unit 2
Cluster 3	"	"	Unit 3
Cluster 4	"	"	Unit 4
Cluster 5	"	"	Unit 5
Cluster 6	"	"	Unit 6

5.4.1 Comparison of plan areas

The plans of the areas covered by Units 1 and 2, i.e. Light Green and Dark Green units, were compared with the plans of the areas covered by Clusters 1 and 2, respectively. To be able to carry out the comparison, the plans from the Mine were digitised and reproduced at the same scale as those of Clusters 1 and 2 generated with program BKPLOT.

Figure 5.6a shows the area in plan of Cluster 1. If this figure is compared with Figure 5.6b, which shows the area covered by vein Unit 1, i.e. Light Green unit., it can be observed that, in both cases, the veinage area is divided into two along Panel 6. Again in both figures, waste blocks are distributed in the region of the four corners of the study area.

However, there are differences. Figure 5.6a is not completely divided into two; towards the southern part, Blocks P6D21 and P6D23 form a bridge between the two halves (See Figure 4.5 for block nomenclature). In general, not all ore blocks shown in Figure 5.6a appear as ore blocks in Figure 5.6b. These discrepancies are also due to the fact that the Mine's plans are based on widths of single veins whilst the clusters identified in this project are based on the vein concentration. The blocks forming the bridge for example, have a vein concentration of at least 0.18m, but were excluded from the Mine's geological reserves.

Out of the total of 72 blocks in each of the two plans, 23 whole ore blocks and 20 whole waste blocks (from each plan) overlap, making a total of 43 whole blocks or 60% of the total number of blocks overlapping. There are 23 blocks in the Mine's plan which are partially indicated as ore. Of these, 12 overlap with ore blocks, and 11 overlap with waste blocks, in the Project's plan. Thus, 23 blocks or 31.9% of the total number of blocks (from each plan) partially overlap.

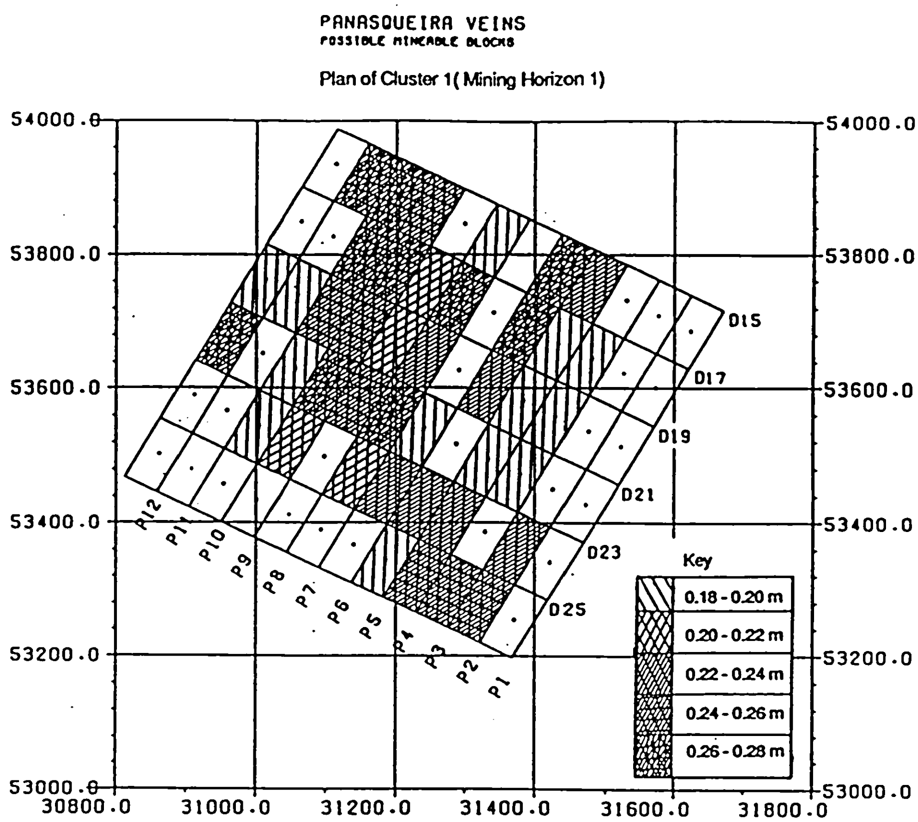


Figure 5.6a. Plan of project cluster 1 showing the area occupied by blocks with vein concentration ≥ 0.18 m..

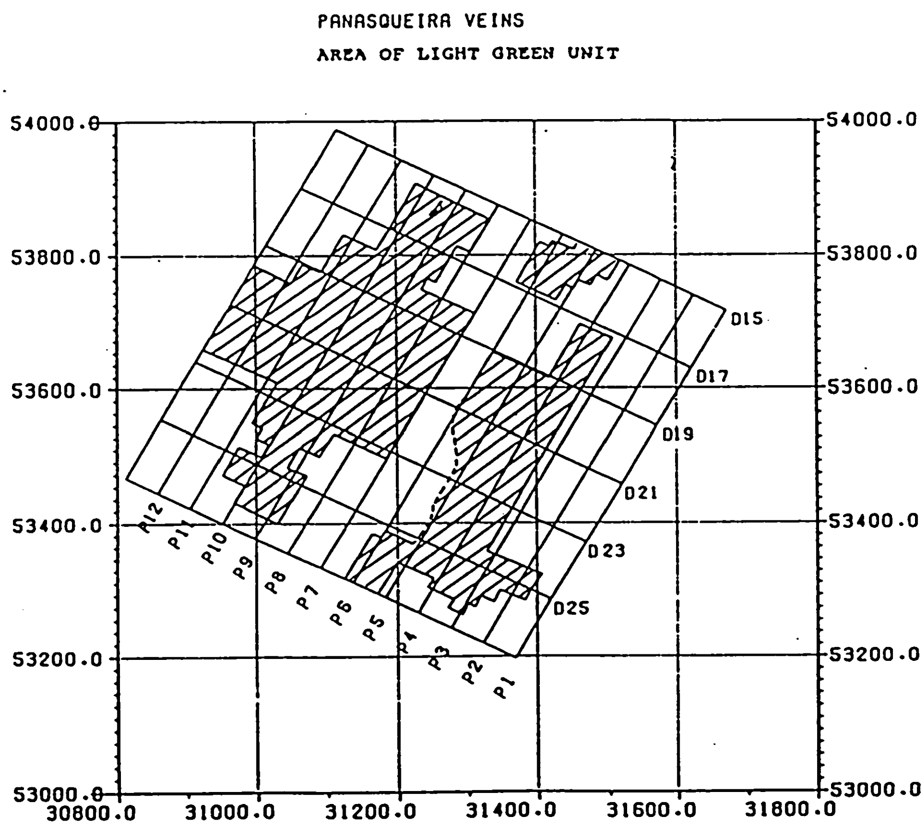


Figure 5.6b. Plan of mine vein unit 1 showing the area occupied by mine geological reserves (vein widths ≥ 0.18 m).

Only 6 whole blocks or 0.1% of the total number of blocks are indicated either as ore or waste in the Mine's plan, but vice-versa in the Project's plan.

One way of comparing two sets of spatially distributed variables, such as the two plans showing the location of ore and waste blocks, is to find the coefficient of correlation between them. To be able to do this, the following values were assigned:

1.0 — to a block which is wholly ore;

0.5 — to a block partially indicated as ore; and

0.0 — to every whole waste block.

The coefficient of correlation between the two plans was then calculated using the one-to-one correspondence of these values. The coefficient of correlation was found to be 0.81, indicating a good correlation between the two plans.

Figures 5.7a and 5.7b show, respectively, the areas covered by Cluster 2 and Unit 2 (the Dark Green Unit). Again, there is similarity between these two plans: the SE and NW corners in both pictures are occupied by waste blocks. There is also a difference: more ore blocks are indicated in Figure 5.7a than in Figure 5.7b. Again, this discrepancy may be accounted for by the fact that, at the Mine, any single veins with widths less than 0.18 m are not considered; or that mineral measurements did not justify the inclusion of some of the blocks in the reserves.

Thirty one out of the total of 72 blocks in each of the two plans, comprising 21 whole ore blocks and 10 whole waste blocks from each plan, or 43.1% of the total number of blocks, overlap. There are 37 blocks in the Mine's plan partially indicated as ore. Of these, 23 overlap with ore blocks, and 14 with waste blocks, in the Project's plan. Thus, 51.4% of the total number of blocks partially overlap. Only 4 complete blocks or 5.5% of the total of 72 blocks are indicated as ore or waste in the Mine's plan, but vice-versa in the Project's plan.

By assigning 1.0 to a whole ore block, 0.5 to a block partially indicated as ore and 0.0 to a whole waste block, the coefficient of correlation between the two plans was calculated to be 0.52.

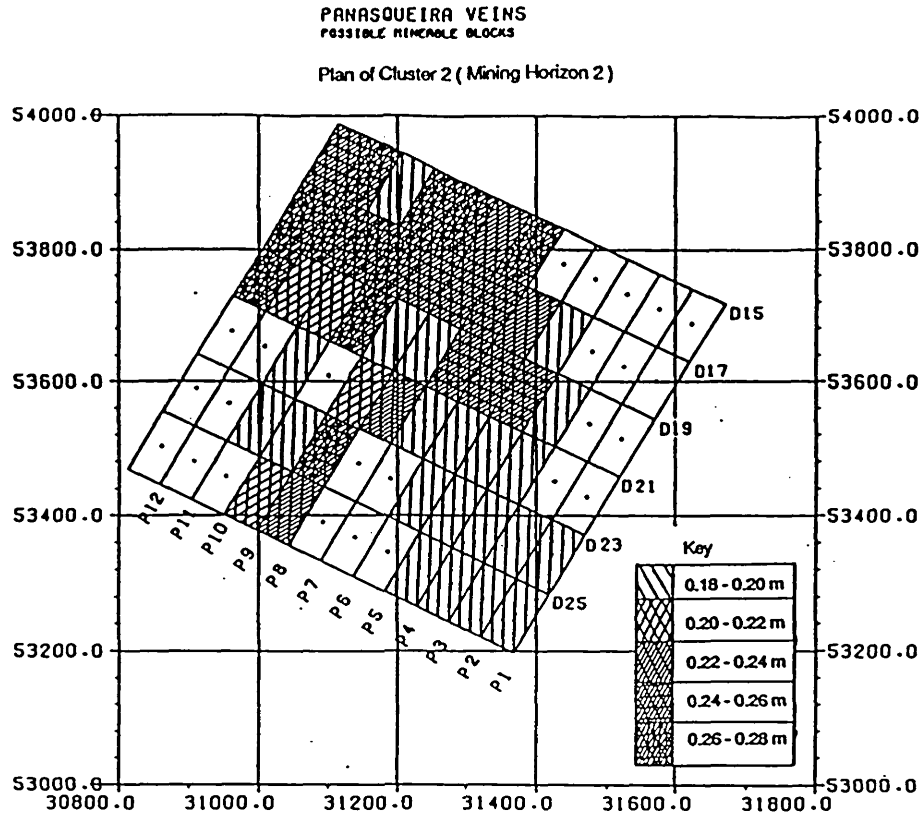


Figure 5.7a. Plan of project cluster 2 showing the area occupied by blocks with vein concentration ≥ 0.18 m.

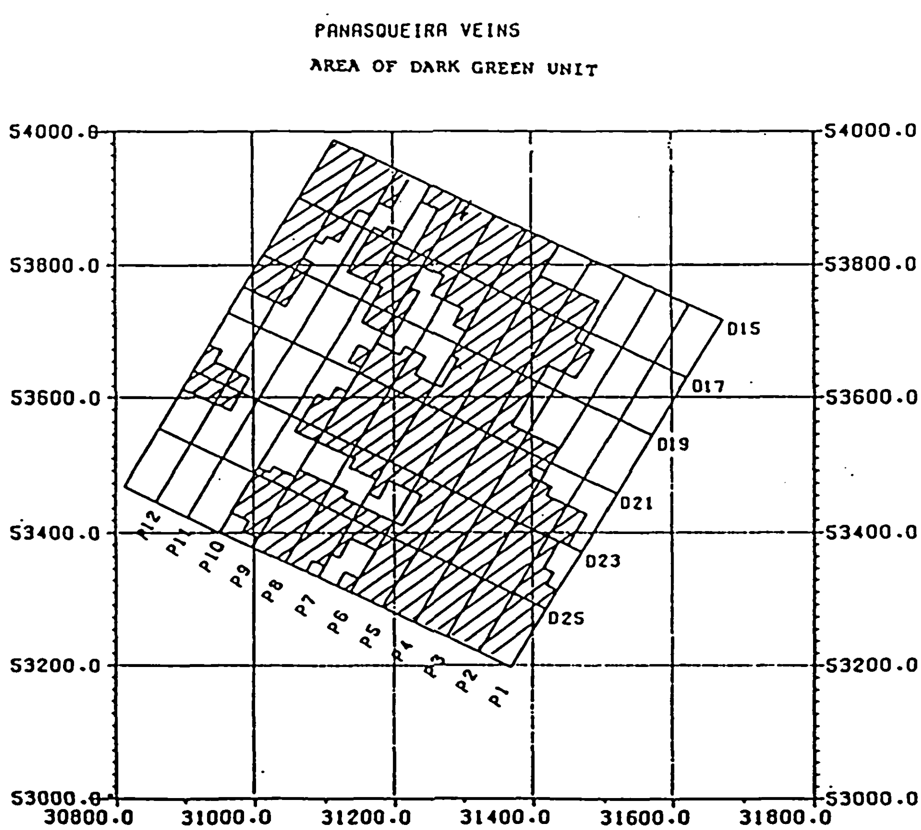


Figure 5.7b. Plan of mine vein unit 2 showing the area occupied by mine geological reserves (vein widths ≥ 0.18 m).

5.4.2 Comparison of tonnages

The total tonnages of quartz within the study area, as determined by the mine using stope plans and tables gave the following results for each vein unit:

(i) area and average width of quartz already stoped;

(ii) area and average width of quartz eliminated from the initial geological reserves due to low grade (kg/m^2) or inaccessibility;

(iii) the area and average width of quartz remaining as pillars, after Phase 1 stoping – this is the Mine's 'Reasonably Assured Reserves (RAR)'; and

(iv) the area and average width of quartz identified as possible reserves but not developed yet – this is the Mine's 'Expected Additional Reserves (EAR)'.

From the information above, the tonnage was calculated as:

$$tonnage\ of\ quartz = plan\ area \times\ average\ width \times density\ of\ quartz$$

The sum of the tonnages, classified from (i) to (iv), gave the total tonnage for the vein unit.

The average widths mentioned above varied from 0.20 m to 0.45 m, with an overall average of 0.30 m. It must be emphasised that although Panasqueira includes all veins with widths at and above 0.18 m in the geological reserves, only those veins with widths of at least 0.20 m are currently considered mineable. The figure 0.20 m is therefore the mining cut-off width and 0.30 m represents the average width above the cut-off. Therefore, to find comparable figures from this Project's estimates it has been necessary to estimate the tonnages of quartz corresponding to the cut-off of 0.20 m for each of the 6 horizons. The values obtained can then be compared with those of the mine.

The results of this exercise are presented in Table 5-2. The calculated coefficient of correlation between the Mine's tonnages and the Project's estimates is, in fact, 0.95. It can be seen that apart from Horizon 6, the Project's estimates are greater than those of the Mine. This observation does not mean, however, that the quartz veins are overestimated in this project. The variable estimated in this research project is the vein concentration. In contrast, the Mine's figures are based on the widths of single veins, meaning that several veins with widths below the cut-off are neglected; it is not inconceivable that some of those veins can add up to meet the Mine's requirements. If the ground mined is considered to be payable when it contains a single vein with a width of 0.18 m, then obviously the same ground must be payable if it contains a number of small veins with an aggregate width of 0.18 m since the Mine's axiom holds that the more quartz mined the greater the quantity of wolfram obtained. It can be concluded therefore that Panasqueira's interpolation and delineation procedures underestimate the tonnages of quartz and hence the reserve potential of the ground.

Table 5-2. Expected Tonnages from Mine Units of Veins, and from Project Clusters of Veins.

	Tonnage of Quartz [Mt] at 0.2 m cut-off		% Increase
Horizon	Project Clusters (PC)	Mine Units (MU)	$(\frac{PC-MU}{MU} \times 100)$
1	0.272	0.138	97.1
2	0.288	0.189	53.4
3	0.266	0.163	63.2
4	0.398	0.196	103.1
5	0.132	0.111	18.9
6	0.116	0.128	- 9.4
Total	1.474	0.925	59.1

5.5 Effect of Reducing the Cut-off

The effect of reducing the cut-off value of 0.18 m on the areas identified as containing the maximum quantity of quartz was studied by generating sections at 0.16 m and then at 0.14 m cut-offs. Figures 5.8 and 5.9 are the respective sections along Drive D19 and are typical of all the sections generated along the other drives.

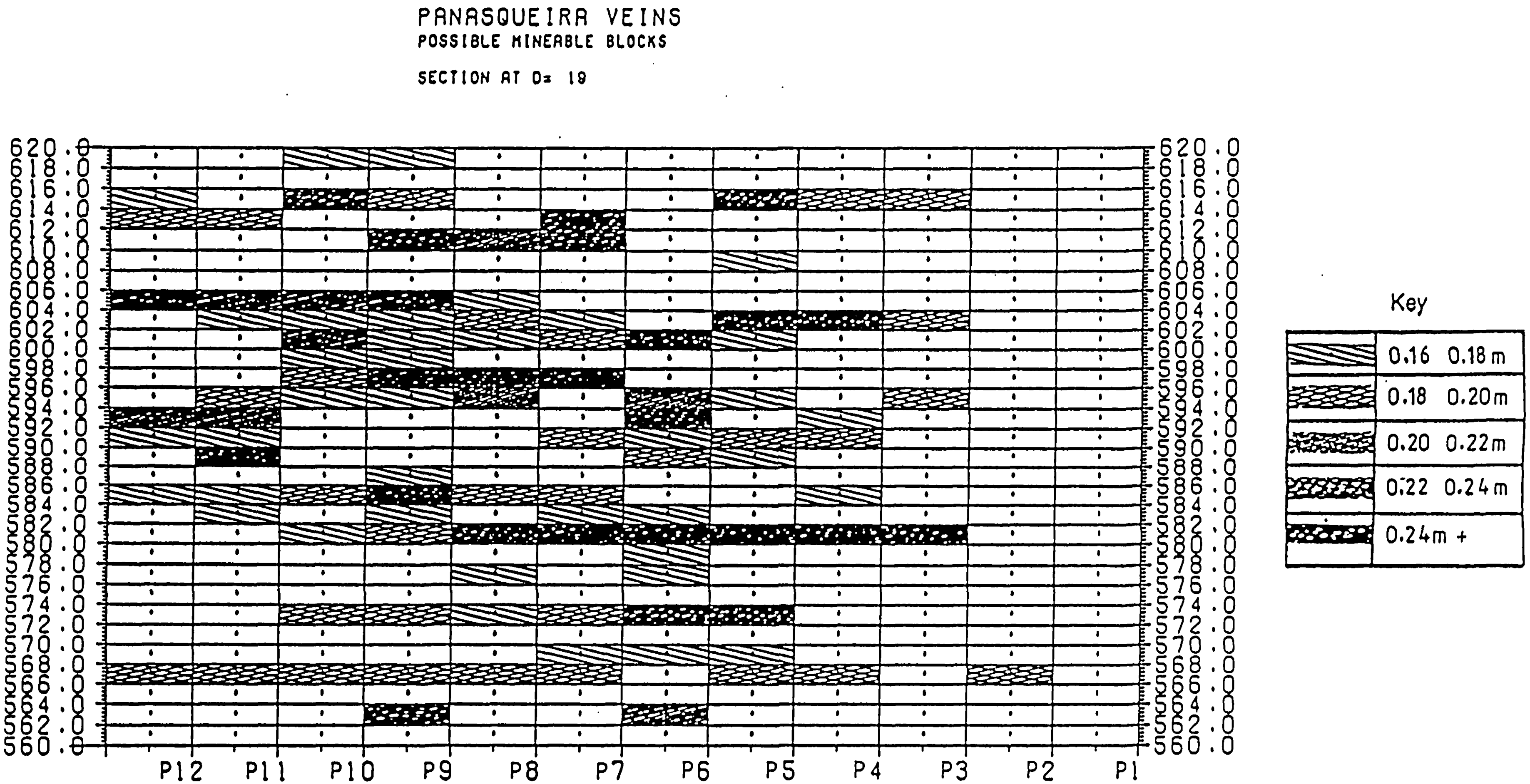


Figure 5.8. Section along Drive D19 showing mineable blocks at 0.16 m cut-off vein concentration.

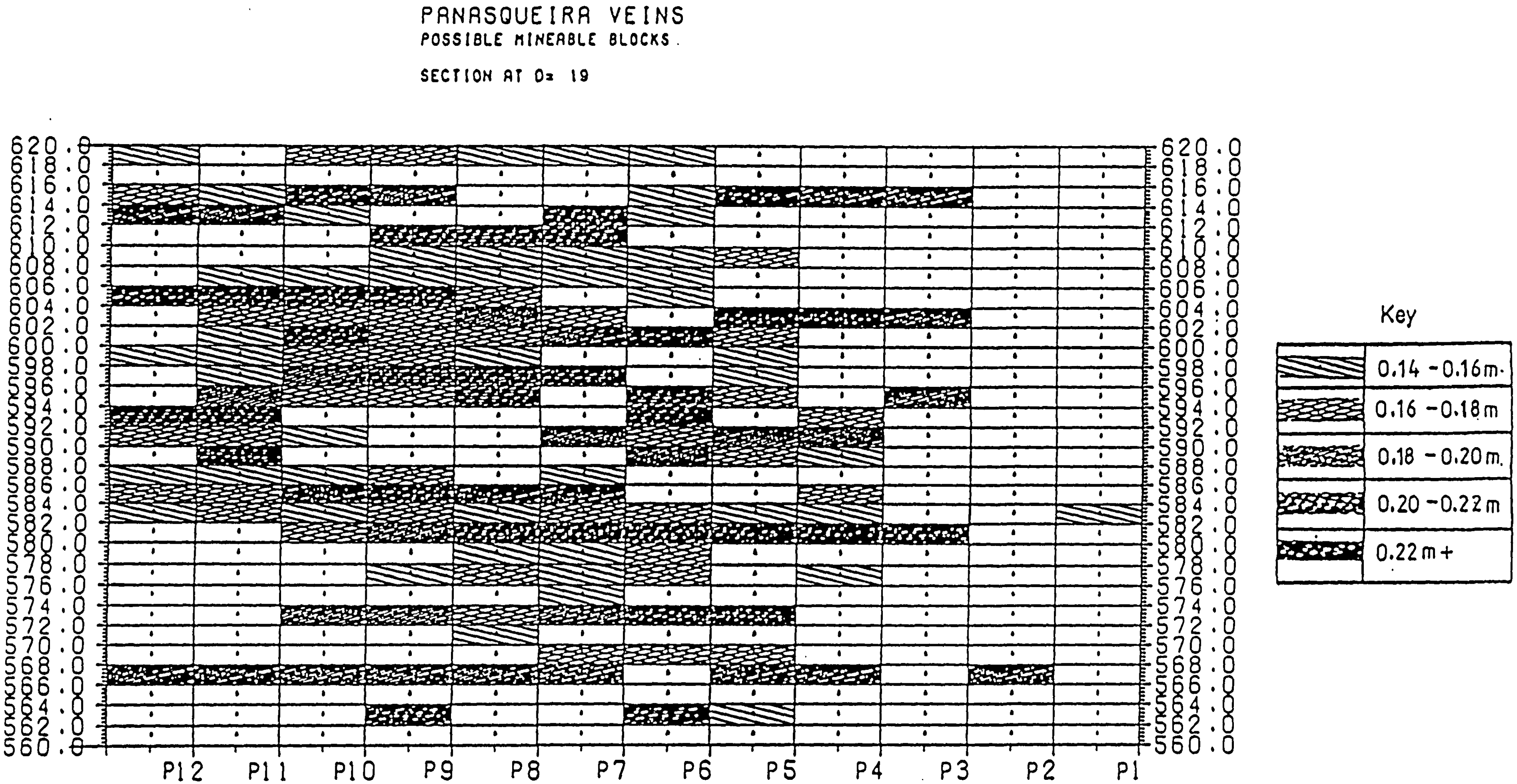


Figure 5.9 Section along Drive D19 showing mineable blocks at 0.14 m cut-off vein concentration.

In Figure 5.8, it can be seen, as expected, that lowering the cut-off from 0.18 m to 0.16 m has increased the number of potentially mineable blocks, but the following observation can also be made :-

The number of potentially mineable blocks in Horizon 1 (from 620 m to 610 m) has increased and forms an area that might be mined as a unit. Blocks in Horizons 2 and 3 (from 610 m to 590 m) have almost joined together to form another area that might be mined as a unit. The blocks in Horizon 4 have also increased, but stand out together to form a possible mining unit. Blocks in Horizons 5 and 6 are closer and together make a possible mineable unit. Thus, four new units on four new horizons appear to be forming.

The following names were associated with these 4 new units and horizons so as to distinguish them from the 6 units and 6 horizons previously identified in the sections using the 0.18 m cut-off:

Unit A on Horizon A corresponding to Unit 1 on Horizon 1;

Unit B on Horizon B corresponding to Units 2 and 3 combined on Horizons 2 and 3;

Unit C on Horizon C corresponding to Unit 4 on Horizon 4; and

Unit D on Horizon D corresponding to the combined Units 5 and 6 on Horizons 5 and 6.

The observation of these units, i.e. Unit A, Unit B, Unit C and Unit D, becomes clearer in the section shown in Figure 5.9 where the cut-off is further reduced to 0.14 m. It can be seen that the number of possible mineable blocks has further increased. Units A and C still stand out as separate possible mineable units, and Units B and D become more identifiable as individual areas which are possibly mineable.

While it is unlikely that Panasqueira will consider changing the current room and pillar mining system, it is noteworthy that the new mining units observed to be present in the study area suggest the possibility of employing a sub-level caving method. Each of the four units could be mined as a sub-level starting from the top horizon and mining down. In Figure 4.5, it can be seen that at 0.14 m cut-off, 19.22 Mt of 'ore' is available in the study area containing 3.92 Mt of quartz. The average vein concentration above this cut-off is 0.2 m. There is therefore a significant increase in both the tonnages of 'ore' and quartz, although there is a concomitant increase in dilution, compared to the 5.66 Mt of 'ore' containing 1.472 Mt of quartz with an average vein concentration of 0.28 m at the cut-off value of 0.2 m. The increased tonnages could satisfy the higher production rate which is achievable with a sub-level caving method. However, whilst cavability is likely to be satisfied, the increased production tonnages and the added dilution will require a larger and a more efficient processing plant – these are issues that require further investigations.

5.6 'In-vein' Development

In the previous sections, the tonnages of potentially mineable material and quartz obtainable from each of the 6 mining horizons in the study area were established. Also, cross-sections and plans were generated to show the locations of all selected blocks and to indicate the areas to be developed. The objective of this section is to make a schedule for the yearly marginal development work necessary to expose the quartz veins using this information, given the current room and pillar mining method.

It can be recalled from Chapter 2 (Section 2.4) that the Mine is developed from an orthogonal system of haulages, i.e. drives and cross-cuts (panels), laid on a 100 m grid. Separate levels are connected by 60 m vertical raises at 50 m centres along the drives to serve, eventually, as orepasses and service-ways for compressed air and electricity supplies, and to act as part of the ventilation network. There are also ramps to provide access for equipment to individual veins between the levels. From these raises or ramps, reef drives known locally as "inclines" are developed to expose the quartz veins for subsequent mineral measurements. The following features of the operation are noteworthy:-

(i) The orthogonal system of haulages provide access as well as necessary sites for diamond drilling which establishes the vein occurrences. Therefore the haulages must be completed as part of the long-term infrastructural capital development.

(ii) Since ventilation and other services are needed during this period, the raises are also a necessary part of the capital development.

It follows that the developments necessary to ensure the correct sequence of quartz vein exposure for subsequent mineral measurement are:

(i) *The ramp* – usually the ramp is also considered as part of the long-term infrastructural capital development and therefore developed at about

3.2 m/day in two blasting rounds, but it is assumed in this study to be part of the medium-term secondary development in order to include its marginal advance in the schedule.

(ii) *The reef drives* – the development of the reef drive is, in fact, the start of the room and pillar mining system. The broken material resulting from the reef drive development is therefore assumed here to be 'ore'.

This development must be completed before sampling can take place. Mining can start only after enough blocks have been sampled and short term-planning has been completed. Thus, the development must be ahead of sampling and sampling must precede stoping.

5.6.1 Development Model

At Panasqueira, the objective is to produce an average of 2000 t/day run-of-mine, working for 21 days in a month throughout the year. To achieve this goal, enough blocks must have been sampled with the mineral measurement method and must have been proven to contain at least 13 kg/m^2 of wolfram.

A recent problem in production is to determine how much advance of development work must be undertaken each year in order to maintain the production target, assuming the following conditions:-

(i) The total yearly production is from the 6 mining horizons identified in the study area.

(ii) 80% of all blocks included by this project in the reserves will have the required grade of at least 13 kg/m^2.

(iii) Mine loses as floor fines will constitute 8% of the reserves.

(iv) The mine will keep within the recommended stoping height of 2 m and therefore an extraction ratio of 84.6% is obtainable (see Chapter 6).

(v) The maximum allowable inclination of the ramp is 10°.

The yearly production YP can be calculated as:

$$\begin{aligned} YP &= 200t/day \times 21days/mth \times 12mth/yr \\ &= 504000t/yr \end{aligned}$$

If AER is the extraction ratio, L the mining loses as floor fines, RM the proportion of selected blocks that will be mined, then under the assumed conditions the total quantity of ore TT in place that must be made available to maintain the yearly production YP is:

$$\begin{aligned} TT &= \frac{YP}{AER \times RM \times (1 - L)} \\ &= \frac{504000 t/yr}{0.85 \times 0.80 \times 0.92} \\ &= 806000\ t/yr \end{aligned}$$

or 0.806 Mt/yr less the 'ore' tonnage resulting from the reef drive development.

Table 5-3 gives the length of drive that must be developed in order to make a tonne of potential 'ore' available for sampling and mining on each of the 6 horizons. These lengths and tonnages were estimated from the plans and sections of each horizon and can be seen to differ according to the number of selected blocks and how rich they are in quartz.

Table 5-3. Drive Development Yield

Horizon	Quantity of Ore [Mt]	Total Length of Drives [km]	Development Yield [km/Mt]
1	1.764	3.15	1.79
2	1.736	3.15	1.82
3	1.964	3.25	1.66
4	2.320	3.50	1.51
5	0.812	2.15	2.65
6	0.952	2.25	2.36

A program DPLAN was developed to calculate the amount of drive development that should be completed on various horizons during each year of mining. The program also calculates the length of ramp that should be developed to provide access for equipment in good time. Since the program is interactive it can be re-run with different parameters, i.e.

(i) number of mining horizons and the total tonnage on each horizon,

(ii) yearly production and a time lag before mining can start,

(iii) mine losses in percentage, and

(iv) the maximum allowable inclination of the ramp.

The program computes the following for each year up to when all development work should finish, i.e. when all the reserves are developed:

(i) the required length of drive on each horizon;

(ii) the required length of ramp; and

(iii) the total length of development.

For the criteria set above, the results are presented in Figure 5.10; it shows, for example, that 1.44 km of reef drive must be completed on Horizon 1 in Year 0 before mining can start. In the first year of mining, another 1.44 km of reef drive must be completed to replace the 'ore' that will be mined and 0.06 km of ramp must be developed to provide access between Horizons 1 and 2. In the second year, 0.27 km and 1.46 km of reef drive must be completed on Horizons 1 and 2 respectively.

The ramp must, at the latest, be developed to join Horizons 2 and 3 in the fourth year because in the fifth year Horizon 3 must be accessed for drive development. In general, ramp development to a lower horizon must, at the latest, precede the year in which drives on that horizon are developed.

Drive development is at a minimum in Years 6, 7 and 8 because during these years development is mainly on Horizon 4 which has the highest tonnage and consequently the highest tonnage yield per metre development. In Years 9, 10 and 11, when Horizons 5 and 6 are to be developed, the greatest length of drive must be completed per year to make enough ground available for sampling because these two horizons have the least tonnage of reserves.

From this information disposition of labour and machinery can be planned, but this aspect of planning does not form part of this research.

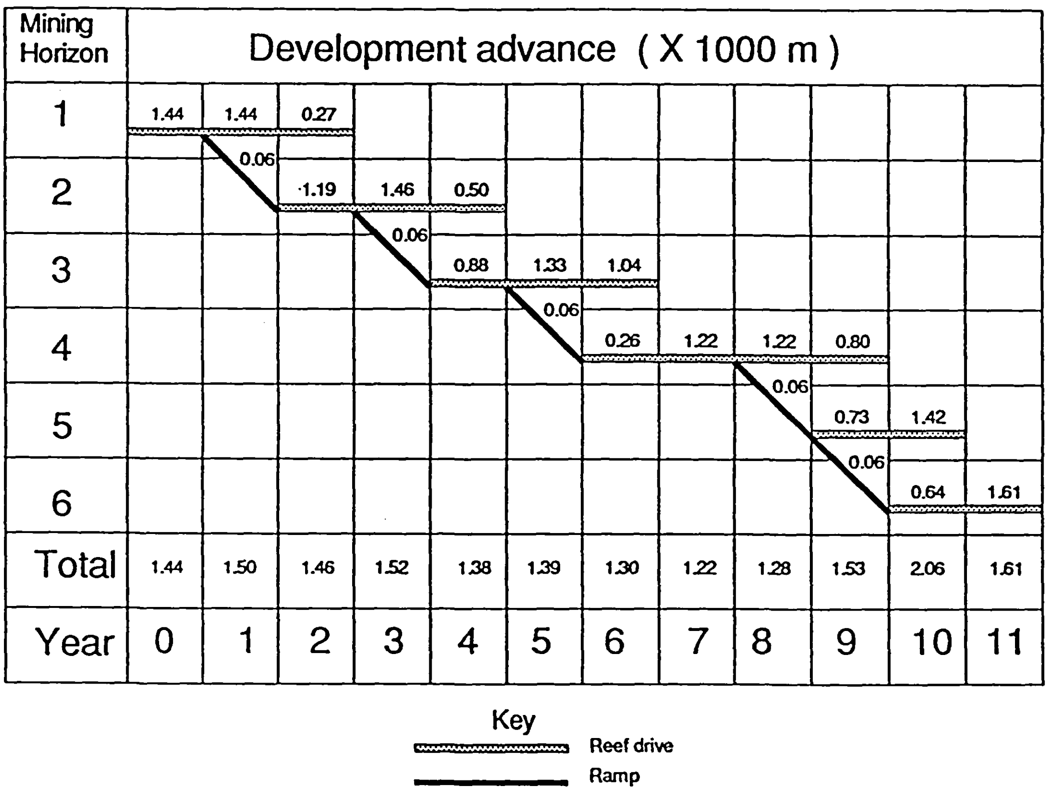


Figure 5.10. Diagram showing the yearly reef drive and ramp development.

5.7 Summary

In this Chapter, it has been shown that within the study area blocks with a vein concentration of at least 0.18 m form clusters which define 6 mining horizons. There is evidence, from the sections generated to show the spatial arrangement of these blocks, to suggest that these individual mining horizons are regularly placed at approximately 10 m vertical intervals.

Based on the sections and plans, it has been possible to delineate areas which contain contiguous blocks, that can be conveniently extracted using a room and pillar mining method, indicating where reef drives can be developed to expose the quartz veins for subsequent sampling with the mineral measurement method.

Plans showing the distribution of mineable blocks differ from horizon to horizon. In some cases, e.g. Horizon 1, the richer blocks are spread evenly over the whole area under study. In other instances, e.g. Horizon 2, the richer blocks are mainly to be found in the NW quadrant of the area. Generally, however, it is observed that blocks along the eastern boundary are mostly waste, especially around the NE corner.

The 6 mining horizons identified in this project are in line with ideas developed at Panasqueira, but the tonnages of quartz estimated in this project, for each of the horizons, are greater than those determined at the Mine.

The techniques used in this project for the estimation and delineation of mineable ground provide a creditable alternative to those employed at the Mine; they are scientifically based and therefore provide objective and reproducible results.

Since mining can start only after the veins have been exposed and sampled, it is suggested that the reef drives must be developed ahead of mining; at least the 'ore' made available by yearly development advance must be sufficient to replace the yearly production. Development work needs to be intensified on those horizons with least reserves.

An interesting observation is that at cut-off values lower than 0.18 m the sections show the mineable blocks forming four clear horizons that could be mined as separate units, indicating the possibility of employing a sub-level caving mining method to extract the quartz veins.

CHAPTER 6

ANALYSIS OF ROOM & PILLAR LAYOUT

6.1 Introduction

The effectiveness of any Room and Pillar Mining System (RPMS) depends on the optimal selection of room and pillar dimensions to ensure maximum ore extraction while maintaining safe working conditions. Larger rooms give high ore recovery but at the expense of the factor of safety of pillars. The objective of this Chapter is to analyse the room and pillar layout at Panasqueira and so find out if the room dimensions, especially the stoping height approximate to an optimum for ore extraction. The outline of this work is as follows:-

(i) A model for the initial design of a room and pillar layout is developed using existing design theories.

(ii) This model is then applied to the geomechanical environment prevailing at Panasqueira to check on the possibility of increasing the current room and pillar dimensions.

6.2 Room and Pillar Mining System

In Room and Pillar Mining part of the ore is mined out to create rooms which can also serve for the purposes of access and ventilation. The remaining material (comprising ore and waste) is left as pillars to provide support. The pillars can be circular, or rectangular, or as longitudinal walls. The arrangement of rooms and pillars can be regular or irregular, although the former is preferred in order to simplify design, planning and mining activities.

RPMS is normally applied to orebodies with flat or near horizontal dip, but variations of the method can be used in orebodies with an inclination not exceeding 30 degrees (Bullock (1982)). Successful RPMS implementation requires suitable geomechanical conditions such as: stable hanging wall, usually low rock pressure and few or no cross jointing in the immediate roof rock. Uniform orebody thickness greatly enhances mechanisation but local variations are acceptable.

RPMS has been applied successfully to bedded deposits of sedimentary origin including coal and copper-mineralised shales, and industrial minerals such as limestone, salt and potash. Some gold deposits have been mined with this method (Eichmeyer *et al.* (1986)). In all cases, the main objective is to extract as much ore as possible while maintaining stable rooms. Geomechanical factors such as the strength of ore and country rock and thickness of the deposit impose a limit on room dimensions. Increasing the pillar size or number of pillars and reducing the room width or height can compensate for poor ground conditions. This however means reducing the ore recovery since more ore is left in the pillars. On the other hand, small size pillars that are theoretically stable may not always be safe in practice due to local geological conditions. It is therefore necessary to combine theory and practice in order to achieve optimal dimensions of rooms and pillars.

6.3 Design Approach and Limitations

An analytical method for designing the layout of RPMS can be undertaken using one of two alternative approaches.

Assuming that rock is an elastic, transversely isotropic material, a theoretical model can be developed to calculate the total stresses and mining induced displacements around excavations, including surface deflections. Boundary element and finite element analysis are two methods that can be used. There are programs for such analysis but, apart from being computationally expensive, they are appropriate only when the orebody has been exposed and immediate near field stresses have been measured over a period of time. In the particular case of this project, such techniques could not be used because near field stresses have not been measured.

For an initial design, a more pragmatic approach has involved the use of the Tributary Area Theory (TAT) for pillar design, and the Plate Theory (PT) for the design of the roof of stable rooms. This approach is suggested because the required data, e.g. uniaxial compressive strength of the rock, density, internal angle of friction, *etc.* are available (Mendes *et al.* (1987)). Moreover, the method provides a satisfactory solution without incurring high computational costs in the analysis. TAT was outlined by Obert and Duvall (1967), and Salamon (1967) utilised the idea to design RPMS layouts in S. Africa. The limitations of this approach have been outlined by Brady and Brown (1985) as follows:

(i) The average axial pillar stress (which is regarded as representing the state of loading a pillar in a direction parallel to the principal direction of confinement) is not simply related to the variation in the state of stress in the pillar, which can be better determined by detailed stress analysis.

(ii) TAT focuses attention on the pre-mining normal stress component directed parallel to the principal axis of the pillar support system. This implies that stress components parallel to the orebody of the pre-mining stress are assumed to have no effect on pillar performance. This may not generally be true.

(iii) The effect of pillar location within the ore body is ignored.

PT also has a disadvantage: generally, a rock mass has very low tensile strength due to existence of joints, cracks or other planes of weakness. The common empirical approach, erring on the side of safety, is to assume the tensile strength to be zero. PT, however, assumes that rock strata in the immediate hanging wall of openings have a finite tensile strength. Because of the difficulty of assessing the actual value of tensile strength, the usual tendency is to use a high Factor of Safety (FOS) in the design procedure. Based on observed performances of immediate hanging wall strata *in situ*, Obert and Duvall (1967) suggest that the FOS value against plate failure in the roof should be between 4 and 10.

Despite the shortcomings discussed above, the design approaches based on TAT and PT have been successfully employed in several prominent mining areas, e.g. South Africa. They provide a useful and easy way of examining various possible extraction strategies, as a guide in determining the sensitivity of extraction ratio to changes in rock type and properties, mining layout, etc. It is therefore suitable for the consideration of the general viability of ore extraction with RPMS. However, if, due to specific location or rock properties, the layout requires special attention, then it might be appropriate to follow up the initial analysis with a more complex design technique, as previously mentioned, using the finite element analysis or boundary element analysis techniques.

6.4 Design Procedure

The design procedure adopted here considers four possible modes of failure:

(i) pillar failure;

(ii) bearing failure in roof;

(iii) bearing failure in floor; and

(iv) plate failure in roof.

The suggested approach is to select a suitable value of FOS, based on empirical knowledge or otherwise, for each mode of failure, to provide a basis for the design. Normally two FOS must be satisfied, i.e. the pillar strength FOS and bearing capacity FOS, but the analysis is based on three design criteria:

(i) the strength of the proposed pillars and, therefore, their ability to withstand the axial stress imposed on them;

(ii) the bearing capacity of the immediate roof and floor; and

(iii) the ability of the roof span to support itself.

It should be mentioned that there is another type of roof failure known as wedge failure. This arises when an opening intersects a plane or planes of weakness which may result from stratification in sedimentary rocks or may be due to joints or faults in other types of rocks. When anticipated, wedge failure can be prevented with rock bolts. Hoek and Brown (1980, p. 247) discuss the selection of rockbolts to suspend wedges which are free to fall from the roof of an underground excavation, and to reinforce wedges against sliding. The recommended practice is to select a rockbolt such that the total bolt load gives a factor of safety of 1.5-2. At Panasqueira, there are a series of joints parallel to the vein system. Despite these, Mendes *et al.* (1987, p. 1148) describe the rock mass as "having such characteristics that usually there are no problems in opening linear voids even when the sections are large. As a rule the roofs of such voids stand without support, sometimes with occasional bolting or some other kind of discontinuous support where the jointing seems unfavourable." Because the joints are parallel to the vein system, the planes of weakness are normal to the axis of pillars and therefore do not lower the pillar strength significantly (Obert and Duvall (1967, p. 549)). These observations do not exclude the need for more detailed analysis taking rock discontinuities into account when appropriate data are available.

6.4.1 Pillar Strength

A generally accepted method for calculating pillar strength, as proposed by Hardy and Agapito (1977), relates pillar strength to the uniaxial compressive strength of a sample of known dimensions — $1m^3$, for example. The resultant relation which can be applied as a scaling law is shown below:

$$S_p = S_s \left(\frac{V_s}{V_p}\right)^A \left(\frac{W_p}{H_p} * \frac{H_s}{W_s}\right)^B \tag{6.1}$$

where

S_p = strength of pillar;

S_s = sample's uniaxial compressive strength;

V_s = volume of sample;

V_p = volume of pillar;

W_p = width of pillar;

W_s = width of sample;

H_p = height of pillar;

H_s = height of sample;

and A and B are empirical constants which cannot be determined for an initial design. However, from a study on oil shale pillar performances the values suggested by Hardy and Agapito (1977) are $A = 0.118$ and $B = 0.833$. These are obviously not applicable at Panasqueira.

An alternative relation for determining the strength of pillars, which was used by Salamon and Munro (1967) is expressed by the empirical formula:

$$Sp = G{W_p}^T {H_p}^R \tag{6.2}$$

where G, T and R are constants. For square pillars, Salamon and Munro suggest the values of these constants to be 7.18, -0.66 and 0.46 respectively. The constants given so far must not be applied arbitrarily since different geomechanical environments in different locations can have different constant values. It is prudent to determine these values for the particular environment under consideration. *As far as Panasqueira is concerned, Mendes et al. (1987) have derived values for these constants, based on studies carried out at the mine (see*

Section 6.5.1). Thus, it is possible to incorporate Equation (6.2) in the design model.

Using the TAT, the average pillar stress for square pillars is given as:

$$\sigma_p = P_z \left(\frac{W_p + W_o}{W_p} \right)^2 \tag{6.3}$$

where

$$\begin{aligned}
\sigma_p &= \text{average pillar stress;} \\
P_z &= \text{vertical normal component of pre-mining field stress;} \\
W_p &= \text{width of pillar; and} \\
W_o &= \text{width of room.}
\end{aligned}$$

Figure 6.1 explains the geometry for the tributary area analysis. The Area Extraction Ratio (AER) can be defined as the ratio of area mined to the total area of the orebody:

$$\begin{aligned}
\text{AER} &= \frac{(W_o + W_p)^2 - W_p^2}{(W_o + W_p)^2} \\
&= 1 - \frac{W_p^2}{(W_o + W_p)^2} \\
&= 1 - \frac{P_z}{\sigma_p}
\end{aligned} \tag{6.4}$$

The pillar strength FOS is given by the following relation:

$$FOS = \frac{S_p}{\sigma_p}$$

and it is recommended that $1 < FOS < 2$, based on several investigations carried out on the stability of pillars by Salamon (1967). The commonly used value of the pillar strength FOS is 1.6 (see Figure 6.2).

6.4.2 Bearing Capacity of Roof and Floor

The load applied by a pillar to both the footwall and hanging wall in a stratiform orebody can be compared directly with a distributed load applied on the surface of a half span. A useful method of calculating the load bearing capacity was proposed by Brinch (1970). This method expresses the bearing capacity in terms of pressure or stress and assumes that the average axial pillar stress is equivalently applied as a uniformly distributed normal load to the adjacent

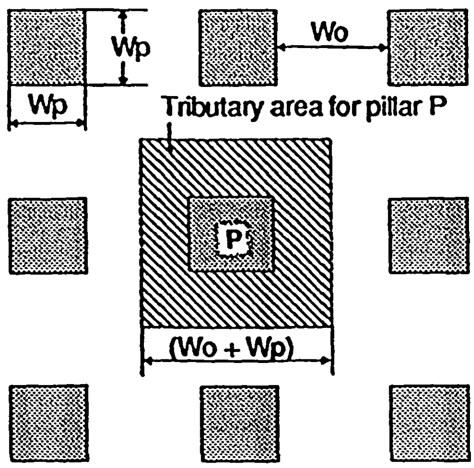


Figure 6.1. Geometry for tributary area analysis of pillars in uniaxial loading (after Brady and Brown (1985)).

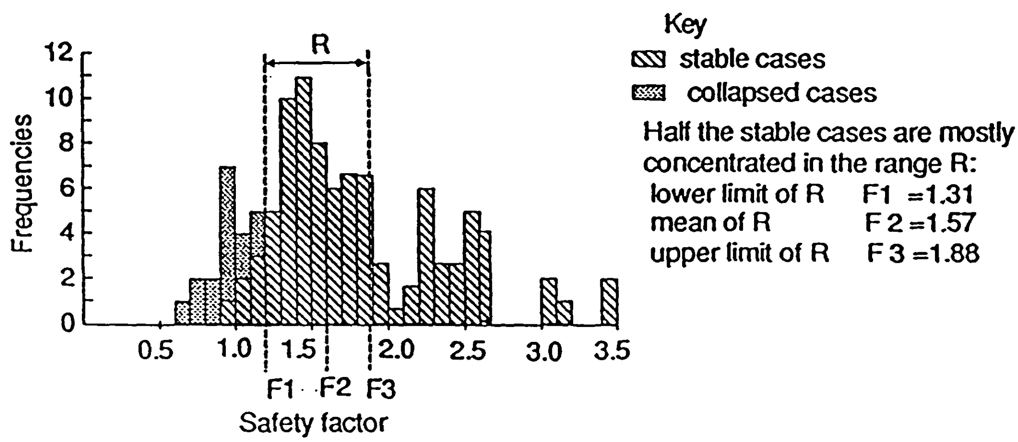


Figure 6.2. Histogram showing frequencies of intact pillar performance, and plliar failure (after Salamon and Munro (1967)).

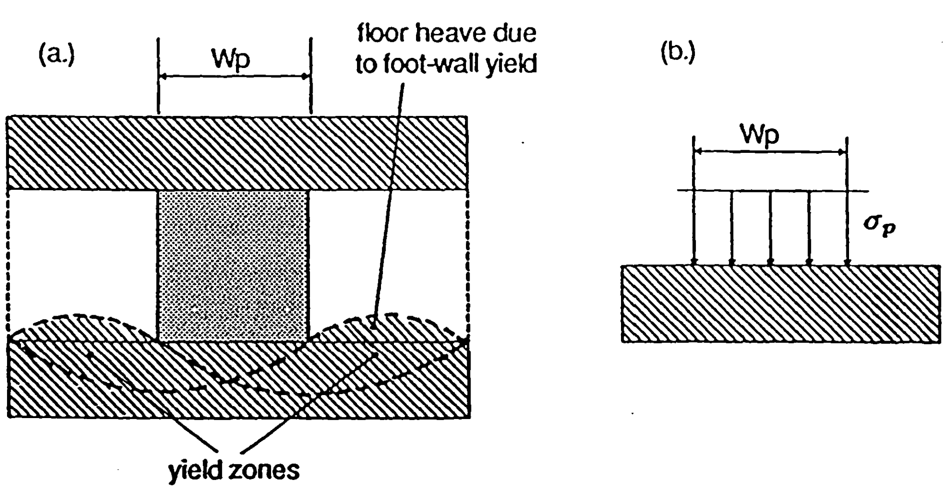


Figure 6.3. Model of yield of : (a.) country rock under pillar load and (b.) load geometry for estimation of bearing capacity (after Brady and Brown (1985)).

rock. Schematic and conceptual representations of the problem are illustrated in Figure 6.3. Under the above assumption, the bearing capacity, as given by Brady and Brown (1985), is:

$$q_b = \frac{1}{2}\left[\gamma W_p N_\gamma S_\gamma + 2C \cot\phi (N_g S_g - 1)\right] \tag{6.5}$$

where N_γ and N_g are strength factors and S_γ and S_g are shape factors and given by:

$$\begin{aligned}
N_g &= e^{\pi \tan\theta} \tan^2(45 + \frac{\phi}{2}),\\
N_\gamma &= 1.5(N_g - 1)\tan\phi,\\
S_\gamma &= 1.0 - 0.4(\frac{W_p}{W_l}),\\
S_g &= 1.0 + \sin\phi(\frac{W_p}{W_l}),
\end{aligned}$$

where

$$\begin{aligned}
\gamma &= \text{density of floor rock},\\
W_p &= \text{width of pillar},\\
W_l &= \text{length of pillar},\\
\phi &= \text{angle of friction, and}\\
C &= \text{cohesion}.
\end{aligned}$$

The FOS against bearing capacity failure is:

$$FOS = \frac{q_b}{\sigma_p} > 2.0$$

6.4.3 Stability of Opening Room Span

The estimation of a maximum stable room span is based on the assumption that the immediate roof consists of laminae which can therefore be treated by means of the PT. The approach is suitable for seams of moderate thickness as the assumption may not hold for very thick seams. It is thus directly applicable at Panasqueira where the stopes are of limited height.

Using the PT, Obert and Duvall (1967, p. 552) derive that the maximum tensile stress due to gravity at the centre of a plate clamped at all edges is given by the expression:

$$\sigma_t = \frac{6\beta\gamma S^2}{t} \tag{6.6}$$

where β = a constant related to the shape of the plate and can be taken as 0.0513 for square pillars, γ = unit weight of the plate material i.e. density, S = span, t = thickness of immediate roof or plate laminae. If a stope is inclined such that the angle between the roof and the horizontal is θ, then the expression for σ_t must be multiplied by $\cos\theta$. For $\theta \leq 10^0$, as is the case at Panasqueira, the stope can be considered to be horizontal and Equation (6.6) is valid.

As a very general guide, the modulus of rupture can be taken to be: $R_o = \frac{1}{10}$ of the uniaxial compressive strength. The expression for the FOS against plate failure is given as:

$$\text{FOS} = \frac{R_o}{|\sigma_t|}$$

For reasons discussed earlier, the value of FOS against plate failure is recommended to be between 5.0 and 8.0

6.5 Features of the Design Model

Using the concepts discussed in the preceding paragraphs, a model (a computer program PILDESN) for the initial design of a room and pillar layout was developed to analyse the current room and pillar layout at Panasqueira. A simplified flowchart of the model is shown in Figure 6.4. It has two main parts (Mireku-Gyimah *et al.* (1988)):-

(i) The model attempts to calculate pillar dimensions on the basis of the required pillar strength FOS. For this an iterative solution is necessary and the Newton-Raphson method (Fermer (1974)) is employed. Using the pillar dimensions now available, the bearing capacity FOS is calculated and compared with the required bearing capacity FOS. If the outcome is satisfactory, the recovery ratio is calculated and results are printed out.

(ii) If the result is unsatisfactory, the second part of the model calculates pillar dimensions on the basis of the required bearing capacity FOS, again employing the Newton-Raphson iterative method. The resulting pillar dimensions are used to calculate the pillar strength FOS which is then checked against the required value. If the outcome is satisfactory the recovery ratio is computed.

In all cases, the stability of roof span is verified using the relation expressed by Equation (6.6). The following input data are required:

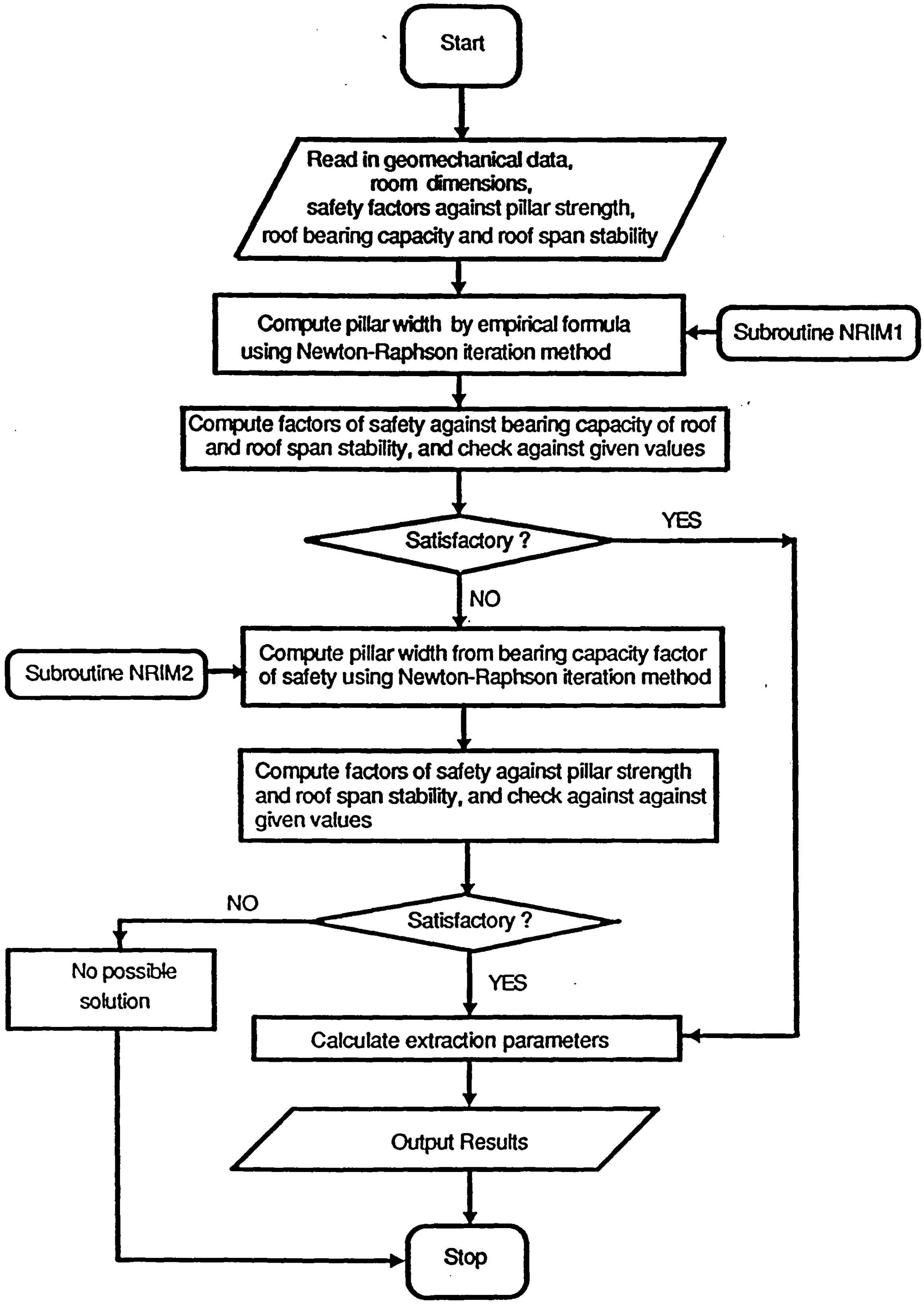


Figure 6.4. A simplified flowchart of the room and pillar layout design model PILDESN.

(i) depth of mine;

(ii) density of rock;

(iii) compressive strength of rock;

(iv) initial room dimensions;

(v) pillar strength FOS;

(vi) values of constants G,T and R (as in Equation 6.2);

(vii) angle of friction;

(viii) rock cohesion;

(ix) bearing capacity FOS; and

(x) roof span FOS.

If experiments have been conducted on a sample to determine its compressive strength, then the following may also be supplied otherwise the program automatically uses Equation (6.2) as the alternative relation to compute the pillar strength:

(i) dimensions of sample;

(ii) uniaxial compressive strength of sample;

(iii) density of sample; and

(iv) values of constants A and B (as in Equation (6.1)).

The analysis also requires input of the desired room span and, in order to allow a wide variety of conditions to be examined, the model has the facility to vary the following:

(i) room width;

(ii) pillar height; and

(iii) pillar strength FOS.

6.6 Model Application

The room and pillar layout design model was applied to the Panasqueira ore deposit. To date, no *in situ* tests have been undertaken to permit the use of boundary or finite element methods to study the state of total stress and induced displacements around rooms. However, Mendes *et al.* (1987) have carried out several tests and studies, and derived useful goemechanical characteristics of the rocks. From this and other sources, values were taken for the present analysis and can be summarised as follows:

(i) uniaxial compressive strength averages at about 100 MPa;

(ii) density is 0.0275 MN/m^2;

(iii) internal angle of friction is $40 - 57^0$;

(iv) the depth of the mine is 300 m;

(v) referring to equation (6.2), the constants G, T and R have the values 62, 0.46 and -0.66 respectively (Mendes *et al.* (1987)); and

(vi) cohesion was taken to be $50MPa$ as suggested by Kulhaway (1975).

Since all other necessary input values can be varied, the program was run for each of the following cases to find the corresponding extraction parameters:

(i) varying the pillar height at constant room width and pillar strength FOS;

(ii.) varying the pillar strength FOS at constant room width and pillar height; and

(iii.) varying the room width at constant pillar height and pillar strength FOS.

The results of the analysis are summarised in Figures 6.5, 6.6 and 6.7. Only the pillar strength FOS is featured in the results, but the reader is reminded that the values shown in the graphs also satisfy the recommended factors of safety against roof and floor bearing capacities, and against plate failure in the roof, these verifications being part of the design model's features as described previously.

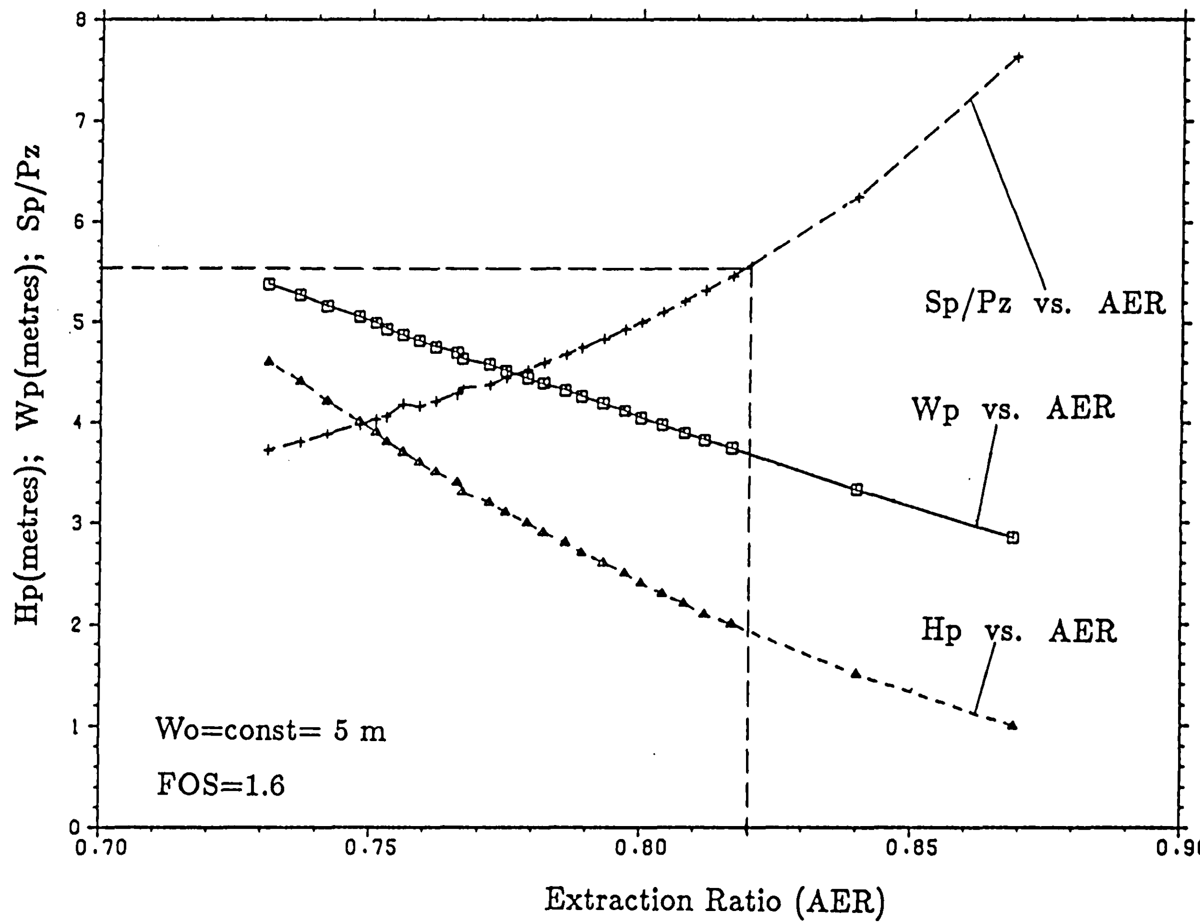


Figure 6.5. Variation of pillar height (Hp), pillar width (Wp) and stress concentration factor (Sp/Pz) with extraction ratio (AER) at constant room width (Wo) and pillar strength factor of safety (FOS)

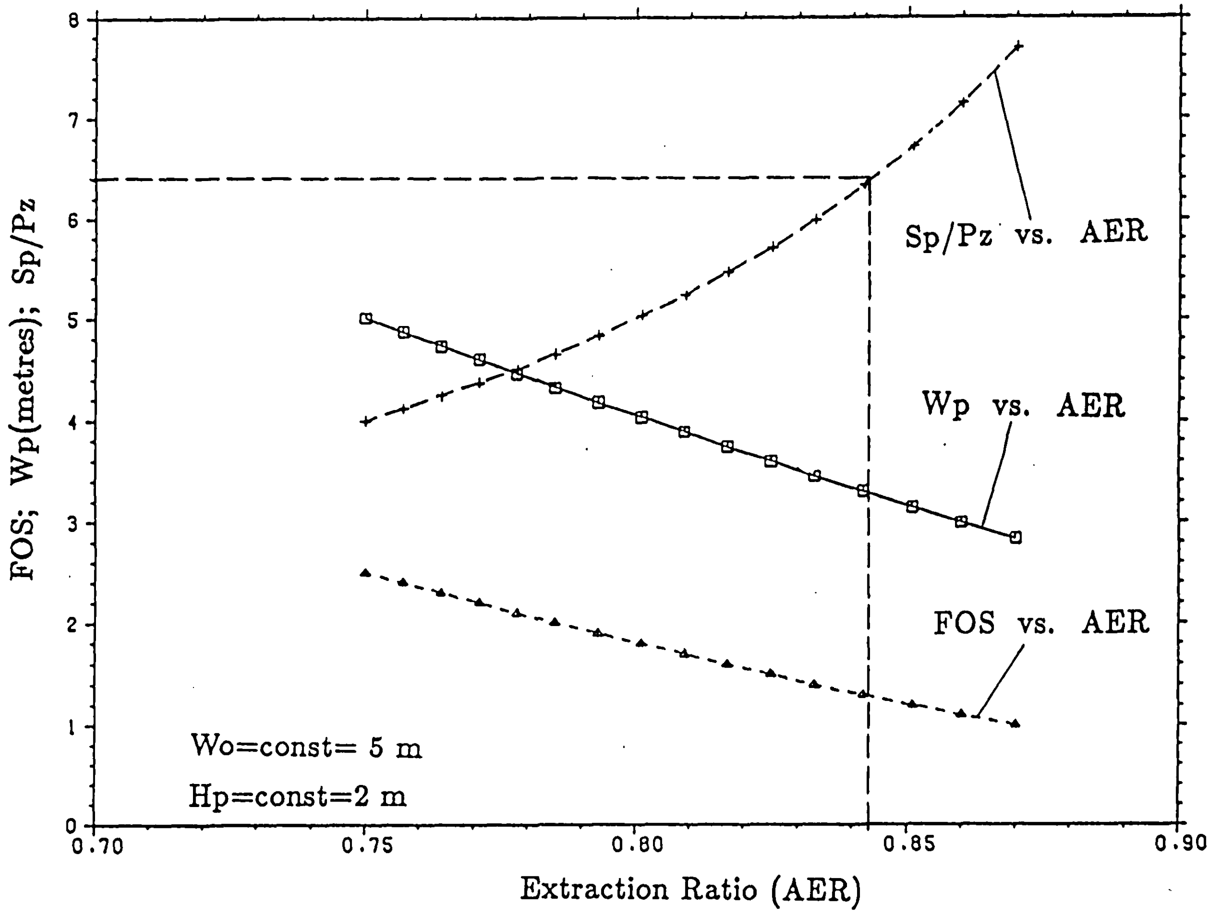


Figure 6.6. Variation of pillar strength factor of safety (FOS), pillar width (Wp) and stress concentration factor (Sp/Pz) with extraction ratio (AER) at constant room width (Wo) and pillar height (Hp).

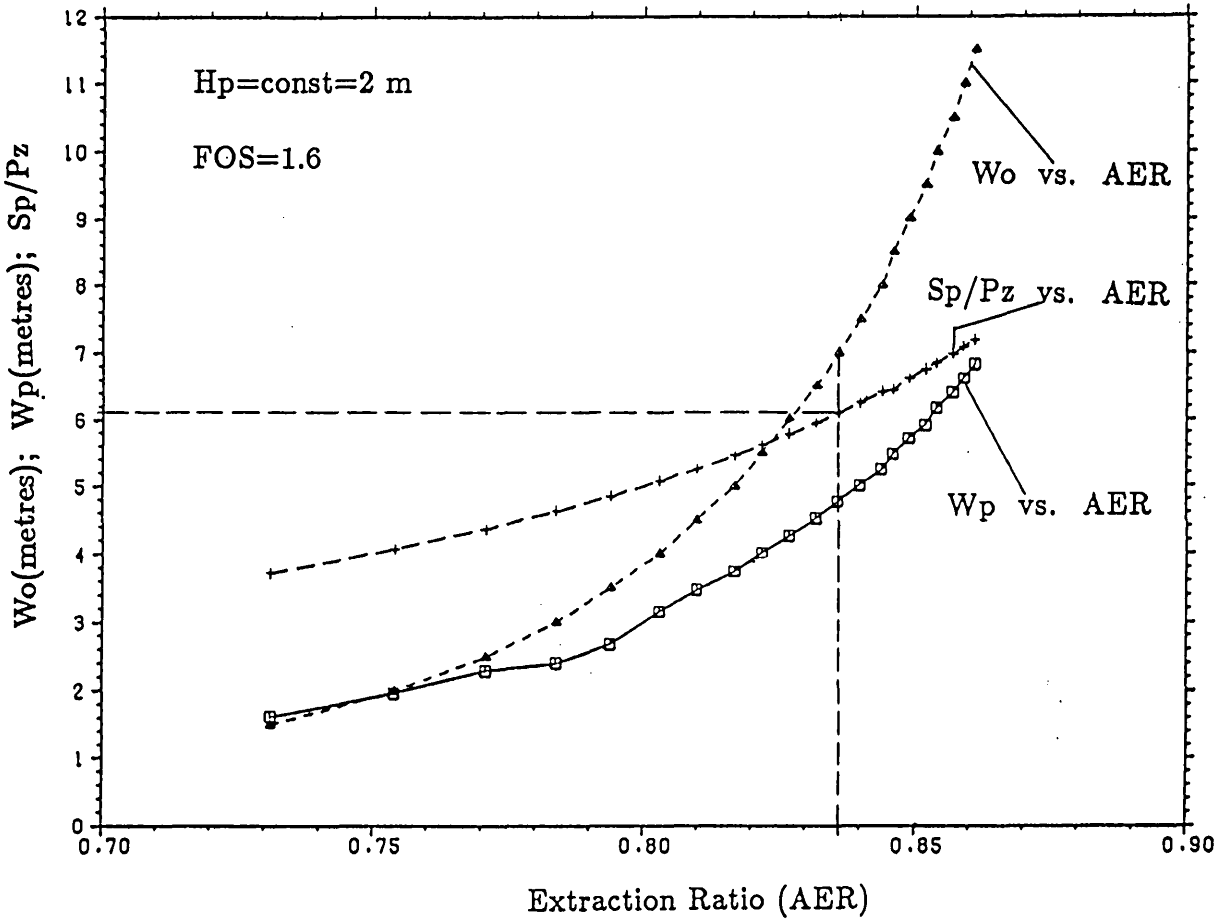


Figure 6.7. Variation of room width (Wo), pillar width (Wp) and stress concentration factor (Sp/Pz) with extraction ratio (AER) at constant pillar height (Hp) and pillar strength factor of safety (FOS).

Figure 6.5 illustrates the variant where the room width and the pillar strength FOS were kept constant at 5 m and 1.6 respectively while varying the pillar height. In this case, it is observed that at smaller pillar heights and widths, the extraction ratio is high, but the stress concentration factor is also high and the pillar could fail under stress. Increasing the pillar height will enhance mechanisation, but requires a corresponding increase in pillar width to maintain pillar stability with consequent reduction in the extraction ratio. When the pillar height is kept at 2 m, a pillar width of 3.74 m is necessary to maintain the safety factor of 1.6, the extraction ratio is 82% and the stress concentration factor is 5.5 which is nominal. Increasing the pillar height to say 2.3 m requires that the pillar width must be increased to 3.97 m to maintain the same factor of safety of 1.6 and the extraction ratio falls to 80%. It is interesting to note that at Panasqueira 3 m by 3 m pillars are left when the room width is 5 m and the stoping height is 2 m or even 2.5 m. This, of course, means that the pillar strength factor of safety is lower than the recommended value of 1.6.

In Figure 6.6, the room width and pillar height are kept constant at 5 m and 2 m respectively while varying the pillar strength factor of safety. Even at the minimum recommended pillar strength factor of safety of 1.31 (see Figure 6.2), a pillar width of 3.3 m is required to maintain the pillar strength. Under this situation, the stress concentration factor is 6.4 and the extraction ratio is 84.2%. The 3 m square pillars currently in use at Panasqueira corresponds to a factor of safety of 1.10 which is the lowest acceptable safety factor for pillar stability. Therefore, the pillars cannot be considered to be very stable unless supported by some strong evidence, e.g. practical observation. Any further increase in stoping height above 2 m without an increase in the pillar width is bound to reduce the factor of safety to a value below the 1.10 limit, into the region of instability (see Figure 6.2).

In Figure 6.7, the pillar height and the pillar strength factor of safety are kept constant at 2 m and 1.6 respectively while varying the room width. In this case, it would appear that higher and higher extraction ratios can be achieved simply by increasing the room width since an increase in the room width requires a relatively small increase in pillar width to maintain the pillar strength factor of safety of 1.6. However, there is a limit to this increase in room width because the stress concentration factor also increases. For example, when the room width is increased to 7 m, a pillar width of 4.77 m is enough to maintain stability. The corresponding extraction ratio is 83.6%. A further increase in the room width to say 11.5 m requires the pillar width to be increased to 6.6 m to maintain the factor of safety at 1.6. The extraction ratio increases to 85%, but the stress concentration factor is nearing 7 and the room span stability is being sacrificed.

If the mine were completely undeveloped with no experience of the natural behaviour of the rocks, it would be appropriate to start with any of the combinations of room width, pillar width and stoping height which satisfy a factor of safety of 1.6, and then modify these parameters as experience builds up. In the particular case of Panasqueira, some experience has been gathered over the years and before suggesting a suitable combination of the room and pillar layout design parameters, it is necessary to take this experience into account.

6.6.1 Practical Experiences at the Mine

Prior to 1975, long-wall stoping was the principal mining method employed at Panasqueira (see Chapter 2). Timber packs provided support for the back of the stope and a system of pack-walls parallel to the face served as a means of minimising loss of fines during blasting. Hand-held drilling machines on air-leg supports were used. The stoping height was kept around 1.5m. This long-wall mining method was abandoned in favour of RPMS because of the increasing cost of timber and labour. However, a useful observation had been made: no roof deformation occurred due to discontinuities in the rocks during stoping operations. However, deformation was observed after a critical span of about 50 m, which developed gradually into the middle of the span.

The recent room and pillar mining method started on a test basis with 5 m wide rooms and pillars measuring 15 m by 15 m which were later reduced to 5 m by 5 m, corresponding to an extraction ratio of 75%. In this method the stoping height was increased to approximately 2 m and low profile trackless tramming equipment and electric-hydraulic jumbo drilling rigs were employed. Over the years, the pillars have been reduced to 3 m by 3 m in an attempt to increase the extraction ratio. According to Mendes *et al.* (1987), these 3 m by 3 m pillars are observed to be stable. However, during the author's visit to the mine, some pillars on Level 2 were seen to have failed and roofs had fallen, denying access to certain working places.

Occasionally, either to extract more than a single vein or to further facilitate equipment movement, the stoping height is increased to as much as 2.5 m. The analysis carried out in this Chapter shows that at a stoping height of 2.3 m, the pillar width must be increased to 3.97 m to maintain the pillar strength factor of safety of 1.6. If the pillar width is not increased, then the pillar strength factor of safety is bound to fall below the lowest limit of 1.10 at which pillars are observed to be stable.

For the purposes of mine planning, the deposit has been divided into a series of contiguous blocks measuring 100 m by 50 m, these being considered to be the practical mining units (see Chapter 4). This arises from the fact that the wolframite mineral occurs erratically as individual crystals or groups of crystals within the quartz veins, making normal chemical assay methods impracticable. Measurements of the areas of exposed crystals in the side walls of mining faces or development ends are used for grade and hence ore reserve estimation. Smaller blocks would require a major increase in the extent of reef drive development necessary for sampling and reserve estimation. It is obvious therefore that the final design of room and pillar layout for the exploitation of the veins should take these pre-ordained blocks into account.

6.6.2 Design Parameters

On this framework and with the results from the model analysis, it can be shown that, 5 m wide rooms can be supported by 3.74 m square pillars, when the stoping height is kept at 2 m, to maintain the recommended pillar strength factor of safety of 1.6. Under these circumstances, an extraction ratio of 82% is achievable. However, since the rocks are clearly stable, the current pillar size of 3 m by 3 m can be maintained, but it is suggested that extra pillars be left whenever waste material occurs within the mining area, or, indeed, at distances not exceeding the critical span beyond which the 3 m by 3 m pillars may fail under shear stress — according to Mendes *et al.* (1987), this is about 200m.

To safeguard against chances of the 3 m by 3 m pillars failing, and for purely geometrical reasons associated with the 100 m by 50 m mine blocks, it is suggested that the peripheral pillars of a block should be increased in size as shown in Figure 6.8 which is the proposed layout design for a block. This layout gives an overall extraction ratio of 84.6%.

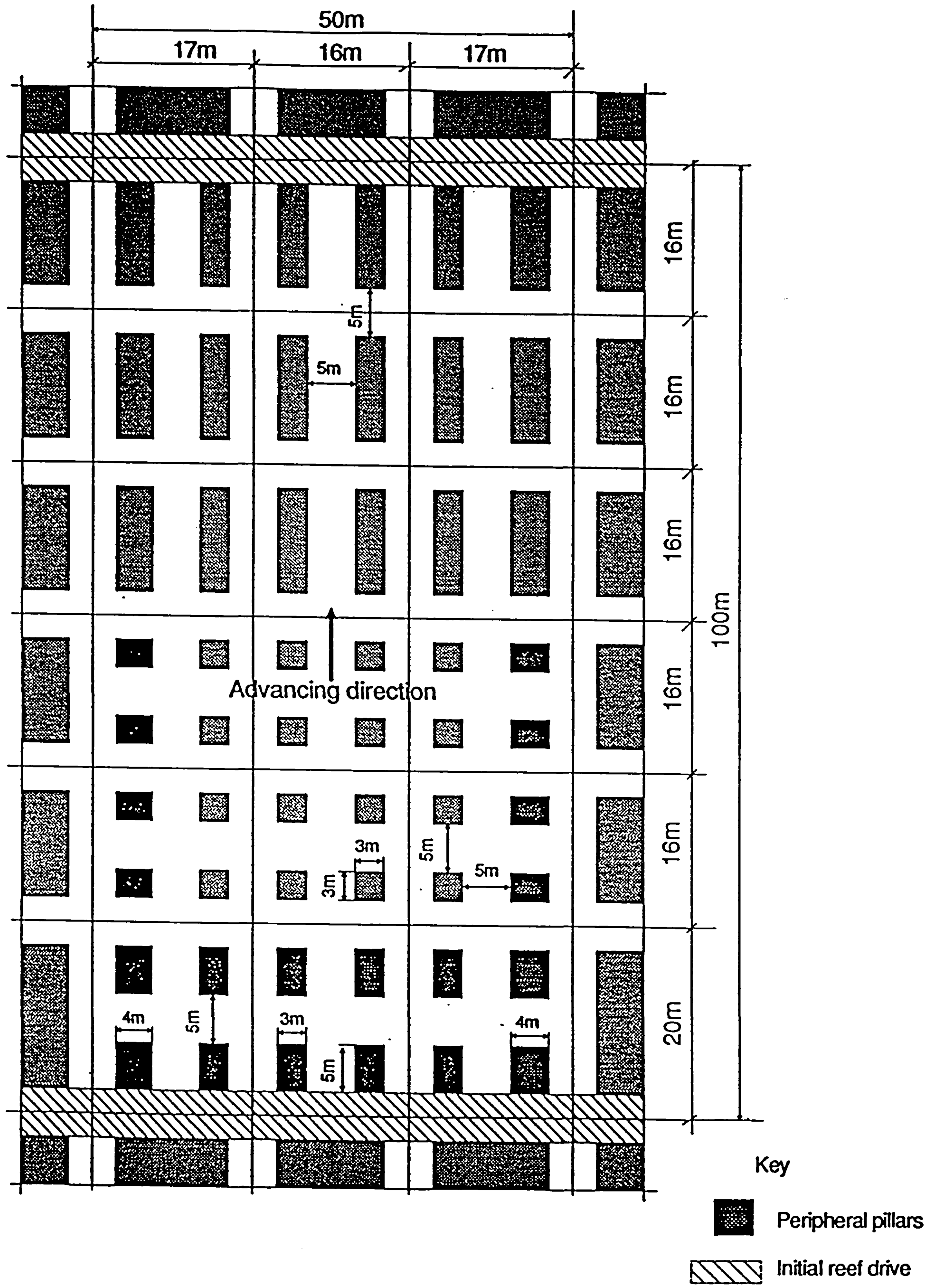


Figure 6.8. Design of the room and pillar layout for a block.

6.7 Summary

In this Chapter, a model (program PILDESN) for the preliminary design of a room and pillar layout has been developed from existing design theories. The model has shortcomings which arise from assumptions made within the TAT and PT. Again, like all models, the quality of the results obtained from its application depends on the validity of the input data. Given sensible input values, however, the model does provide a good guide for the initial design of a room and pillar layout.

In the case studied, the model indicates that, the 5 m wide rooms at Panasqueira can be supported with 3.74 m by 3.74 m pillars, when the stoping height is 2 m, to maintain the recommended pillar strength factor of safety of 1.6. The corresponding extraction ratio is 82%. However, practical experiences at the Mine suggest that the pillar size can be reduced to 3 m by 3 m. Even so, it is suggested that, to minimise the risk of pillar failure and for purely geometrical reasons, the size of peripheral pillars of the pre-ordained blocks must be increased.

The extraction ratio in that case is 84.6%. It will be necessary to review the analysis as the mine becomes deeper since the average pillar stress increases directly with depth.

Increasing the stoping height would require an increase in pillar width to maintain the recommended pillar strength factor of safety of 1.6. This in turn reduces the extraction ratio. For example, increasing the stoping height to 2.3 m requires that the pillar width be increased to 3.97 m. Consequently, the extraction ratio reduces to 80%. Again since quartz material currently mined constitutes only an average of about 15% of the material being mined, an increase in stoping height will introduce an unnecessary dilution likely to introduce problems at the processing plant, and an added cost in ore production. It is therefore suggested to maintain the stoping height of 2 m since this can allow equipment movement.

In the next chapter, a summary of all the observations made and the conclusions drawn during the course of this research project are presented. Suggestions for further research at Panasqueira are also made.

CHAPTER 7

SUMMARY AND RECOMMENDATIONS

7.1 Summary and Conclusions

The Panasqueira mine in Portugal is the major producer of tungsten in Europe. The principal ore mineral (wolframite) occurs erratically as individual crystals or clusters within a series of near horizontal, hydrothermal quartz veins in a schistose country rock — see Figure 2.5. This has made conventional sample assay methods impractical. Measurements of areas of the wolframite crystals exposed in the side walls of mining faces or development ends have been utilised to estimate ore reserves, using an empirical factor known as the Mineral Evaluation Factor — see Chapter 1. Any meaningful mine design and planning work therefore depends primarily on the ability to expose the quartz veins, and hence provide reliable estimates of grade and ore reserves.

To date it has not been possible, geologically, to connect, interpolate or correlate the individual vein intersections observed in diamond drill cores and thereby delineate possible mineable areas. In practice, a single promising vein is developed and followed, its direction being inferred from past experience and information at the face. The complexity of the Panasqueira vein system and the concomitant difficulty of interpolating the vein intersections have led to much development in unprofitable ground.

The primary problem which this research has therefore addressed is the possibility of evolving a fast and reliable process, with a sound scientific support, based on the vein intersections, to delineate the ground which can be developed to expose the maximum quantity of quartz veins.

Due to the increasing cost of labour and timber, Panasqueira replaced its long-wall mining method with a mechanised room and pillar mining system twelve years ago. To support 5 m wide rooms, 15 m by 15 m pillars were initially left and eventually reduced to 5 m by 5 m pillars. Over the years, the pillar size has been reduced to 3 m by 3 m, in order to increase the ore extraction ratio. To facilitate mechanisation, the stoping height has been increased from 1.8 m to 2 m. Occasionally, either to be able to mine more than a single vein or to further facilitate mechanisation, the stoping height is increased to as much as 2.5 m.

Another problem, which has been addressed in the latter part of the research, is to examine the stability of the present 3 m by 3 m pillars and to find out whether the advantages of increasing the stoping height outweigh the associated disadvantages.

In order to provide solutions to the problems mentioned previously, the aim of this research has been: firstly, to develop a computational methodology to identify areas of optimal quartz content, and hence delineate possible mineable horizons for subsequent planning of development work that will expose the veins; and secondly, to analyse the current room and pillar layout design. The research has been based mainly on diamond drill logs supplied by the Geology Department at Panasqueira, as well as available geomechanical properties of the rocks. The logs consist essentially of vein widths and the x-, y-, z-co-ordinates at which they were intersected.

In the absence of distinguishing lithological or mineral characteristics by which the vein intersections can be extrapolated, or by which zones of common characteristics can be delineated, the procedures and some of the subjective judgements used by the mine geologists to identify areas to be developed have been translated into simple rules and coded into a program — MBANDS. This is an interactive program that generates a section showing the vein intersections. The user is allowed to select vein intersections which are then checked against a set of rules that may or may not permit their connection to enable possible mineable areas to be delineated. Thus, the program emulates the interpolation and delineation process at the mine; its advantage lies in the fact that it is fast and can be used to quickly define possible mineable ground and hence assist in mine planning.

However, due to the capricious nature of the vein occurrences, and because not all intuitive judgements can be qualified, different users of the program may arrive at different ways of connecting the vein intersections in the same section. Clearly, therefore, there is the need for a more scientifically based interpolation process. In this aspect, geostatistical techniques have been employed, in Chapters 4 and 5.

In Chapter 3, it has been demonstrated that the vein occurrences have Markovian properties and hence, by superimposing a grid on the sections showing the vein intersections, it is possible to identify matrices representing the different geological states, i.e. the quartz veins and the schistose country rock. The resulting matrix is diagnostic of the pattern of vein occurrences in the plane of the section and can be analysed to establish the probabilities of vein occurrences in a given direction at distances corresponding to the resolution of the grid. With the statistics from a known section, it is possible to generate a simulated pattern of veins that represent the likely vein occurrences in a different section. The program, MAK2, used for this analysis, is based on Switzer's concepts of extending the Markov phenomenon to planar patterns.

The results of such an analysis carried out during the research show that once a vein is intersected there is a 0.5 probability of it continuing to be present through a distance of 12 m, or that another vein will be intersected through that distance, in the near horizontal direction, but only a 0.07 probability in the vertical direction. This observation is very useful in that it provides a good guide as to when to abandon the development of a vein which suddenly disappears — an event that is not uncommon at Panasqueira. Upon this evidence, it is suggested that if, for example, during the development of a reef drive the quartz vein should suddenly disappear, the development work should continue for at least 12 m before being abandoned.

The vertical separation between successive veins has been derived, analysed statistically and found to be log-normally distributed. Using Sichel's t-estimator, the mean vertical separation has been estimated to be 6.6 m with a lower limit of 6.4 m and an upper limit of 6.9 m. This information is valuable at Panasqueira where a room and pillar mining method is employed in extracting the veins. Veins that are very close can be mined together. On the other hand, the vertical separation between veins that will be mined separately must be enough to provide the necessary support in the form of crown and sill pillars. A mean vertical separation of 6.6 m suggests, for example, that room and pillar mining, as pertains at Panasqueira, will be successful if and only if crown and sill pillars measuring 6.6 m or less are stable.

The vein widths intersected within the study area have also been statistically analysed and found to be approximately, log-normally distributed. Again using Sichel's t-estimator, the mean vein width has been estimated to be 0.165 m with a lower limit of 0.157 m and an upper limit of 0.175 m. If the veins are to be mined individually, as is the case at present, then an economically important aspect of mineralisation is the average quantity of the quartz vein which is available at the expected grade. Since at Panasqueira the grade is unknown until the veins are exposed, and the average quantity of veinage is directly proportional to the vein width, the mean vein width becomes a dominant factor.

It has been noted in Chapters 2, 4 and 5 that only veins with widths of 0.20 m or more are currently mined at Panasqueira. Moreover, the average width of veins mined is reported to be 0.30 m. These two facts clearly indicate that the veins selected for mining are picked from the tail of the positively skewed vein width distribution (see Chapter 3). Thus, it can be concluded that the majority of veins are left unmined simply because they are thin. The fact still remains, however, that if these veins can be mined together, then their aggregate width, and hence the quantity of quartz obtainable, can be equal to, or even greater than, that obtained by mining single veins.

A more understandable variable that has been derived and considered as constituting a measure of mineability is the *vein concentration,* defined as the vein width accumulation per 2 m height of ground. The word concentration is, therefore, not meant to suggest grade or any unit measure of the mineral content, i.e. saleable product content of the ground. However, an *a priori axiom* of the Mine holds that the more quartz mined the more wolframite obtainable. Consequently, the aggregate quartz content of the ground mined is of prime importance, at least, until the grade is known.

The vein concentration has been statistically analysed and also found to be approximately, log-normally distributed. The mean value of the vein concentration has been estimated to be 0.178 m $\approx$ 0.18 m with a lower limit of 0.170 and an upper limit of 0.186 m. Since the minimum width of veins included in the Mine's geological reserves is 0.18 m (see Chapter 5), there is no reason why the ground containing a vein concentration of at least 0.18 m cannot also be included in the geological reserves. Thus, more ground is expected to become mineable if the vein concentration is considered as the measure of mineability rather than the vein width.

In the subsequent semi-variogram analysis, carried out in Chapter 4, it has been shown that for both attributes of the deposit, i.e. the vein width and the vein concentration, there is a well-defined geostatistical structure.

In the case of the vein width, the semi-variogram has features conforming to that of the spherical model with a nugget variance of 0.015 m^2, a transition variance of 0.007 m^2 and a range of 47 m.

The features of the semi-variogram for the vein concentration also conform to that of a spherical model with a nugget variance of 0.017 m^2, a transition variance of 0.009 m^2 and a range of 48 m.

The high nugget variance associated with these semi-variograms has been attributed to the fact that Panasqueira veins can thin off, or split up abruptly, giving rise to erratic vein widths. This capricious behaviour has genetic origins: the random dilation of pre-existing joint fractures caused by irregular movements of rocks under a series of tectonic and hydraulic forces.

The well-defined structure observed in the deposit, with respect to both attributes, i.e. the vein width and the vein concentration, justifies the use of geostatistical methods to estimate either property. However, since the vein concentration is a better representation of the material mined, it is this variable that has been estimated on a block by block basis. A linear kriging technique, which is incorporated in the MINPAK library of geostatistical routines, has been used in the estimation process because it is accepted in the mineral industry to give the best unbiased estimates.

Each of the estimation blocks measures 100 m by 50 m by 2 m. These blocks are pre-ordained by the Mine and considered to represent the level of resolution that is vital for effective control in mine planning. Each block's estimate, in turn, has been calculated using linear kriging on samples identified as falling within the bi-conical search volume for that specific block (see Chapter 4). This ensures that samples selected for estimation do not belong to vertically adjacent veins.

The resulting vein concentration model has been used as the basis for a step by step selection of possible development horizons, using the interactive graphics program, BKPLOT, that has been specially developed for this purpose. The user of the program aims at satisfying two criteria:

(i) a selected cut-off value of 0.18 m, this being the mine's figure for declaring geological reserves and used here so that results can be compared with the mine's plans; and

(ii) the contiguity of selected blocks in the horizontal plane to enable a smooth application of the mechanised room and pillar mining method currently in use at the mine.

Consequently, it has been shown that within the study area blocks with vein concentration of at least 0.18 m form clusters which define 6 mining horizons. There is evidence, from sections generated to show the spatial arrangement of these blocks, to suggest that within every drop of 10 m in elevation, starting from the top at 620 m to the bottom at 560 m, one of the 6 clusters of blocks is encountered, forming a well-defined mining horizon.

Based on these sections, it has been possible to delineate areas which contain contiguous blocks indicating where reef drives can be developed to expose the veins for subsequent sampling by the mineral measurement method. It is suggested that, during development, pilot holes should be drilled to further establish the continuity of the veins. It is not recommended that development be abandoned simply because a single thick vein splits up into thin ones, or even thins off, because it is likely that a thick vein or several thin ones can be encountered even in the same block.

Plans showing the distribution of blocks differ from horizon to horizon. In some horizons, e.g. Horizon 1, blocks with higher quartz content are spread evenly over the whole area under study. In others, e.g. Horizon 2, the blocks with higher quartz content are mainly in the NW quadrant of the area. On the whole, however, it is observed that blocks along the eastern boundary are mostly waste, especially around the NE corner. The expected tonnages of ore and quartz increase from Horizons 1 to 4 and then fall significantly in Horizons 5 and 6. Horizons 3 and 4 are observed to contain the highest tonnages of ore and therefore at the same rate of mining they should remain longer in production. Horizons 5 and 6 have the lowest tonnages of ore and should be expected to last for shorter times at the same rate of mining unless further drilling or development reveals more vein occurrences or unless the grade is so high as to warrant a low run-of-mine throughput.

The 6 mining horizons identified are in line with recent ideas developed at Panasqueira, but the tonnages of quartz estimated in this project for each of the horizons are greater than those determined at the mine. The reason for this lies in the fact that while the Project has estimated *vein concentration* the Mine's figures refer to vein widths. It is therefore suggested that the Mine underestimates the mineable potential of the ground.

Finally, it has been demonstrated that it may be possible to plan secondary development work to expose quartz veins, based on the vein concentration estimates. The program, DPLAN, developed to calculate the yearly development work is based on the understanding that mining can start only after the exposed quartz has been sampled and found to have a grade value of at least 13 kg/m^2. Consequently, the development should be well in advance of mining. It has been assumed here that tonnages resulting from development of the reef drives constitute part of a year's production, that the total production is from the study area, and that the yearly development advance must make enough blocks available to replace a year's production. Again, the ramp, which provides access for equipment from horizon to horizon, has been assumed to be part of the secondary development so as to enable the calculation of the marginal development necessary for the study area. In this way it has been possible to work out a simple development schedule to meet the production objective. Development work is expected to be intensified on those horizons with the least tonnage of quartz.

The techniques adopted in this project for the estimation and delineation of mineable ground offer a creditable alternative to those employed at the mine. They are scientifically based and provide objective and reproducible results. Moreover, they are versatile enough to allow engineering judgements to be exercised.

It must be emphasised that if the steps taken in this project to identify areas of optimal quartz content are to be successfully employed at Panasqueira, then it is imperative that all vein widths, however small, be considered to be equally important since the measure of mineability has been based on the aggregate vein width per 2 m height of ground, i.e. the vein concentration.

In the latter part of the project, the current room and pillar layout at the mine has been investigated. The model PILDESN used for this analysis has been developed on the basis of existing design theories, i.e. the Tributary Area Theory (TAT) and the Plate Theory (PT). The model has some short-comings due mainly to several assumptions within the original theories. It does, however, provide a good guide for initial design of room and pillar layout.

In the case studied, the model indicates that the 5 m wide rooms at Panasqueira can be supported with 3.74 m by 3.74 m pillars when the stoping height is 2 m, to maintain the recommended pillar strength factor of safety of 1.6. The corresponding extraction ratio is 82%.

Practical evidence at the Mine suggests that the pillar size can be reduced to 3 m by 3 m. It is suggested that, to minimise the risk of any failure and for purely geometrical reasons, the peripheral pillars of the pre-ordained blocks must have a larger size as shown in the suggested layout design (see Figure 6.8). The overall extraction ratio in this case is 84.6%. It will be necessary to review this analysis as the mine becomes deeper since the average pillar stress increases directly with depth.

With regard to increasing the stoping height, the analysis carried out in this project has shown that such an increase requires a corresponding increase in pillar width to maintain the recommended pillar strength factor of safety of 1.6. This in turn reduces the extraction ratio. For example, increasing the stoping height to 2.3 m requires that the pillar width be increased to 3.97 m as a consequence of which the extraction ratio reduces to 80%. With a 2 m stoping height and assuming a 0.2 m cut-off vein concentration, quartz constitutes an average of 15% of the ground mined. An increase in stoping height serves only to introduce unnecessary dilution which may cause problems at the processing plant and raise the unit cost of production. It is suggested therefore that the stoping height of 2 m be maintained since this affords adequate head room for most underground equipment.

7.2 Recommendations for Further Research

(i) A useful line of research could involve an investigation into the possibility of using an expert system driven program to connect the individual vein intersections. An approach could be to set up a questionnaire on the behaviour of veins observed. This questionnaire must be answered by the geology and mining staff. In this way, it is possible to build over the years a database of rules which can then be used by the expert system.

(ii) The possibility of employing a sub-level caving mining method at the mine is another area of useful research. An approach may be to compare the cost of a bigger and more efficient processing plant against the gains from less development work and lower mining cost per tonne of ore. If the results prove satisfactory, rock cavability can be tested or even induced caving can be employed. The division of mining levels into appropriate sub-levels can then be investigated along the lines indicated in Chapter 5 of this thesis. A good description of the sub-level caving method has been given by Cokayne (1982) and the economics of this method has been discussed by Nilson (1982).

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APPENDIX 1

HISTORY OF THE MINE

The history of the Panasqueira mine is one of a widely fluctuating fortune caused by the vicissitudes of wolfram demand and price. This history is recounted in a number of published and unpublished works, which include the following: Allen *et al.* (1947), Reis (1971), Stopford-Sackville (1977), Smith (1979), McNeil (1982), Hebblethwaite (1983), Williams (1985) and Polya (1987). The brief account given below is taken from these sources.

Although said to have been worked as early as the Roman times, the mine was only recognised as a source of wolfram in 1886. Little is known of the activities prior to 1911, when underground mining began under the ownership of Smelting Co. Ltd. which later became known as Beralt Tin and Wolfram Ltd. in 1927.

Wolfram production actually began in 1912 when the first cross-cut was driven into the veins of the Panasqueira Mining Concession at an elevation of 2600 m. The cut allowed the miners to follow the ore down the dip of 10° for 500 m. The production of this very labour-intensive mining reached 200 tons per year with cassiterite being an important by-product.

The two World Wars affected the mine both advantageously and adversely, as a result of a sudden high demand for tungsten followed by the inevitable gluts. Although the mine's production reached a peak in 1943, operations were suspended by the Portuguese authorities in order to prevent the concentrate falling into the hands of the Axis powers.

The price of wolfram which fell sharply after the Second World War, rose again in 1950 following the Korean conflict and its concomitant demand for tungsten. Consequently, mining activities saw significant intensification and improvements. The labour-intensive hand-mucking was replaced by mechanical scrapers and shovel loaders, and mules by locomotives. Throughout the fifties, the company kept its production at a steady rate of 2200 tons of concentrate per year. Exploratory work was carried out in the Vale da Ermida area in search of additional sources of high grade wolfram ore. To supplement wolfram revenue, an intensive search for tin led to the re-opening of the veins at Barroca Grande, Corga Seca and Alvoso.

In 1962, wolfram price plunged again and production at Panasqueira had to be curtailed to cut costs. Charter Consolidated Plc. acquired a major interest and became the company's technical advisors in 1965 and by 1967 the company once again become profitable. A steady rise in wolfram price permitted the implementation of an expansion programme in 1970, but the price fell again leading to significant losses and forced the company to stockpile its product. The Portuguese national bank (Banco Nacional Ultramarino) became part-owner with 20% of the equity. The company changed its name to Beralt Tin and Wolfram Portugal SARL.

Infusion of capital brought a new development programme which established higher grade reserves. The period since 1974 has seen dramatic changes in the operations of the mine. Firstly, a raise borer was introduced to develop raises from drives and haulages. Secondly, the long-wall stoping was replaced by a mechanised room and pillar mining system. Thirdly, the development of a new inclined shaft commenced in 1977. This shaft was commissioned in 1982. This gave machines direct access to the haulage ways and permitted the transportation of ore, materials and men on conveyors. Fourth, an underground crushing station and surge bins were constructed.

In 1987, the price of wolfram fell from an average *Metal Bulletin* price of US$143.48 per metric ton in 1981 to US$53.50.

Comment

If the mine is to maintain its operations, then more efficient ways of delineating mineable horizons will be one way to reduce costs of development.

APPENDIX 2

THE MINERAL MEASUREMENT FORMULA

A2.1 Introduction

The mineral measurement formula was originally developed by Mendes (1958). Since then, other authors including Mather (1972), Hebblethwaite (1981), Hanvey (1982) and Williams (1985) have either tried to improve upon the formula through practical and theoretical reasoning, or given more meaning to the original logic by explaining the ideas from a different angle of reasoning.

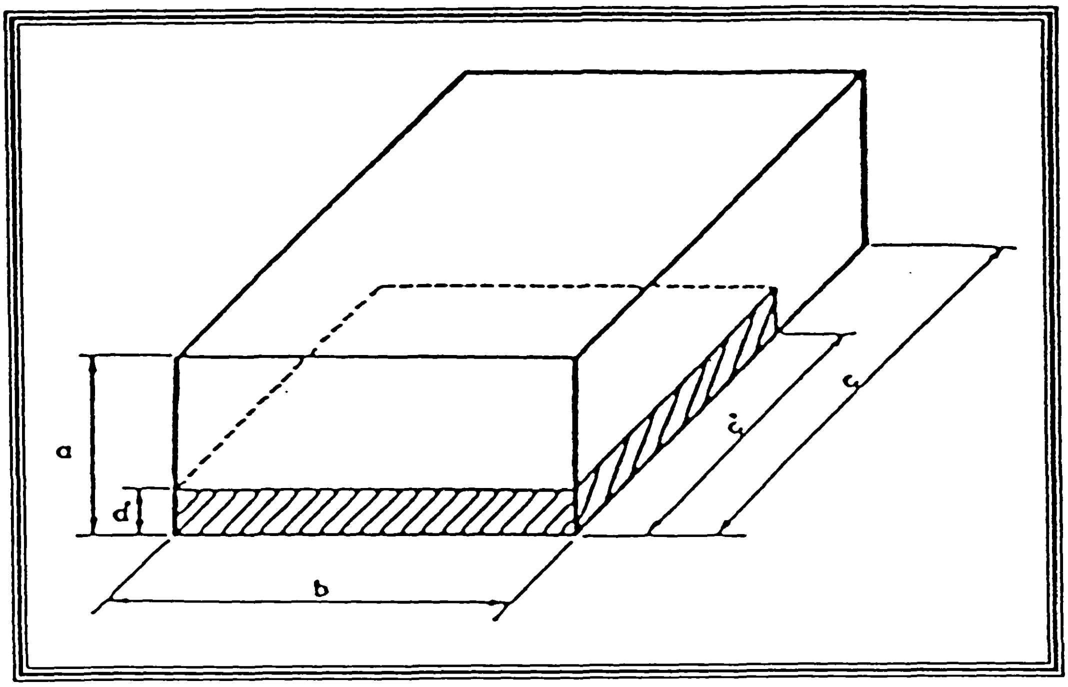
The fundamental principle behind the mineral measurement formula is that *the areal properties of an exposed vein in a plane is a measure of its volumetric properties. This plane can be a vertical section through the vein, and is practically represented by the face of a stope*

In more familiar terms, it means a big wolframite crystal at the stope face is more likely to extend further into the face than a small crystal will, and the more the number of crystals or the bigger the crystals at the face, the greater the possibility of there being crystals at some horizontal distance into the face.

A2.2 Mendes' Formula

Mendes' explanation was that an exposed crystal, in the stope face, is a sample. The area of this sample is a measure of the horizontally projected grade that can be extended into the face, to a certain distance, to establish an "area of influence" of the sample. The resultant volume of ground will then become payable or not depending on the sample's grade and some cut-off criterion.

In order to establish the ultimate grade of ground as would be realised eventually in terms of the weight of bagged concentrate at the mill, Mendes involved other factors including metallurgical recoveries, number of veins and the width of the stope. Obviously, his main difficulty was the decision as to how far the grade of the exposed crystal extends into the face. Figure A2.1 is a reproduction of Mendes' original nomenclature for the mineral measurement



S	=	sum of the areas of wolframite	(cm^2)
a	=	face height	(metres)
b	=	face width	(metres)
c	=	face depth	(metres)
a'	=	vein height (thickness)	(cms)
b'	=	vein width (length)	(metres)
c'	=	vein depth	(metres)
ρ	=	specific gravity of wolframite	7.5 gms/ml.
α	=	ratio of c':c	
V	=	Value of vein	(kg/m^2)

Figure A2.1. Mendes' nomenclature for mineral measurement parameters (after Reis, 1977) parameters. Referring to this figure, the value (V) of vein in kg/m^2 is given by:

$$V = \frac{\frac{S \times 100 \times \alpha \times c}{1000} \times 7.5}{b \times c} \qquad (A2.1)$$

$$V = 0.75 \frac{\alpha S}{b} \qquad (A2.2)$$

Mendes then made an allowance for the purity of the concentrate which should have been 76% of WO_3, but had actual assayed value of 70% at that time. He therefore introduced the ratio (D) of WO_3 in the wolframite to WO_3 in the concentrate: $D = \frac{70}{76}$, and therefore

$$V = 0.75(\frac{70}{76})\frac{\alpha \times S}{b} \qquad (A2.3)$$

The α factor (included as the ratio of vein depth to vein width) was initially assigned a value of 0.5 by Mendes, but was later changed to 0.61 after some bulk sampling tests had been carried out, bringing the final equation to:

$$V = 0.488 \frac{S}{b} \qquad (A2.4)$$

The Mineral Evaluation Factor (MEF) is simply the reciprocal of the numerical coefficient in equation (A2.4):

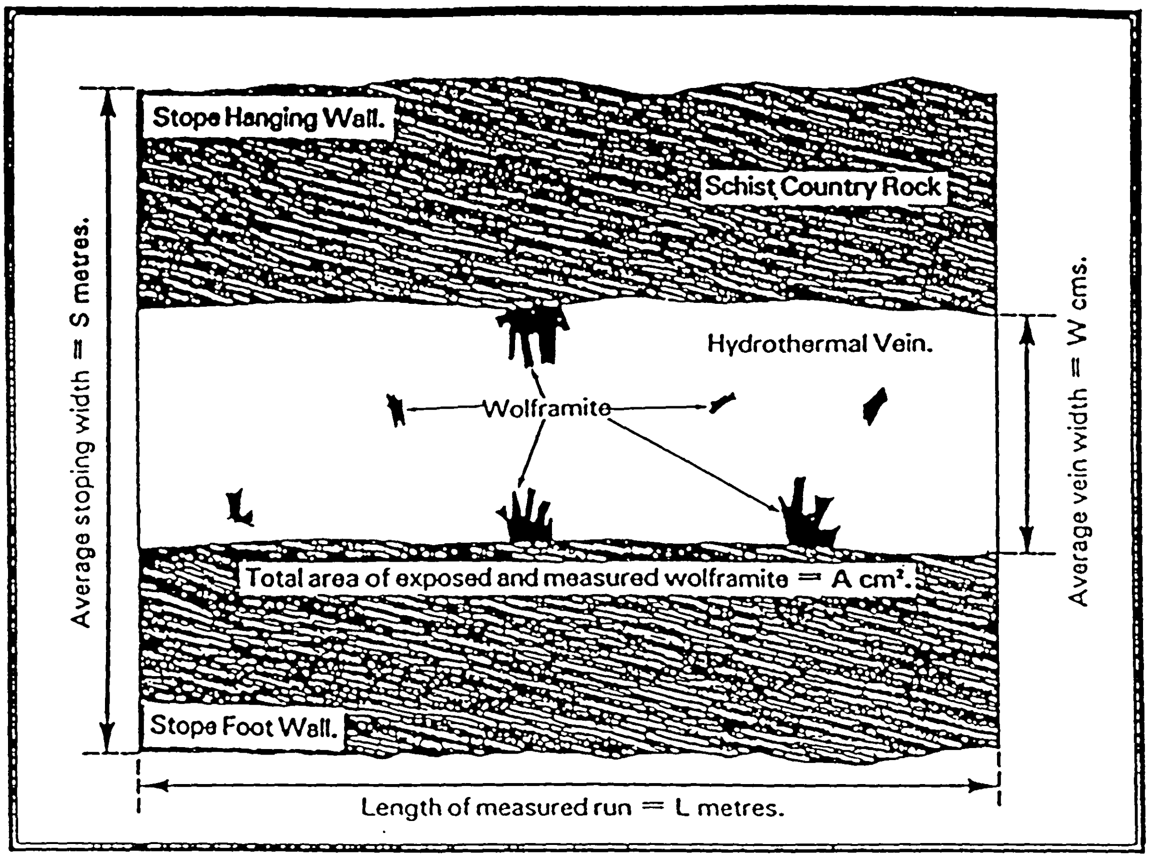
$$\text{MEF} = \frac{1}{0.488} = 2.176 \qquad (A2.5)$$

Of the several contributions made to the later development of this final formula, perhaps the most significant is that of Hebblethwaite (1981). He carried out a regressional analysis of the grades of predicted ore, against sampled run-of-mine, and against the final bagged concentrate. Based on this analysis, he recommended that the MEF should be changed. The value was consequently reduced from 2.17 to 1.8, which is the current value in use.

A2.3 Williams' Deductions

In 1985, Williams used Delesse's principle (Williams (1985, p.104)) to explain how the "wolframite tenor" of quartz *in situ* can be arrived at using the mineral measurements, without any reference to mining methods or metallurgical recoveries. He defined the wolframite tenor as the volume density of quartz with respect to the wolframite it contained. Figure A2.2 shows his nomenclature.

According to Delesse's principle, the areal fraction of a section of rock covered by transections of a certain component is equal to the fraction of the rock



L = length of measured run	(m)
S = average stoping width	(m)
W = average vein width	(cm)
ρ_w = specific gravity of wolframite	(7.5)
ρ_v = specific gravuty of quartz vein	(2.7)

Figure A2.2. Williams' nomenclature for mineral measurement parameters (after Williams, 1985) volume occupied by this component. In other words, if a structure containing some objects a is sectioned randomly, then the areal density A_A of profiles of a on the section is equal to the volume density V_V of the objects in the structure i.e.

$$V_{V_a} = A_{A_a}$$

where

$$V_{V_a} = \frac{V(a)}{V(c)}$$

where V(a) is the volume of the subset a and V(c) is the volume c symbolising the structure or containing space, and

$$A_{A_a} = \frac{A(a)}{A(c)}$$

is the ratio of the area of all a profiles to the area of the section throughout the structure c.

Obviously, this statement can be true if and only if the 'objects of phase a' are randomly distributed in all directions (i.e. isotropically) within the 'structure'. Williams assumed that mineralisation within the veins is horizontally isotropic, and that the section planes can be considered to be random. Applying Delesse's principle, he stated that:

$$\text{Wolframite Tenor}(\%\ \text{volume}) = \text{Area}\ \%\ \text{wolframite}.$$

He then substituted equation (A2.5) in (A2.4) to arrive at

$$\text{Grade}(kg/m^2) = \frac{A(cm^2)}{L(m) \times \text{MEF}}$$

In order to dissociate himself from the implications of the MEF (the MEF having connotations of mining method, metallurgical recoveries, etc.), he replaced it with W(cm) to give the equation the meaning of areal density:

$$\text{Areal Density}(\text{area}\%) = \frac{A(cm^2)}{L(m) \times W(cm)}$$

$$\text{Tenor}(\text{volume}\%) = \frac{A(cm^2)}{L(m) \times W(cm)} \qquad (A2.6)$$

In the more familiar units of kg/m^3,

$$\text{Tenor}(kg/m^3) = \frac{A(cm^2) \times \rho_w \times 10}{L(m) \times W(cm)} \qquad (A2.7)$$

$$\text{Tenor}(kg/t) = \frac{A(cm^2) \times \rho_w \times 10}{L(cm) \times W(cm) \times \rho_v} \qquad (A2.8)$$

where ρ_w is the specific gravity of wolframite, taken to be 7.5, and ρ_v is the specific gravity of the quartz vein taken to be 2.7

The grade used at the Panasqueira mine (kg/m^2) is effectively the product of the wolframite tenor and the vein width and therefore

$$\begin{aligned}\text{Grade}(kg/m^2) &= \frac{\text{Tenor}(kg/m^3) \times W(cm)}{100} \\ &= \frac{A(cm^2) \times W(cm) \times 10 \times \rho_w}{L(m) \times W(cm) \times 100} \\ &= \frac{A(cm^2) \times \rho_w}{L(m) \times 10} \\ &= \frac{A(cm^2)}{L(m) \times 1.33} \qquad (A2.9)\end{aligned}$$

A2.4 Remarks

Williams' final equation illustrates that the constant 1.33 corresponds exactly to the MEF and it is in fact ten times the reciprocal value of the specific gravity of wolframite i.e.

$$\text{MEF} = 10 \times \frac{1}{\rho_w} = \frac{1}{7.5} = 1.33$$

This conclusion, therefore, which is based purely on theory without any consideration of mining or metallurgical recoveries, means the grade value of the veins is constant — the bigger the aggregated value of the area of the veins mined (and therefore the greater the volume of the veins mined) the greater the quantity of wolframite obtainable. It must be mentioned that William's MEF value of 1.33 does not conflict with that of 1.8 suggested by Hebblethwaite. The former relates to 100% *in situ* recovery of wolframite; the latter assumes 74% *in situ* recovery due to loss of floor fines. However, his conclusion means that the value of 1.8 can be reduced if some floor fines could be recovered.

It would appear from Hebblethwaite's analysis, and from observations made elsewhere by Whyte (1972), that the whole evaluation process based on Mendes' formula is governed by "the regression of calculated reserves against yield and not by the value of the ore *in situ*" (Williams (1985)). Whatever the merits and demerits, the evaluation process based on the mineral measurements and the MEF seems to be appropriate for the Panasqueira mine. Since the mine has objected to bulk sampling techniques for reasons stated in Chapter 2, the author has no better suggestion to offer. His contention is that the mine will benefit a lot if it were to follow the steps recommended for its development programmes in order to expose an optimal quantity of the quartz veins for subsequent mineral measurement.

APPENDIX 3

DATA FILES AND OUTPUTS OF MBANDS

A3.1 BORHO File

The BORHO file contains information on the boreholes and raises held as individual records which are 89 characters long. The definition and description of the data fields are summarised in Figure A3.1. Table A3-1 is a short list of the file formatted with LISTB. The information includes the following:

(i) A six figure sequential reference number used as the file key in the data handling.

(ii) An identification number 1, 2, or 3 to indicate respectively a raise, surface drill hole, or underground drill hole.

(iii) The company's name for a borehole or raise. The raises are logically named LxDxxRxxC/G where x indicates a number. L stands for Level, D for drive and R for the Raise index i.e. the panel number. C or G indicates whether the raise serves as an access (i.e. C) or as an orepass (i.e. G). Boreholes are named xxxxZ where xxxx is a sequential number and Z is a letter (A to M and X) designated to each individual drill rig.

(iv) Collar co-ordinates. In the case of boreholes, collar co-ordinates are calculated by the Survey Department and recorded in the logs. In the case of raises, the co-ordinates are estimated by manual measurements from existing plans.

(v) Collar elevation as calculated by the Survey Department and recorded in the log books. The method of calculating the elevation at the start of the raise logs varies with the type of log: when the log starts from a haulage raise there is usually a survey elevation at the point; where the log measurements are taken from the footwall of a stope which the raise intersected, the Geology Department is able to estimate the elevation of the point by reference to nearby survey points, the accuracy being within 10 cm.

BORHO

Fields	Type	BTWP Name	RSM Variable	Description
1-6	I6	KEY	KEY	Computer Reference Number
7	I1	KT	TYPE	Raise or Surface/Underground Borehole
8	A1	N1	NAME1	The Company's name for the
9-12	A4	N2	NAME2	raise or borehole.
13-16	A4	N3	NAME3	
17-23	F7.2	CE	COORDE	Eastings
24-30	F7.2	CN	COORDN	Northings
31-34	F4.2	DT	DATE	Date of start of drilling.
35-40	F6.2	ELV	ELEV	Collar elevation metres a.o.d.
41-44	A4	IS1	BITYPE1	The first bit type.
45-47	A3	IT1	BIDEPT1	The depth to which it was used.
48-51	A4	IS2	BITYPE2	The second bit type.
52-54	I3	IT2	BIDEPT2	The depth to which it was used.
55-58	A4	IS3	BITYPE3	The third bit type.
59-61	I3	IT3	BIDEPT3	The depth to which it was used.
62-63	I2	INA	STRTDIP	Starting dip of an inclined hole.
64-66	I3	INT	STARTAZ	Azimuth of an inclined hole.
67	I1	LO	UPDOWN	1 = uphole, 2 = downhole.
68-69	I2	MV	MINVEIN	Minimum vein recorded (ie 1 or 5cms).
70-89	5A4	LOC	LOCATED	Description of location.

Figure A3.1. BORHO file data locations and field descriptions.

Table A3-1. Specimen of BORHO file.

10236	3	236-I	30743.52	53289.52	10.83	630.70	EXL	35	0	0	90	0	1	5	L1-P6 X D31
10237	3	237-I	31449.90	53639.30	5.84	579.41	EXL	55	0	0	90	0	1	5	L2-D19-R3A-I5N
10238	3	238-I	31449.90	53639.30	5.84	576.44	EXL	16	0	0	90	0	2	5	L2-D19-R3A-I5N
10239	3	239-I	31467.50	53669.15	5.84	573.50	EXL	15	0	0	90	0	2	5	L2-D19-R3A-I5N
10240	3	240-I	31467.50	53669.15	5.84	576.30	EXL	48	0	0	90	0	1	5	L2-D19-R3A-I5N
10241	3	241-I	31418.60	53583.40	6.84	576.81	EXL	15	0	0	90	0	2	5	L2-D19E-R3A-I5S
10242	3	242-I	31418.60	53583.40	6.84	579.24	EXL	55	0	0	90	0	1	5	L2-D19E-R3A-I5S
10243	3	243-I	31287.00	53755.00	6.84	588.50	EXL	35	0	0	90	0	2	5	L2-D17-R5A-I5S
10244	3	244-I	31287.00	53755.00	7.84	591.05	EXL	35	0	0	90	0	1	5	L2-D17-R5A-I5S
10246	3	246-I	31405.80	53660.10	7.84	576.74	EXL	15	0	0	90	0	2	5	L2-D17-R3-I5S
10247	3	247-I	31405.80	53660.10	7.84	579.34	EXL	50	0	0	90	0	1	5	L2-D17-R3-I5S
10248	3	248-I	31502.03	54129.06	10.84	588.60	EXL	30	0	0	90	0	2	5	L2-D11-R5A-I2N
10251	3	251-I	31247.61	53291.06	8.85	582.40	EXL	20	0	0	90	0	2	5	L2-D25-R3A-I4S
10252	3	252-I	31263.50	53319.10	9.85	582.55	EXL	45	0	0	90	0	1	5	L2-D25-R3A-I4S
10253	3	253-I	31267.50	53326.10	10.85	580.27	EXL	20	0	0	90	0	2	5	L2-D25-R3A-I4S
10254	3	254-I	31453.70	53446.20	10.85	571.15	EXL	55	0	0	90	0	1	5	L2-D21-R2A-I6S
10255	3	255-I	31407.40	53466.70	10.85	572.30	EXL	55	0	0	90	0	1	5	L2-D21-R2-I6S
10256	3	256-I	31404.90	53462.40	11.85	570.30	EXL	10	0	0	90	0	2	5	L2-D21-R2-I6S
10257	3	257-I	31393.10	53442.00	11.85	571.90	EXL	10	0	0	90	0	2	5	L2-D21-R2-I6S
10258	3	258-I	31393.10	53442.00	11.85	574.50	EXL	55	0	0	90	0	1	5	L2-D21-R2-I6S
10259	3	261-I	31358.70	53309.60	1.86	589.74	EXL	35	0	0	90	0	1	5	L2.D25.R2.I3E
10262	3	262-I	31358.70	53309.60	1.86	587.42	EXL	20	0	0	90	0	2	5	L2.D25.R2.I3E
10263	3	263-I	31431.84	53287.56	1.86	602.88	EXL	25	0	0	90	0	1	5	L2.D25.RAMPA L2-L1
10264	3	264-I	31205.27	53315.78	2.86	576.73	EXL	30	0	0	90	0	1	5	L2.D25.R3.I5S
10265	3	265-I	31386.45	53281.48	2.86	597.98	EXL	30	0	0	90	0	1	5	L2.P4.D25E (RAMPA)
10266	3	266-I	31386.45	53281.48	2.86	595.24	EXL	30	0	0	90	0	2	5	L2.P4.D25E (RAMPA)
10267	3	267-I	31466.31	53261.25	3.86	568.50	EXL	60	0	0	90	0	1	5	L2.D25E.P1-A
10268	3	268-I	30992.61	53601.33	4.86	625.80	EXL	57	0	0	90	0	2	5	L1.P4.D23W
10269	3	269-I	31083.65	53556.78	5.86	625.30	EXL	58	0	0	90	0	2	5	L1.P4.D23W
10270	3	270-I	31002.63	53666.81	5.86	624.30	EXL	60	0	0	90	0	2	5	L2.D19.R5.AW20

(vi) The attitude of the borehole at the collar (raises are vertical). The zenith is corrected from the magnetic grid to the national grid. The magnetic correction in use at the mine is N 10^0 W of grid north. The azimuth is recorded as the dip away from the horizontal.

(vii) A single character, 1 indicating 'up' or 2 for 'down' to fix the azimuth. This method, rather than simply stating the azimuth as positive or negative, is adopted because it complies with the traditional convention in the drill logs.

(viii) A series of entries for the drill bit types. The conventional alphanumeric characters used by manufacturers of diamond drill crowns is followed: BXL – surface drills producing 42 mm diameter core; EXT – Boyles underground drills producing 22 mm diameter core; XRT/XL – Victor underground drills producing 19 mm diameter core; T46 – Diamec underground drills producing 32 mm diameter core; TT46 – Diamec underground drills producing 35 mm diameter core. Sometimes the drill string length in metres to which the bit type is used is also indicated.

(ix) A number to indicate the minimum vein width recorded. This number is either 1 or 5 and indicates whether the original log had been compiled by recording all quartz vein intersections over 1 cm or those over 5 cm.

(x) Twenty alphanumeric fields are reserved for the description of the borehole location. This will include the drive and panel numbers where applicable.

A3.2 LITHO File

The LITHO file contains a block of data intended primarily to serve as a library record of the observations on the drill cores. Figure A3.2 summarises the descriptions and the locations of the data fields in the records which are 42 characters long. Table A3-2 is a short list of the LITHO file formatted by using the program LISTL. The data fields contain the following:

(i) A six figure sequential reference number used as the borehole key in the data handling.

(ii) Serial number of the observation.

LITHO.				
Fields	Type	BTWP Name	RSM Variable	Description
1-6	I6	KEYL	KEYLITH	Computer Reference Number
7-8	I2	IPL	SERIAL	Serial number of observation.
9-13	F5.2	ST	START	String distance to start of observations.
14	A1	L1	LOSS	Severity of core loss.
15	A1	L2	SCHIST	i.e. Slate.
16	A1	L3	GREY	Darkness of grey colour.
17	A1	L4	BANDED	Severity of banding.
18	A1	L5	ARGIL	Argillaceous.
19	A1	L6	AREN	Arenaceous.
20	A1	L7	SPOT	Spottiness of slate.
21	A1	L8	ALTRD	Degree of alteration.
22	A1	L9	SXBR	Seixo Bravo veining.
23	A1	L10	FAULT	Fault encountered.
24	A1	L11	CARB	Carbonate filled.
25	A1	L12	QTZ	Quartz.
26	A1	L13	GREIS	Greisen.
27	A1	L14	APL	Aplite.
28	A1	L15	GRAN	Granite.
29-42	3A4	KREL	REMARKS	Remarks.

Figure A3.2. LITHO file data locations and field descriptions.

Table A3-2. Specimen of LITHO file.

```
1Ø236  1    .ØØ Ø 6 4 Ø Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø236  2   9.5Ø Ø 6 4 Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø236  3  11.ØØ Ø 6 4 Ø Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø236  4  12.82 Ø 6 4 Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø236  5  2Ø.35 Ø 6 4 Ø Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø236  6  2Ø.74 Ø 6 4 Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø236  7  33.ØØ Ø 6 4 Ø Ø 3 Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø236  8  34.28 Ø 6 4 Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø236  9  35.4Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø END
1Ø237  1    .ØØ Ø 6 4 Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø237  2   9.1Ø Ø 6 4 Ø 4 Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø
1Ø237  3  12.4Ø Ø 6 4 Ø Ø 3 Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø237  4  17.ØØ Ø 6 4 Ø Ø 4 Ø Ø 2 Ø Ø Ø Ø Ø Ø
1Ø237  5  19.2Ø Ø 6 4 Ø Ø 3 2 Ø Ø Ø Ø Ø Ø Ø Ø
1Ø237  6  23.52 Ø 6 4 Ø Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø237  7  33.4Ø Ø 6 4 Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø237  8  36.1Ø Ø 6 4 Ø 4 Ø Ø Ø Ø 3 Ø Ø Ø Ø Ø
1Ø237  9  36.54 Ø 6 4 Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø237 1Ø  38.15 Ø 6 4 Ø Ø 4 Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø237 11  55.87 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø END
1Ø238  1    .ØØ Ø 6 4 Ø 3 Ø Ø Ø Ø Ø Ø Ø Ø Ø Ø
1Ø238  2   6.ØØ Ø 6 4 Ø 3 Ø 3 Ø Ø Ø Ø Ø Ø Ø Ø
```

(iii) Depth below or above the collar of the observation.

The description of the most commonly occurring lithological features are given in the form of codes: the number 1 is normally used to indicate the presence of a feature and 0 is used to indicate the absence of the feature. The intensity of the occurrence of the feature is designated by numbers 2 to 6 i.e.

2 = trace,

3 = slight,

4 = moderate,

5 = severe,

6 = total.

The selection of these figures is subjective as it depends on the judgement of the person compiling the data. The following are the features or variables which the intensity values describe:

(i) Core loss. Conventionally, the number 6 indicates 'all', 2 means 20% and so on.

(ii) Schist. This is the accepted description of the country rock. The number 6 is used to describe the rock rather than 1 when the log does not contain a suitable entry.

(iii) Grey. Conventionally the number 5 indicates 'light grey' and 6 means 'black'.

(iv) Banded i.e. laminations of different rock types.

(v) Argillaceous.

(vi) Arenaceous.

(vii) Spotted.

(viii) Altered. Normally this refers to a hydrothermal alteration of the schist near the veins, rather than regional contact or contact metamorphism.

(ix) Seixo Bravo – quartz segregations.

(x) Faulted. The number 6 indicates an observed fault in the core.

(xi) With carbonate. Normally this refers to a fault fill.

(xii) With quartz.

(xiii) Greisen. In some cases the veins are described as greisen, rather than hydrothermal.

(xiv) Aplite.

(xv) Granite – an important observation indicating the intersection with the underlying granite batholith.

(xvi) Remarks – 14 alphanumeric fields are provided for descriptions of the lithology not covered by the terms above.

The LITHO file also contains the record of the final depth to which the borehole was drilled.

A3.3 VEINAGE File

The VEINAGE file contains the vein intersections and a description of the minerals observed. Figure A3.3 is a summary of the description and location of the data fields in the records which are 45 characters long. Table A3-3 is a short list of the VEINAGE file formatted by using the program LISTV. The following are contained in the VEINAGE file:

(i) Serial number of the vein intersection.

(ii) The distance in metres from the collar of the drill hole, along the core, to the start of the vein intersection.

(iii) The width of the vein in metres measured along the core.

The strength of the mineralisation is codified as follows:

1 = unknown intensity (mineral present),

2 = trace (F),

3 = slight (FR),

4 = moderate (R),

5 = abundant (B), and

VEINAGE				
Fields	Type	BTWP Name	RSM Variable	Description
1-6	I6	KEYV	KEYVEIN	Computer Reference Number
7-8	I2	IPV	VEINNUM	Serial of vein intersection.
9-13	F5.2	STA	STARTS	Drill string length to start of vein.
14-16	F3.2	WID	WIDTH	Width of vein in metres.
17	A1	A	QTZ	Quartz.
18	A1	B	WOLF	Wolframite.
19	A1	C	CASS	Cassiterite.
20	A1	D	CHALC	Chalcopyrite.
21	A1	E	PYR	Pyrite.
22	A1	F	ASPY	Arsenopyrite.
23	A1	G	SPHAL	Sphalerite.
24	A1	H	GAL	Galena.
25	A1	I	MARC	Marcasite.
26	A1	J	SID	Siderite.
27	A1	K	CARB	Carbonates.
28	A1	L	MICA	Mica (usually Muscovite).
29	A1	M	APAT	Apatite.
30	A1	N	FLUOR	Fluorite.
31	A1	O	TOPAZ	Topaz.
32				Spare.
33				Spare.
34-45	3A4	KREV	REMARKS	Remarks.

Figure A3.3. VEINAGE file data locations and field descriptions.

Table A3-3. Specimen of VEINAGE file.

1Ø236	1	5.45	.Ø6	6	Ø	Ø	Ø	2	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	IPLITE 2
1Ø236	2	7.86	.Ø8	6	Ø	Ø	2	3	Ø	3	Ø	Ø	Ø	Ø	Ø	Ø	Ø	3	
1Ø236	3	11.25	.Ø9	4	Ø	Ø	2	3	Ø	3	Ø	Ø	Ø	Ø	Ø	Ø	Ø	5	
1Ø236	4	16.16	.24	6	Ø	Ø	2	2	Ø	2	Ø	Ø	Ø	Ø	Ø	Ø	Ø	2	
1Ø237	1	5.17	.Ø8	6	Ø	Ø	2	2	Ø	Ø	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	
1Ø237	2	13.46	.21	6	Ø	Ø	Ø	2	2	Ø	Ø	Ø	Ø	Ø	3	Ø	Ø	Ø	
1Ø237	3	18.44	.56	6	Ø	Ø	3	3	3	Ø	Ø	Ø	Ø	3	3	Ø	Ø	Ø	
1Ø237	4	36.87	.1Ø	2	Ø	Ø	2	Ø	2	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	Ø	
1Ø237	5	39.55	.26	6	2	Ø	2	2	2	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	
1Ø237	6	48.7Ø	.22	6	Ø	Ø	Ø	2	2	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	
1Ø237	7	54.12	.Ø5	6	Ø	Ø	2	2	2	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	
1Ø238	1	.ØØ	.11	6	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	
1Ø238	2	13.36	.22	6	Ø	Ø	2	2	3	Ø	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	
1Ø239	1	.93	.35	6	Ø	Ø	2	3	3	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	
1Ø239	2	9.62	.Ø5	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	6	Ø	Ø	Ø	
1Ø24Ø	1	4.37	.1Ø	6	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	Ø	Ø	3	Ø	Ø	Ø	
1Ø24Ø	2	5.19	.Ø6	6	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	2	4	Ø	Ø	Ø	
1Ø24Ø	3	21.57	.Ø5	6	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	
1Ø24Ø	4	21.63	.35	6	Ø	Ø	Ø	Ø	4	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	Ø	
1Ø24Ø	5	26.2Ø	.2Ø	6	Ø	Ø	2	4	4	Ø	Ø	Ø	Ø	2	Ø	Ø	Ø	Ø	
1Ø24Ø	6	38.41	.21	2	Ø	Ø	5	Ø	5	3	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	
1Ø24Ø	7	46.Ø4	.Ø9	6	Ø	Ø	2	Ø	2	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	Ø	IPLITE
1Ø241	1	4.Ø5	.4Ø	6	2	Ø	2	Ø	1	Ø	Ø	Ø	Ø	Ø	1	Ø	Ø	Ø	

6 = very abundant (MB).

Conventionally, quartz is assigned an intensity value of 1 and assumed to be present in all veins unless its absence is specifically noted. Wolframite and cassiterite are also assigned an intensity value of 1. The intensity values describe the following minerals:

(i) Quartz – almost always present.

(ii) Wolframite.

(iii) Cassiterite.

(iv) Chalcopyrite.

(v) Pyrite.

(vi) Arsenopyrite.

(vii) Sphalerite.

(viii) Galena.

(ix) Marcasite.

(x) Siderite.

(xi) Carbonates – these usually refer to calcite or dolomite.

(xv.) Mica – this usually refers to muscovite.

(xii) Apatite.

(xiii) Fluorite.

(xiv) Topaz.

(xv) Two spare columns for future use.

(xvi) Twelve alphanumeric fields are provided for remarks which generally indicate minor minerals not included in the above list such as tourmaline, chlorite, triplite or pyrrhotite. Occasionally, the 'angle of contact' (i.e. the angle of intersection of the plane of the vein with the perpendicular to the core axis) is recorded here.

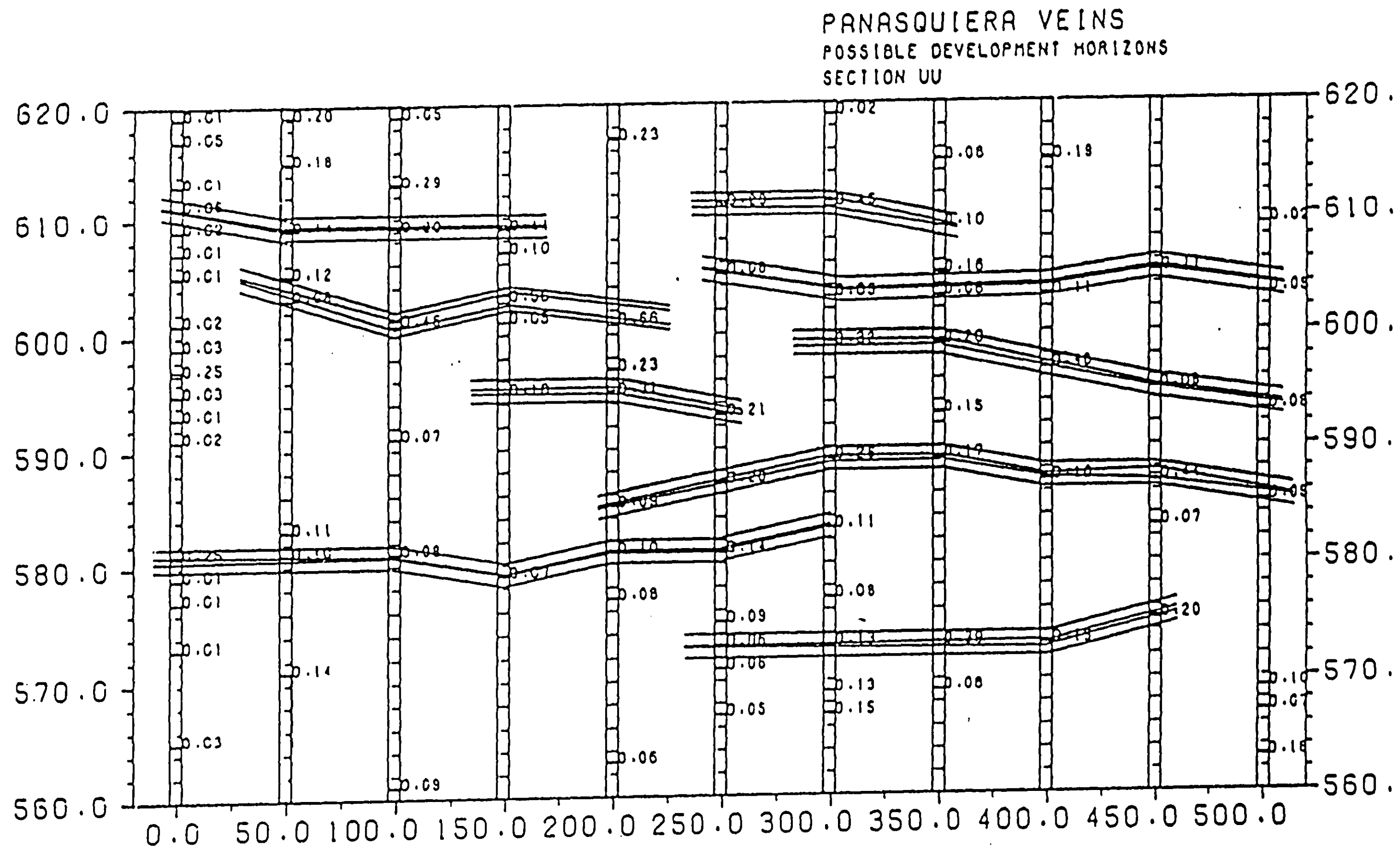


Figure A3.4a. Output of MBANDS showing possible development horizons in Section U-U.

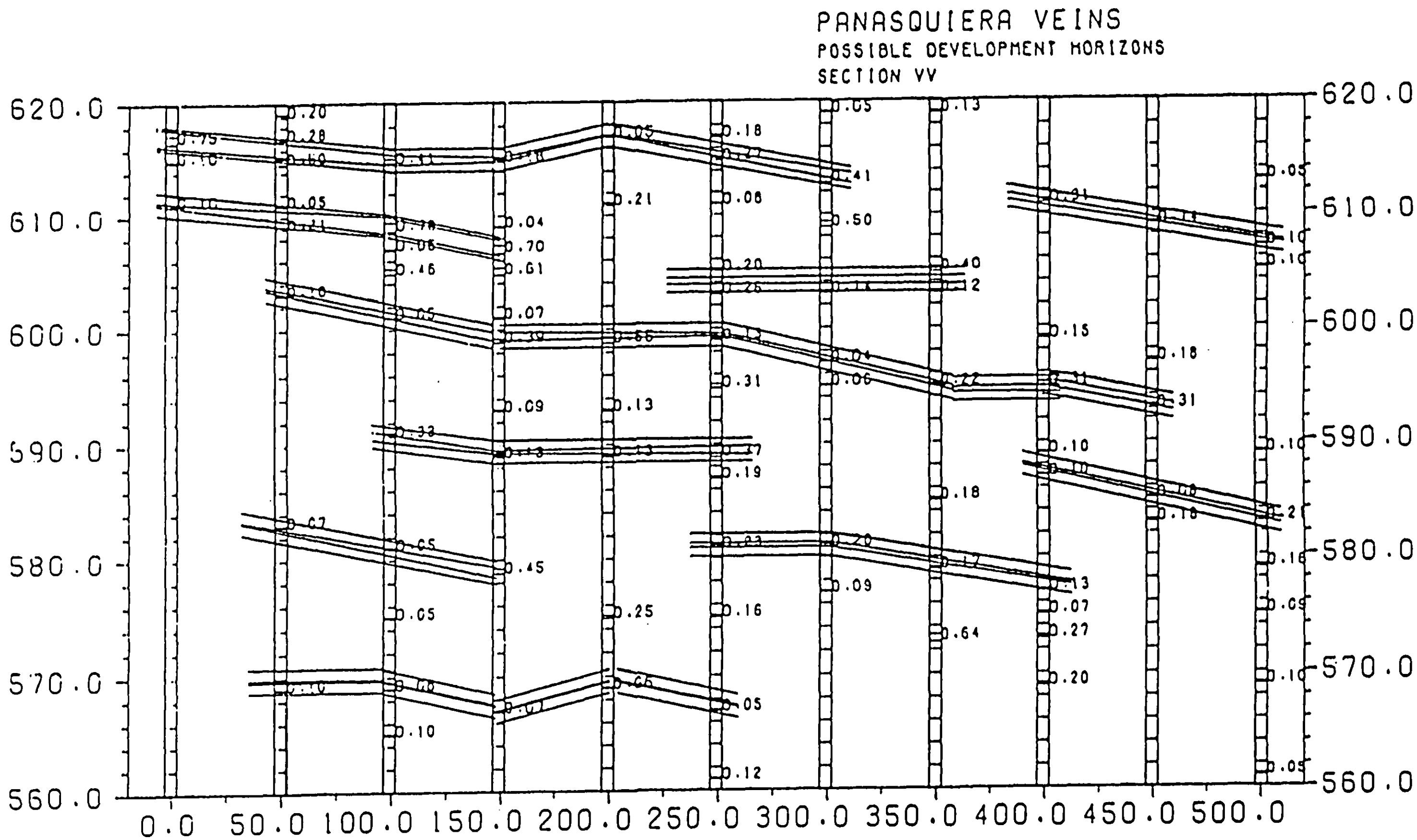


Figure A3.4b. Output of MBANDS showing possible development horizons in Section V-V.

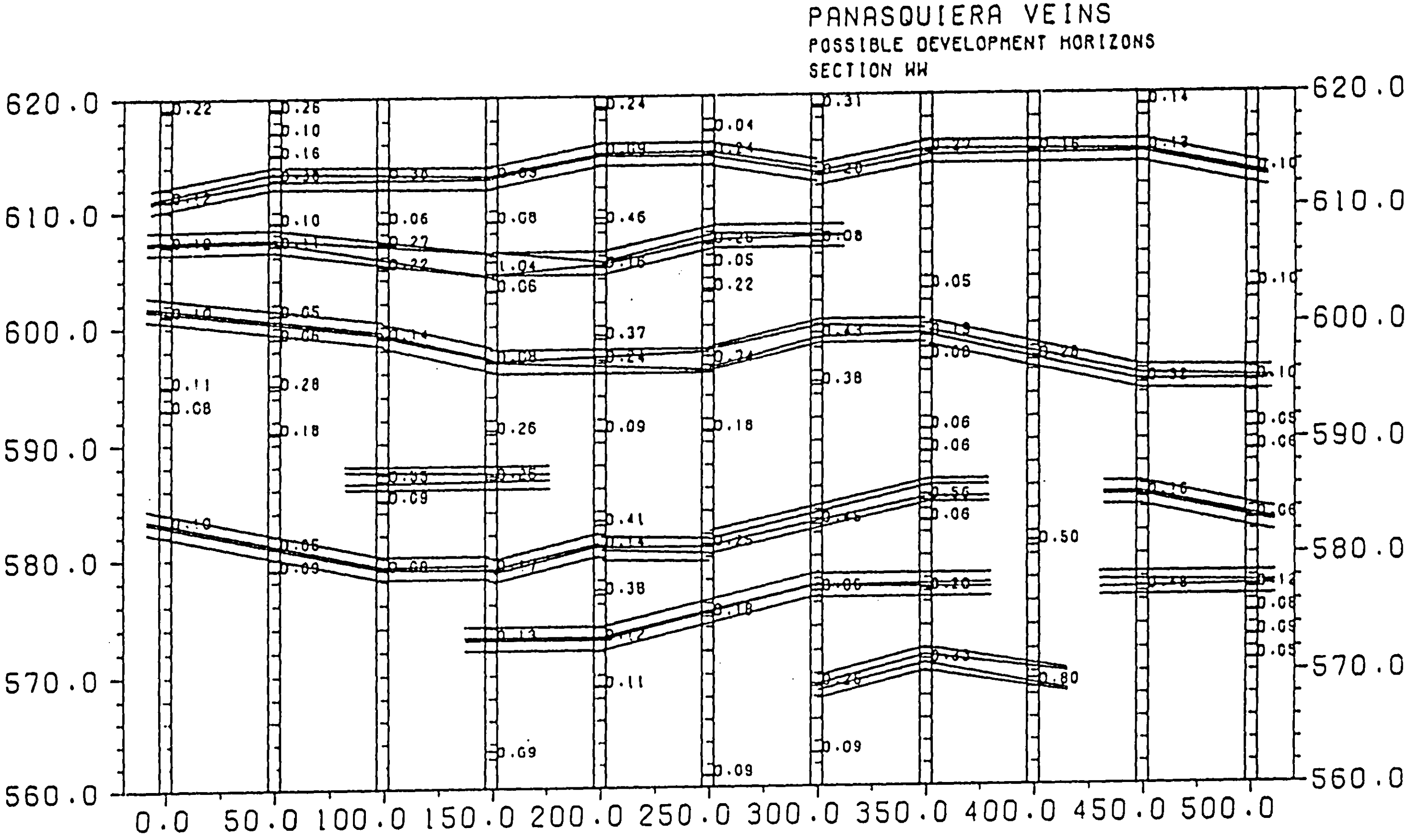


Figure A3.4c. Output of MBANDS showing possible development horizons in Section W-W.

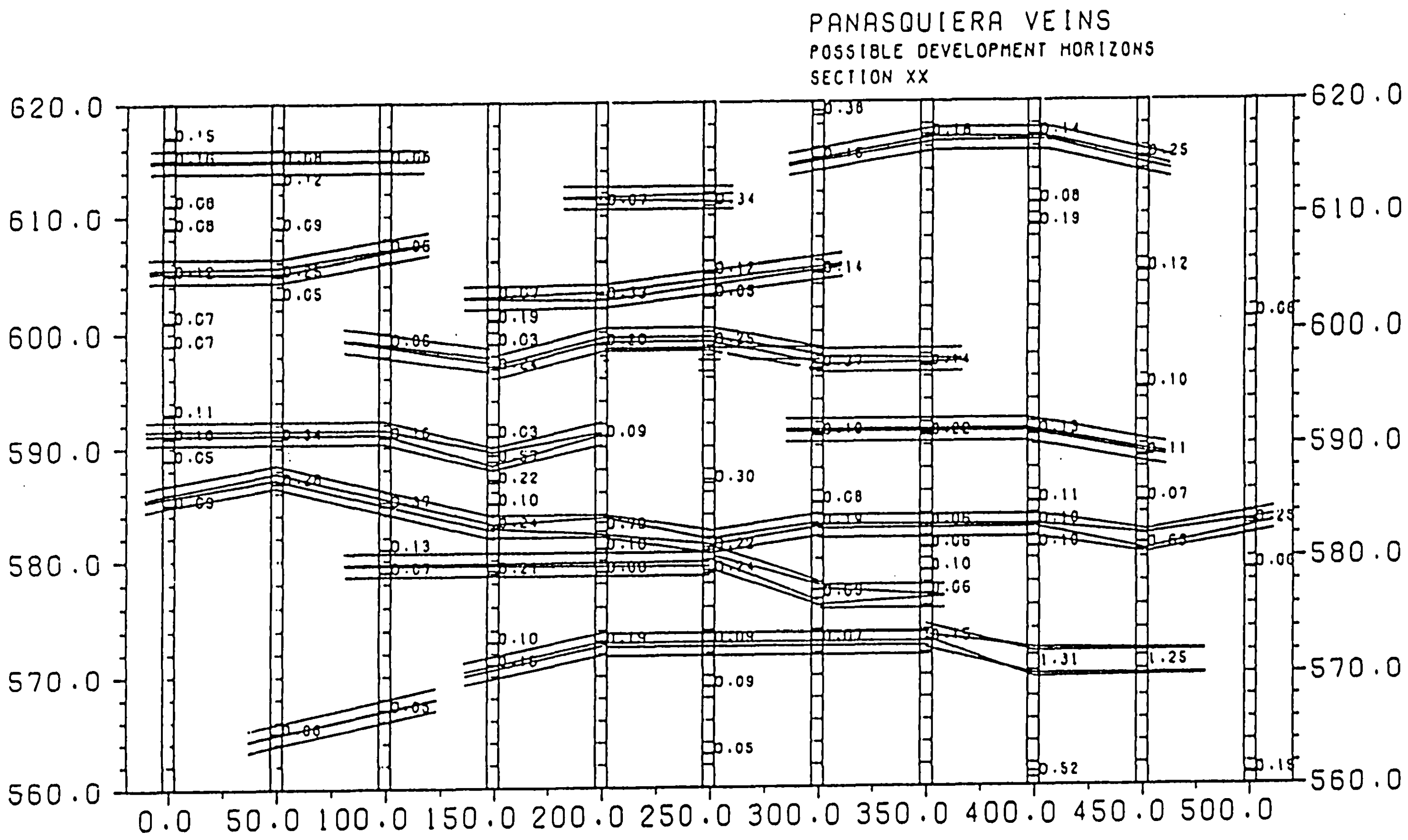


Figure A3.4d. Output of MBANDS showing possible development horizons in Section X-X.

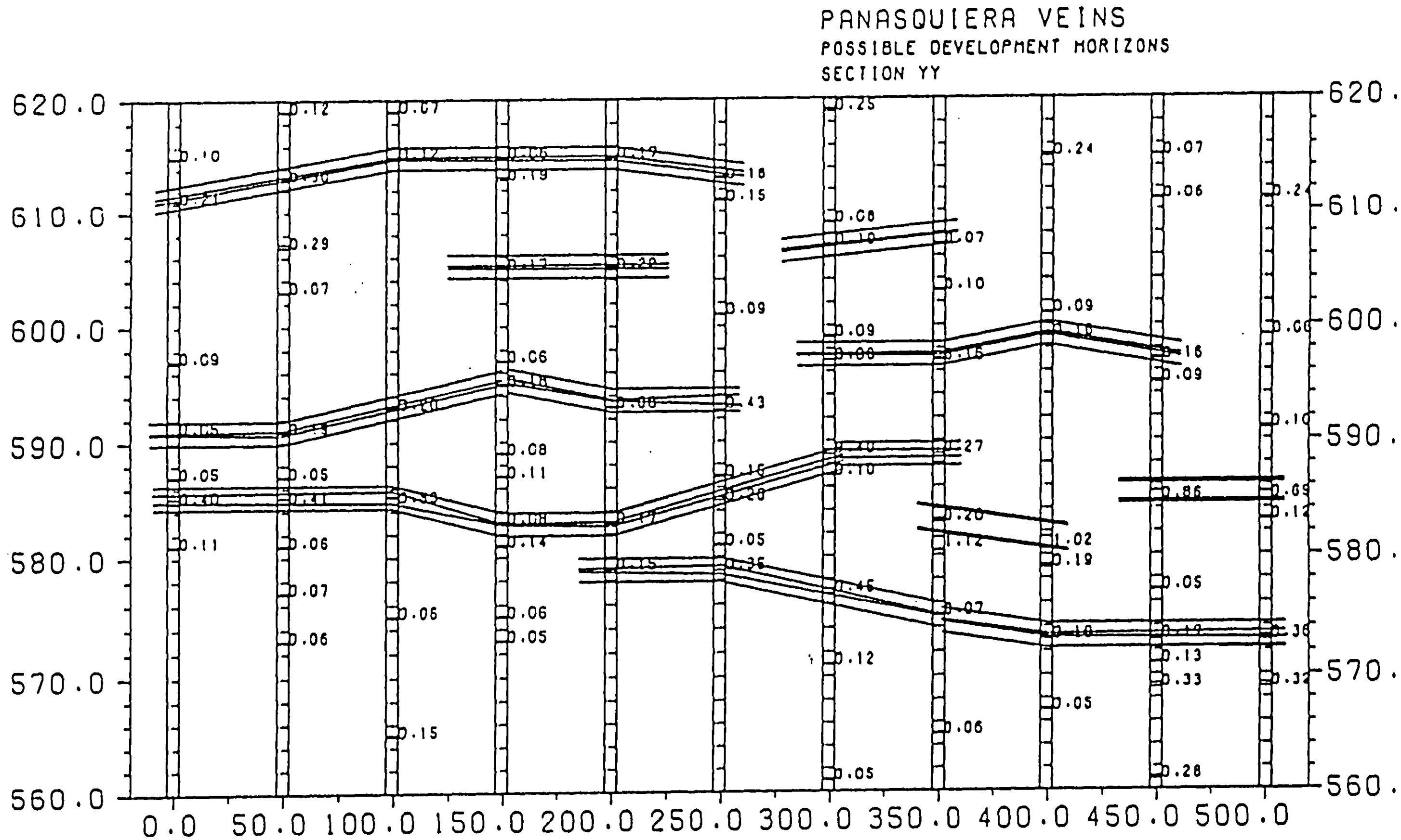


Figure A3.4e. Output of MBANDS showing possible development horizons in Section Y-Y.

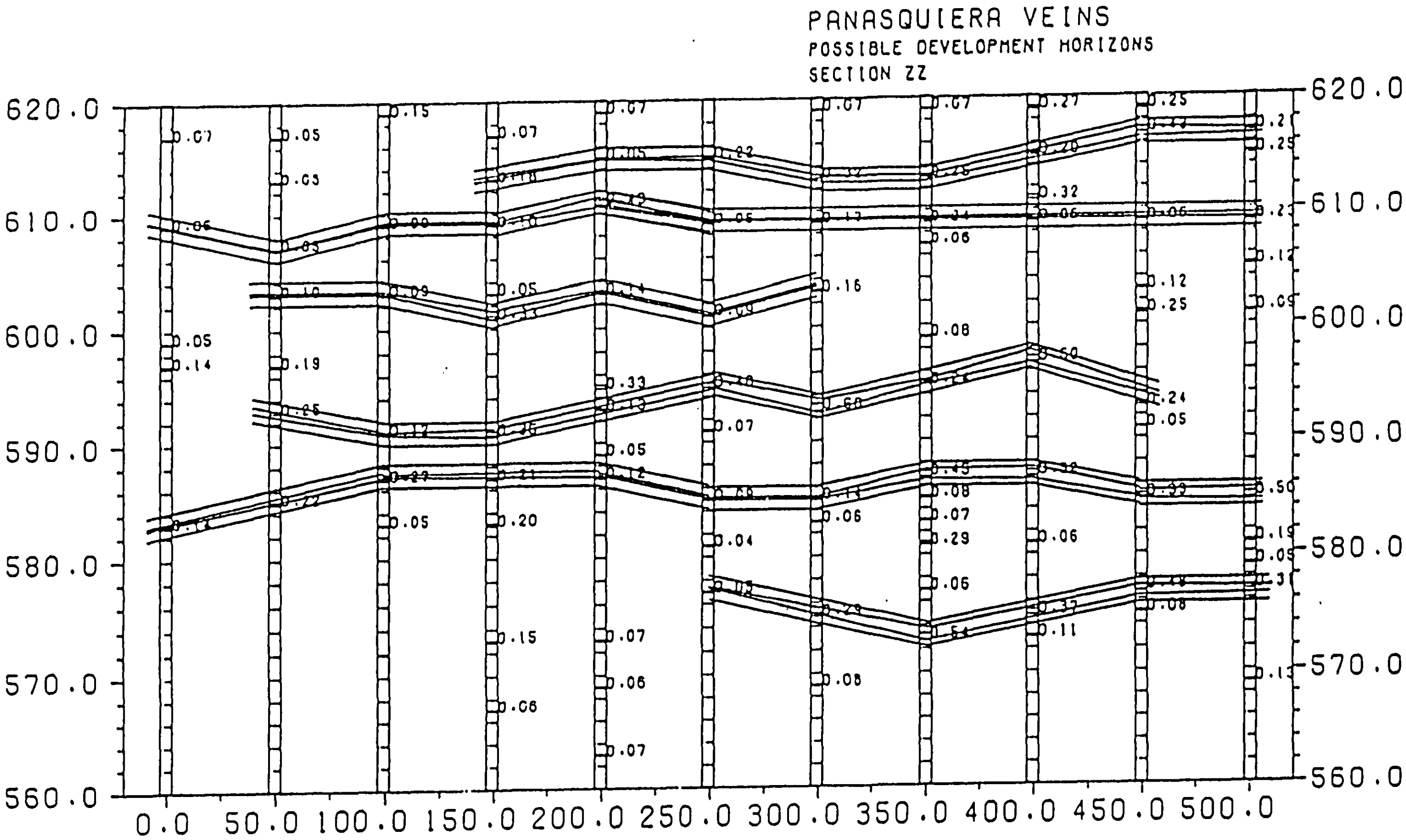


Figure A3.4f. Output of MBANDS showing possible development horizons in Section Z-Z.

APPENDIX 4

DIRECTIONAL SEMI-VARIOGRAMS OF LOG. VALUES

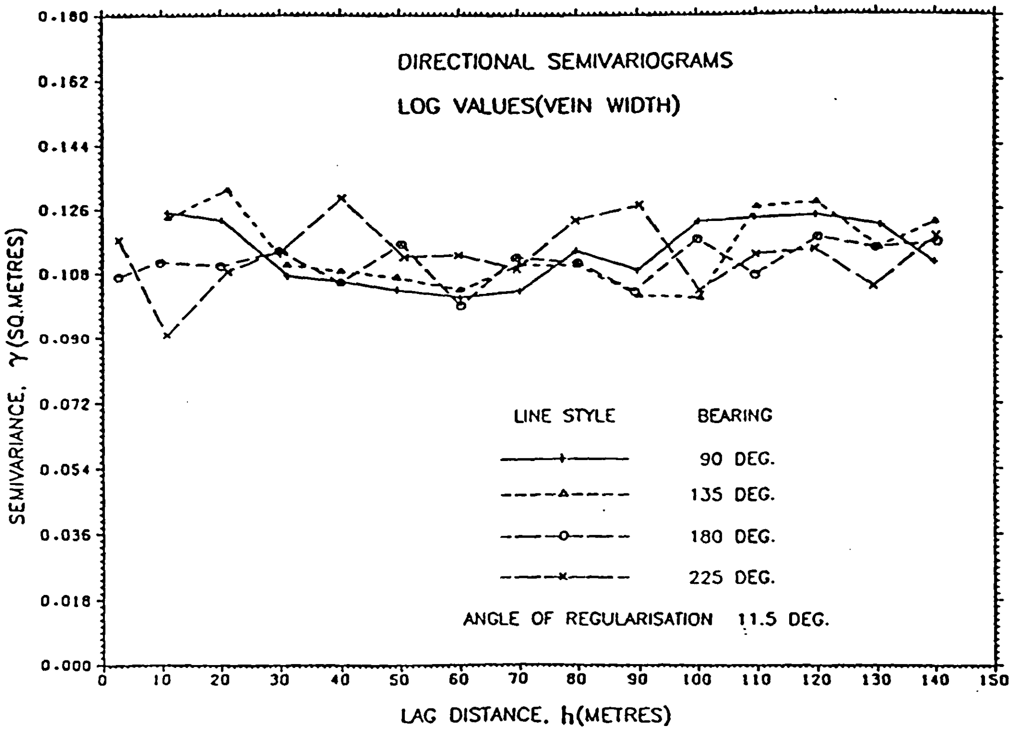


Figure A4a. Directional Semi-variograms of Logarithmic Values of Vein Widths.

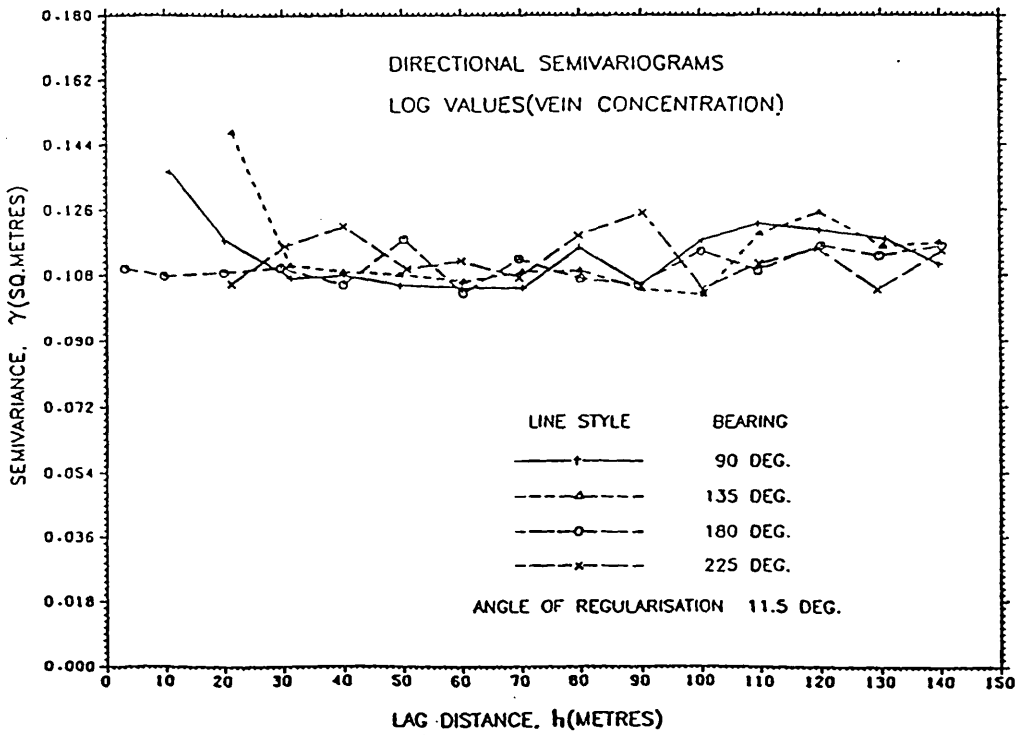


Figure A4b. Directional Semi-variograms of Logarithmic Values of Vein Concentration.

APPENDIX 5

SECTIONS SHOWING DELINEATED AREAS

Figures A5a, A5b, A5c, A5d, A5e and A5f, in this Appendix, are the outputs of program BKPLOT showing delineated areas in sections along Drives D15, D17, D19 D21, D23 and D25 respectively. For clarity, these areas have been traced separately onto transparency overlays, as shown, respectively, in Figures F1, F2, F3, F4, F5 and F6.

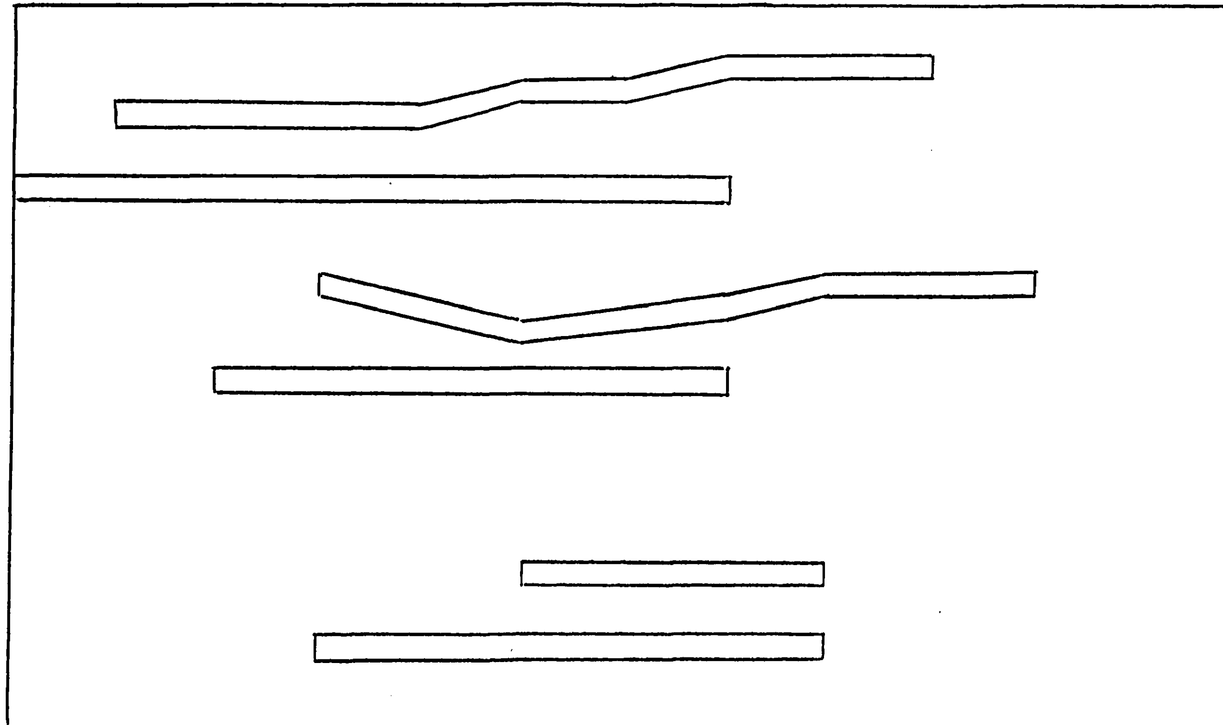


Figure F1. Section along Drive D15 showing delineated areas.

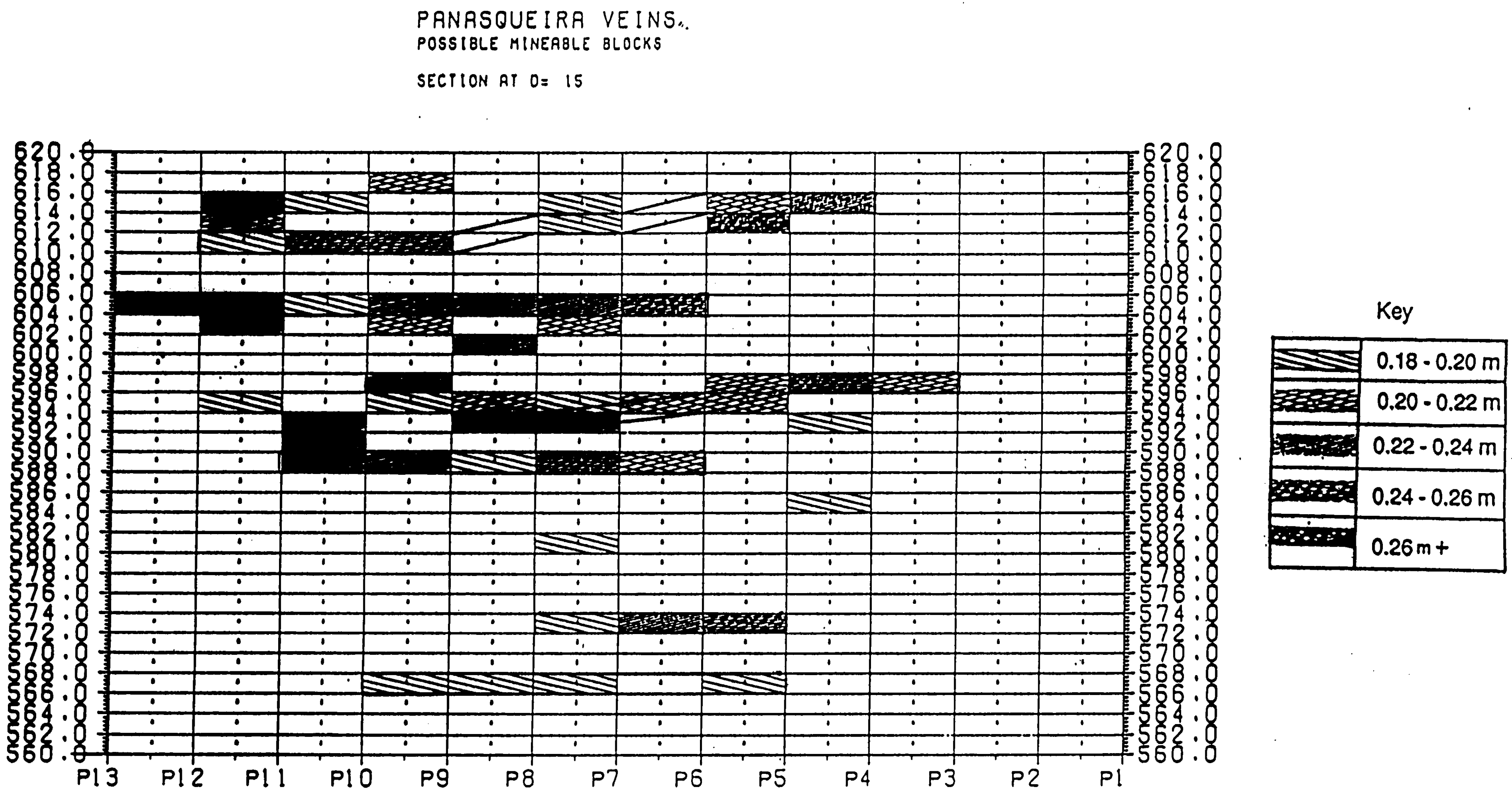


Figure A5a. Section along Drive D15 showing delineated areas for develop-

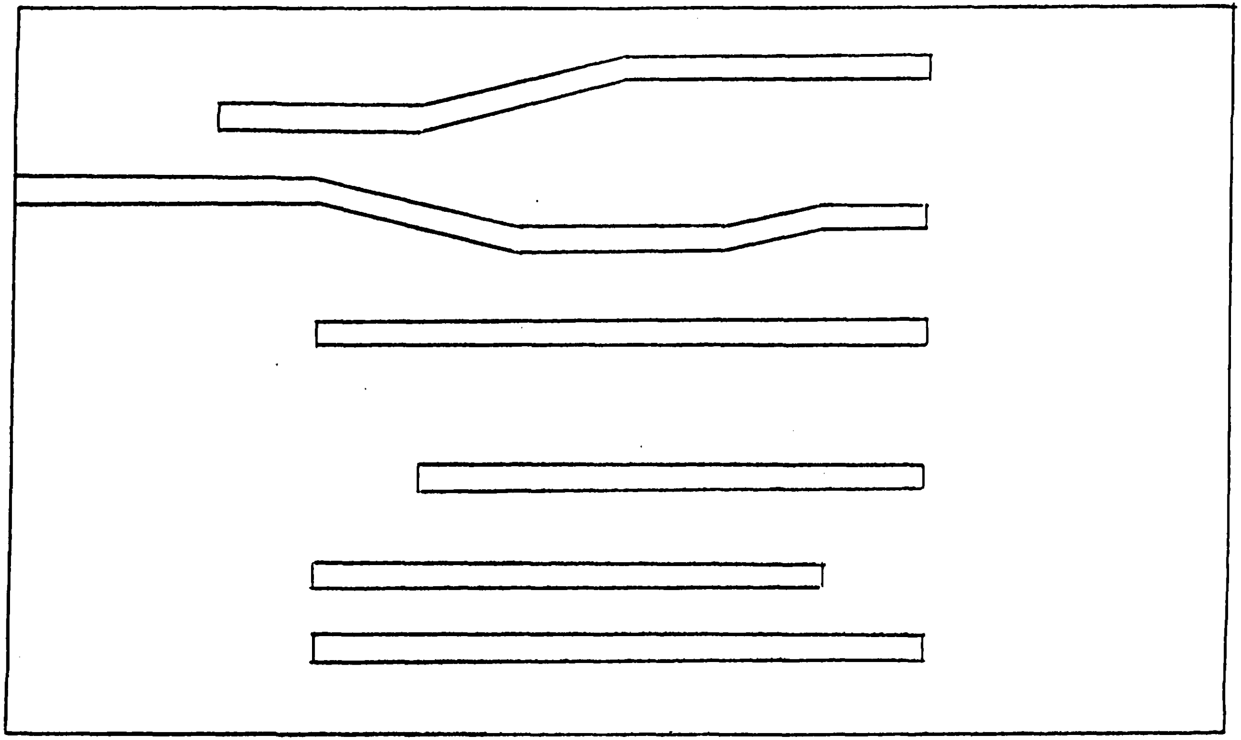


Figure F2. Section along Drive D17 showing delineated areas.

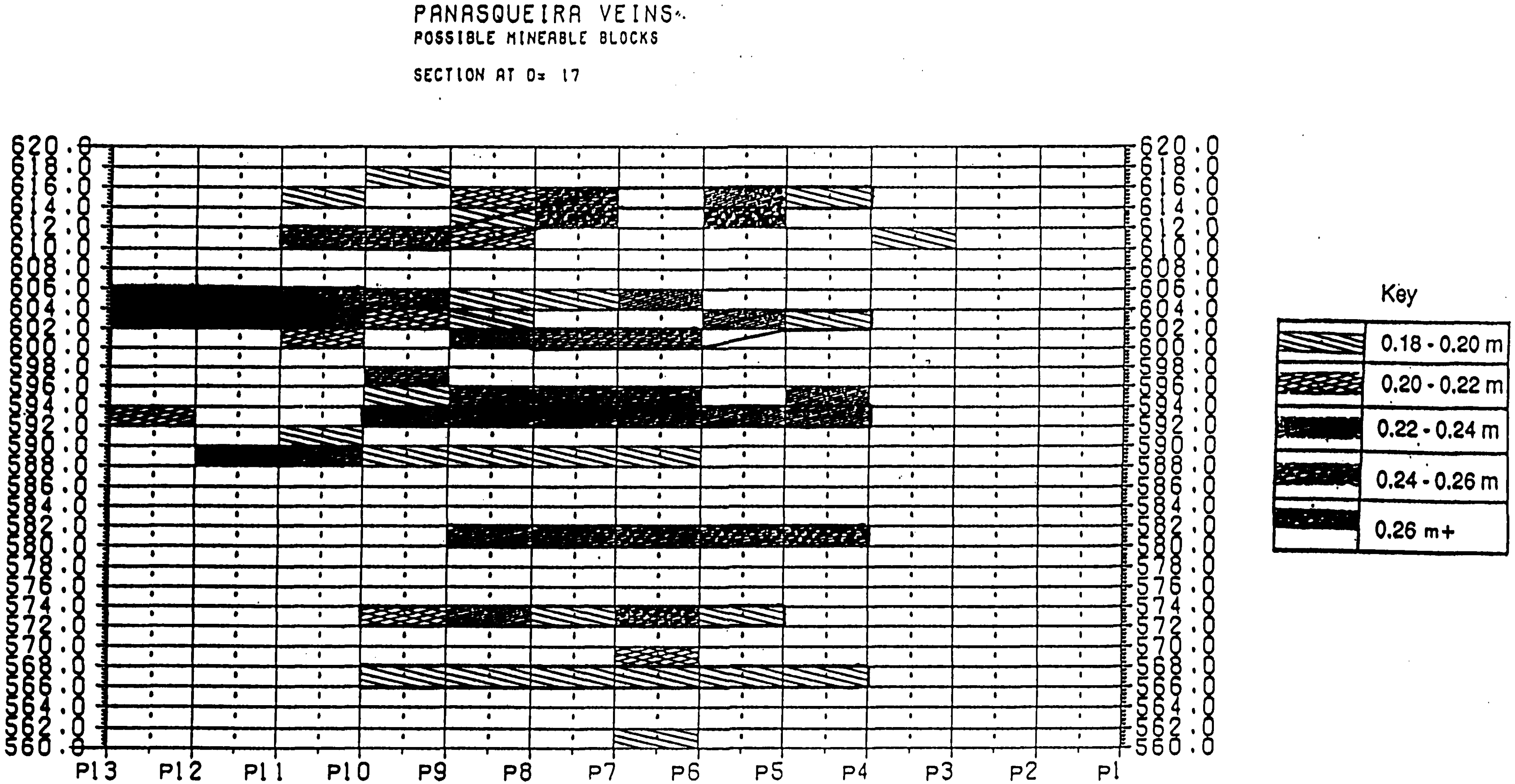


Figure A5b. Section along Drive D17 showing delineated areas for develop-

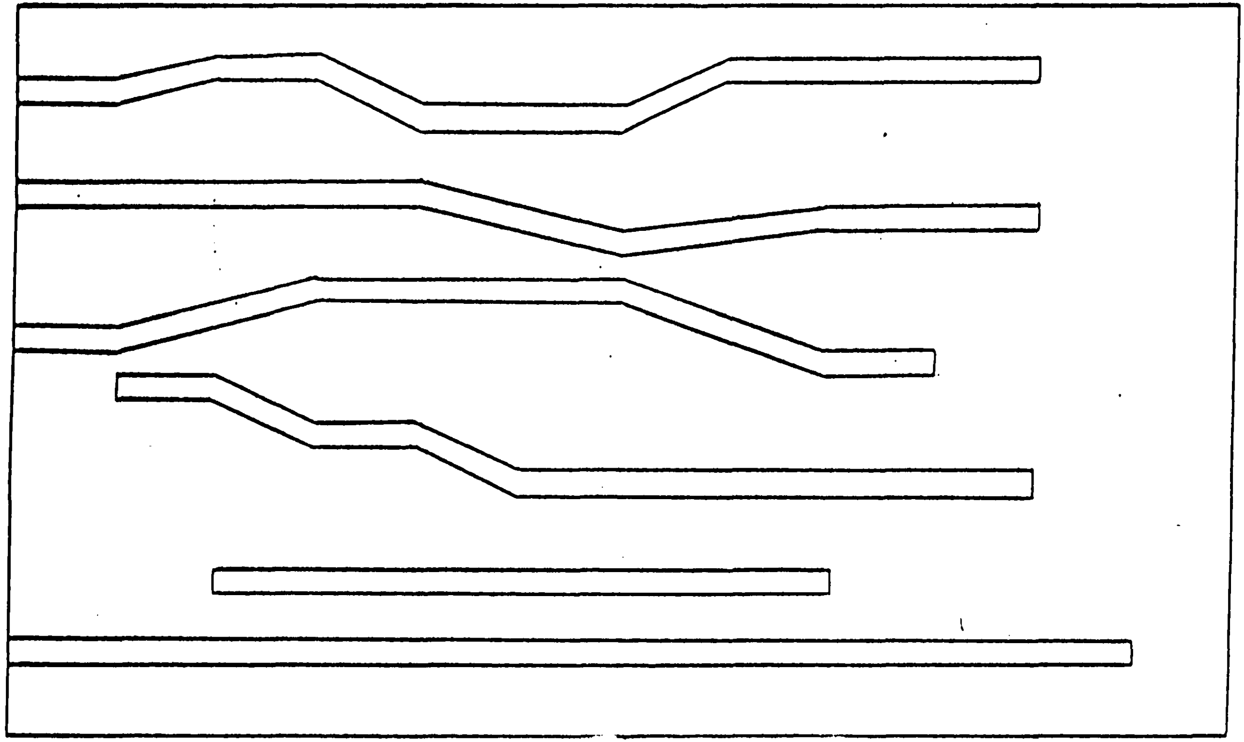


Figure F3. Section along Drive D19 showing delineated areas.

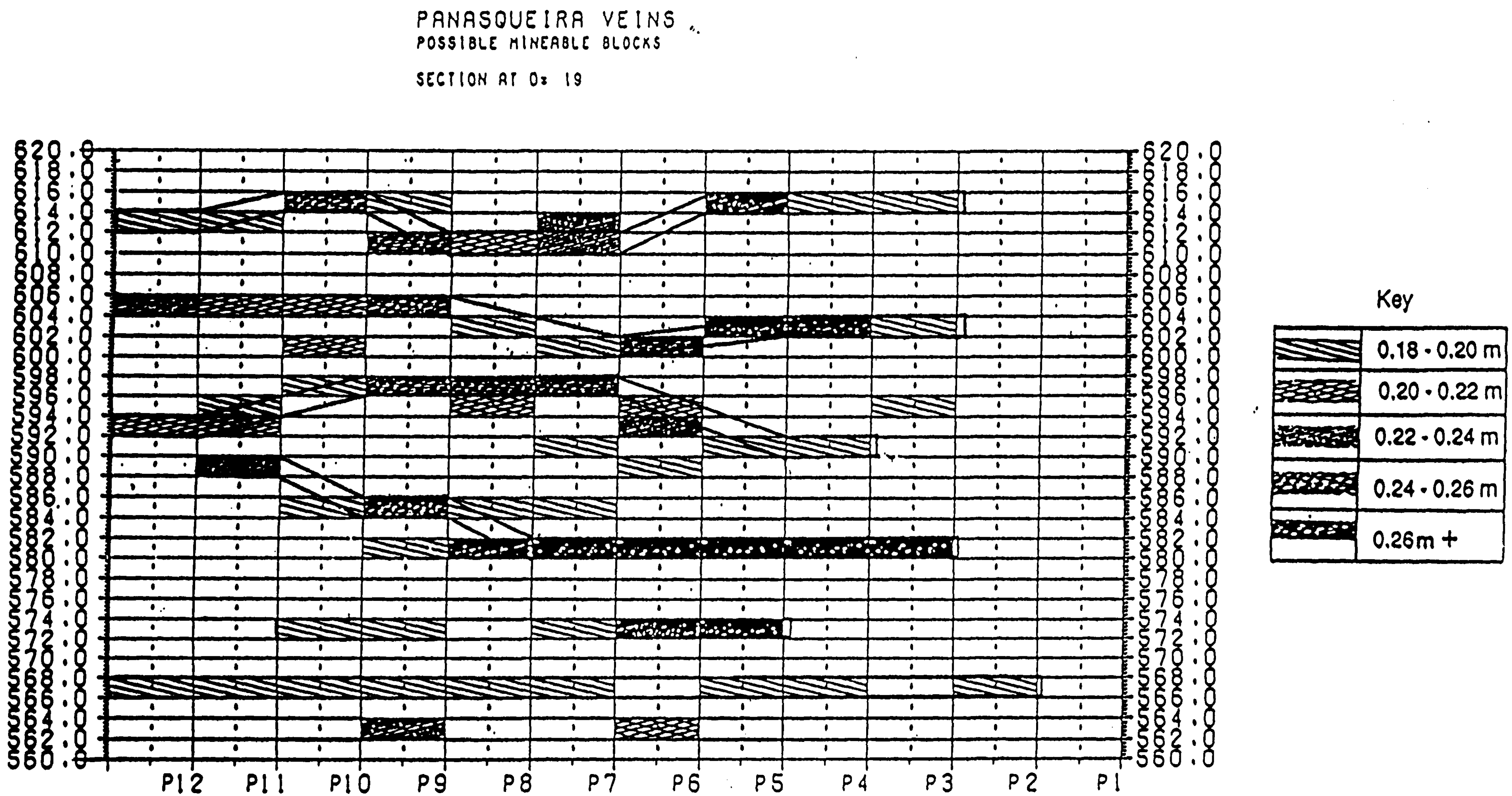


Figure A5c. Section along Drive D19 showing delineated areas for development.

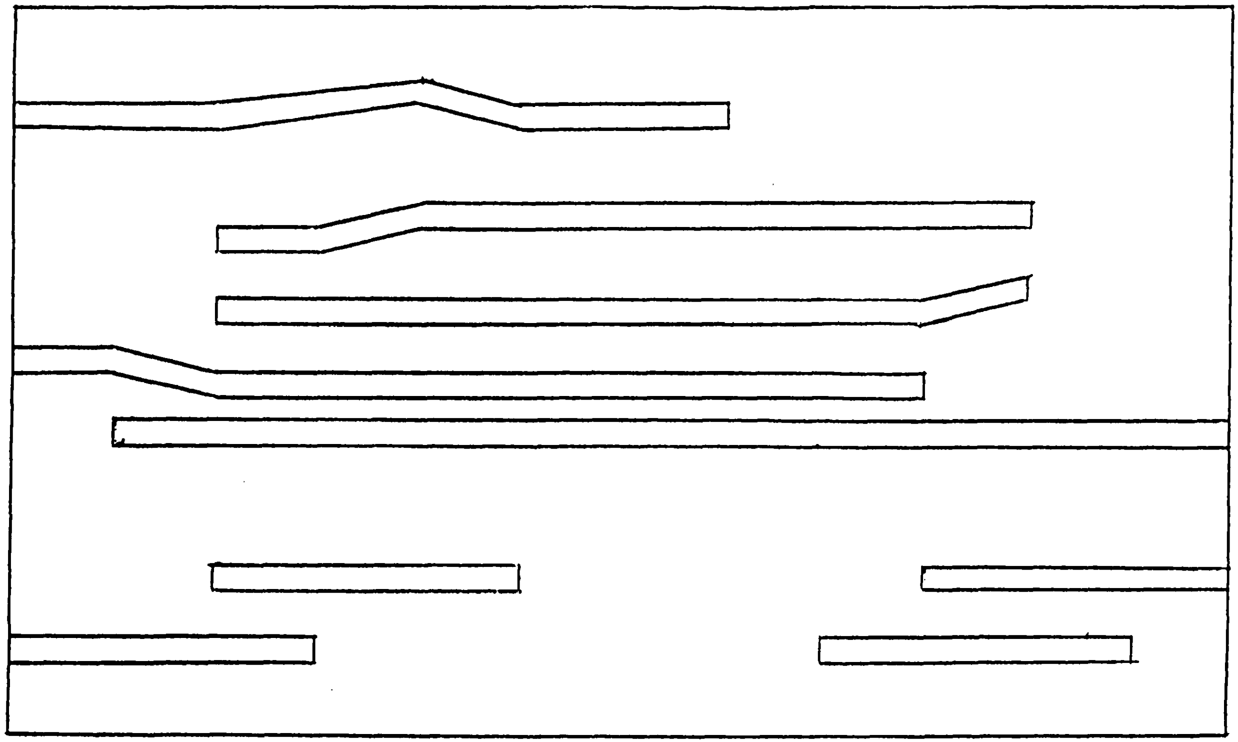


Figure F4. Section along Drive D21 showing delineated areas.

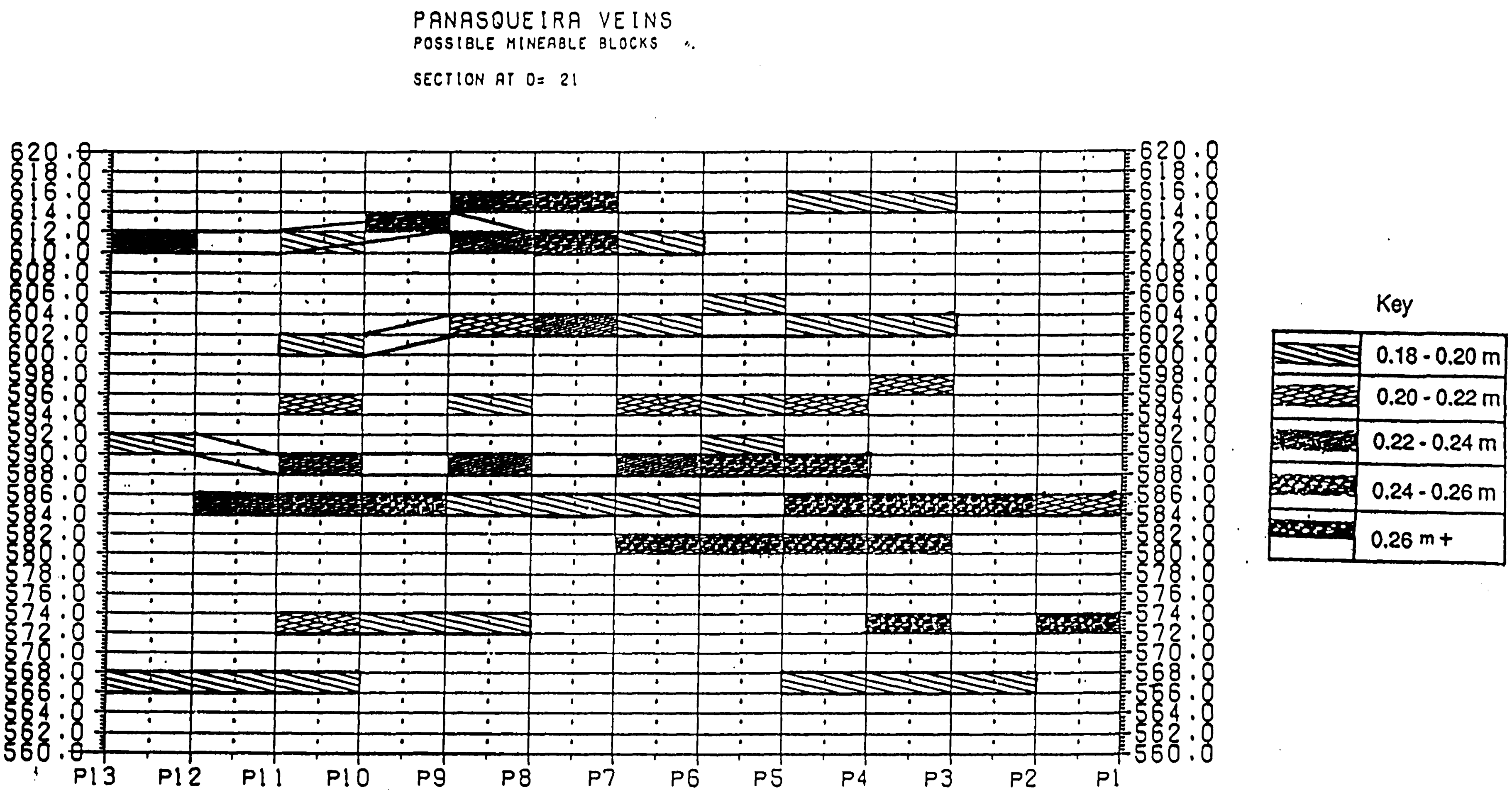


Figure A5d. Section along Drive D21 showing delineated areas for development.

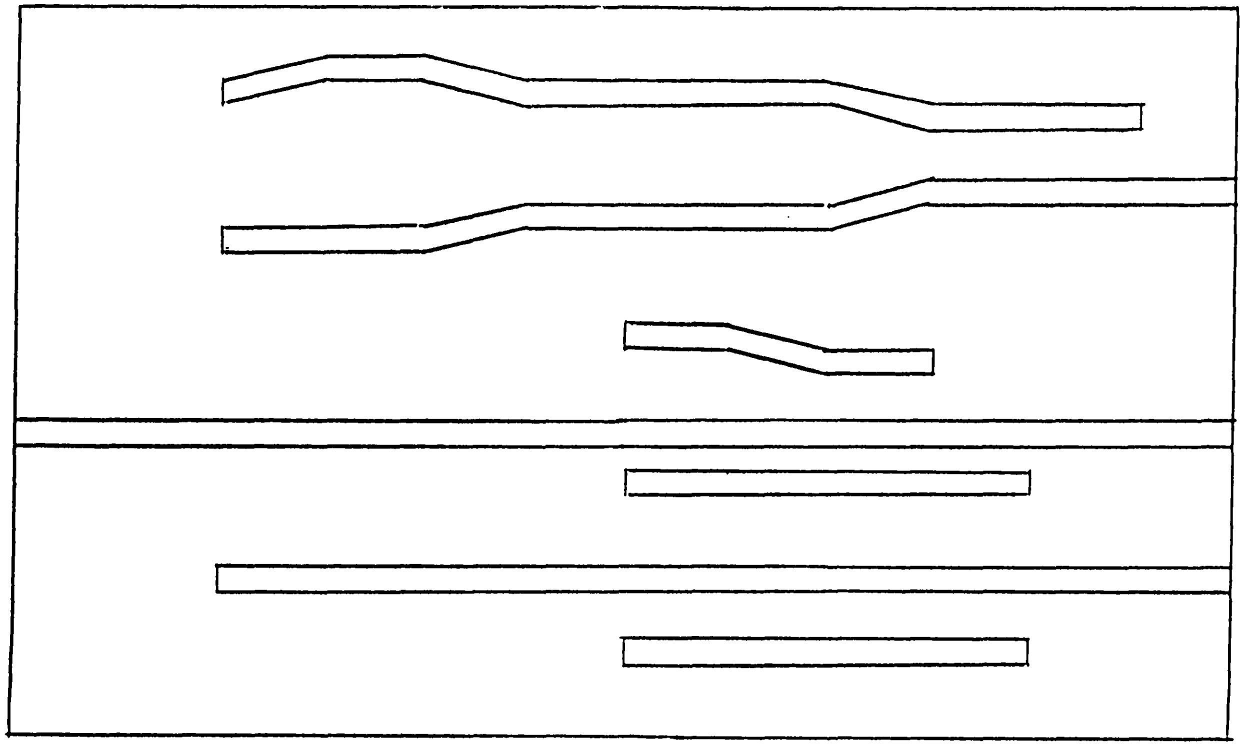


Figure F5. Section along Drive D23 showing delineated areas.

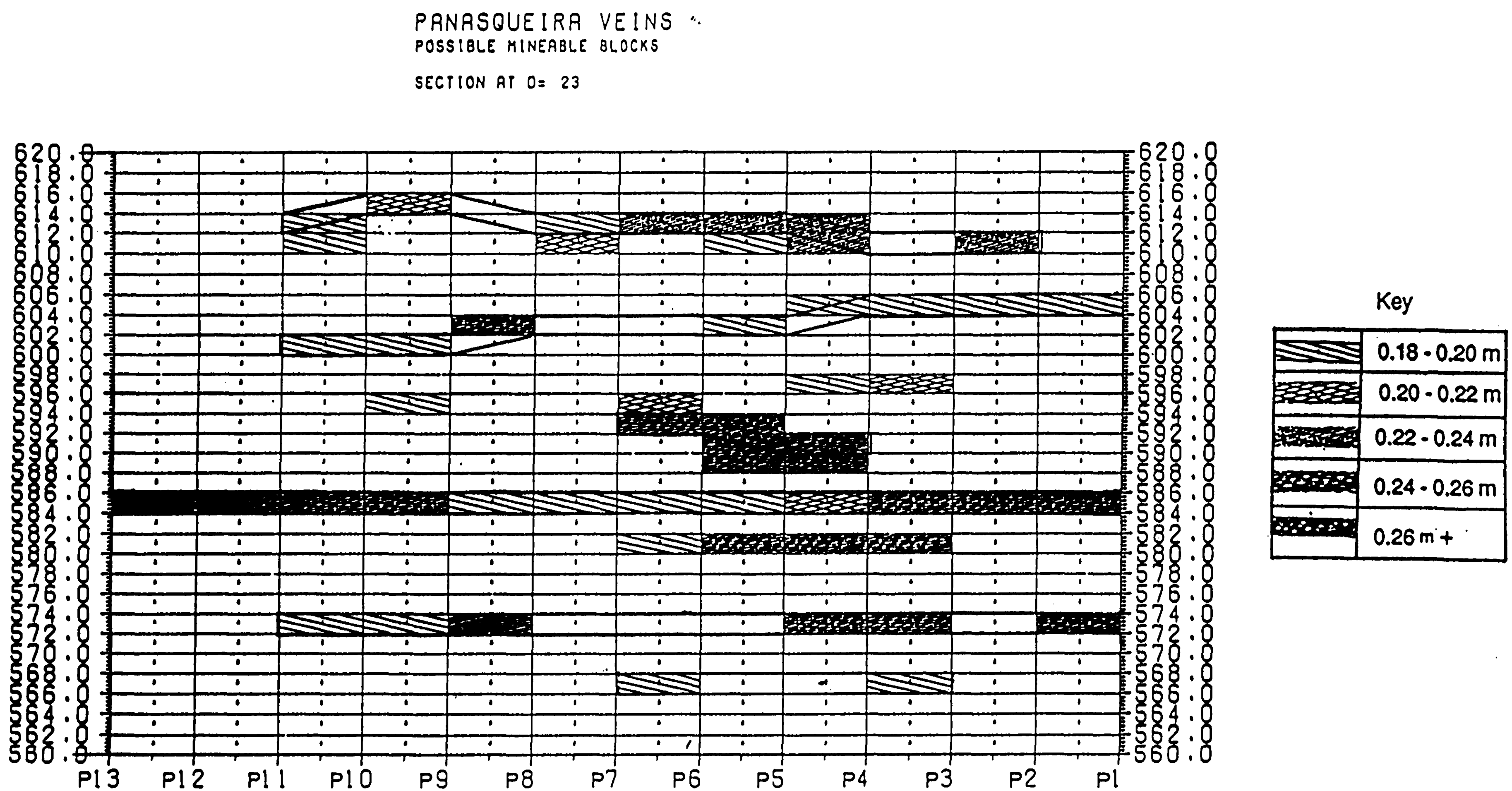


Figure A5e. Section along Drive D23 showing delineated areas for develop-

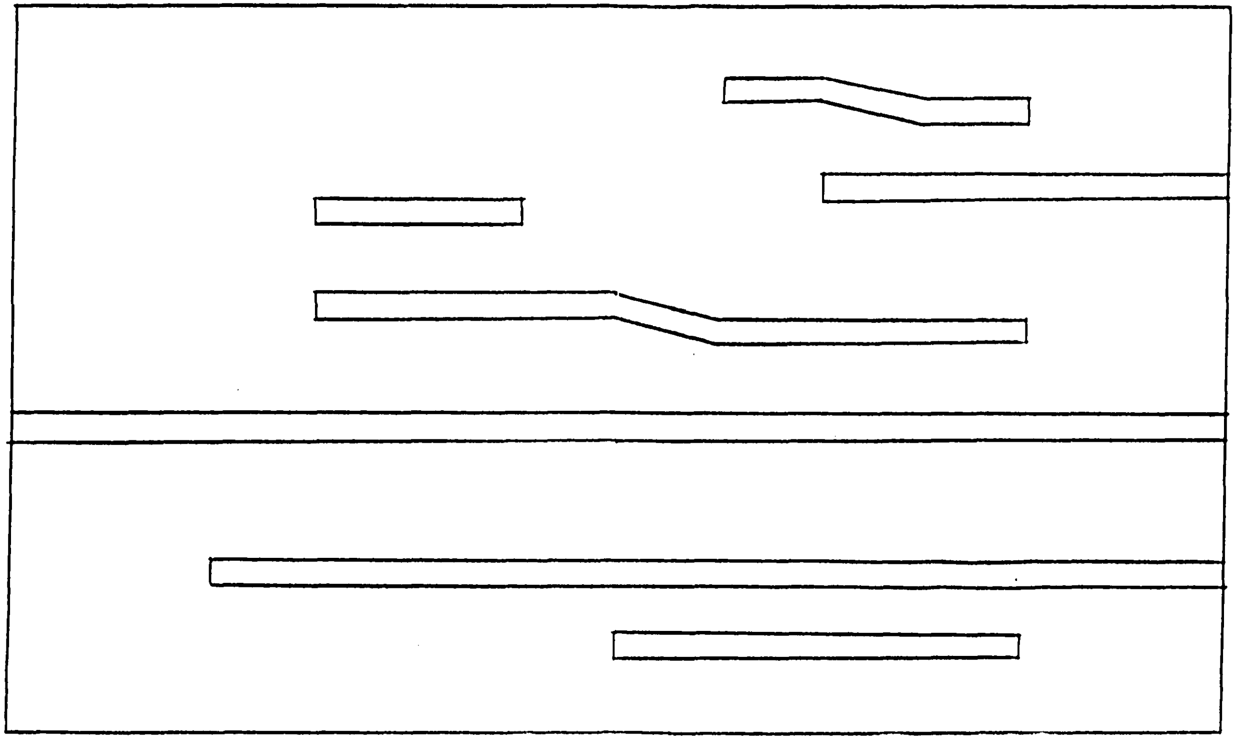


Figure F6. section along Drive D25 showing delineated areas.

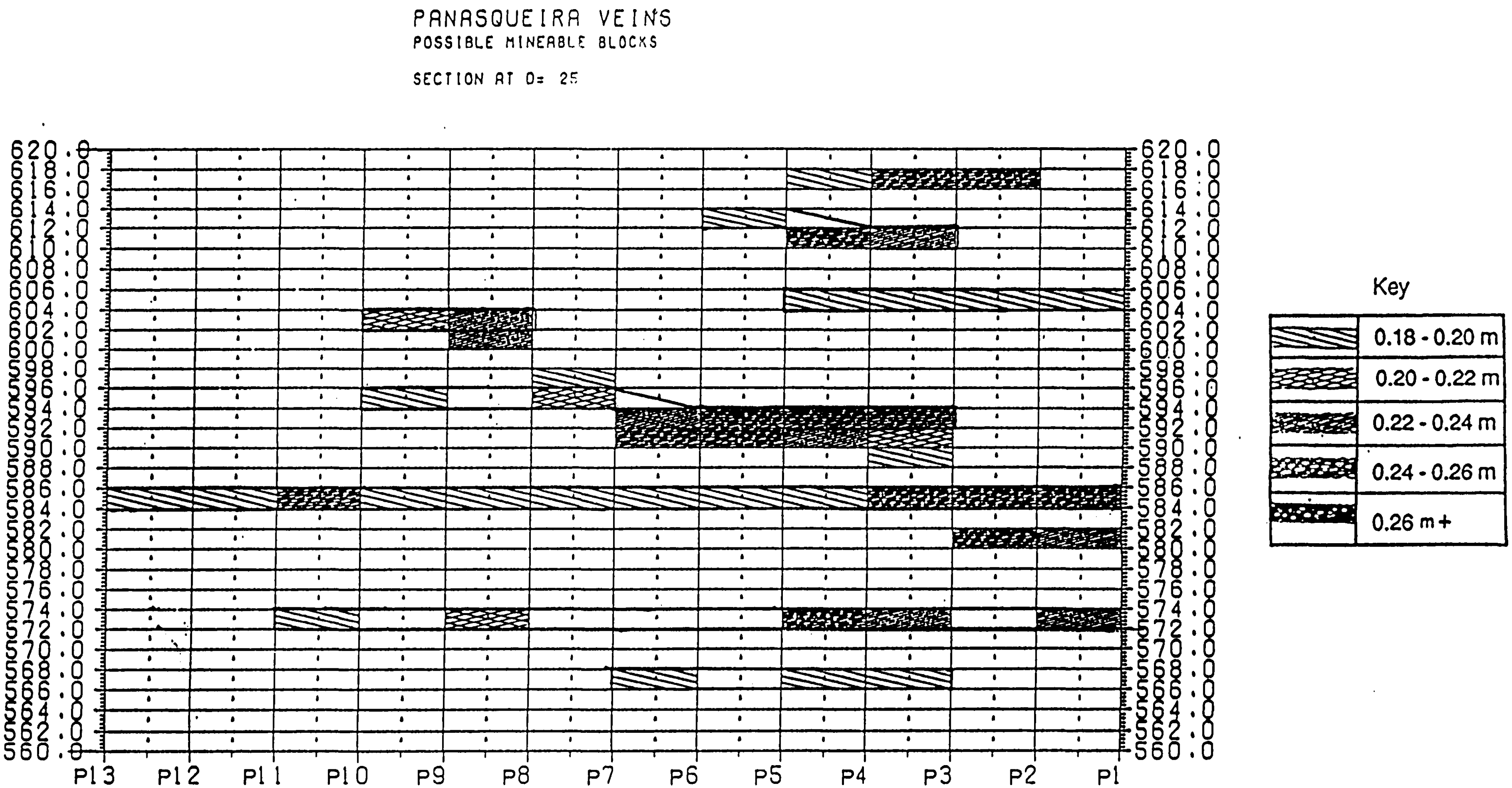


Figure A5f. Section along Drive D25 showing delineated areas for develop-