GEOLOGICAL CONTROLS FOR COAL EXPLORATION AND MINING

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ABSTRACT

The identification and interpretation of geological features is essential for the planning and ultimate success of any mining venture. Examples of geological features significant for mining are presented, and their identification during exploration discussed. In particular, the importance of coal qualities, seam thickness and seam elevation are emphasised in relation to longwall mining.

Geostatistical analysis provides a powerful tool for improving the prediction and decision-making capabilities of both exploration and mine geologists. The availability of geostatistics, and the benefits resulting from its application, are demonstrated using actual data for calorific value, seam thickness and seam elevation.

Contamination of run-of-mine coal is a common problem on highly-mechanised collieries. The problem generally arises from over-cutting of the designated mining horizon. A practical system for monitoring and controlling contamination on a mechanised bord-and-pillar and longwall colliery is presented. The results and benefits of applying such a system are cited for an actual longwall colliery.

Numerical geological predictions are not always reported in terms of the reliability of such estimates. Many of these values can be reported in terms of confidence limits, particularly for routine grade control purposes. The methods and benefits of such reporting are described and illustrated by way of examples for calorific value and contamination levels.

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OPENING STATEMENT

It is common in industry for experience to be promoted. The impact of this on exploration is that the less-experienced geologists are often those who log and sample drill-core. While this practice is not necessarily incorrect, it may result in significant "gaps" in the geological database, due either to the non-recognition of some features, or failure to appreciate the significance of these features. These "missing" features may be critical to the planning, marketing and operation of any mining venture.

The dissertation addresses the following four aims:

1. To remind the coal explorationist of the significance of various geological factors for mining.
2. To illustrate the availability of geostatistics as a tool for improving sampling efficiency and prediction capability to exploration and mining geologists alike.
3. To present a system for monitoring contamination on a longwall colliery.
4. To demonstrate the benefit of reporting errors of estimation and confidence limits for routine geological or grade control estimates.

The dissertation begins with an overview of the main coal-mining methods. This is followed by a description of various geological factors affecting the performance of these methods, particularly in underground collieries. The identification of these features during various exploration phases is also discussed.

Geostatistical analytical techniques are then described and applied to a set of real data for calorific value, seam thickness and seam elevation. The aim is to provide a practical guide of the analytical steps involved, and the results available, that may be followed by a "non-geostatistician".

The improved estimation reporting provided by geostatistics is then incorporated in a grade and contamination control example for a longwall colliery. The contamination control system described was developed and implemented over a period of more than three years. Simple and effective, the system is believed to be unique. The routine calculation and application of "errors of estimation" and "confidence limits" are discussed and illustrated.

1. GEOLOGICAL FACTORS AFFECTING MINING

1.1 INTRODUCTION

A brief overview of various mining methods employed in the coal industry is presented to remind the explorationist of the pen-ultimate application of the whole prospecting effort. The significance of various geological factors for mining, and their recognition during exploration, is also discussed.

The ultimate prospecting application in this context is the sale of the raw, or processed (beneficiated), coal product. The driving force behind both prospecting and mining is the marketing and sale of the product. However, mining and marketing are inextricably linked. The volume and quality of the sales product may depend as much on the mining method and its performance, as the parameters of the *in situ* coal reserve.

Both mining and marketing depend on the actual physical and chemical characteristics of the coal reserve. These actual characteristics are never all known with 100% certainty, and it remains the responsibility of the exploration and mining geologist to predict the "actual" reserve characteristics as accurately as possible, or as may be required, given limited financial and time resources. The reliability (accuracy or confidence) of predictions is discussed in Chapters 2 and 3, with specific emphasis on calorific value and seam thickness.

1.2 MINING METHODS

Various coal mining methods are described and then compared with respect to average costs, productivity and production trends.

1.2.1 Open-cast methods

(Based on MacGillivray, 1984).

Open-cast mining proceeds by completely removing overlying soil and/or rock (overburden) to expose a seam or number of seams for extraction. Three obvious advantages of open-cast to underground mining are: safety; high extraction rates (about 90%); and reasonable flexibility, providing sufficient coal is exposed. The methods are also relatively labour un-intensive.

The methods are generally only viable where little overburden exists, or where the stripping ratio ("saleable" coal:overburden) renders the operation economical (typically a ratio in the order of 1:5, with overburden of less than 50m). Open-cast strip mining may also be used to recover shallow coal left behind as pillars or barriers in old underground mining areas.

OPEN-PIT MINING

Open-pit mining may be used to extract very thick coal seams down to relatively great depths. Excavation proceeds in a number of steps or benches, and overburden material is dumped outside of the pit itself. The bench dimensions determine the angle and stability of the pit-walls, and depend on the engineering properties of the soil and rock to be excavated.

The largest open-pit colliery in South Africa is Iscor's Grootegeluk Mine at Ellisras, where a 60m thick coal seam and 15m of overburden are worked with five benches.

STRIP MINING

Strip mining proceeds with overburden removed in the form of long, narrow trenches. Individual trenches may be tens of metres wide (30m to 80m) and hundreds or thousands of metres long. Overburden ahead of a mined-out trench is generally cast into the void left by this trench, thus exposing a new strip of coal. For multi-seam or selective strip-mining, the depths of the individual trenches may vary along their lengths or between trenches, with strips of different stratigraphic horizons exposed at any one time.

1.2.2 Underground methods

(Based on Fauconnier and Kersten, 1982; Cloete *et al.*, 1984).

The two major underground mining systems in use involve methods that either support or cave the roof strata. The surface is left essentially intact with the first system (bord-and-pillar), while the roof strata in the latter systems (pillar-extraction or longwalling) are caved in a controlled manner, invariably with some surface effects. Open-stoping or sub-level caving (not discussed here) may be used for mining dipping seams of more than 10m in thickness.

Having selected a mining system, the next consideration is the method of actually breaking or winning the coal. The decision is generally between blasting or machine-cutting methods. The latter are generally more productive than blasting operations, and are commonly employed in roof-caving mining systems.

In addition to the geology of the reserve, the choice between the various mining methods depends on other factors such as: technology and equipment availability; skilled-labour availability; surface and sub-surface considerations (legal and environmental); capital availability; and cost and market considerations, such as price and quality (including sizing) requirements.

BORD-AND-PILLAR MINING

Bord-and-pillar mining is commonly employed in a wide range of seam thicknesses, but is accompanied by a rapid decrease in extraction as the seam thickness and/or depth of mining increase. For a 3m coal seam, 75% extraction at a 50m depth reduces to only 35% extraction at a depth of 200m. The workings are generally designed around the Salamon-formula safety factor, although recent design improvements, with improved extraction benefits, have been possible by application of the "squat-pillar" formula (Madden, 1991).

A machine-cutting (continuous miner) bord-and-pillar section is illustrated in Figure 1.1. Seams thicker than the primary capability of the equipment (of up to 4.0m to 6.0m), may be mined by employing the secondary-mining operations of top and/or bottom coaling.

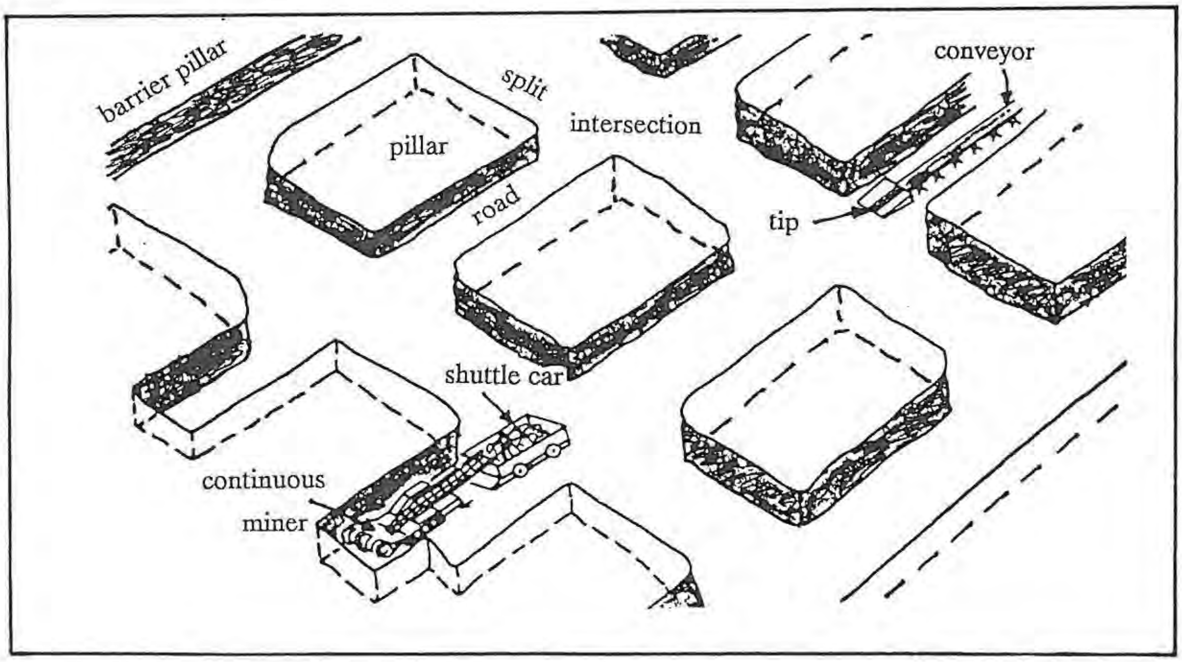


Figure 1.1 *Components of a bord-and-pillar (continuous miner) section (after Cloete et al., 1984).*

PILLAR-EXTRACTION MINING

Conventional pillar-extraction is practised with large pillars left during primary bord-and-pillar advance. These pillars are then extracted during a secondary phase of mining on retreat. An in-panel extraction of around 90% is possible. Pillar-extraction is not commonly practiced in South Afica, although Usutu Colliery and thin-seam mines in Natal have employed the method.

Rib-pillar extraction refers to a series of methods designed to extract a strip or rib of coal situated between development roads and the goaf (area of caved roof strata), with a solid block of coal (the balance of the panel) providing the major support. Rib-pillar extraction has been successfully employed at various SASOL collieries and is popular in Australia (Wagner, 1989). An in-panel extraction of up to some 93% is possible.

Pillar-extraction mining methods are not generally regarded as the safest of the "total-extraction" techniques, primarily due to roof-support problems in the working areas.

SHORTWALL AND LONGWALL MINING

Shortwall and longwall methods extract all of the coal over the width of a panel face in successive slices or cuts, with the roof strata allowed to goaf behind powered, self-advancing supports.

The shortwall method operates conventionally with cutting by continuous miners and loading by shuttle cars or continuous haulage systems. In South Africa, the term "shortwall" has been used to describe a shortened-longwall at New Denmark Colliery (S. Afr. Mining World, Sep. 1990), which is the only "shortwall" system employed in the country. This shortwall has also demonstrated the ability to extract pillars from previously mined bord-and-pillar areas (Amcoal Annual Report, 1991).

Longwall mining makes use of a shearer to cut the coal and an armoured conveyer system to move the coal from the face. "Retreat" longwall mining entails the pre-development of a panel with main-gate and tail-gate developments, after which the coal is mined on the retreat from the boundary back to the main development (Figure 1.2). "Advance" longwall mining, which takes place simultaneously with gate-road development away from the main development, is not practiced in South Africa. Retreat longwall mining in South Africa is currently employed at various SASOL collieries, Matla Colliery (No. 5 Seam), and New Denmark Colliery.

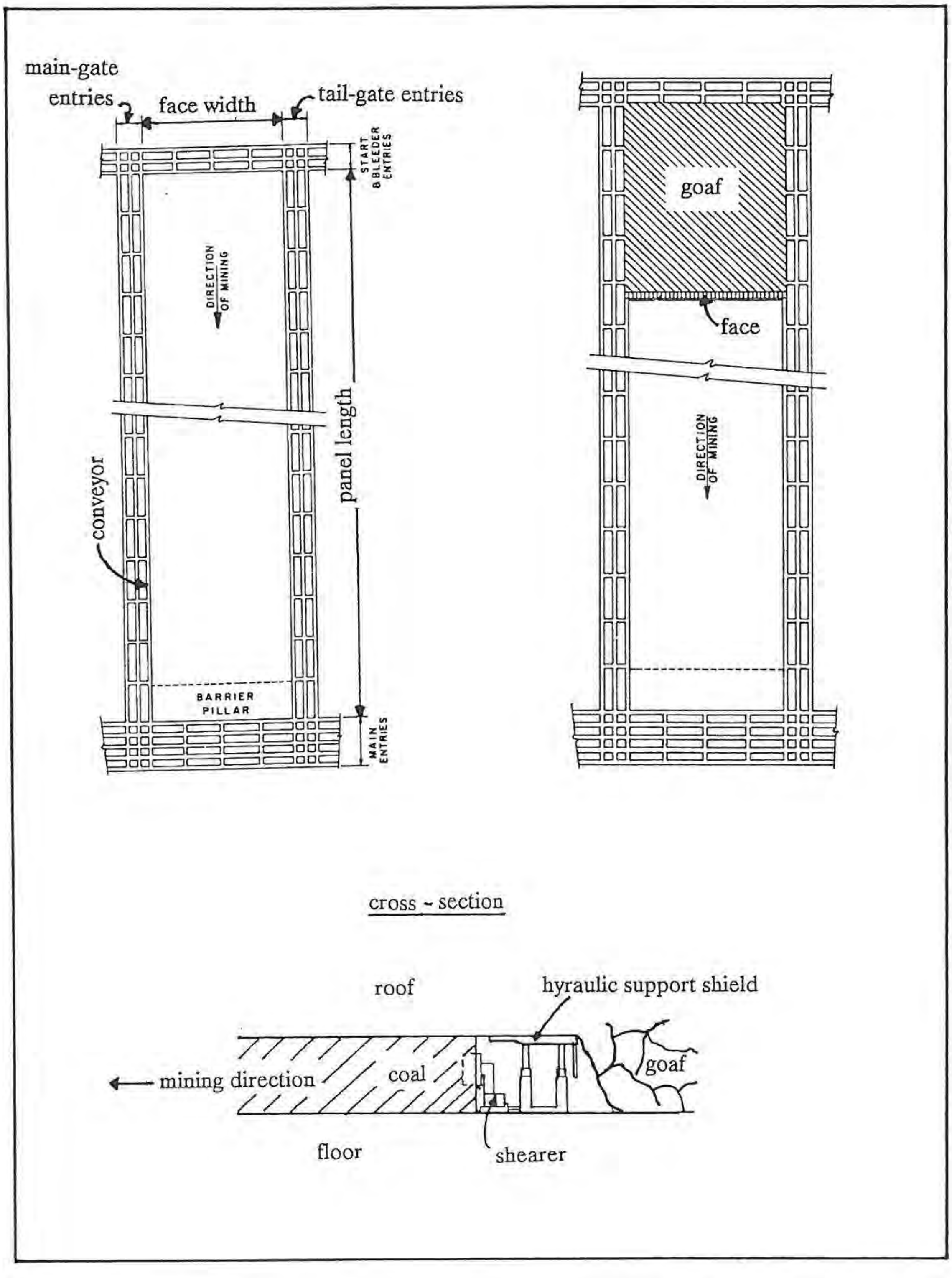


Figure 1.2 *Components of a retreat longwall section (after Trent and Harrison, 1982; Ziolkowski, 1989).*

The gate-road developments generally consist of two or three roads in a bord-and-pillar section. Some of the gate-road pillars, or portions of them, may be extracted during longwalling to facilitate caving. Longwalls are generally applied in seams varying in thickness from 1m to 4.5m and operate with a face width or length of about 200m. The panel length is generally in the order of 1000m to 3000m to minimise section moves. An individual longwall panel will normally be one of a number of parallel panels in a planned longwall area, the overall dimensions of which will depend on various geological conditions, such as seam continuity, seam elevations (dips), and dolerite structure.

Longwall mining may achieve in-panel extraction rates of some 90%, compared to 30% to 60% for bord-and-pillar mining, at depths of between 100m and 200m in similar seams. Retreat longwalling is also potentially the most cost-effective method for bulk-tonnage mining, while its benefits in terms of labour productivity and safety are indisputable (Wagner, 1989).

1.2.3 Comparison of mining methods

Underground mining of hard coal on a world-wide scale has become increasingly important during the 1980's, while in South Africa, open-cast mining has become more firmly established over the same period (Table 1.1). The importance of underground mining, at between 60% and 65% of total world production, is expected to increase, or at least remain static, during the current decade. Table 1.2 highlights the increasing importance of longwalling as an underground mining method, and summarises typical productivity levels for various mining methods.

The relative breakdown of actual in-section operating costs per mining system are illustrated in Figure 1.4.

Table 1.1 *Underground versus open-cast coal production for the World, South Africa and the USA.*

	WORLD		SOUTH AFRICA+		USA[oo]	
	1975*	1989[x]	1977	1988	1975	1988
% UNDERGROUND OF TOTAL	45.2	65.0	81.5	61.5	44.6	40.0

* - Beasley, 1978.
\+ - Plaistowe et al., 1989.
x - Spooner and Chadwick, 1990.
oo - Wagner, 1989.

Table 1.2 *World, South African and USA coal-production and productivity by mining method.*

	WORLD* 1975	SOUTH AFRICA+ 1977	1988	USA° 1978	1988	PROD.°°
Continuous Miner and Machine Loading	94.7	85.0	85.0	95.0	80.0	3100
Longwalling	3.6	3.4	12.9	5.0	20.0	7000
Other[x]	1.7	11.6	2.1	-	-	-
Underground						3200
Opencast						4200

* - Chironis, 1977.
\+ - Plaistowe et al., 1989.
x - Includes hand-loading and dump-reclaimation.
o - Wagner, 1989.
oo - Productivity in tons per underground worker per year, average for South Africa, Australia and USA. (Mining Survey, 1988).

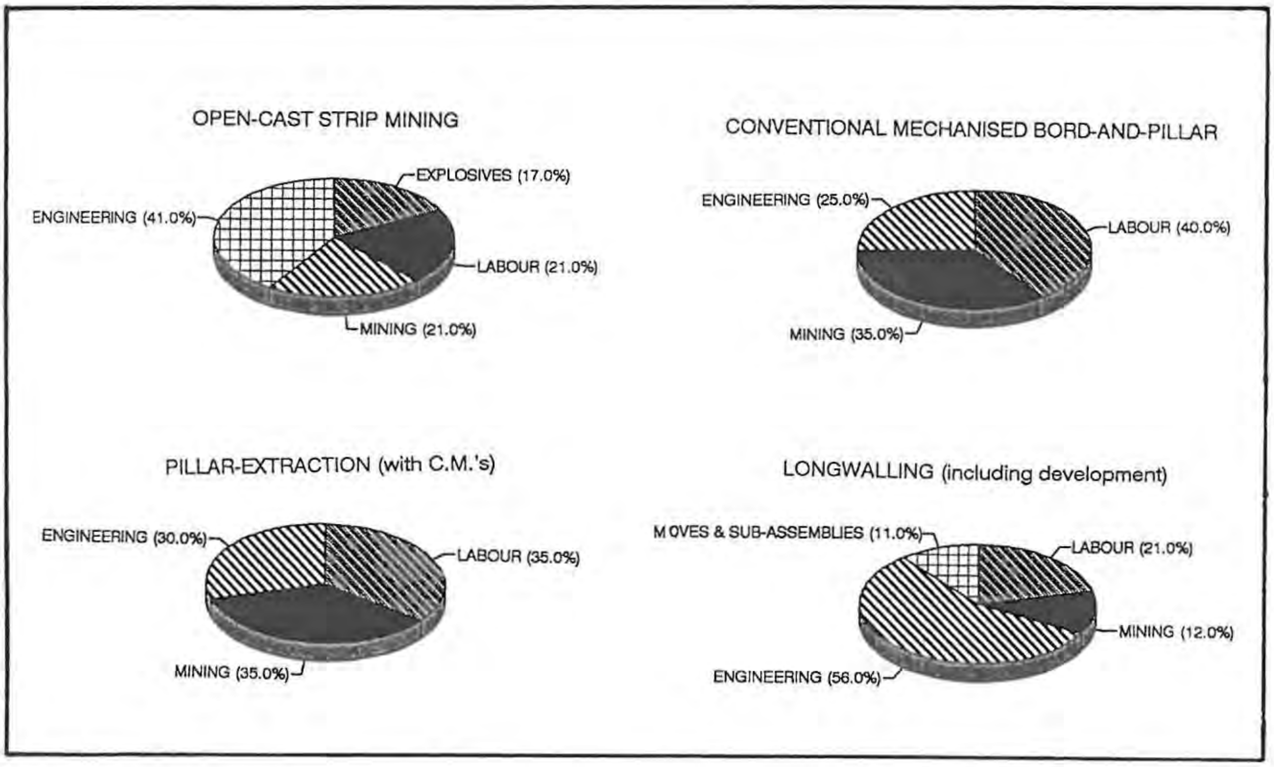


Figure 1.3 *The breakdown of operating cost by coal mining system (MacGillivray, 1984; Fauconnier and Kersten, 1982).*

Table 1.3 summarises the relative operating costs and capital outlays for underground mining systems for given production capacities.

Table 1.3 *Relative operating and capital costs for selected underground coal mining systems (Fauconnier and Kersten, 1982).*

	RELATIVE OPERATING COST	MONTHLY PRODUCTION (tons)	RELATIVE CAPITAL OUTLAY
Continuous miner (bord and pillar)	1	45000	1
Continuous miner (pillar-extraction)	1	34000	1.35
Longwalling (including development)	1.2	120000	2.25

Recent estimations indicate the relative capital outlay of a longwall to be three times that of a C.M. section , while a typical unit production cost (including engineering maintenance) for a pillar-extraction section in 1989 was some R9 per ton (De Beer *et al.*, 1991).

Figure 1.4 illustrates the approximate breakeven situation between open-cast, longwall and bord-and-pillar mining for various seam depths and thicknesses (overall longwall extraction assumed constant at 65%). In addition, the bord-and-pillar system is limited to operating slopes of less than some 18^0, while longwalling may be applied in seams dipping at up to 75^0, providing the equipment has been so designed (Sleeman, 1977).

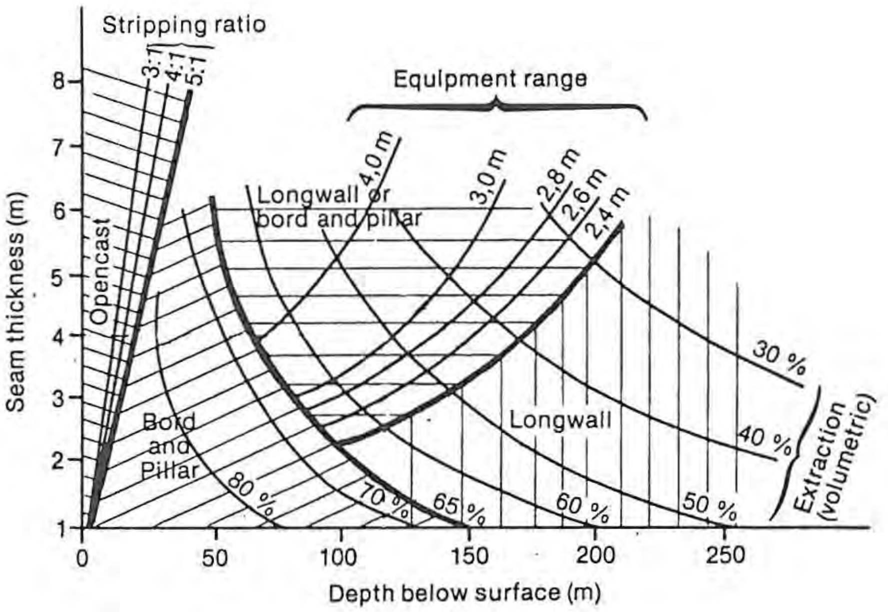


Figure 1.4 *Approximate breakeven conditions between open-cast, longwall and bord-and-pillar coal mining systems (Fauconnier and Kersten, 1982).*

1.2.4 The need for increased extraction

The narrow economic operating range of opencast mining is highlighted in Figure 1.4. The increase in world underground production compared to opencast production further highlights the increasing depth of remaining World reserves. As for any non-renewable resource, the drive should be towards maximum utilization of the coal resource. In gently-dipping seams, at depths greater than about 100m (bord-and-pillar extraction rates <65%), the optimum method of mining, depending on the capital available and local conditions, is either longwalling or rib-pillar extraction.

South Africa's recoverable reserves of some 58Bt have an expected life of 240 years at a 5% annual production growth rate. The average life for the balance of the world's hard coal producers with recoverable reserves of 630Bt, would be in the order of 150 years (Noppe, 1991).

In addition to the generally increasing depth of remaining reserves, any remaining shallow reserves are generally of lower quality than those currently mined. Maximising extraction from the better quality reserves must be a primary aim to prevent the "sterilisation" of coal in unnecessary barriers and pillars. The impact of the "total extraction" undergound mining methods of pillar-extraction and longwalling on groundwater and the surface remains an important issue. These effects can sometimes be accomodated for their duration, but require continual monitoring together with delicate negotions with those affected.

1.3 GEOLOGICAL FACTORS

1.3.1 Factors

PHYSICAL FEATURES

Physical geological features and parameters affecting coal mining are summarised in Table 1.4, together with an idea of their relative significance for different mining systems. Many of these features are self-explanatory and require no further discussion at this level.

QUALITY PARAMETERS

The term "coal quality" encompasses a wide range of physical, chemical and engineering analyses and tests. These tests are initially aimed at determining the ultimate uses for the coal in a given deposit. End uses may include gasification, coke-production, direct reduction or steam generation.

The analyses aim at simulating the behaviour of the coal under its practical conditions of use. Not all of the analyses are adequate for this purpose, and research into improved analytical techniques is ongoing, for instance into measuring coal abrasiveness.

Table 1.4 *Geological factors affecting coal mining.*

GEOLOGICAL FACTOR	RELATIVE IMPORTANCE TO GIVEN MINING SYSTEMS				EXAMPLE OF SIGNIFICANCE
	O/C	B & P	P-EX	L/W	
SURFACE					
Topography, hydrology	H	M-L	M	M-H	Pit dewatering, inflow into U/G, subsidence.
Soil and clay types, distribution	H	M	L	L	Surface stability and trafficability.
OVERBURDEN					
Weathering -depth, extent, controls	H	M-H	L	L	Shallow seam quality, O/B strength.
Rock types: thicknesses & engineering properties	H	M-H	H	H	O/B removal (blasting), caving characteristics,
Sedimentary structures and classification	M-L	M-H	H	H	U/G roof stability, facies interpretations.
Matrix, cement and, secondary minerals	M	H	H	H	Weathering behaviour (U/G oxidation).
Structural features	H	H	H	H	Potential discontinuities.
Gas potential (emission)	L	M	H	H	With caving - ventilation needs.
Spontaneous combustion potential	M-L	M-L	M-H	M-H	Pit-wall or spoils, or the goaf - ventilation.
In situ stress regime	L	M	H	H	Mine design.
FLOOR STRATA					
As for OVERBURDEN	H	H	H	H	Behaviour with heavy equipment, water, etc.
GROUNDWATER					
Aquifer location, controls Capacity, recharge	H	H	H	H	Water inflow into workings.
Water quality	H	H	H	H	Environmental impact.
Water pressure	L-M	M	H	H	U/G safety.
Borehole yields	M	M	H	H	Replacing farmer supplies.
STRUCTURAL FEATURES					
Faults, joints	M-H	M	M-H	H	Throw of 1/2 seam thick. can stop L/W.
Coal slip-planes	M	M	M	M	Stability of coal face.
Coal cleats	H	H	H	H	Bulk coal behaviour.
Intrusions, dykes	M	M	M	H	Mine layout.
sills	H	H	H	H	
Dips: regional	M-L	H	H	L	Equipment performance,
local	M	H	H	H	mine design, dewatering.

Table 1.4 *Geological factors affecting coal mining (cont.).*

GEOLOGICAL FACTOR	RELATIVE IMPORTANCE TO GIVEN MINING SYSTEMS				EXAMPLE OF SIGNIFICANCE
	O/C	B & P	P-EX	L/W	
MINEABLE COAL					
Depth of seam/s	H	M-H	M	M	Mining method & *in situ* stresses.
Thickness variation	M-L	M	M	H	Dilution, equipment selection.
Sub-outcrop position	H	M-H	M	L	Pit limits, U/G stability.
Roof/floor contacts ease of parting nature, e.g. gradational	H	H	H	H	Behaviour with mining, actual recognition during mining.
Intra-seam partings types thickness/es seam position lateral extent engineering characteristics	H	H	H	H	Seam correlation, equipment specification, mining horizon control, quality predictions.
Seam stratigraphy/zonation vertical & lateral on visual & quality basis	H	H	H	H	Selective mining horizon selection & correlation.
Structural features (as above)					
Mechanical behaviour	H	H	H	H	Degradation with mining, dust, sizing-recovery, cuttability.
Gas potential (emission)	L	M-H	H	H	Safety, ventilation.
Spontaneous combustion potential	M	M	M	M	Mine & stockpile design.
Intrusion-related features devolatilisation, induration, gas, slickensides & weakened ground	M-H	M-H	H	H	Coal quality, support requirements, mining layouts, gas drainage.

ABBREVIATIONS

B&P	bord-and-pillar	H	High degree of importance
P-EX	pillar-extraction	M	Medium degree of importance
L/W	longwalling	L	Low degree of importance
O/C	open-cast		
O/B	overburden		
U/G	underground		

Once the coal has been characterised and its potential product or products identified, then the analyses conducted on a routine exploration (drill-core) samples and underground channel samples may be reduced to a selected few analyses. Occasional samples may still be analysed for the full range of initial tests to check the consistancy or variability of the deposit. Examples of the typical analyses are given in Table 1.5, together with an indication of whether the analyses tend to be conducted routinely, or only for selected or "special" samples.

A given coal seam may also be sub-divided into a number of plies, or laterally extensive zones with specific characteristics. These plies should be sampled seperately to determine the seams suitability for selective mining. This would add a further constraint on the choice and design of mining systems and methods.

The true value for a given physical or chemical parameter is seldom known, and hence it may be difficult to assess the accuracy of the results for a given test, even if the precision of the test is well-known. In general, standard analytical methods and equipment have been designed for the common analyses, for example the British analytical standards (BS). The "true" value may then be assumed to be the best estimate of the mean of the results from a number of laboratories. This would account for analytical errors, including the "human" factor, and may allow laboratories to compare the reliability of their results. For some analyses, such as calorific value, it may be standard procedure to conduct two analyses and report the average, thus reducing to some degree the sample-preparation and analytical error, while checking the repeatability (precision) of the test. In this case, the within-laboratory results for two sub-samples prepared from an original sample should be within 0.12 MJ/kg (BS S1016-5). The between-laboratory agreement, for the same standard, should be less than 0.30 MJ/kg.

It is important to be aware of the range of analytical errors associated with various results, particularly those on which contracts, and hence grade control systems, will be designed. Where necessary, namely with small numbers of sample and/or relatively large analytical errors, such errors should be included when reporting results or estimates of "average" grades or other values. The importance of reporting estimation errors is discussed in Chapter 3 (Grade and Contamination Control).

Table 1.5 *Some analytical coal properties and their importance (Schmidt, 1976; Moodie, 1975; Falcon, 1978).*

PROPERTIES	Routine / Special	EXPLANATION/ IMPORTANCE
INHERENT PHYSICAL PROPERTIES		
Density (relative)	R	Measure directly or estimate from ash content.
Gross heating value (calorific value, MJ/kg)	R	Combined effect of physical and chemical properties.
CHEMICAL PROPERTIES		
Proximate Analyses		
Moisture content (%)	R	Surface and inherent moisture.
Volatile matter (%)	R	Decreases with increasing rank.
Ash content (%)	R	Measure of non-coal or mineral matter content.
Fixed carbon (%)	R	(100-moist.-vol.-ash.)
Ultimate analyses		
Coal composition in terms of % C, H, N, S, O.	S	C increases with increasing rank. H varies with coal type and rank. O decreses with increasing rank. S may be divided into forms: organic and inorganic (sulphide or sulphates).

Table 1.5 *Some analytical coal properties and their importance - cont.(Schmidt, 1976; Moodie, 1975; Falcon, 1978).*

PROPERTIES	Routine / Special	EXPLANATION/ IMPORTANCE
Ash analyses (oxide %)	S	Elemental composition (major and minor elements). Ash behaviour in furnaces.
Forms of silica (elemental salt %)	S	Including P, Cl, Fe and B. Ash behaviour in furnaces.
ENGINEERING PROPERTIES		
Swelling index	R	Coking propensity
Roga index	R	Coking propensity
Ash fusion temperatures (^{o}C)	R	Ash behavior in furnaces.
Abrasive index (mgFe)	R	Abrasiveness of the coal.
Hardgrove grindability index	R	Ease of pulverisisation.
BENEFICATION/PHYSICAL BEHAVIOUR		
Sizing	R	Amount of fines generated (< 0.5mm) and possibly "lost".
Float and sink properties	R	Benefication potential.
PETROGRAPHIC ANALYSIS		
Organic constituents Macerals, micro-lithotypes vitrinite reflectance	S	Coal type (origin) and rank, together with estimates of expected behavior and uses (characterisation).
Mineral matter	S	Type, distribution/ occurrence, grain shape and size. Potential effects on coal properties.

Certain analyses require a minimum mass of sample, for example the current abrasive index test requires a 4kg sample. This may dictate the minimum drillcore size or channel sample width to be taken from a given seam. While this is not necessarily the optimum manner for deciding on sample sizes (sample support), such restrictions must be included when applying the principles of sampling theory. For example, if pyrite framboids measuring 30mm across are spaced at some 300mm along a particular horizon, then there is a 5% chance that a 60mm drill-core will sample the framboid, even though the framboid only makes up some some 1% of the area being sampled.

1.3.2 Identification during exploration and mining

The recognition and interpretation (prediction) of geological features during coal exploration, including on-mine investigations, is of utmost importance. The typical phases of an exploration program are outlined below, together with their aims and an idea of investigation methods that may be applied. Although a number of exploration techniques may be mentioned, this does not mean they are all applied simultaneously. Such "saturation prospecting" can result in costly and unnecessary repetition of results. Instead, the methods mentioned represent a selection of those available to the industry. The discussion is based on works by Adams (1976), Wier (1976), Sleeman (1977) and Noppe (1991b).

The following phases and aims are typical in any exploration-mining program:

PHASE	AIM
MANAGEMENT-EXPLORATION	
-Objective and goal setting.	Set realistic goals.
EXPLORATION	
-Planning (and literature review).	Establish resource potential.
-Regional reconnaissance.	Define inferred reserves.
-Target identification and investigation (preliminary evaluation).	Define inferred-indicated reserves.

-Prospective selection and investigation (evaluation).	Define indicated reserves.
-Final feasibility investigation and evaluation.	Define indicated-proven reserves.
EXPLORATION-MINING	
-Pre-mine investigation and technical report, including detailed mine design.	Further define indicated and proven reserves.
MINING	
-Production build-up investigations. -Full-production involvement. -Production-decline involvement.	Move towards proven reserves over the whole reserve or lease area.

For the purpose of this discussion, a reserve is considered an economically mineable resource, while the terms inferred, indicated and proven (or measured) reserves indicate improved confidence in the delineation of the reserve. The amount of data by way of sampling and other investigations required to place the reserves in the different catagories will vary with company, application, and with the type of mining system planned for the deposit.

OBJECTIVE AND GOAL SETTING

The relative abundance of coal resources, together with their relatively restricted geological time and stratigraphic settings, means the coal exploration objective must be more specific than simply "finding coal". The objective must be expanded to finding and developing the best reserves for a specific application.

The entire exploration drive is directed towards identifying and proving reserves that are, or will soon be, economically mineable. This requires the early development of a risk analysis simulation model to allow projects to be ranked relative to each other, and to monitor the economic potential of developing exploration projects. In addition, the coal quality parameters necessary for coal marketing and characterisation must also be defined at an early stage.

EXPLORATION PLANNING

Planning involves more than the "routine" processess of budgeting, staffing, and contract setting. Perhaps the most important function during this phase is the review of existing literature and available data, particularly since initial reconnaissance will generally be conducted in areas of known potential. "Literature" may include available geophysical and remote-sensing images, as well as aerial photography, and topocadastral and orthophotograph plans. These items can be essential for later detailed interpretation and planning purposes.

The literature review also includes the establishment of mineral and/or property ownership in the areas of interest. If no open ground, or potentially open ground, of suitable dimensions exists, then the initial area of interest may already be rejected.

Basic exploration methodology may also be formulated once a regional area-of-interest has been identified. In particular, the techniques to be applied at various stages may be decided on in principle, for example:

Drilling:	Open-hole or coring.
	Core-sizes.
	Configuration (spacing and pattern of holes): regular grid, or based on developing knowledge and requirements.
	Logging and sampling criteria.
	Data (including core) storage and recording decisions.
Geophysics:	Airborne, surface, or down-the-hole methods to be applied, and when to apply.

The end result of the planning stage will be the identification of an area with the potential to contain the reserves of interest.

REGIONAL RECONNAISSANCE

The aim of the reconnaissance phase is to determine the broad extent and quality of coal in an area, thus moving the resource to the status of an "inferred reserve". Reconnaissance begins with the compilation of geological base plans from the available data. Surface mapping or photogeological-, geophysical- and remote-sensing studies (using available material) may further refine such plans. Planned drilling may then be based on interpretations of the existing data. Regional geophysical surveys may also be conducted to interpret broad basinal structures, such as sub-outcrop and basin palaeotopography.

Open-hole drilling may be conducted initially in an area with limited pre-existing information (no previous drilling or sampling results available). However, if the area shows promise, full-core drilling should be considered at an early stage, since the detailed information available from borehole core is invaluable to later feasibility studies. Any extensive non-core drilling should be accompanied by down-the-hole geophysical logging to improve the reliability of the hole records. The additional cost associated with the geophysical logging may make full-core drilling more appropriate, although a combination of full-core drilling and down-the-hole logging may be appropriate in very thick seam areas, where geophysical logs can aid seam-zone correlations and sampling decisions.

Drilling may be carried out on a wide-spaced grid, or be selectively placed depending on the interpretations of existing information. The reconnaissance drilling will probably be spaced at between 3km to 5km, and as such will provide limited information on structural continuity, such as thickness trends, faulting and dolerite activity. Borehole collars may also not be surveyed at this stage, thus further limiting correlation ability.

Coal quality information may also not be correlatible between holes. Having said that, money spent on one or two detailed coal analyses at an early stage will determine the typical coal character of the different seams or sub-seams present, thus indicating the potential industrial applications of the deposit. This knowledge is fundamental for deciding whether the deposit may meet the initial exploration objectives or not. Detailed coal analyses should accompany detailed sampling of practically-sized and recognisable plies (sub-samples).

The regional reconnaissance phase will be concluded with the identification of specific prospects or targets for more-detailed follow-up work.

PROSPECT INVESTIGATION

Detailed prospect investigation generally implies closer-spaced drilling and possibly more-detailed or better-defined logging, sampling and analytical requirements. The aim is to improve the confidence in the geological reserve from that of an "inferred reserve" to that of an "indicated reserve".

This improved cofidence generally requires a drilling or sampling spacing in the order of 1km or closer. The actual spacing will depend on the variability of the geologically significant features, such as dolerite sills, seam thickness and elevations, and quality parameters. The selected spacing is often determined from the experience gained from prospecting and/or mining similar deposits. Initial drilling may be planned on a broadly-defined grid or series of fence lines. Closer-spaced follow-up drilling may be completed in specific areas of interest, such as to define seam sub-outcrop positions, coal-sill intersections, or steeply-dipping and faulted zones. Borehole collars will either be routinely surveyed or surveyed before final geological structural interpretations are made.

The drilling spacing at this stage is generally too wide for detailed or definitive structural interpretations. For example, faulting with a throw of 10m or less cannot be interpreted. The same holds for seam undulations with a wavelength in the order of the drill spacing or less. Seam washouts, splits and local variations can also not be suitably identified or projected. The definitive interpretation of dolerite structures is generally also not possible (Prins *et al.*, 1976).

This is where geological experience becomes important. Knowledge of similar deposits, possibly combined with sedimentary facies interpretations, may greatly assist geological predictions with regard to primary depositional features, such as seam floor, seam roof, and in-seam channel or horizon behaviour. Various coal qualities, particularly ash distributions, may also be predicted with greater confidence if based on sound sedimentological interpretations (Cadle *et al.*, 1991).

Geophysical techniques may also be applied to further resolve features difficult to interpret from available drill data, or to provide more accurate down-the-hole measurements in areas of poor core-recovery. The benefits of down-the-hole geophysical logging for seam correlation in very thick, complex seams has also been mentioned. Refraction seismics may delineate coal-basin topography and stratigraphic continuity and structure, including that of sills and major faults. Magnetics (airborne) may be considered to delineate dykes, sill-edges and basement-lithologies. Basement lithology and structure may in turn control the coal-basement floor palaeotopography, and hence impact on coal seam distribution.

The prospect investigation stage generally ends when "sufficient" data, physical and chemical, has been collected to define the deposit in terms of an "indicated reserve". In terms of exploration, the reserve may actually be considered "proven", but additional more-detailed information may still be required from a mining point of view for mine planning purposes.

The concept of "sufficient" data will be addressed in Chapter 2 (Some Geostatistical Applications), while classical statistical distribution curves for the various data will also assist in this regard. In particular, the "completeness" of distribution curves for various values, such as for quality or thickness, may indicate the reliability of the data for estimation purposes. However, classical statistics will not indicate the degree of correlation between nearby samples. Geostatistics, on the other hand, provides information on the degree of correlation between nearby samples.

The prospect investigation stage concludes with the preparation of a comprehensive geological report summarising results and recommending specific future investigations. Such a report may form the basis for an overall feasibility report, or may be used for raising mine-development capital.

PRE-MINE INVESTIGATION

Pre-mine investigations follow the decision to actually "go-ahead" with a mining venture. Specific investigations may be carried out to assess some additional quality or structural variable, or to locate suitable sites for the mine shaft, the initial open-cast box-cut, administration buildings, workshops, plant and stockpiles. Site investigations may include rock engineering tests as well as detailed drilling to evaluate initial short-term mining conditions.

Various geophysical techniques may also be considered at this stage. These may include: borehole-to-borehole in-seam seismics, tomography, or radio imaging to identify discontinuities; ground resistivity or natural gamma to characterise open-cast overburden; and airborne magnetics to interpret dolerite structures (BPB, 1981; Bulletins of the Australian Coal Research Laboratories - ACRIL).

The pre-mine investigation stage terminates with the compilation of an in-depth feasibility and technical report, encompassing aspects of geology, environmental issues, mine and personnel planning, and financial and marketing-related considerations. Initial shaft-sinking or pit construction may also have begun during the latter stages of this period. From a geological veiwpoint, inconsistencies in the original feasibility report would have been addressed, preliminary mining conditions would have been assessed, and the reserves for the initial mining area would have been moved into the "proven reserve" category. In addition, attention would have been given to future grade control and reserve reconciliation needs, such as sampling needs and estimation accuracy requirements.

MINING

Geological input and investigations continue through the three main periods of a mine's life-cycle, namely : production build-up; mature production; and declining production. The geological input is aimed primarily at ensuring the optimum extraction and utilisation of the coal reserve.

During production build-up, geology is concerned with the reconciliation of mining data with pre-mine estimates, and the modification of models, mine planning, and procedures to suit these realities. In addition, geology must continue to prove-up reserves and assess the presence of geological difficulties such as faults, intrusions, other discontinuities, and safety-related factors such as pillar and mining-roof conditions (Ziolkowski, 1989).

Investigation techniques available to the mine geologist include the following: underground or highwall mapping; in-seam (longhole) and surface drilling (core and non-core); in-seam seismics and radio imaging; close-spaced channel or blast-hole sampling (including the use of down-the-hole geophysical probes); and production or control reports on adverse geological conditions encountered during mini-ng.

The wealth of information available from the above investigations will be "lost" if not translated into some usable form. In most cases this implies the preparation of plans and sections compatible with mine-planning and production-plan scales. Such up-to-date plans, combined with a computerised database of quantifiable data, forms the information base from which the mine geologist operates.

The above procedures are continued through the remainder of the mine's life, but should be continually assessed, and updated if necessary. The full-production period on a high-technology high production colliery requires quick and reliable evaluations and estimations from geology. Traditional estimation techniques may generally be adequate. However, geostatistical techniques may be readily applied to provide confidence limits for estimations and to improve the efficiency of sampling campaigns (see Chapter 2: Some Geostatistical Applications).

1.2.3 Significance for longwall mining

The advanced technology and high production associated with longwall mining ensures it is subject to many controlling factors. A typical longwall face of 200m length may consist of 3500 tons of equipment and cost some R50million (Ziolkowski, 1989). The cost of unplanned longwall stoppages and face moves, due to encountering unexpected geological difficulties, is immense. Clearly, the geological factors controlling longwall mining need to be suitably defined and predicted to ensure that longwall panels and blocks are correctly placed and planned. Some of the important geological factors have already been summarised (Table 1.4), while many of the references cited in this chapter include discussions specific to longwalling.

The interdependency of three of these factors is discussed below, namely: seam thickness: calorific value (C.V.); and seam elevation. The aim is to highlight the importance of these three factors for prediction and contamination purposes, and to provide some background for their application later in this study (Chapters 2 and 3).

SEAM THICKNESS AND CALORIFIC VALUE

Longwall equipment, in particular the hydaulicly-operated support shields, may be designed to operate up to an effective height of some 4.5m. However, their design includes a minimum height down to which the shields may be lowered. For example, a face may be designed to operate in a range from say 1.60m to 2.60m. In practice, the face may be found to be unproductive at the minimum height of 1.60m, and the minimum height for production purposes may be increased to say 1.80m. Should the seam in the given mine area actually vary from say 1.30m to 3.30m, then the range of the equipment may render a large proportion of the area inaccessible to that particular set of longwall equipment.

Seam thickness variations are generally controlled either by original seam floor palaeotopography and/or by the palaeo-erosion associated with the overlying sediments. The latter tend to result in the more drastic rates-of-change, including such features as "scour channels" or "washouts" in the coal seam. These features may vary in width from 0.5m to 4000m and in length from 5m to 40km, and may have eroded up to tens of meters of underlying sediments.

When seam thicknesses of less than the minimum height of the equipment are unexpectedly encountered in a panel, there are two choices, namely:

1. Mine roof and/or floor material, depending on their respective hardnesses and behaviour.
2. Stop and move the equipment around the thin-coal area.

The first choice damages equipment and morale, and contaminates the run-of-mine (ROM) coal, resulting in at least an increased ash content and abrasiveness, and a reduced C.V.. The second choice is also extremely costly, with between two and six weeks of production being lost. In addition, if this thin-coal area occurs near the end of a panel, the decision may actually be to abandon the coal in the remainder of the panel and move to a new panel. Such a decision can seriously affect mine planning, placing pressure on development sections to open up new areas, and reducing the *in situ* reserves.

For the above example, 0.10m of stone (with C.V. = 0MJ/kg, ash = 100%, RD = 2.35) added to 1.70m of coal (with C.V. = 24.70MJ/kg, ash = 22.0%, RD = 1.50), results in the following contamination effects:

	in situ	Contaminated
% Contamination (by mass)	0.0%	9.2%
Ash content	22.0%	29.2%
C.V.	24.70MJ/kg	22.43MJ/kg

The contamination calculation formulae are given in Appendix 1.1. These calculations assume that the *in situ* values and the added contamination are known with some degree of accuracy, a problem that is addressed in section 3.3 (Routine reporting of estimation errors).

The amount of contamination is not generally known, but is calculated (estimated) from the ROM coal quality (generally C.V.) and the predicted *in situ*, uncontaminated coal quality. Since ROM coal is generally sampled before stockpiling or sale, this provides a global contamination value for the whole mine, but does not differentiate between individual production sections. Since most of the longwall mines in South Africa supply steam coal to power stations, C.V. is a frequently-quoted and analysed parameter. A system for measuring, estimating and reporting longwall contamination is presented in Chapter 3 (Grade and Contamination Control for a Longwall Colliery).

Management must decide what amount of deliberate contamination can be tolerated by both the coal supply contracts and by the mining equipment. For example, would it be cost-effective and sensible for a longwall face to mine through a 100m-wide zone of thin coal, the equivalent of some one to two weeks of production (about 56 000 tons, including 9% contamination)?

If management decide that a 100m wide zone of thin coal is the maximum that can be handled by the system, then geological investigations should be able to identify, or predict, thin coal zones down to at least that scale. A 1000m x 1000m drilling grid, or possibly even a 500m x 500m grid, would apparently be insufficient for predicting such occurrences. The "optimum" drilling configuration to satisfy such requirements may be determined by applying geostatistical estimation techniques. Such techniques also allow the proportion of any given mining block falling below some cut-off value to be predicted, and thus the probability of encountering thin coal may at least be estimated (chapter 2: Some Geostatistical Applications).

SEAM ELEVATION

The seam floor elevation may not display rates-of-change as dramatic as those for seam thickness, depending on the deposit characteristics and the nature of the seam mined. In general, the seam may vary over three specific scales or ranges, namely:

1. The seam may display a regional dip over tens or hundreds of kilometers.
2. Super-imposed on this may be more local undulations, with wavelengths from 1km to 5km.
3. Smaller-scale "floor rolls", with wavelengths of between 2m and 100m may also occur.

Mine layouts are normally controlled by the 1km to 5km-scale features. Longwall faces and panels in particular are generally designed to allow water to drain into the goaf, and mining advances (or retreats) up-dip where possible. In addition, the main-gate (along which coal extracted, control equipment placed, and personnel travel) will also be positioned up-dip.

Knowledge of the smaller-scale features is more important for predicting short-term, within-panel conditions than for annual or longer-term estimates. This resolution is generally beyond geological drilling or pre-mine investigation control, and tends to be assessed only as mine development progresses. Geostatistical estimation techniques may however be used to simulate this within-panel behaviour, as will be discussed in Chapter 2 (Some Geostatistical Applications).

2. SOME GEOSTATISTICAL APPLICATIONS

2.1 INTRODUCTION

2.1.1 General

The use of geostatistics in the coal industry is constantly gaining support, and the results of geostatistical analyses are applied particularly towards improving reserve estimations. This is achieved by determining the best possible weighting for samples used in the estimation so as to produce the lowest possible error of estimation. The analyses are commonly conducted using historical data (already completed boreholes and/or channel samples) and applied to very large mining blocks, equivalent to annual production areas (Clark, 1991).

Geostatistics has not apparently been generally used to aid the selection of sample configurations, nor to provide some degree of confidence (error of estimation) on predictions for short-term production requirements. Geostatistics in fact provides the only estimation technique that quantifies the error associated with making such estimates.

In general, geologists have relied on their experience and large geological databases to emperically design "ideal" sample or drilling configurations. Alternatively, sampling may be conducted on some historically determined and unquestioned grid pattern, the reasoning for which has long since been forgotten. Both of these "techniques" may provide data that allows relatively accurate predictions to be made of, say, monthly production parameters.

Historically, reconciliation of predictions against run-of-mine results has generally been good, since any significant inconsistancies would have resulted in a revision of prediction techniques, including sample patterns and laboratory procedures. However, it is not unusual for unexplained "problems" to occur, particularly when trying to predict some additional parameter, or when reducing the size of the "block" (time-scale of production) being estimated. In addition, with the need for improved efficiency, can the geologist be certain an area is not being over-sampled, at the expense of time, effort and money? Could the sampling be reduced without significantly affecting the quality of results and predictions?

2.1.2 Objectives

This section highlights some applications of geostatistical techniques to the coal industry by attempting to resolve the following questions :

For an operating or planned mine:

1. What is the error associated with estimating mean values for different sized mining blocks (reflecting annual, monthly, weekly, or daily production) for different sampling configurations?
2. What sample configuration (boreholes and/or channels) will provide "adequate" results for practical estimation purposes at the lowest (possible) costs?
3. Given that a mining method has a minimum mining height, can the probability of a given block mean thickness being above cut-off be predicted?
4. Can the proportion of the above block expected to be below "cut-off" also be estimated? Similarly for other structural or quality parameters.
5. What are the advantages of such predictions?

For an exploration prospect:

1. How early in an exploration program can geostatistics be applied towards determining an optimum sample spacing and pattern?
2. What are the associated costs and benefits?

The above questions are discussed with respect to seam thickness (width), calorific value (C.V.)-specific energy or heating value-, and seam floor elevation. The principles may well apply to other parameters such as ash content, volatile content, sulphur, washability yields, etc.

2.1.3 Layout and data

This section does not attempt to provide a detailed account of geostatistical theories and procedures. Instead, a real set of data will be used to illustrate the potential of some geostatistical techniques for solving the questions set out above. Further, more rigorous treatment of the data could result in more refined models and results than those presented. The idea is rather to highlight the availability and advantages of the various techniques to all geologists, no matter their level of experience or expertise. A brief overview of geostatistical theory is presented simply to serve as a reminder of the background for what follows in the section.

The database used consists of actual borehole and channel sample results from an area of the Number 4 Seam of the Highveld Coalfield. The data consists of a total of 227 full-seam borehole results and 171 full-seam channel sample results. The channel samples provide control at closer spacings than is generally available from the boreholes. The data has been adjusted to protect the confidentiality of the original data.

2.1.4 Computer programs used

The data was sorted and analysed for classical statistical results and initial semivariogram behaviour using spreadsheets within "Quattro-Pro" (version 3.00) and a program called "XYZ". The latter is a public-domain geostatistical program written by Lucien Verrezen of GENMIN (1990). The bulk of the geostatistical analyses were completed using "GEO-EAS", short for "Geostatistical Environmental Assessment Software" (Englund and Sparks, 1988), while kriging estimations were conducted with "KRIGTEST/PLAYKRIG" (supplied by the Ore Evaluation Department of A.A.C.).

2.2 BACKGROUND THEORY

The following discussion is based on publications by Clark
(1979 a,b,c) and Brooker (1979).

2.2.1 Semivariogram construction and modelling

The semivariogram graph (or formula) describes the difference in value between pairs of samples with respect to their relative spatial arrangement. In fact, geostatistics assumes the difference between sample values to be purely a function of the direction and distance between the respective samples (the "intrinsic hypothesis" of geostatistics). This permits the "structure" of the data to be described mathematically, and allows for a wider range of applications than are available from classical statistical distribution parameters.

The theoretical (or "true") semivariogram (γ) is defined as one half of the mean squared differences in sample values between points separated by a distance "h". This may be represented in mathematical notation as follows:

The semivariogram result at a distance h is given by:

$$\gamma(h) = \left[\frac{SUM(g_i - g_{i+1})^2}{n}\right] x \frac{1}{2}$$

where g_i = value of the sample at point "i".
n = total number of sample pairs separated by a distance h.

For a large number of sample pairs the semivariogram result is equivalent to the variance of the given samples, which would be calculated as follows:

$$\text{Variance, } v^2 = \frac{SUM(g_i - \bar{g})^2}{n}$$

where $\bar{g}$ = mean value of all samples at a distance h apart.

In practice only a limited number of samples are available and this allows only an estimation of the mean difference in grade to be made. Consequently, an experimental ("estimated") semivariogram (γ^*) is constructed by plotting γ^* versus h for the available samples. The various aspects of a semivariogram curve for the spherical model are illustrated in Figure 2.1.

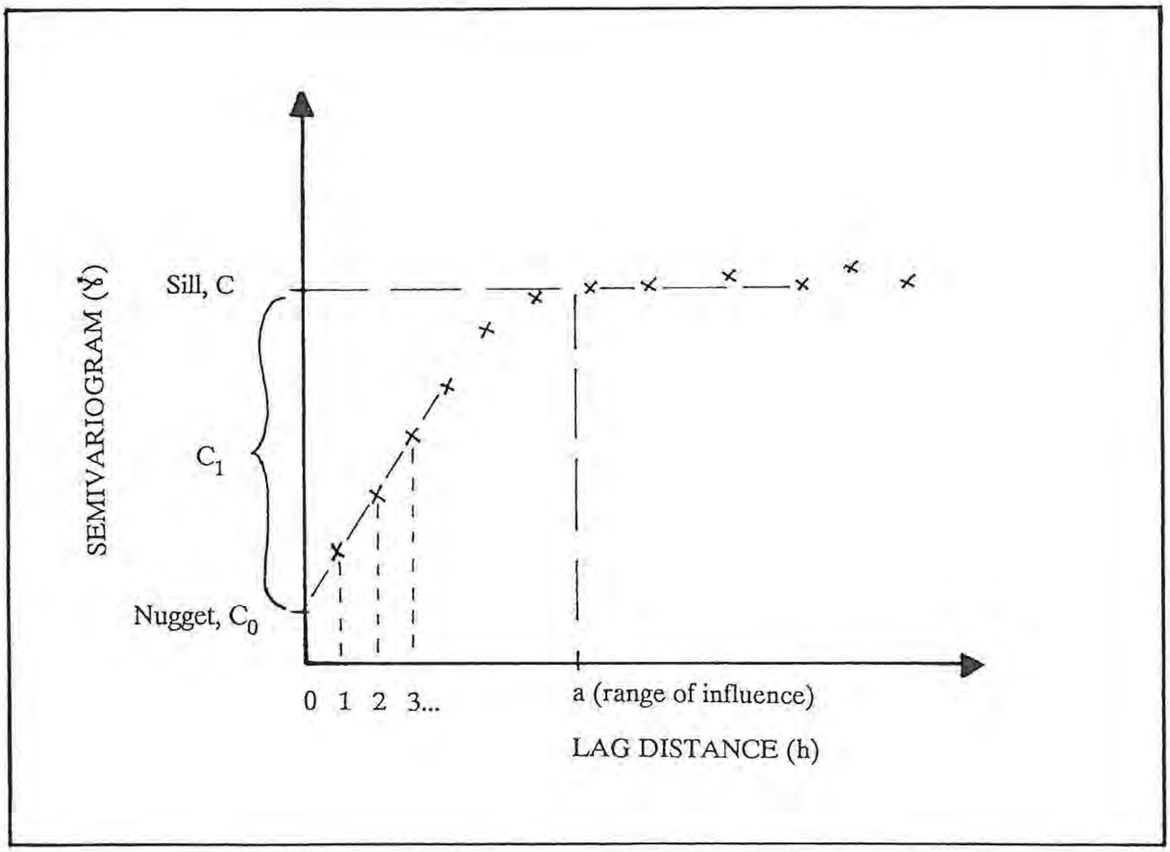


Figure 2.1 *Semivariogram features for the spherical model.*

The maximum restriction on distances between points used in calculating γ^* is generally between one quarter and one half of the total extent of sample information. The number of data pairs used to calculate γ^* generally decreases as the distance (h) between pairs increases. However, there is often a lack of close-spaced data in coal, and the number of pairs tends to increase and then decrease as h increases. The greater the number of pairs used, the more reliable the semivariogram (γ^*) point calculated.

A line drawn through the first two points of the semivariogram may not always pass through the origin (zero), but may intersect the γ^* axis at some positive value (for point samples). By definition, the difference in values between a point and itself must be zero if analytical error is ignored. If a difference in values exists for samples taken very close together, then this is known as the nugget effect, and it reflects the naturally occurring randomness of the data. The identification and estimation of the nugget effect depends on the sample size and the sampling interval - a wider sampling interval may either over-estimate the nugget effect (increased unpredictability) or not identify it at all.

Sample support (size, shape, mass, and analytical method) also affects the semivariogram. In general, any sample that differs markedly from a "point"-sample results in smoothed or averaged results (sample variance lower than actual variance), and variances between samples may decrease, with even "negative" nugget effects being implied. Depending on the deposit type, a sample size of a few kilograms or a limited length may be able to be treated as a point value. Alternatively, the point-sample semivariogram model may be estimated from a sample-defined semivariogram model. Any semivariogram should be based on samples with the same support.

Assuming samples are available on a regular grid for an essentially two-dimensional deposit, say a narrow coal seam, then semivariograms may be constructed in any direction within the deposit plane. Such omnidirectional semivariograms provide valuable information regarding continuity across a deposit. For a reasonably large deposit, semivariograms should be constructed across different parts or areas of the deposit to test for the presence of geostatistically different domains. In fact, the boundaries between domains may be readily identified by dividing the area into a number of equidimensional blocks, and then comparing the individual block means and variances for regularised data (see below) across the deposit.

Sampling is rarely conducted on a perfect grid and certain approximations are generally made to assist semivariogram construction. The data may be "forced" onto a grid or regularised before the semivariogram calculation. In such a case, it is important to check that the regularised data maintains the statistical distribution properties of the original data. Alternatively, most programs allow search windows to be specified for both direction, width and distance, thus effectively regularising the data. For example, a search direction may be 25^0 $^+$-10^0, with a maximum width for this window of $^+$-50m, while the distance range may be specified as say 50m $^+$-20m. Both methods allow sufficient pairs to be located from the original data for the semivariogram calculation.

Once a semivariogram has been established for a deposit, it must be modelled with an appropriate model type. Since it is hoped the semivariogram defines a sill, the most commonly used models are the exponential and spherical models. Both models assume there is some distance (range of influence "a") beyond which samples are unrelated and independent of one another. Identification of the model parameters is essential for completing kriging and other estimation procedures.

The principles and models used to construct a semivariogram are sufficiently simple that a spreadsheet may be set up within say Lotus123 or Quattro-Pro, to calculate at least two-directional experimental semivariograms from regularised data (Mallinson, 1991). Such spreadsheets may be particularly useful for modelling the semivariogram results. Similarly, personalised semivariogram calculation programs may be written or acquired. While this may seem unnecessary given the wide range of commercially available packages, such methods have the advantage of being robust and user-definable, as well as requiring little in the way of computer hardware. Such a set-ups may also be useful to obtain a "feel" for the data before turning to the more extensive, "black-box"-type programs. The major limitations of the personalised programs include data volume limitations and possibly the need to use regularly-spaced data.

2.2.2 Practical semivariogram considerations

Some difficulties in matching models to good, "stable" experimental semivariograms include the following :

1. A regional trend or zonation (non-stationary situation) to the data (determines the type of kriging to apply).
2. Anisotropy (zonal - variance differs with direction; geometric - range of influence differs with direction).
3. Identification of the nugget effect.
4. Differing scales of correlation over different but specific distances (for example, seam thickness and elevation - see section 1.2.3), possibly with varying statistical structures, requiring combined or stacked models.

5. Limited number of pairs used (experimental semivariogram confidence).
6. Sample spacing and support (size, shape, mass, analytical procedure) - deviation from a point-sample.

Anisotropy, particularly geometric anisotropy, may be inferred from the original data with knowledge of the deposit-type, and may be quantified by directional semivariogram construction. Trends on the other hand, may only affect the semivariogram beyond the local range of influence that is of interest. This is not always the case for coal deposits, and more sophisticated analyses are then required to treat the data.

With regard to sample support, this should be the same for all samples used in constructing an experimental semivariogram. The actual point semivariogram needed for kriging or grade-tonnage calculation may then be inferred, and the point variance (dispersion variance) will then allow variance estimation for mining blocks of any geometry. In addition, the semivariogram between samples with different supports, such as boreholes and mining blocks, may also be derived. This may be achieved by applying standardised tables and curves, or by using specially designed computor programs.

2.2.3 Geostatistical estimation techniques

Geostatistical estimation techniques (kriging) allow the accuracy of mining block estimates to be quantified. This is achieved by applying the relevant semivariogram model to determine the variance of the estimation error distribution. In fact, geostatistical estimation is the only estimation technique that takes the actual variability of the ore-body into account. The kriging procedure also allows a set of sample weighting coefficients to be determined for a given block and sample configuration, so as to minimise the block estimation variance ("best linear unbaised estimator"). These weighting coefficients take account of the variability of the deposit, as defined by the semivariogram model and expressed as the sill, nugget, range and anisotropy parameters.

The main requirements for geostatistical estimation theory are the following:

1. The blocks to be estimated must lie within the same area (geological domain) for which the semivariogram has been calculated.
2. Sufficient samples (data pairs) are used so that the experimental semivariogram reliably reflects the variability in the area.
3. The area consists of homogenous mineralisation with no distinct trends in the data (stationary condition).

Should a trend be present in the data at the scale of the blocks to be estimated (non-stationary situation), then special kriging techniques must be applied. If the data are not on a regular grid then an approximation technique (random kriging) must also be applied, or the data must be regularised. Any changes in sample spacing or semivariogram structure in different areas of a deposit will affect the estimation variance.

It is also possible to estimate between-block variance ($\sigma^2_{v/V}$), block mean estimation variance (kriging variance), and sample weighting coefficients for any given block and sample configuration. This may be done without knowing the "actual" mean value for the block by using "ordinary kriging".

Knowing the between-block variance and the point-sample variance ($\sigma^2_{o/V}$), allows the within-block variance ($\sigma^2_{o/v}$) to be calculated from the variance additivity relationship:

$$\sigma^2_{o/V} = \sigma^2_{v/V} + \sigma^2_{o/v}$$

The point-sample variance is constant for a given geological domain, and is equivalent to the sill (C) of the model semivariogram for that area. In general, the point-variance is quoted as C_1 (C - C_0) with the nugget (C_0) quoted separately.

The between-block variance is constant for a given block anywhere within the given geological domain, and it tends towards the sample variance as the size of the block reduces to sample-size dimensions.

The within-block variance describes the distribution of sample values within the block. This value obviously tends towards zero (or possible the nugget value, if present) as the block size approaches sample-size dimensions.

The errors (or kriging variance) associated with estimating the mean value for a block (actual mean - estimated mean) are generally normally distributed, and thus may be readily interpreted in terms of confidence limits for the estimation.

For example, if the kriging variance (v^2) for the estimation (e^*) of a block mean ($\bar{e}$) is 0.50 (units)^2, then the kriging standard deviation or error (v) is 0.71 units. The actual mean value for the block may then be interpreted as falling within specific limits for a given confidence limit, as follows:

90% confidence:

$$[\ (e^{*} - 1.64x0.71) < \bar{e} < (e^{*} + 1.64x0.71)\ ]$$

95% confidence:

$$[\ (e^{*} - 1.96x0.71) < \bar{e} < (e^{*} + 1.96x0.71)\ ]$$

In classical statistics, the above confidence limits would be calculated using "t-tables" if less than about 25 samples were used for the estimation. The kriging variance incorporates both the number of samples used in making an estimate as well as the samples' spatial arrangement with respect to the block being estimated, and therefore no further "t-table" correction is needed.

The above technique permits the estimation error associated with different sample configurations to be quantified. Such a calculation then provides an objective criterion for deciding on "suitable" sampling patterns.

In addition, the probability of a block average exceeding a certain cut-off value may be estimated from the kriging variance and the estimated mean for that block. Going a step further, knowing the within-block variance and the estimated block mean (and its variance), allows the proportion of a given block exceeding a cut-off value to be estimated. For example, if there is a 95% probability (certainty) of the seam thickness exceeding a certain minimum value throughout a given longwall block, then management will commit the longwall equipment to that block with a greater degree of confidence than if the probability were only 70%.

2.2.4 Conclusion

The semivariogram is the only way of determining whether geostatistics or classical statistics should be applied to a given deposit. The experimental semivariogram achieves this by defining the extent of correlation between data points. The experimental semivariogram should, therefore, be as much a part of ore reserve estimation and evaluation as the construction of data histograms, and geological maps and sections.

Obtaining an experimental semivariogram at an early stage of an exploration program can greatly aid the design of future sampling and drilling procedures, with the benefits of improved estimation and better applied financial resources. An initial sampling program to define the semivariogram need not be "blindly" determined, but could be based on results from other similar deposits. Obviously, any such early semivarigram would be based on very few sample pairs and would have to be checked and updated as more information became available. The same is true for a developing mine - the validity of existing semivariogram models should be checked regularly, and updated if necessary.

A well-defined semivariogram forms the basis for geostatistical estimation, a crucial technique for maximising exploration and grade control resources by optimising sampling campaigns and realistically reporting results.

2.3 ANALYTICAL PROCEDURE

The following procedure is typical of any geostatistical analysis. In fact, the first step (2.3.1) would already be conducted for any routine evaluation of geological data.

2.3.1 Data distribution

Describe the data used, including the variables of interest, which constitute the database with regard to the following: origin; data collection; analytical procedures; and obvious sources of error.

Produce a plan or plot of sample localities, and contour plots (hand or computer generated) of the applicable variables.

Complete a classical statistical analysis of the respective variables, including: sample variable frequency curves; distribution types; and estimates of sample parameters such as the mean, range, and variance.

Separate the area into statistically (and geologically) specific and homogenous domains and repeat classical statistical analysis for each domain.

2.3.2 Semivariogram construction and modelling (statistical structural analysis)

Construct experimental semivariograms (two-dimensional for coal), bearing the following points in mind: introducing cut-offs may improve the semivariogram, but must be reintroduced for later estimation purposes; varying lag spacing and search ranges can improve the semivariogram appearance; the presence of anisotropy and trends must be tested for by repeating the construction in at least four directions, say 0°, 45°, 90° and 135°.

The final stage is to actually fit models to the experimental semivariograms by selecting a model type and the model parameters, namely the sill, nugget and range-of-influence. Multiple or stacked models may be required to match the experimental semivariogram, and this may complicate later interpretation and estimation procedures, particularly with regard to ranges of influence.

2.3.3 Cross-validation

The validity of the semivariogram model may be checked or verified by using the model to estimate the values at specific sample points (point-kriging cross-validation or "jack-knifing"). The model parameters (nugget, sill, range, anisotropy orientation) are then varied until the following criteria are satisfied:

1. Good graphical fit to experimental data points.
2. Mean ratio of estimated to kriging variance (Z-score) close to "0", with a standard deviation close to "1" (say 1 ± 0.05).
3. Mean error of estimation close to "0", and a minimised mean kriging standard deviation (variable - estimate).

2.3.4 Estimation

Kriging estimation of block values may then be conducted with or without actual sample values. Estimates for different sized blocks and various sample configurations may be produced together with kriging variances (variance of estimated block mean from the expected or actual mean value). These results may then be analysed in terms of confidence limits together with the simulation of various practical scenarios. This may aid decision-making, while kriging techniques may also be applied to improve contouring.

2.4 PRACTICAL EXAMPLE

2.4.1 Data distribution

DATA

Both borehole and channel sample position plans are shown in Figure 2.2. The parameters available for these samples were: X-, Y- and Z-co-ordinates; seam thickness; and air-dry calorific value (C.V.).

No problems were originally foreseen in combining borehole and channel results for seam thickness and seam floor elevation analysis, since these parameters are generally measured as "point-values". However, the channel sample elevation values are actually "average" values, inferred from the nearest surveyed points (pegs), and as such may represent a far larger than expected sample support. Similar problems may occur for borehole elevations if the collar elevations have been inferred from topographic maps and not actually surveyed.

In the case of calorific value (C.V.), the impact of different diameter cores and channel widths on the semivariogram was uncertain. The sample mass collected over the full seam thickness is generally similar, and it was believed a channel width of some 300mm would not produce significantly different C.V. results to those from borehole core diameters of between 45mm and 60mm. This assumption could be easily tested in the field if necessary.

Preliminary statistical analyses of channel and borehole results showed similar classical statistical distributions (see below) for each of the three variables (C.V., thickness and elevation) from each data set. Preliminary experimental semivariograms also appeared fair.

Combining borehole and channel sample results for C.V. resulted in a large variation at short lag distances (sample intervals). Since the channel and borehole samples were analysed at different laboratories, this variation was concluded to be due to analytical differences. However, the effect of different support sizes could not be ruled out without being tested.

In the case of parameters such as C.V., it may be more correct to analyse the cumulative C.V. (C.V. x thickness) together with the thickness, and to then predict block qualities from the two sets of results. This is possibly more important in the case of log-normal quality distributions, as for example on the gold fields.

For the purpose of this illustration, it was decided to continue the analysis using only the channel sample data for all three variables. The channel data also include more close-spaced samples and the data is generally on more of a grid than are the borehole data. The channel data is presented in Appendix 2.1.

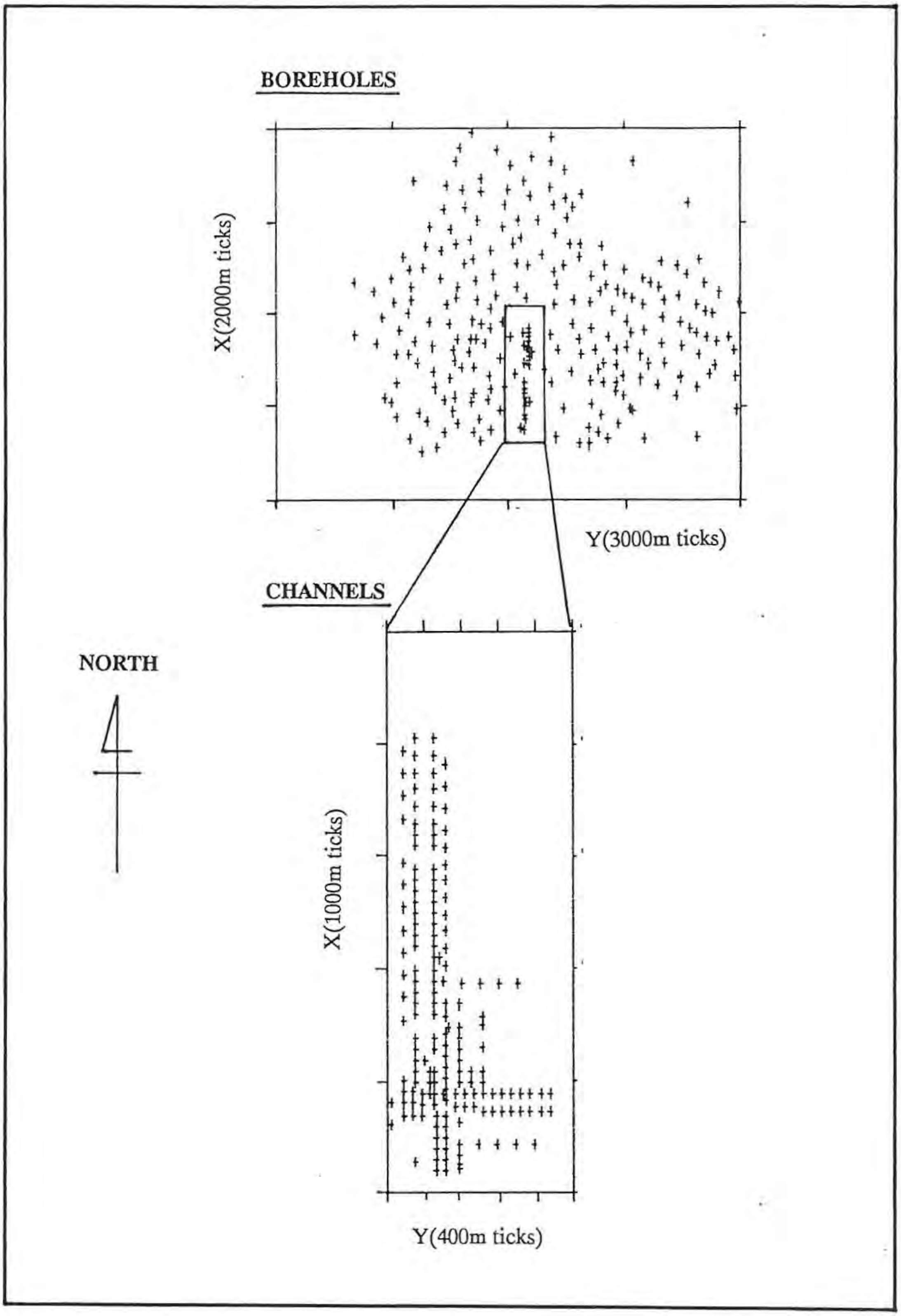


Figure 2.2 *Sample position plots for borehole and channel data.*

CLASSICAL STATISTICAL ANALYSIS
(Using "Quattro-Pro" and "XYZ")

The histograms and cumulative frequency curves illustrating the omnidirectional statistical distributions for C.V., thickness and elevation are shown in Figure 2.3 (a,b,c), together with summary statistics.

The larger sample variances associated with the borehole data may be explained by the larger area sampled by the boreholes (7km x 10km), compared to that of the channels (2km x 4km). The almost bimodal trend to the distribution of seam floor elevation for the borehole data indicates the presence of at least two distinct structural domains with respect to elevation. This is known to be the case, with a roughly 3km x 3km area of low-lying elevations to the east of the channel sample area.

Besides the occurrence of a few outliers, the distribution of channel sample data for all three variables appears to approximate the normal distribution.

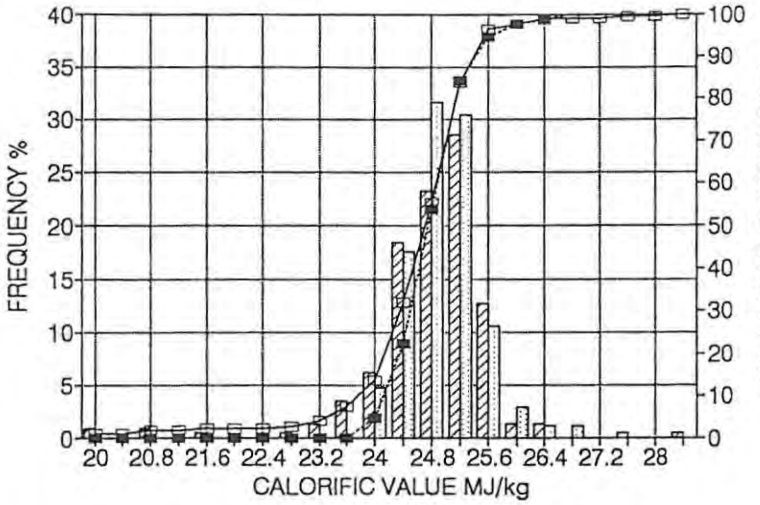
2.4.2 Semivariogram analysis
(Using "GEO-EAS")

CONSTRUCTION

The relatively thin seam allows the deposit to be treated as a two-dimensional body.

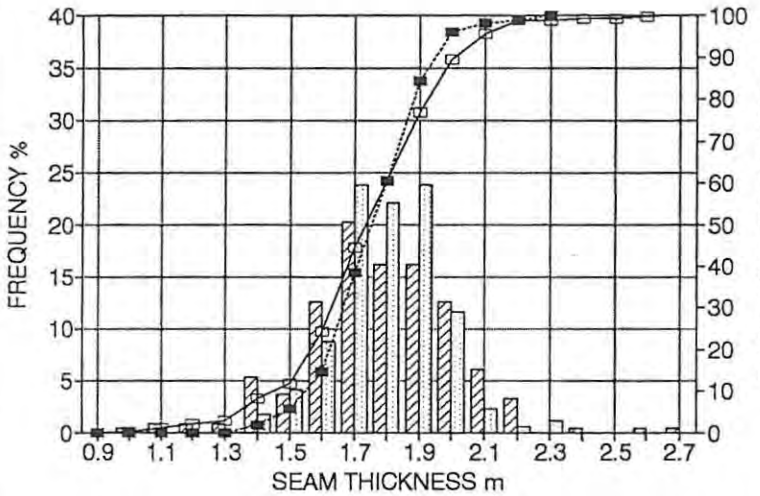
Restricting the maximum search distance to about 1500m ensured that the limitations of GEO-EAS were not exceeded (for example, maximum number of pairs allowed = 16384), even when combined borehole and channel data were analysed. The limited amount of close-spaced data presented some problems for confidently identifying the nugget effect, and generally only two or three points were available within the range of influence.

Experimental and modelled semivariograms for the selected orientations (omnidirectional, 0^0, 45^0, 90^0, and 135^0) are shown in Figure 2.4 (C.V.), Figure 2.5 (thickness) and Figure 2.6 (elevation). The modelling is discussed in more detail below, while the full semivariogram and pair results are given in Appendix 2.2.



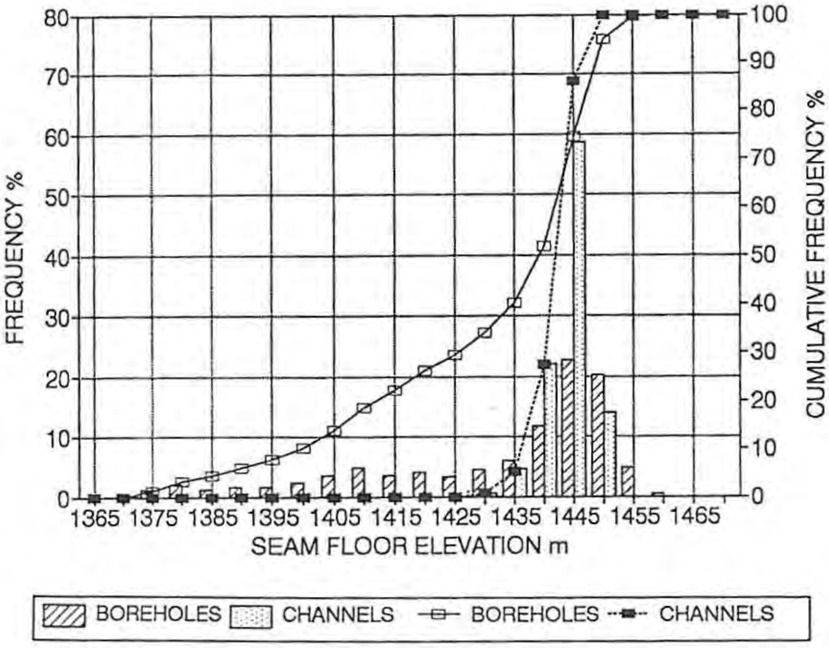
(a) AIR DRY CALORIFIC VALUE

		BOREHOLES	CHANNELS
MEAN	MJ/kg	24.59	24.76
MAX.	MJ/kg	28.13	26.73
MIN.	MJ/kg	17.06	23.61
VAR.	$(MJ/kg)^2$	0.982	0.258
"n"		228	171



(b) SEAM THICKNESS

		BOREHOLES	CHANNELS
MEAN	m	1.74	1.76
MAX.	m	2.63	2.22
MIN.	m	0.93	1.37
VAR.	m^2	0.0543	0.0239
"n"		259	172



(c) SEAM FLOOR ELEVATION

		BOREHOLES	CHANNELS
MEAN	m	1430.58	1441.56
MAX.	m	1456.90	1474.31
MIN.	m	1369.91	1426.15
VAR.	m^2	411.66	12.963
"n"		259	172

Figure 2.3 *Classical statistical distributions and summary statistics for C.V. (a), thickness (b), and elevation (c).*

Note, the above orientations are geographically correct, with bearings or azimuths quoted clockwise from 0^0 (0^0= north, 45^0= northeast, etc.). This corrects the GEO-EAS orientation system which works according to the system: 0^0 = east, 45^0 = northeast, 90^0 = north, and so on, with bearings measured anticlockwise from 0^0 (east).

MODELLING

The individual models and model parameters are shown and listed in Figure 2.4 (C.V.), Figure 2.5 (thickness) and Figure 2.6 (elevation). These models are discussed below and summarised, together with the effects of anisotropy, in Figure 2.7 (C.V.), Figure 2.8 (thickness), and Figure 2.9 (elevation).

CALORIFIC VALUE:

Spherical models were initially fitted to the semivariograms but this required the use of multiple sills and ranges. Single exponential models appeared to best fit the directional experimental semivariograms, with the following range of results:

Nugget (C_0) = 0.01 $(MJ/kg)^2$
Sill (C_1+C_0) = 0.195 to 0.230 $(MJ/kg)^2$
Range (a) = 500 to 550m

The "final" overall model (prior to cross-validation) was interpreted from these results as exponential and isotropic, with the following parameters (see also Figure 2.7):

Nugget (C_0) = 0.01 $(MJ/kg)^2$
Sill (C_1+C_0) = 0.22 $(MJ/kg)^2$
Range (a) = 500m

SEAM THICKNESS

The spherical model proved adequate for modelling the directional semivariograms, with the following range of results:

Nugget (C_0) = 0.005 to 0.008 m^2
Sill (C_1+C_0) = 0.022 to 0.024 m^2
Range (a) = 230 to 650 m^2

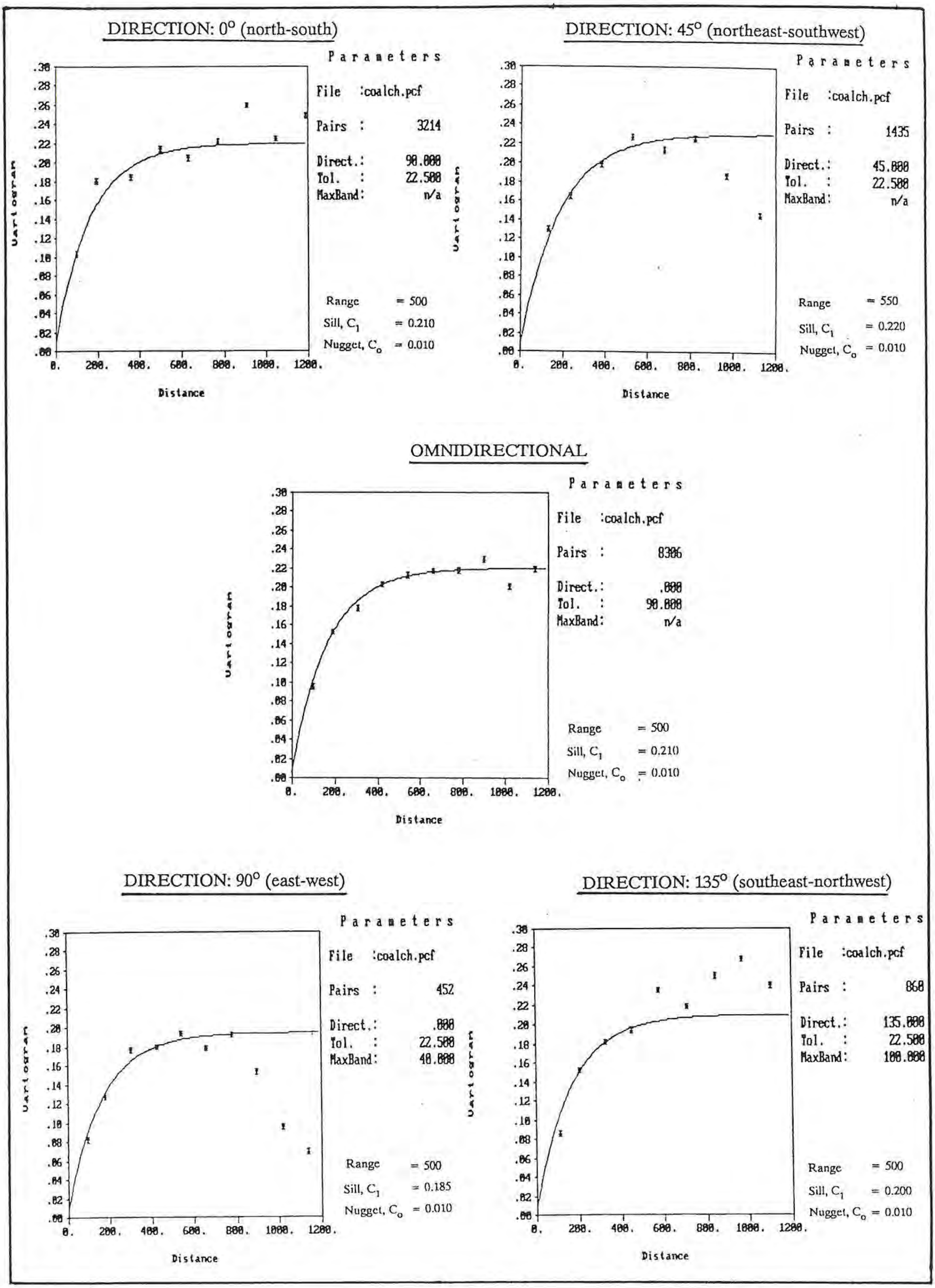


Figure 2.4 *Experimental and modelled semivariograms (exponential model) for calorific value distribution.*

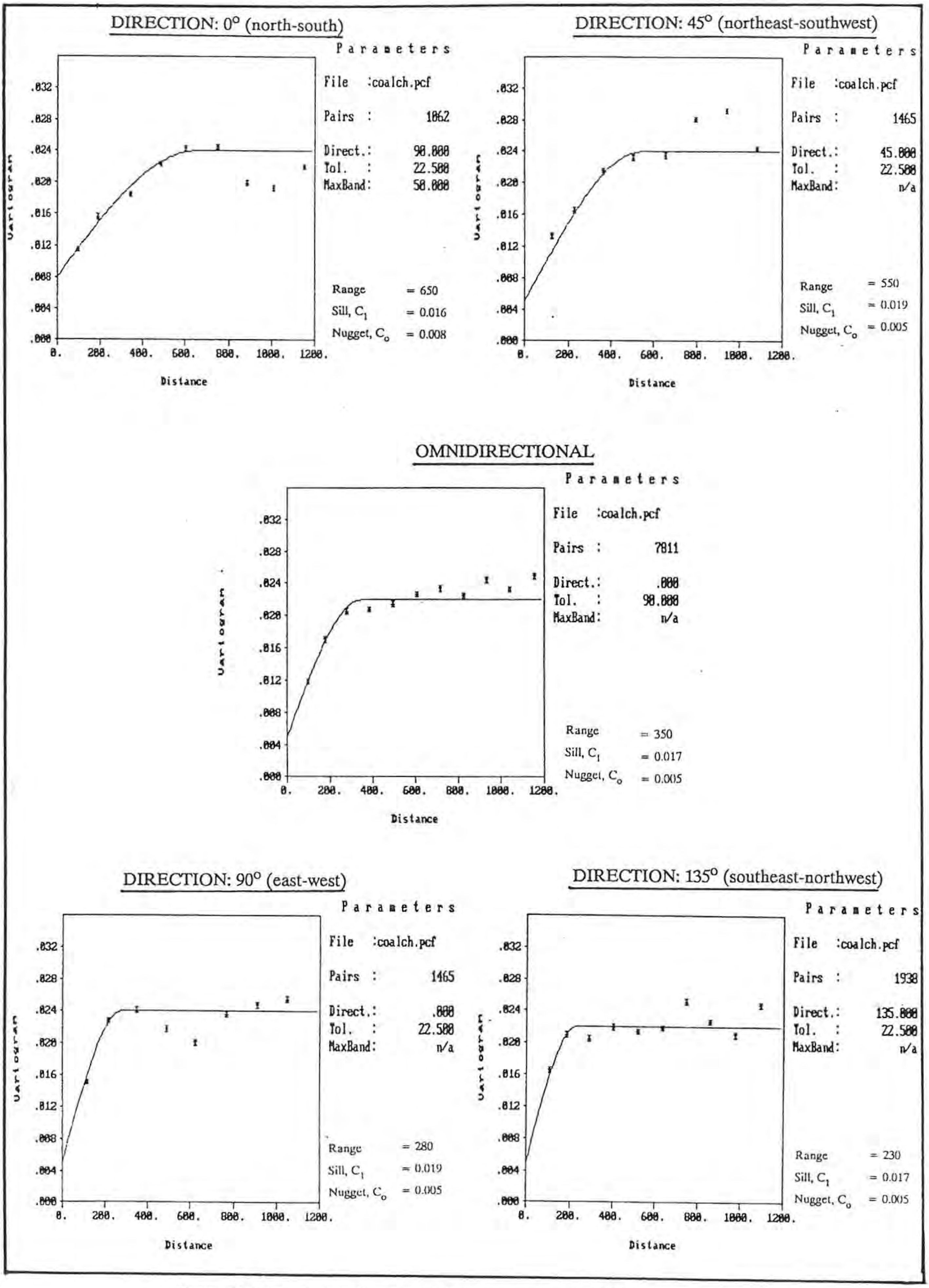


Figure 2.5 *Experimental and modelled semivariograms (spherical model) for seam thickness distribution.*

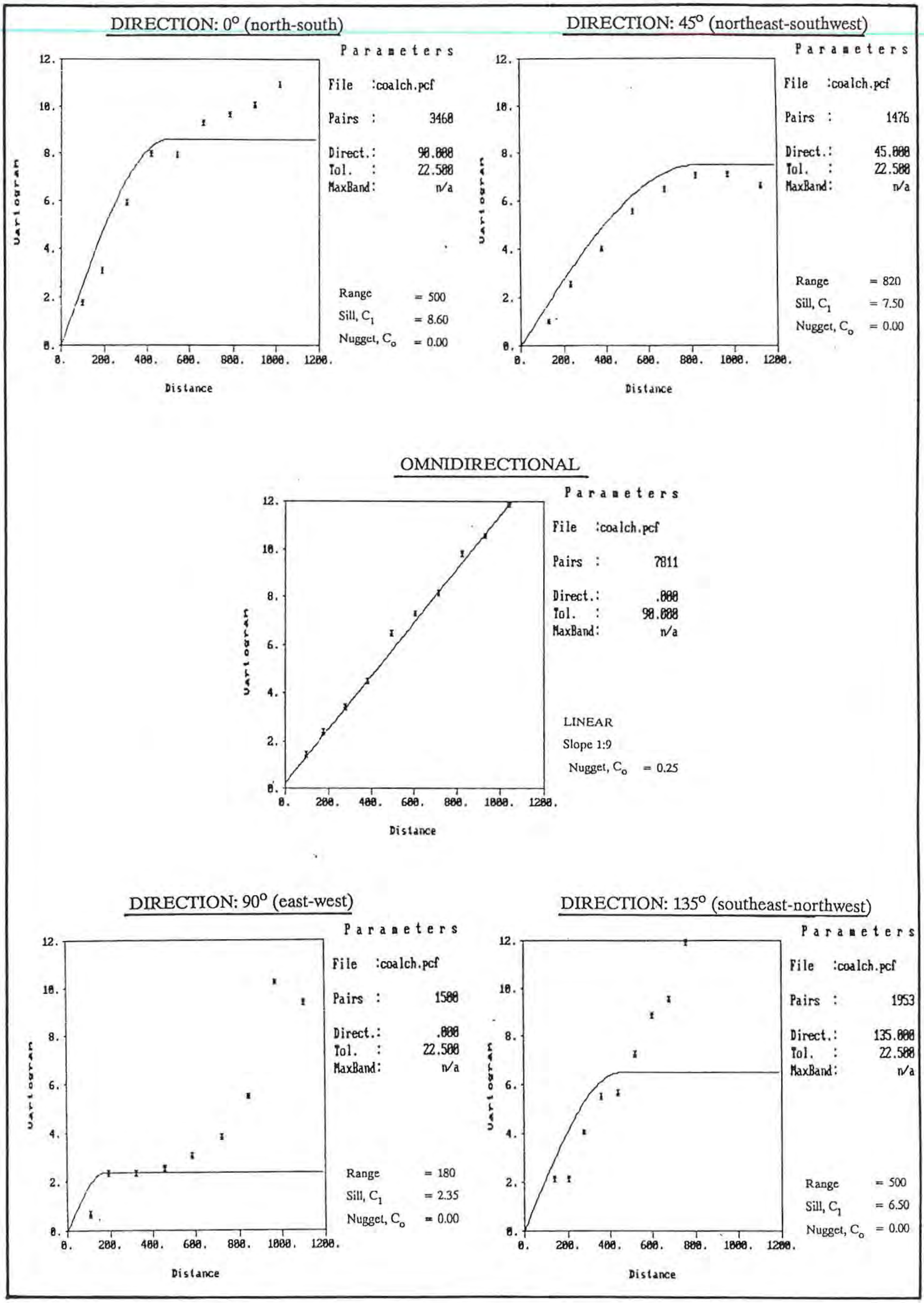


Figure 2.6 *Experimental and modelled semivariograms (spherical and linear models) for seam elevation distribution.*

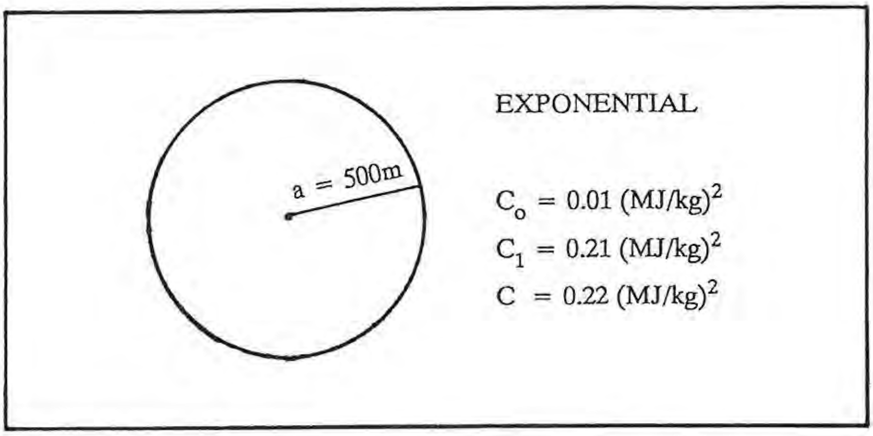


Figure 2.7 *Geostatistical exponential model for calorific value distribution.*

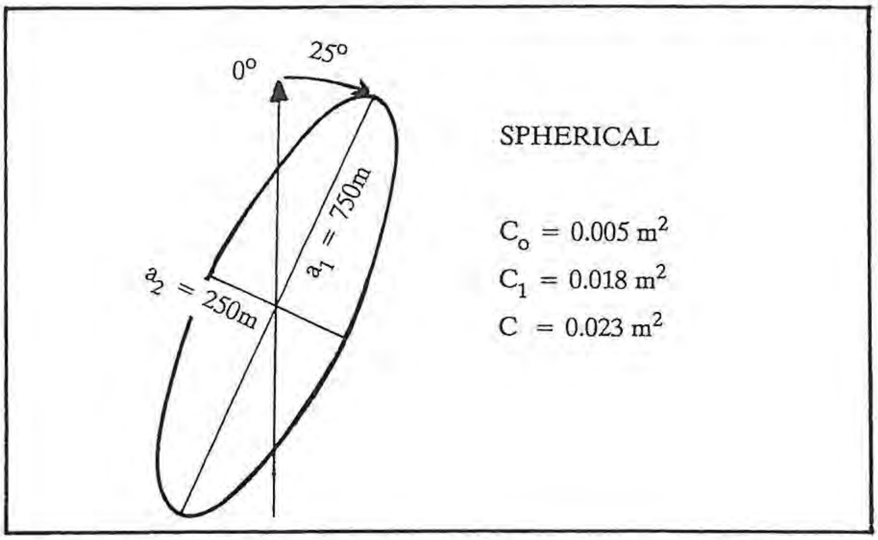


Figure 2.8 *Geostatistical spherical model for seam thickness distribution.*

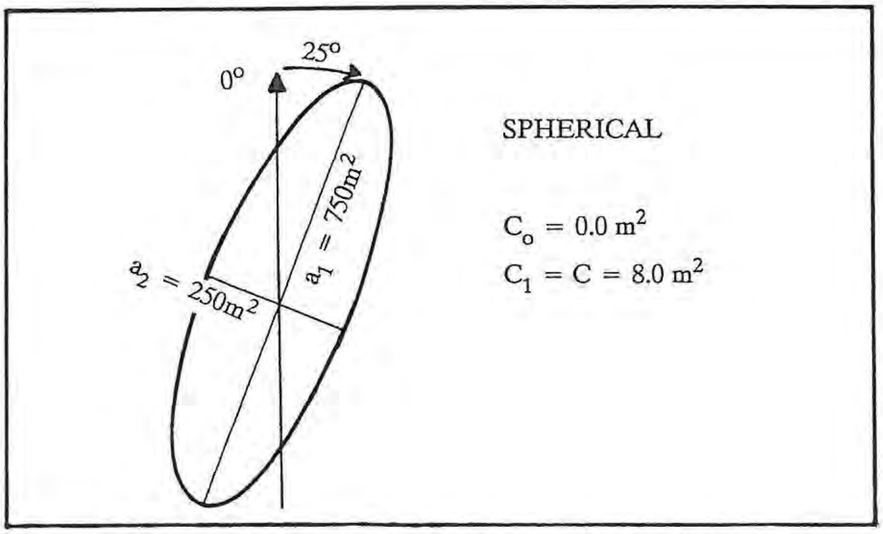


Figure 2.9 *Geostatistical spherical model for seam elevation distribution.*

From these results, the "final" model (prior to cross-validation) was interpreted as spherical and geometrically anisotropic, with the following parameters (see also Figure 2.8):

Nugget (C_0) $= 0.005\ m^2$
Sill ($C_1 + C_0$) $= 0.023\ m^2$
Max. range (a_1) = 750 m (azimuth = 25^o)
Min. range (a_2) = 250 m (azimuth = 115^0)

SEAM FLOOR ELEVATION

The apparently linear models for the experimental omnidirectional and 135^0 semivariograms probably result from the interaction of anisotropic conditions and the presence of a trend. Without applying trend analysis, a sill can be discerned from most of the directional semivariograms at distances less than that affected by the trend (less than about 900m).

The exception is the 135^0 orientated semivariogram which displays a linear behaviour. There is in fact a deep trough to the south and southeast of the channel area, and the linear model may indicate the increasing dip associated with this trough. For the purposes of this illustration, however, the 135^0-orientated semivariogram model is accepted as shown (spherical model).

The following range of results were obtained for the spherical models fitted to the directional semivariograms:

Nugget (C_0) $= 0.0\ m^2$
Sill ($C_1 + C_0$) = 2.4 to 8.6 m^2
Range (a) = 180 to 820 m^2

The "negative" nugget effect in some of the experimental semivariograms probably results from the measurement method for the elevations, namely their estimation (averaging) from the nearest survey pegs.

While not ideal, the model prior to cross-validation was chosen as spherical and geometrically anisotropic for the purposes of illustration, with the following parameters (see also Figure 2.9):

Nugget (C_0) $= 0.0\ m^2$
Sill ($C_1 + C_0$) $= 8.0\ m^2$
Max. range (a_1) = 820 m (azimuth = 30^0)
Min. range (a_2) = 180 m (azimuth = 120^0)

This model was later modified, with the support of cross-validation, to match the ranges and azimuth of the thickness model, namely a_1 = 750m (azimuth = 25^0) and a_2 = 250m.

CROSS-VALIDATION

Cross-validation checks the validity or correctness a chosen geostatistical model. Each model should ideally undergo an iterative process of cross-validation and parameter adjustment, until the model yields the "best" results with regard to the following cross-validation checks:

1. Goodness of graphical fit.
2. Mean error of estimation (variable-estimate) close to "0".
3. Mean kriging standard deviation minimised.
4. Mean Z-score of "0" and standard deviation close to "1".

The cross-validation results for the various models are given in Table 2.1. No further refinements were made to the given geostatistical models for the purpose of this illustration.

Table 2.1 *Cross-validation results.*

PARAMETER	C.V. MODEL	THICKNESS MODEL	ELEVATION MODELS 820mx180m ranges	750mx250m ranges
MEAN ERROR (standard dev.)	-0.002 (0.415)	0.000 (0.133)	-0.010 (1.715)	0.048 (1.723)
MEAN KRIGING STANDARD DEV. (standard dev.)	0.356 (0.045)	0.120 (0.011)	1.892 (0.317)	1.306 (0.259)
MEAN Z-SCORE (standard dev.)	-0.001 (1.165)	-0.001 (1.150)	-0.008 (1.350)	0.011 (2.381)

2.4.3 Discussion and early application of model results

RESULTS

Geostatistical structural models for C.V., thickness and elevation have been interpreted from directional experimental semivariograms constructed from channel sample data. Borehole data were not included in an attempt to reduce the effect of "analytical" errors from complicating the semivariograms, particularly in the case of C.V..

The channel sample data covers a smaller area than that of the boreholes, with corresponding lower total sample variances, for example 0.024 m^2 compared to 0,054m^2 for seam thickness. In addition, the experimental semivariograms have only been constructed (reliably according to the number of sample pairs) up to about 1200m, and thus ignore any trends or statistical structures beyond this range. For this reason, the model semivariogram sill may not correspond to the total sample variance, for example 0.22 $(MJ/kg)^2$ compared to 0.25 $(MJ/kg)^2$ for C.V..

While the channel-based semivariograms may have a lower sill (total variance) than for the borehole-based semivariograms, the ranges of influence are expected to be similar. This was confirmed by constructing a borehole-based experimental semivariogram for thickness. The model parameters for the latter were:

Nugget (C_0) = 0.005 m^2
Sill ($C_1 + C_0$) = 0.035 m^2
Max. range (a_1) = 700 m
Min. range (a_2) = 280 m

The lower variance (and semivariogram sill) for channel data compared to borehole data has an additional implication. Any estimation using the channel-based models will result in lower estimation errors than if a borehole-based model were used. However, the channel-based model may be applied with some confidence in the vacinity of the channel-sampled area, although it may result in an underestimation of errors if applied elsewhere in the reserve area. Similarly, the larger borehole area may well consist of a number of distinct statistical domains with lower sills than that of the "global" model. This could be tested for by dividing the larger area into a number of equidimensional blocks, and then comparing the means and variances of the individual blocks (with regularised data).

Given different sills but similar ranges of influence for the borehole and channel-based models, the channel-based models may still be used to illustrate at least the relative improvement in estimation accuracy associated with, say, different sampling configurations.

A restricted number of close-spaced data points limits accurate identification of the nugget effect. This problem may be reduced if the nugget is known for another similar deposit. The nugget effect is very important for determining estimation variances for point-estimates, but is of less importance when estimating block mean values (David, 1977). This is particularly valid if the nugget effect is small compared to the sill, which is the case in these examples.

APPLICATION

The geostatistical structural models, in particular the ranges of influence, nugget effect and directions of anisotropy, are of utmost importance when designing future sampling campaigns, or when estimating point or mining-block values. Such estimation includes the optimal search ranges and sample-weightings for computer-based contouring packages.

A general rule of thumb when interpreting the range of influence is to ensure the sample spacing for future "optimum" configuration purposes does not exceed 80% of the range of influence. Some authors prefer this value to be more in the order of 67% (Sarkor *et al.*, 1988). The search radius for estimation purposes should equal the range of influence as measured from the point or block centre being estimated. In the case of anisotropy, the geostatistical model would also indicate any preferred orientation to sampling patterns. For example, "ideal" sampling distances would be as follow:

Calorific Value: less than 400m.

Thickness or elevation: sampling distances 480mx200m, with the close-spaced sampling lines (fences) orientated at 115^0.

These generalisations are based on the experience of others and may be accepted as approximations. However, it is useful to know how much better the 67% approximation is than the 80% approximation, or for that matter, what the benefits of sampling within the range of influence actually are. In the case of the C.V. example, the 80% approximation could reduce the amount of sampling by some 30%, from 8.9 samples/Ha (67% of range of influence) to 6.25 samples/Ha (80% of range of influence).

Would such a saving in sampling be off-set by significantly reduced estimation accuracy? What is the actual change in estimation accuracy associated with the different sampling densities and patterns? Such problems may be quantified using kriging techniques. Quantification will in turn allow for more objective decision-making, not only for the selection of the most suitable sample configurations, but also by placing a degree of reliability or confidence on predictions.

2.4.4 Estimation accuracy and sample configurations
(Using "KRIGTEST")

OBJECTIVES

David (1977) comments there is no need to apply geostatistical estimation techniques if the range of influence is less than twice the dimensions of the block being estimated. Presumably, existing estimation procedures, such as inverse distance squared and classical statistics, would then provide as good results.

This section discusses the estimation error associated with the estimation of block mean values for different block and sample configurations. The aim is to illustrate the impact of different sample configurations on prediction accuracy, highlighting the affect of orientating sample patterns to take account of anisotropic conditions.

The following two-dimensional block sizes have been selected as representative of mining blocks for modern, underground high-extraction techniques for which geological predictions may be required:

200m x 200m: Longwall mining block for between two and four week's production (thirty four to sixty eight shifts), depending on seam thickness and productivity. Alternatively, the area mined by a conventional section on primary development for one month (about forty five shifts).

200m x 20m : Longwall mining block advance (or retreat) for between one and two day's production (three to six shifts), or conventional primary development for between two and four days (four to eight shifts).

100m x 100m: Partial block of sufficient dimensions to be critical to longwall or conventional mining design and performance.

10m x 10m : Extreme block size to simulate production (conventional) on virtually an hourly basis.

A 200m x 200m block may represent the monthly production area of one of several production sections. Any monthly estimate for the whole mine would then be a weighted combination of the individual block estimates. Likewise, the variance associated with the monthly estimate for the whole mine would be some combination of the estimation variances of the individual blocks. The same reasoning holds for predictions associated on any time-scale, say equivalent to daily or weekly production areas. The combination of blocks estimates to produce global average estimates will be dealt with in Chapter 3 (Grade and Contamination Control).

Sample configurations were selected to approximate the reality of channel sampling or drilling on a mine with the given production-block dimensions. Some of the sample and block configurations selected for simulation are illustrated in Figure 2.10.

METHOD

KRIGTEST was used to generate the results using ordinary kriging (mean of block to be estimated is unknown). For this purpose, KRIGTEST operates as follows :

SAMPLE CONFIGURATION

1. Sample localities defined in space (X,Y,Z co-ordinates).

BLOCK CONFIGURATION

1. A single rectangular-shaped block defined in space (maximum and minimum X,Y,Z co-ordinates).
2. Discretisation (number of points) defined for computing the block estimates (X=4, Y=4: 16 points; produced similar results to X=8, Y=8: 64 points, and far more rapidly).

SEMIVARIOGRAM MODEL

Define the:

1. Number of structures.
2. Nugget effect (C_0).
3. Model type (exponential, spherical, etc.).
4. Range/s of structure/s (longest range if geometrically anisotropic).
5. Sill/s of structure/s ($C - C_0$).
6. Anisotropism and its orientation (azimuth and coefficients of anisotropy-ellipse axes), for example longest range (X) =1 (750m); shortest range (Y) =0.42 (0.33x750m=250m).

KRIGING METHOD

1. Select ordinary kriging.

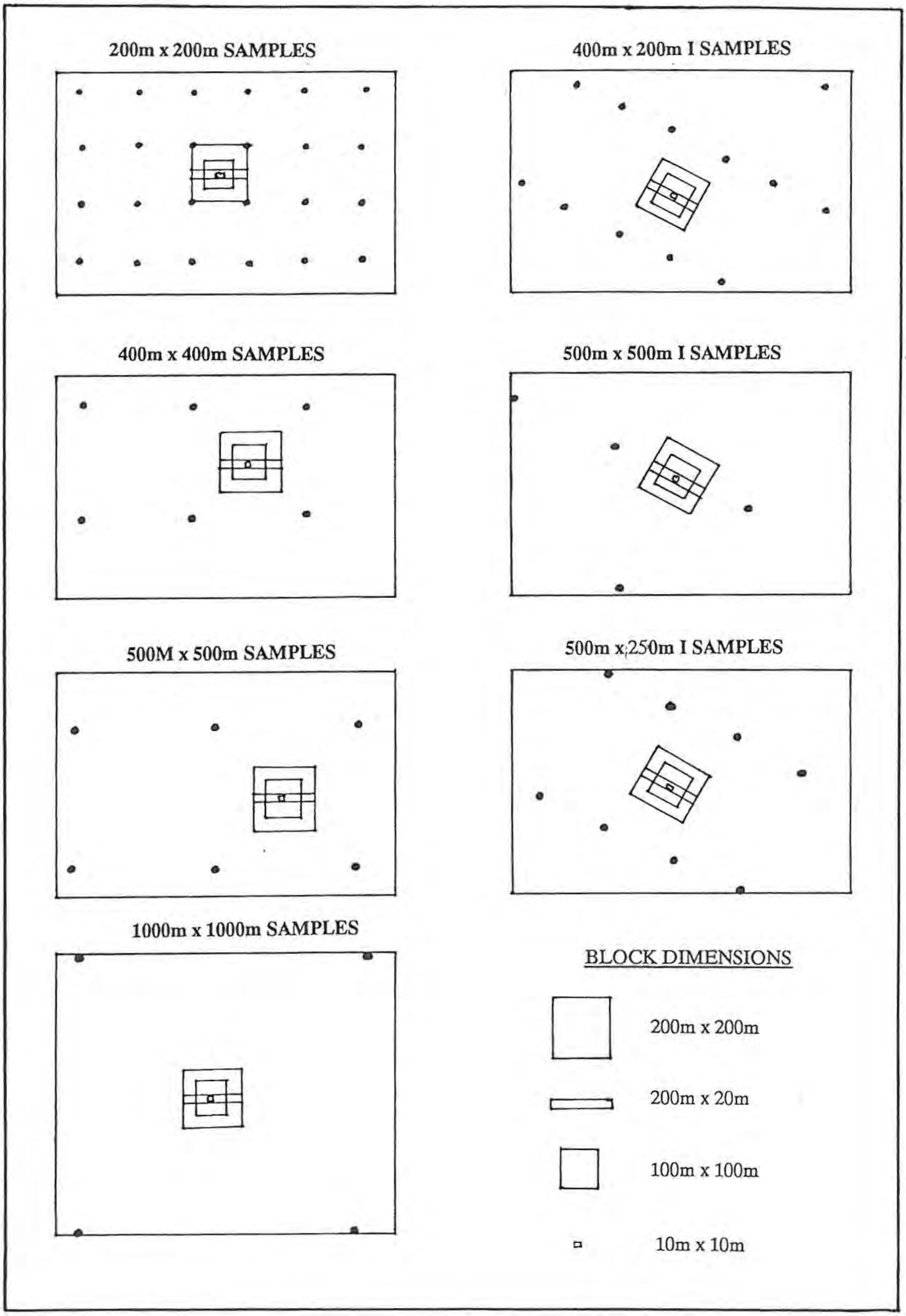


Figure 2.10 *Example sample and block configurations for kriging variance estimation.*

RESULTS

The following results are available:

1. Individual sample weighting coefficients.
2. Kriging variance: (estimated - expected mean)^2.
3. Between-block variance: $\sigma^2_{v/V}$.

The blocks were selected so as to produce approximately the maximum possible estimation error [(kriging variance)$^{0.5}$] for a given block-sample configuration. This was achieved by placing the block as far as possible from all sample positions, with no samples falling within the block boundaries (Figure 2.10).

Knowing the sample variance ($\sigma^2_{o/V}$) within the specified domain or population (sill: $C = C_1 + C_0$), together with the between-block variance (KRIGTEST result: $\sigma^2_{v/V}$), the within-block variance ($\sigma^2_{o/v}$) may be calculated as follows:

$$\sigma^2_{o/v} = \sigma^2_{o/V} - \sigma^2_{v/V}$$

The within-block and between-block variances may also be determined from standardised geostatistical tables and curves for the *F*-function. The *F*-function defines the within-block variance for given models and blocks. However, the block variances can be sensitive to the relative orientation of anisotropy axes, and this is more easily accounted for in programs like KRIGTEST (see later under "RESULTS").

The estimation error of the block mean may be calculated from the kriging variance as follows (assuming a normal distribution for the error):

With 95% confidence:

Estimation Error = $\pm$ (1.96)x(kriging variance)$^{0.5}$

With 90% confidence:

Estimation Error = $\pm$ (1.64)x(kriging variance)$^{0.5}$

This estimation error may then be expressed as a percentage of the block mean (estimated or actual), or the percentage estimation error at a given confidence level.

RESULTS

The full results of the above estimation exercise for C.V., thickness, and elevation are summarised in Appendix 2.3, and illustrated graphically in Figure 2.11. The latter graphs plot the estimation error of the mean (at 95% confidence) for the different block sizes and different sampling configurations.

The sample densities per hector (Ha) associated with each error are also plotted. The latter may be directly interpreted as relative sampling costs, providing the same method of sampling or drilling is assumed throughout. When interpreting these graphs it should be borne in mind that for sampling densities of around ten samples per hectare, a reduction in sampling of only one sample per hectare will save ten per cent of the sampling (and analysing) budget!

The volume-variance relationship for the various blocks is summarised in Table 2.2.

Table 2.2 *Volume-variance relationship: between-block and within-block variances.*

BLOCK	C.V. $(MJ/kg)^2$		THICKNESS m^2		ELEVATION m^2	
mxm	Between-	Within-	Between-	Within-	Between-	Within-
200x200	0.173	0.047	0.015	0.008	6.870	1.130
200x20	0.185	0.035	0.016	0.007		
100x100	0.190	0.030	0.017	0.006	7.432	0.568
10x10	0.208	0.012	0.018	0.005	7.943	0.057

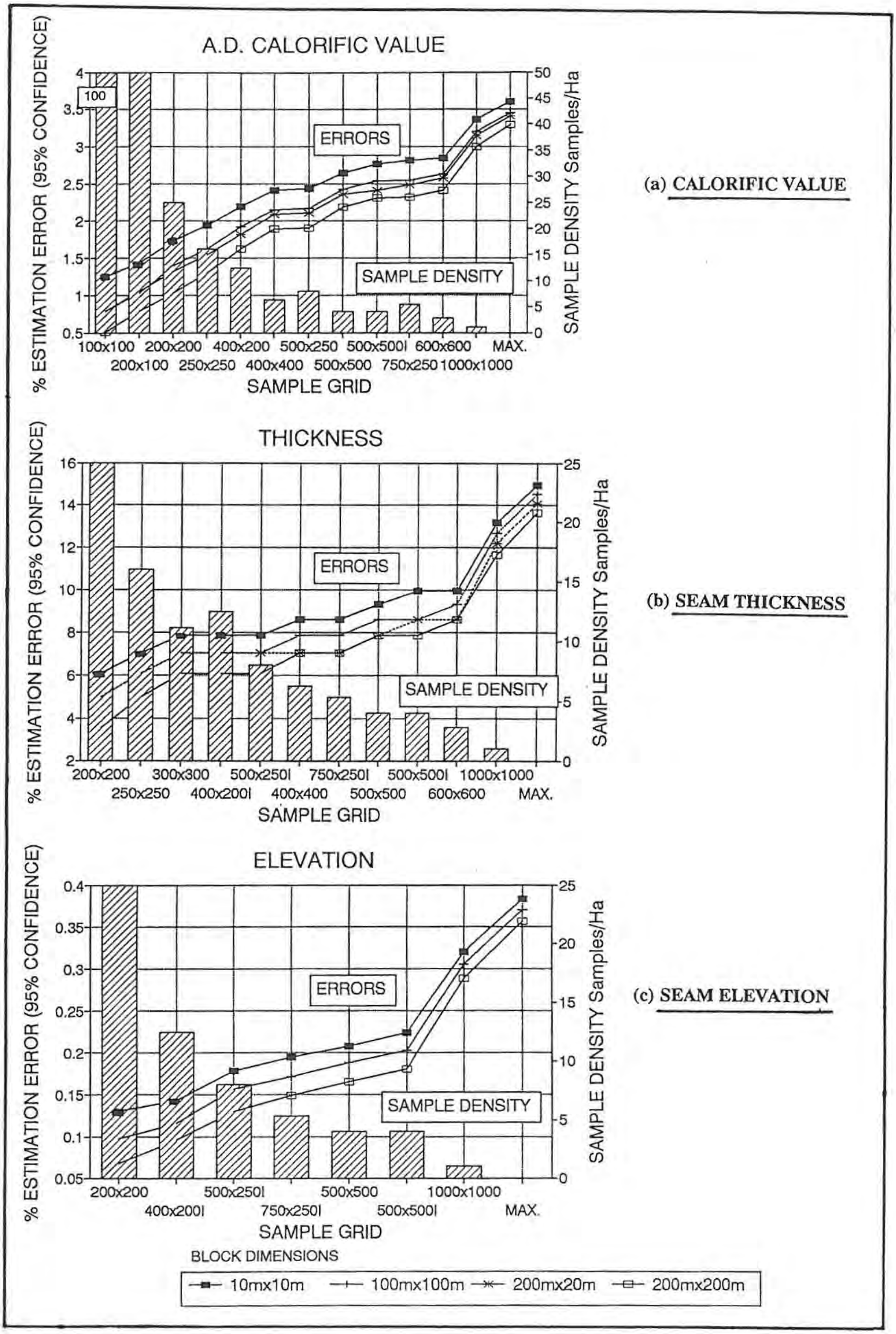


Figure 2.11 *Estimation error of the block mean for various block and sample configurations, for C.V. (a), thickness (b), and elevation (c).*

Note that standardised F-tables would return a value of 4.512 m^2 for the between-block variance for elevation (200m x 200m block). However, this does not take the orientation of anisotropy into account, and the same configuration in KRIGTEST (anisotropy axes parallel to block edges) returns a result of 4.794 m^2. These results are virtually half of the between-block variance reported in Table 2.2, and highlight the effect of block-orientation relative to anisotropy.

Table 2.3 and Figure 2.12 further summarise the results for C.V., thickness and elevation for 200m x 200m blocks. Such a summary can assist in the selection of "ideal" sampling configurations to suit specific requirements. Note, the sample density in Figure 2.12 is shown on a log scale to highlight the differences at low sampling densities, while the estimation errors have also been adjusted for illustration purposes.

Table 2.3 *Summary of sample configuration and estimation error results for 200m x 200m blocks (95% confidence).*

GRID	SAMPLE DENSITY	ESTIMATION ERROR FOR THE MEAN					
		AS UNITS			AS % OF THE MEAN		
		C.V.	THICK.	ELEV.	C.V. 24.77	THICK. 1.76	ELEV. 1441.60
Xm x Ym	/Ha	MJ/kg	m	m	MJ/kg	m	m
200x200	25.0	0.26	0.06	0.98	1.05	3.41	0.07
400x200I	12.5	0.40	0.11	1.38	1.61	6.25	0.10
500x250I	8.0	0.47	0.11	1.87	1.90	6.25	0.13
400x400	6.2	0.47	0.12	2.02	1.90	6.82	0.14
750x250I	5.3	0.57	0.12	2.15	2.30	6.82	0.15
500x500	4.0	0.54	0.14	2.38	2.18	7.95	0.17
500x500I	4.0	0.57	0.14	2.60	2.30	7.95	0.18
600x600	2.8	0.60	0.15	2.72	2.42	8.52	0.19
1000x1000	1.0	0.74	0.21	4.16	2.99	11.93	0.29

"I" implies the grid is inclined parallel to anisotropy axes (25^0), and staggered in the case of 500x500I.

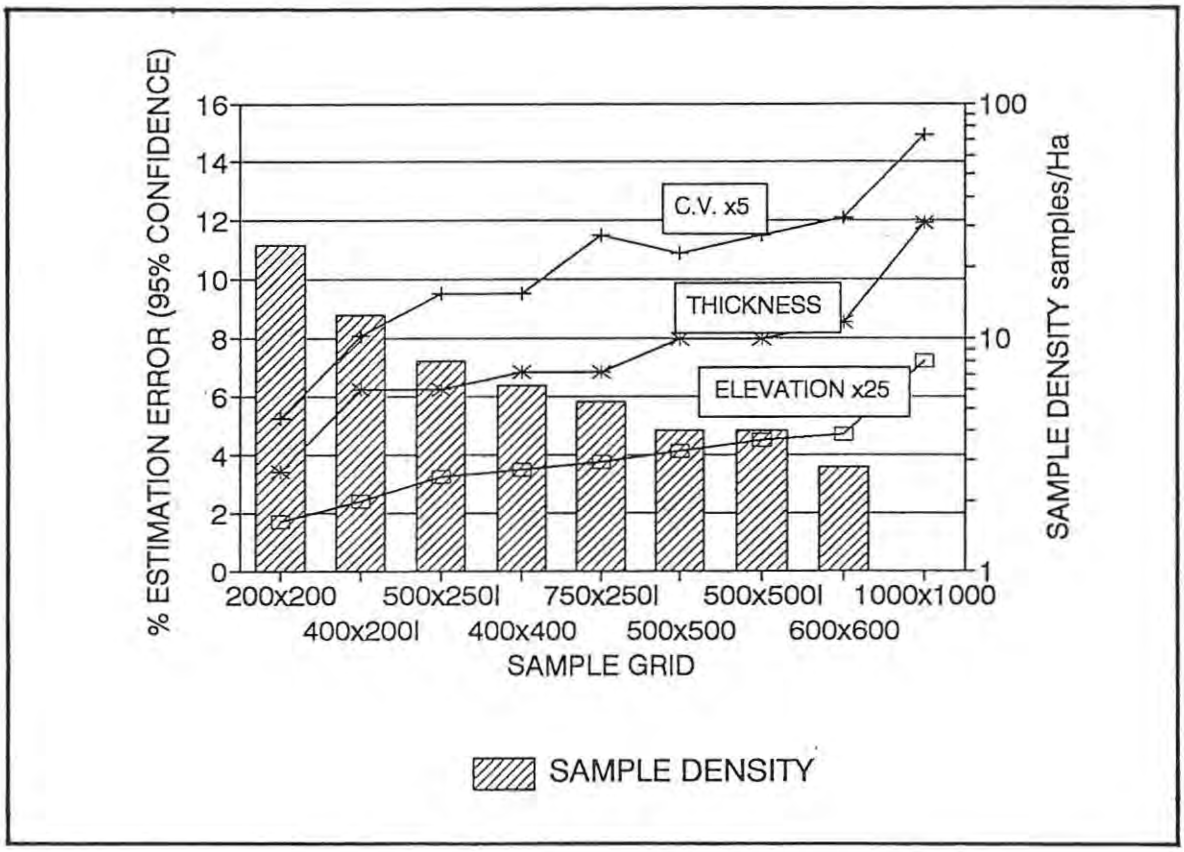


Figure 2.12 *Summary of sample configuration and estimation error results for 200m x 200m blocks (95% confidence). Sample density on a log-scale and errors enhanced as shown).*

DISCUSSION AND ESTIMATION EXAMPLES

The geological planner may now use the above results to compare the improvements in accuracy associated with various sampling configurations to the costs of sampling. In addition, the planner should be familiar with the levels of accuracy required for different variables at various exploration or mining stages, such as for tender purposes, or for short- and long-term mine planning.

For example, the estimation errors associated with a 500m x 500m staggered drilling pattern, with fence lines parallel to the direction of maximum variation (or minimum range of influence), are generally higher than for a similarly orientated but square grid. For this example, the estimation error for the mean (for a 200m x 200m east-west, north-south orientated block) increases by some 6% for C.V., 9% for elevation, and remains essentially unchanged for thickness. Doubling the sample density by drilling at a 250m spacing along every second fence line of the 500m x 500m inclined pattern, would reduce the estimation errors for the mean of the three variables by between 18% (C.V.) and 28% (elevation).

In practical terms (at the 95% confidence limit), the 500m x 250m inclined sample pattern improves the accuracy of the estimate for the 200m x 200m block mean, compared to the 500m x 500m inclined, staggered pattern, as follows:

C.V. $\pm$0.10 MJ/kg.

Thickness $\pm$0.03m.

Elevation $\pm$0.73m.

Such improvements may not be worth the additional sampling cost, depending on the application of the estimated means. If a C.V. difference 0.1 MJ/kg can make or break a contractual consignment, then the additional expense will obviously be considered.

The minimum value for the actual mean of a block may be estimated as follows:

95% confidence:

Estimated mean - (1.96)x(kriging variance)$^{0.5}$

90% confidence:

Estimated mean - (1.64)x(kriging variance)$^{0.5}$

Following on from the above discussion concerning the estimation accuracy for a 200m x 200m block with 500m x 500m and 500m x 250m sample configurations: The 500m x 500m inclined, staggered sampling may return an estimated mean for a block of 24.50 MJ/kg. At 95% confidence, the minimum value for the actual mean would be 23.93 MJ/kg. This minimum would increase to at least 24.03 MJ/kg with the 500m x 250m sampling (assuming the same estimated mean was also returned, which may not be the case with additional samples falling within the search radius for estimation). These predicted minimum C.V. values would increase to 24.02 MJ/kg and 24.11 MJ/kg respectively, if the degree of confidence were reduced from 95% to 90%.

A similar estimation may be made for the minimum average thickness expected for a block. This has important consequences for longwall mine-planning. In fact, the probability that an estimated block mean will exceed a given cut-off may be readily determined using indicator kriging (Van Zyl, 1990). Since the emphasis is on block means, these estimates are generally used for long-term mine planning (five years or more).

For short-term planning or production (say monthly), the within-block behaviour is of prime importance, and the proportion of a block expected to fall below a certain cut-off may also be predicted. The requirements are the within-block variance and the estimated mean thickness and its variance (the prediction is dependent on the combined variances as shown), for example:

-Mean thickness = 1.80m (kriging var. = 0.008m^2)

-Within-block variance = 0.015m^2

Minimum within-block thickness (95% confidence):

$1.80m - 1.96x(0.008)^{0.5} - 1.96x(0.015)^{0.5}$

$= 1.38m$

or, arranged differently:

$1.80m - 1.96x[(0.008)^{0.5} + (0.015)^{0.5}]$

$= 1.38m$

There is a 2.5% probability of seam thicknesses within the given block being below 1.38m.

The proportion of the given block expected to fall below a cut-off, or the probability of encountering values of less than the cut-off within the block, may be estimated as follows (applying statistical "z-tables" for the cumulative standardised normal distribution):

Probability of encountering within-block thickness of less than 1.60m:

"z" is given by: $\dfrac{1.60m - 1.80m}{(0.008)^{0.5} + (0.015)^{0.5}}$

$= -0.94$

From the F(z) tables, there is a 17% probability of encountering seam thicknesses of less than 1.60m in the given block.

The benefit of quantifying the estimation errors associated with different sample configurations and mining blocks are obvious. It allows objective decisions to be made between different sampling patterns for different purposes, which could save many millions of rands over the life of a project or mine. In addition, knowledge of the error of estimation for short-term grade control purposes can greatly assist forecasting and hence mining efficiency and performance, with the added advantage of increased client confidence. Some further applications to grade control systems will be discussed in Chapter 3 (Grade and Contamination Control).

2.5 CONCLUSION

The objectives of this chapter were stated as a series of questions for a mining venture and an exploration prospect (section 2.1.2 "Objectives").

The preceding discussion answers all the questions posed for the mining venture, namely:

1. The error of estimation associated with estimating mean values for given mining blocks and sample configurations can be calculated.
2. The confidence limits for the above mean estimations may be calculated and compared for different block and sample configurations.
3. The probability of a mean value for a mining block meeting some cut-off value can be determined.
4. The proportion of a given mining block expected to meet some set criteria may also be determined.

Steps "1" and "2" above provide an objective method for selecting suitable, cost-effective sample patterns for any given purpose or required level of "accuracy". In addition, the "accuracy" of a given estimation may also be assessed.

Steps "3" and "4" are essential for high-quality long- and short-term mine planning and geological predictions. In fact, the performance of geologists, planners or entire mining operations often hinge on the accuracy of predictions, as measured by reconciliations against actual results. Correctly applied, steps "1" to "4" will improve the scientific and professional standing of those who apply them.

The question not yet addressed, is that of the early application of geostatistical methods during prospect exploration, particularly towards optimising sample configurations. This problem is discussed below.

> Geostatistical analyses of a cross of 11 by 11, 100m-spaced samples orientated north-south and east-west were carried out to determine the semivariograms for C.V., seam thickness and

seam elevation. A "vague" spherical structure was discernable for the omnidirectional semivariograms for calorific value and elevation, but no pattern could be discerned in that particular area for thickness. The models to fit the experimental semivariograms were also fairly variable, with ranges of influence possible from 200m to 800m for a single directional experimental semivariogram.

Such early interpretations may be greatly aided by applying geostatistical results from nearby or similar deposits that have been more closely sampled. These existing results could provide an idea of the ranges of influence to be expected, thus aiding initial sampling-density decisions. The nugget effects determined from the existing data may also be applied to the semivariogram models for a new area, thus improving the reliabilty of such early models.

In summary then, it is suggested that experimental semivariogram construction be a standard part of prospect-result analyses. Even a poorly-defined semivariogram provides some idea about the behavior of the sample data, particularly with regard to degrees (and ranges) of correlation between samples. The following example indicates how geostatistical analyses may fit into a prospect investigation sequence:

PHASE 1. Wide-spaced drilling and field mapping conducted to define the broad reserve potential.

PHASE 2. Complete a cross of close-spaced (50m to 100m spacing) drill samples, say 10 holes by 10 holes, to provide initial experimental semivariograms. The semivariograms should be calculated as the data becomes available, and the close-spaced drilling modified until a structure becomes evident. Application of available results from similar deposits may refine these early models. Initial estimates of ranges of influence for the significant variables may be applied to the drilling configuration for later prospecting phases.

PHASE 3. Continue prospect drilling based on the results of PHASE 2. The original semivariogram models should be continually checked against the experimental semivariograms calculated from the incoming data. Additional close-spaced sampling in a different area of the prospect may be considered to check the validity of the models to the whole deposit.

The benefit of correctly designed drilling campaigns reaches beyond the potential cost-savings associated with optimally-designed grids. The improved estimation and contouring ability following from the application of geostatistical techniques will aid geological predictions, mine planning, mine operation, marketing, and plant operation, while boosting client confidence in a mine's ability to meet it's commitments. The smallest improvement in any one of these areas will probably pay for the close-spaced drilling needed to define the initial semivariogram models during early prospecting.

In conclusion, it must be remembered that geostatistics remains simply a tool to improving predictions and assisting with objective decision-making. As such, geostatistics should not be applied blindly, without an appreciation of the geological controls responsible for its attributes.

3. GRADE AND CONTAMINATION CONTROL FOR A LONGWALL COLLIERY

3.1 INTRODUCTION

Grade control, as practiced on any colliery, is concerned with predicting routine values for short- to medium-term mining operations, and the reconciliation of these predictions against "actual" results. For a longwall colliery supplying steam coal, such routine parameters may include the proximate analyses, calorific value, abrasiveness and possibly total sulphur, as well as float and sink analyses if the coal is beneficiated. In addition, grade control is concerned with identifying the mineable horizon (or horizons) in a selective-mining operation, together with monitoring how well these horizons have been extracted or adhered to.

Close-spaced surface drilling can be costly for any mining venture, particularly for longwall collieries operating in the deeper seams. For this, reason much of the data for short-term grade control requirements are acquired during actual bord-and-pillar development.

The underground data are derived from so-called "channel" samples. These samples are cut or chipped from the coal seam, perpendicular to the mining horizon, with dimensions suitable for obtaining sufficient "representitive" sample for analyses. For a 2m thick mining horizon, a channel sample 300mm wide and 50mm deep over the thickness of the horizon would provide some 4.5kg of sample, sufficient for most of the necessary tests (abrasive index test requires 4.0kg of sample). To allow the information to be computer recorded, additional information such as seam thickness, sample co-ordinates, and the elevation of the seam (or horizon) roof or floor, may be required.

Channel sample spacing is partially controlled by the layout of the mine sections, and spacings of between 50m and 200m may be typical for various collieries. Geostatistical techniques can be applied to select "optimal" sample configurations within these constraints (Chapter 2, section 2.4.4), and to ensure that unnecessarily detailed sampling does not take place. In the example quoted in Chapter 2, halving the amount of sampling by increasing the spacing from 200m x 100m to 200m x 200m, increases the estimation error for calorific value (C.V.) by $\pm$ 0.06 MJ/kg (95% confidence) for a single 200m x 200m mining block. A change in C.V. of 0.06 MJ/kg would equate to some 0.24% stone contamination (by mass). The significance of this change in estimation confidence would depend on the application of such estimates.

The traditional grade control practices are generally quite adequate, except for monitoring mining-induced contamination. In the case of a longwall colliery mining the full-seam thickness, every 1% of stone introduced into the ROM coal reduces an *in situ* C.V. of 24.70 MJ/kg by some 0.25 MJ/kg. While the overall contamination for the mine may be estimated by comparing run-of-mine (ROM) coal quality and *in situ*, uncontaminated quality predictions (Appendix 1.1), the actual contaminant types, contamination areas, and reasons for the contamination will remain unclear from such estimations.

Personal experience has shown that this contamination can be readily measured and/or estimated in the underground situation, even for longwall panels where the mined-out area is rendered inaccessible in the goaf. The monitoring system involved is outlined below, together with some actual results for a longwall colliery.

The potential application and reporting of estimation errors in routine grade control reports is discussed and illustrated by way of the longwall colliery example. This includes the all-important combining of variances (errors) when producing global estimates.

3.2 CONTAMINATION CONTROL

Contamination is unavoidable in a longwall colliery extracting the full seam thickness, as has been discussed in section 1.2.3 (Significance for longwall mining). The sources of contamination may be divided into the following three main classes:

1. PRIMARY CONTAMINATION
   a. Cutting of the stone floor or roof (accidental or deliberate) by the longwall shearer or continuous miner (C.M.) machines.

2. SECONDARY CONTAMINATION
   a. Slabbing or break-up of the roof or floor during mining and tramming, and the subsequent loading of this material together with the coal.

3. TERTIARY CONTAMINATION
   a. Non-coal material loaded with the coal during section-cleaning operations.
   b. Incorporation of non-coal material from the longwall gate roads (bord-and-pillar generated material) with the coal mined from the retreating longwall face.

Contamination is not necessarily a problem, providing its amount and affect on ROM coal have been correctly predicted during pre-mine planning. The mine, and the mine's clients, would then be prepared for the contaminated ROM coal, with supply contracts, and plant and machinery designed accordingly. However, experience shows contamination from longwall and continuous miner collieries in the Highveld and Eastern Transvaal coalfields to have been under-estimated by as much as five times during pre-mine planning. Actual monthly contamination levels have been known to run at up to 10%, with levels more commonly in the order of 6% (personal observation; Fourie, 1985; Barker, 1986; Chapman and Falcon, 1991). These under-estimations have occurred despite the application of sophisticated rock testing and experience to the problem.

Contamination is generally only considered a problem when the levels are sufficiently high to impact on a mine's ability to meet its coal supply contracts. There are three main ways in which such problem contamination levels may be addressed, namely:

1. Install a coal treatment plant to seperate the more-dense contaminants from the coal.

   Depending on the volume of ROM coal to be treated, this could add some 20% to the estimated R1 billion capital cost of a new colliery (Fourie, 1985). Washing may also result in significant losses of coal. For example, all of say 10% contamination may be removed together with 10% to 20% of the ROM coal. This could amount to the equivalent of some R24 million lost revenue per year for a colliery producing 5 million tons of steam coal annually. The effects of washing coal to produce power-station feed are discussed by Hand (1991).

2. Re-negotiate coal supply contracts to accommodate the lower-quality coal.

   This may translate to finding a new client, particularly if the mine's product is radically different to the initial estimates. Since this is not possible for a power station-tied colliery, the options are essentially "re-negotiate or reduce". Even if the existing client or clients agree to a new contract, they may not have much confidence in the particular mining company's proposals for future ventures.

3. Reduce the amount of contamination at its source.

   This involves firstly identifying the sources of contamination. This must be done with regard to the types and proportions of contaminants (roof, floor, waste, etc.), and the distribution of contamination areas. Only once this has been achieved, can the mine consider ways of reducing the contamination at its source. Depending on the origin of the bulk of the contamination, methods of tackling the problem may include: education (awareness) of production personnel; redefining cutting and/or loading procedures; minor or major design changes to cutting and/or loading equipment.

The remainder of this discussion is concerned with the control of contamination at its source. The method to be described is based on a system implemented by the author over a period of three years, and the results are typical of those actually achieved. Certain facts and figures have been modified to protect the confidentiality of the data.

The discussion is concerned more with the acquisition and manipulation (interpretation) of data, rather than with details of how the results may actually be used to control contamination levels. The methods used to control contamination may differ from colliery to colliery, depending on the nature of the contamination. Detailed report formats are also not presented, as these too depend on the requirements of individual collieries.

3.2.1 Contamination Measurements

Some form of measuring and monitoring system is required to identify and quantify the sources and levels of contamination. This should include a procedure for checking (reconciling) such measurements or estimates. The following example is directly applicable to a single-seam mining operation, where the total seam thickness is being extracted.

BORD-AND-PILLAR SECTIONS

The amount of stone or floor removed during bord-and-pillar mining can be directly measured in the mined-out areas. However, detailed measurements, of say 1m intervals, are not possible from a logistical or financial point of view. Instead, a relatively wide-spaced measuring pattern must be designed to still provide a reasonable estimate of the contamination for a given area.

The measuring pattern must also take account of the C.M. operating procedure. In this example, the full width of a roadway or split (say 6m to 7m) cannot be cut during a single advance, and is therefore cut during two passes. This may result in a "step" cut into the roof and/or floor, particularly in dipping seams (Figure 3.1).

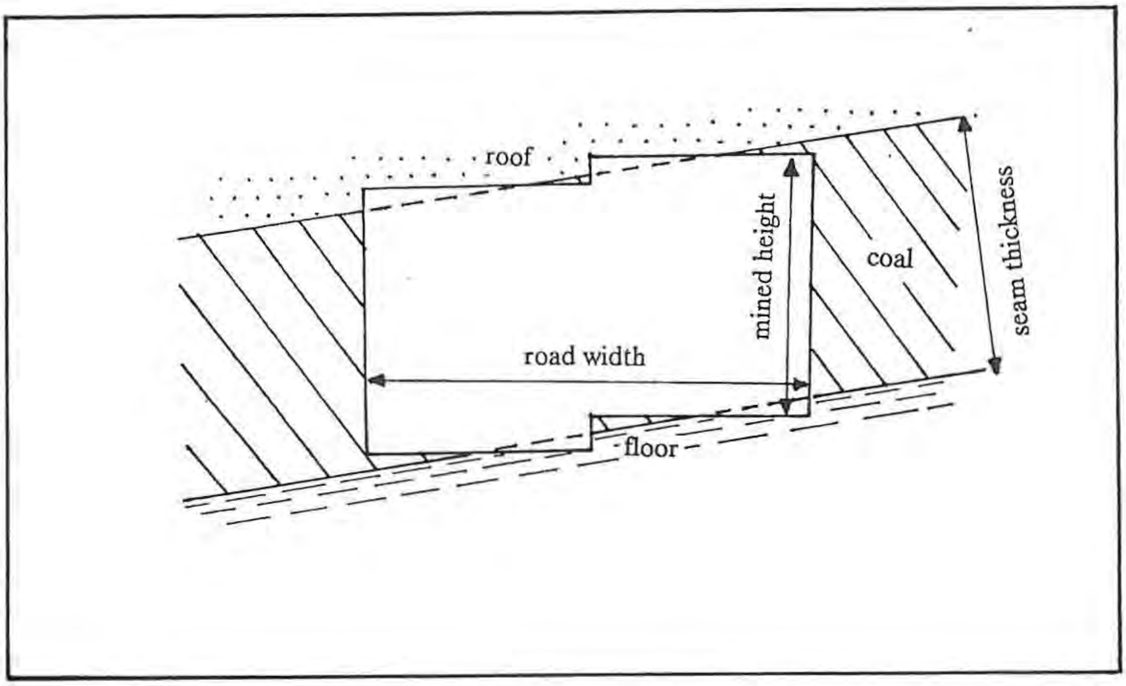


Figure 3.1 *Cross-section of a roadway mined by a continuous miner machine, through a gently-dipping coal seam.*

C.M. and shuttle-car (loader) tramming may also result in the break-up of the floor behind the actual advancing face (potential secondary or tertiary contamination). This is common in the central, travelling way of roadways and splits, particularly at intersections, and must be accounted for by the measuring patterns.

Measurements are taken perpendicularly to the mined horizon (or seam section). The mined height, seam thickness and amount of floor and/or roof material cut or removed are measured at the side of roadways and splits (against solid coal pillars and barriers). Measurements in the centre of roadways are only of the mined height, together with a note of whether roof and/or floor material have been removed. The seam thickness for these mid-road measurements may then be estimated as the average of the two nearest side-road thickness measurements. The amount of floor and/or roof removed at each mid-road point is then calculated as the measured mined height less the estimated seam thickness.

The spacing and pattern of the measurements required is determined largely by trial and error, aided by classical statistical analyses of the results. Geostatistical structural analysis (semivariogram construction and modelling) may also have some application here.

A typical measuring pattern is illustrated in Figure 3.2, together with an example report and calculation form for C.M.-section monitoring.

Primary contamination may be estimated directly from the side-road measurements (Figure 3.2). This is possible because the area immediately against the solid-coal pillar or barrier is generally not affected by secondary floor break-up.

Secondary contamination, derived predominantly from the central portion of the roadways, is readily calculated from the mid-road and intersection measurements (Figure 3.2).

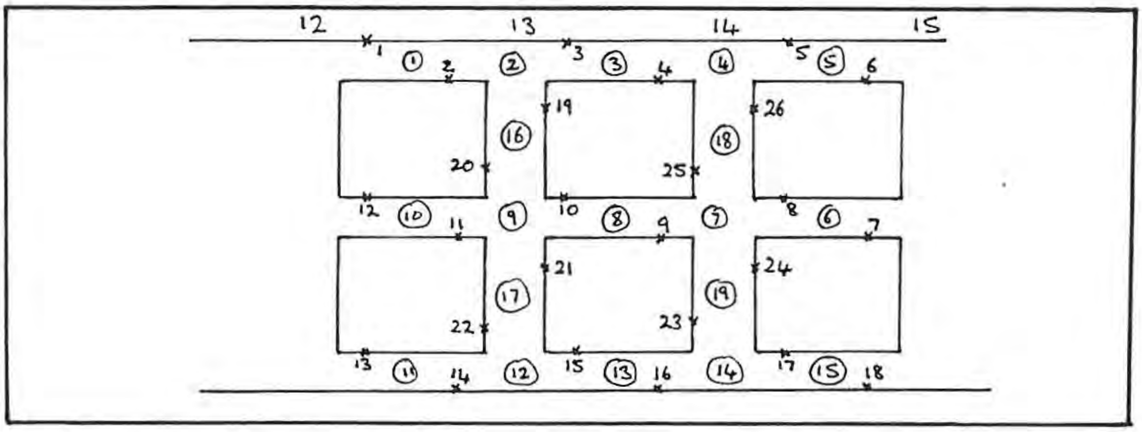
The relative proportions of primary and secondary contamination may be estimated providing the correct ratio of mid-road to side road measurements is used. A ratio of about 1:2, and the pattern shown in Figure 3.2, proved adequate for the given example.

Spacing between measurements may be in the order of 10m to 20m. A typical 3-road C.M. section's monthly advance of between 200m and 300m may then require some 100 to 200 measurements. These measurements may be recorded by one observer in a matter of a few hours, at a rate of some 30 measurements per hour. Height measuring may be facilitated by using readily-available electronic tape measures.

The large amount of regularly-spaced data generated from the measurements lends itself to classical statistical analysis, for storage and manipulation in commercially available spreadsheet packages. The various thickness and height values are typically normally-distributed and arithmetic averages therefore provide adequate estimates of the means of these values.

Calculating the average seam thickness from the mean contamination and mean mined height (as shown in Figure 3.2), produces essentially identical results to those obtained if each measurement were first reduced to its individual components (namely mined height, seam thickness and amount of contamination).

SECTION: WEST 100 SPLIT No.s 12 - 15
DATE: 15 NOV 1991 NAME: G. HOME



SIDE-ROAD MEASUREMENTS		
No.	Mined height	Floor/ roof cut
1	2000	0
2	2050	F 30
3	1950	0
⋮	⋮	⋮
26	2150	F 100
SUM/AVG.	2010	60
STD.DEV.	67	25

MID-ROAD MEASUREMENTS		
No.	Mined height	Floor/ roof cut
1	2050	F
2	2070	F + R
3	2000	F
⋮	⋮	⋮
19	2210	F
	2060	
	73	

CALCULATIONS

AVG. SEAM THICKNESS = avg. mined height - avg. roof/floor cut
= 2010 - 60
= 1950 mm (RANGE = 1800 to 2250)

AVG. MINED HEIGHT $= (SUM_{side} + SUM_{mid}) / (No_{side} + No_{mid})$
= (52260 + 39140) / (26 + 19)
= 2031 mm (RANGE = 1800 to 2300)

PRIMARY CONTAMINATION $= \frac{(\text{avg. roof/floor cut}) \times (RD_{stone})}{(\text{avg. seam thickness}) \times (RD_{coal})} \times 100$

$= \frac{(60) \times (2.30)}{(1950) \times (1.50)} \times 100$

= 2.83 % (CONT. RANGE = 0 mm to 150 mm)

TOTAL CONTAMINATION $= \frac{(\text{avg.mined ht.} - \text{avg. seam thick.}) \times (RD_{stone})}{\text{avg. seam thickness} \times (RD_{coal})} \times 100$

$= \frac{(2031 - 1950) \times (2.30)}{(1950) \times (1.50)} \times 100$

= 6.37 %

Figure 3.2 *Contamination measurement pattern and example report-form for a continuous miner bord-and-pillar section.*

The observer could include additional features of geological interest on the section plan (Figure 3.2), such as floor and roof types, faults, joints, scours (washouts) and dykes. Some of these features are needed to properly interpret the contamination results, and the extra time spent gathering such data is worth the effort.

The C.M. sections can be measured at any given time interval, depending on their individual rates-of-advance, and the manpower available. Fortnightly measurements may be reasonable from a logistical viewpoint, and would also provide contamination trends during the month, thus reducing the amount of measuring required at month-end.

Production personnel, say the miners in each section, are encouraged to measure the primary contamination during each shift. A brief report of these results should be submitted to geology. The report format would be similar to that shown in Figure 3.2, except that measurements would only be along the road or split sides and would be closely spaced, at say 2m intervals. The calculations could also be omitted, but a section for comments on general geological conditions could be included.

LONGWALL SECTIONS

It is impractical, if not impossible, to routinely obtain direct measurements of the roof or floor cut across a longwall face. The coal face is generally unstable and not protected by the support shields, while mining would have to stop, and large volumes of coal be removed to expose the coal-floor contact. In addition, the mined-out area is inaccessible in the goaf. The contamination estimation is therefore less direct than for the C.M. sections.

One parameter that may be measured routinely across the face is the mined height. This may be measured between chock shields (hydraulic support shields), at essentially any spacing or time interval. For example, the mined height may be measured at every tenth shield (15m for 1.5m-wide chocks) at a set time during each shift, or after a specific face advance (retreat) of say every 15m. The production personnel should be involved in these measurements.

A series of such measurements would result in a grid of some 170 mined height values per 200m advance (for a 200m wide panel). As was the case for the mid-road measurements in the C.M. sections, the prediction of contamination requires a value for seam thickness at each grid-point. Since the seam thickness cannot be measured directly, it must be estimated from the available data.

The data available for predicting seam thicknesses within a longwall panel include the detailed contamination measurements along the C.M.-mined gate-road developments, and occasional in-panel borehole intersections. Thickness predictions may also be checked against comments recorded when measuring the mined heights. If a mined height exceeds a predicted seam thickness at a given point, then the difference is inferred as the amount of stone (roof and/or floor) cut or removed. Production personnel may record whether roof and/or floor have indeed been cut (or broken) and loaded at each of the mined height measurement points. A typical recording sheet for a longwall section is illustrated in Figure 3.3. Obviously, independent checks of these measurements may occasionally be required to ensure the correct recording and reporting of the facts.

A simple and effective, although somewhat subjective, method for predicting seam thickness and contamination in a longwall panel is as follows:

1. On an overlay of the relevant section plans, plot the measured seam thickness for individual points, as measured during contamination monitoring of the C.M. sections surrounding the longwall panel.
2. Hand-contour the seam thickness data, extrapolating across the longwall panels, making use of mapped trends for features such as floor and roof rolls (scours), and faults.
3. Superimpose the grid of mined heights over the seam thickness contour plan and calculate the amount of stone cut at each grid-point (mined height less seam thickness). Reported comments of roof and floor cut (or loaded) may be used to update the thickness contour plan.

The C.M.-derived seam thickness measurements lend themselves to the application of computer-based estimations (and contouring) for within-panel seam thicknesses. In practise though, the extrapolation range of 200m or more across the longwall panel may be beyond such programs' capabilities. The hand-contouring method is generally adequate for providing the flexibility and control required.

SECTION: LW1 SPLIT(CHAINAGE) 1250m

DATE: 28 NOV 1991 NAME: L. WALL

No.	CHOCK No	MINED HEIGHT mm	SEAM THICK. mm (completed by geol.)	FLOOR/ROOF CUT OR LOADED
1	MAIN-GATE 1	2100		FLOOR CUT BY C.M
2	16	2130		GRADING DOWN FROM C.M.
3	31	2080		COAL
4	46	2030		FLOOR
5	⋮	⋮		⋮
6				
7				
8				
9				
10				
11				
12	⋮	⋮		⋮
13	TAIL-GATE 130	1950		FLOOR CUT BY C.M.
14				
15				

GENERAL CONDITIONS

ROOF FALL – 5m x 1m x 0.10m (CHOCK 60 – 64) – LOADED INTO GOAF.

Figure 3.3 *Example of a contamination measurement report for a longwall section.*

GENERAL

Monthly contamination values are based on large amounts of data and generally return reasonable (and reconcilable) results. However, even on a daily or shift-level basis, the measurements and estimates may indicate the source of a significant problem.

For example, if a routine ROM sample quality indicates an increase in contamination, the daily or shift reports from the production sections should allow the origin of the problem to be identified. The success, or otherwise, of this system obviously depends on reliable measuring and reporting by production personnel.

Knowledge and confidence of within-panel seam thicknesses and other geological features (faults, joints, floor rolls, etc.), are important for longwall horizon control. This knowledge allows "ideal" mining heights across the face to be determined in advance, while reducing the risk of encountering unexpected geological difficulties.

3.2.2 Reporting results (joint contamination report).

Individual section's contamination results should be recorded in such a way that a cumulative result is available at any time of the production month. The cumulative contamination estimate for each section may then be combined (tonnage weighted) to produce a contamination summary for the whole mine (Figure 3.4).

Additional columns may be included in such a report to compare production personnel contamination measurements (primary contamination) to those determined by geology. A column of target levels for each section may also be inserted.

The total measured contamination for the whole mine may be reconciled by comparison with the contamination calculated from the ROM coal quality and the predicted *in situ*, uncontaminated coal quality (Appendix 1.1). This comparison monitors the reliability of the measurements, providing the various measurement, analytical and estimation errors are known. The application of confidence limits to account for such errors is discussed in section 3.3 (Routine reporting of estimation errors).

CONTAMINATION SUMMARY REPORT DATE COMPILED 29 NOV. 1991

MONTH ENDED - NOV. 1991

SECTION	TONNAGE 28 NOV '91 TONSx1000	MINED HEIGHT mm	MEASURED FLOOR CUT PRIMARY mm	MEASURED FLOOR CUT TOTAL mm	MEASURED CONTAMINATION PRIMARY %	MEASURED CONTAMINATION TOTAL %	DATE OF LAST MEASUREMENT	TARGET CONTAM. TOTAL %
C.M.'s								
AA	20	2010	55	96	4.3	7.7	16/11	6.4
BB	30	1880	37	75	3.1	6.4	17/11	7.1
CC	30	2270	50	82	3.5	5.7	26/11	6.0
DD	25	2410	21	28	1.3	1.8	27/11	2.5
C.M. TOTAL AVG.	105	2142	40	70	3.0	5.4		5.6
L.W.'s								
LW1	130	2200		51		3.6	28/11	4.5
LW2	120	2010		67		5.3	28/11	5.0
TOTAL LWS	250	2109		59		4.4		4.7
TOTAL MINE	355	2119		62		4.7		5.0

CONTAMINATION % CALCULATED FROM CALORIFIC VALUE 5.0

Figure 3.4 *Example of a contamination summary report combining results for the sections of a mine.*

3.2.3 Practical example

BACKGROUND

The colliery in question completed some two and a half years of C.M. bord-and-pillar development, before commissioning its first longwall. Contamination levels during this early period were already higher than initially predicted. The average contamination (calculated from C.V.) during this period was 3.9%, with 95% of the monthly levels below 5.7% (Figure 3.5).

The high contamination levels were investigated and assigned primarily to weaker-than-anticipated floor conditions. Some measurements were taken, but these did not form part of a routine control system, possibly because ROM coal qualities still met the coal supply agreement stipulations. It may also have been hoped matters would improve , or at least remain static, with the introduction of longwalling.

During the eighteen months before a second longwall was commissioned, the average monthly contamination (as calculated from C.V.) was 5.8%, with 95% of the levels below 8.6%. The increase in contamination during this period appears to have been associated with the increasing longwall tonnage (Figure 3.5). A second longwall was brought into production seven months before introducing the contamination control system. The average monthly contamination during these seven months was 7.7%, with a high one month of 9.0% (Figure 3.5).

The effect of virtually doubling the contamination, from the initial monthly average of 3.9% to 7.7%, was to place the ROM coal outside of contractual specifications for more than only C.V..

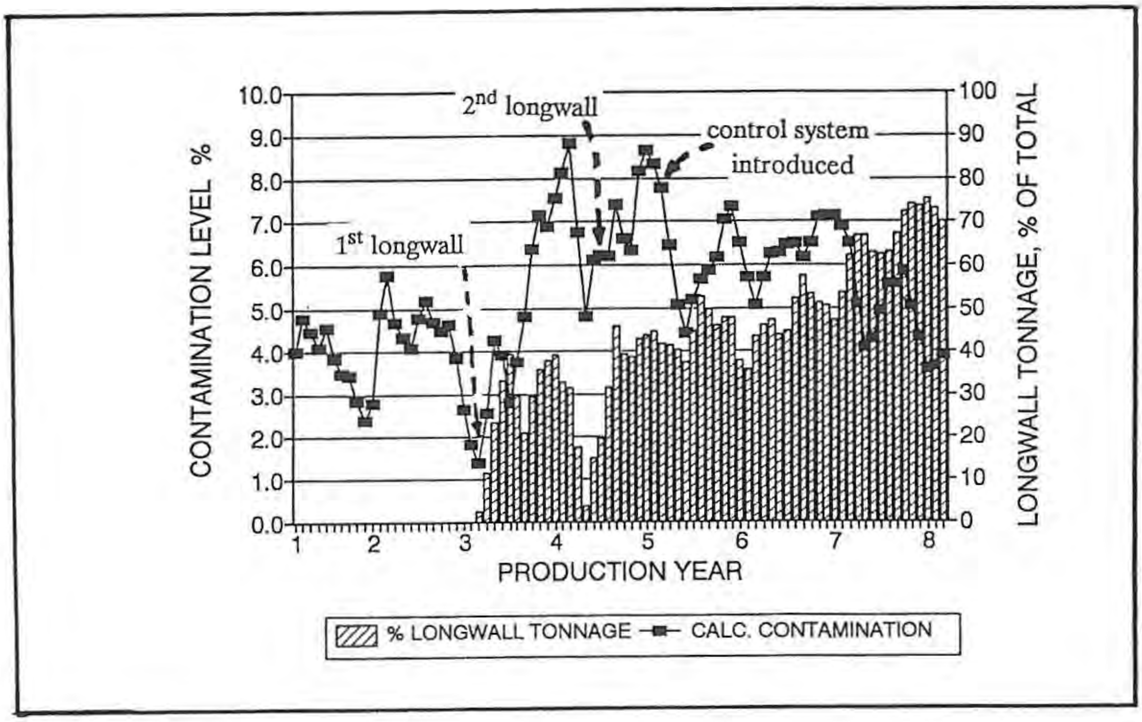


Figure 3.5 *Graph of monthly contamination levels (as calculated from C.V.), illustrating the effect of longwalling, and the impact of the contamination control system (2 month moving average values).*

A contamination measuring system, similar to that outlined above, was introduced in the fifth year of operation. The number of sections and rates-of-advance on the colliery meant that additional man-power was required to run the program. The system currently operates with the full-time services of a geological observer (no formal geological training), and a portion of the time of a geological assistant (formal geological training). The mine geologist takes overall responsibility for the program.

RESULTS

The average monthly contamination level during the first year of the contamination system's operation was 5.8%, with only one month exceeding 6.5%. This was attained despite a continued increase in longwall tonnage (Figure 3.5). The average monthly contamination for the first three years of the system's operation was 5.7%, with 95% of the values below 7.4%.

The monthly average for the contamination measurements are compared to those calculated from C.V. as shown in Figure 3.6. The average monthly difference between these two values is 0.88%, with a trend for the measured contamination to be below the calculated values by some 1%. Since the calculated values are considered "actual" levels, a measured value below the calculated value implies additional ROM contamination to that measured, which may well be expected from secondary or tertiary sources. The interpretation of these differences also depends on the reliability of the C.V. predictions and determinations, as well as the reliability of the contamination measurements, as will be discussed under section 3.3 (Routine reporting of estimation errors). In general, the combined monthly values for measured contamination have an associated error of about 5%, while the error for calculated estimates may be in the order of 10% (90% confidence).

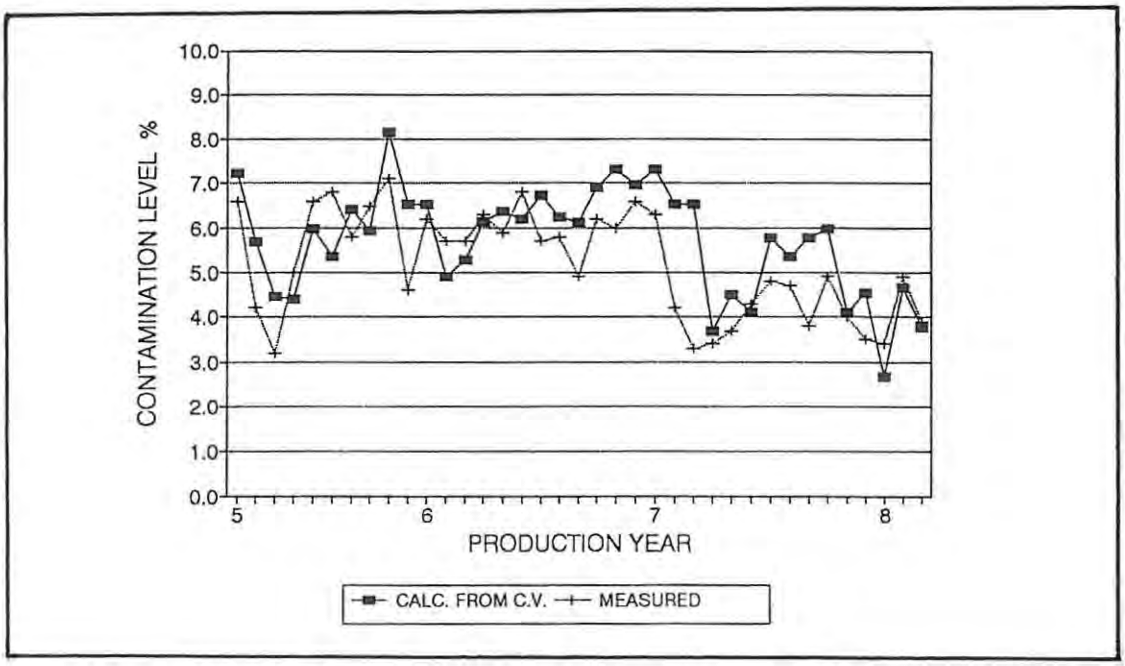


Figure 3.6 *Measured versus calculated monthly average contamination levels.*

BENEFITS

The initial longwall sections appear to have been largely responsible for the significantly increased contamination levels. The results of the contamination measuring system were used to improve long-wall horizon control and planning, and this alone probably resulted in a virtual halving of the longwall-sourced contamination.

The control system also identified floor-stone material as responsible for some 90% of total contamination, with 40% to 60% of this resulting from primary stone cutting. Production personnel awareness, combined with simple modifications or adjustment to cutting procedures and C.M. machines, were largely responsible for improvements in C.M.-related contamination levels.

The detailed measurements generated by the program also provided a vast amount of additional geological information to improve mine planning and prediction capabilities. For example, the seam floor could be divided into zones displaying particular behaviour with respect to both C.M. and longwall mining. Applying this knowledge to borehole core interpretations, allowed contaminated, ROM coal qualities to be predicted with greater accuracy. The improved understanding of contamination sources also allowed the problem of predicting non-linear parameters, such as abrasive index, to be addressed (Noppe, 1990; Chapman and Falcon, 1991).

3.3 <u>ROUTINE REPORTING OF ESTIMATION ERRORS (CONFIDENCE LIMITS)</u>

3.3.1 <u>Introduction</u>

Values used to make grade and contamination predictions all have some error associated with their individual estimations. Errors of estimation were dealt with at length in Chapter 2 (Some Geostatistical Applications). The discussion, however, stopped short of actually combining estimates for individual blocks or sections to produce a global or total-mine estimate.

Measured contamination levels, in the above longwall example, are reported as the arithmetic average of a specific number of approximately equally-spaced measurements. These values may be reported in terms of a variance and specific confidence limits, providing the statistical distributions of the various data populations are known. The ROM calorific value (C.V.) for a specific period, in this case one month, is also the arithmetic average (tonnage weighted) of a specific number of daily sample values, with an associated statistical distribution. These measured and analysed parameters (mined height, seam thickness, contamination, and C.V.) also have associated measurement and analytical errors (human and instrument-related).

This section summarises the statistical rules for combining means and variances, illustrating how these may be used to improve the reporting and interpretation of grade control results. The examples are kept simple, and follow on from the longwall colliery example.

3.3.2 Combining estimates and variances

The coal parameters of interest, namely mined height, seam thickness, contamination, and C.V., conveniently display approximately normal distributions. Some rules for combining variances for normally distributed data are presented in Appendix 3.1 (Suite, 1965; Koch and Link, 1970; Hemala and Stewart, 1979). Note the distribution resulting from the combination of individual normally distributed data will also be normally distributed.

The above rules describe the dispersion (variance) of the new population (total-mine mean estimate), resulting from the combination of the individual populations of the original variables (block or section means). The emphasis in this section, however, is on the reliability of the combined estimates. This reliability may be expressed in terms of confidence limits by combining the estimation variances (error variances) for the individual block or section values.

The important rule for combining estimation variances is as follows (modified from equation "3": Appendix 3.1):

For a weighted average ($\bar{X}_T$) of a number of INDEPENDENT variables (block estimates) with means X_i and variances V_i^2, the resulting variance (V_T^2) is given by:

$$\text{Combined variance, } V_T^2 = \frac{(N_1V_1^2 + N_2V_2^2 + \ldots + N_nV_n^2)}{N_1 + N_2 + \ldots + N_n} \times \frac{1}{n}$$

where n = number of block values combined
and N_i = weighting of a block (area, tonnage, or number of samples).

The blocks or sections are generally separated, by distances greater than the ranges of influence of the parameters of interest. The block means may therefore be treated as independent of each other.

The combined standard deviation (V_T) may be used to estimate confidence limits for the overall mean ($\bar{X}_T$). If more than about 25 individual values (n) are combined, then the calculation is as follows (applying cumulative normal distribution tables):

90% confidence limits for the total-mine estimate (n>25):

$$[(\bar{X}_T - 1.64xV_T) < \bar{X}_T < (\bar{X}_T + 1.64xV_T)]$$

If fewer than 25 individual values (n) are combined, statistical Student's-t tables must be used to determine the factor for multiplying V_T (for n-1 degrees of freedom):

90% confidence limits for the total-mine estimate (n=6):

$$[(\bar{X}_T - 2.02xV_T) < \bar{X}_T < (\bar{X}_T + 2.02xV_T)]$$

EXAMPLE CALCULATIONS

The following calculations illustrate how variances and errors of estimation may be combined to produce confidence limits for total mine estimates.

Calorific value

In situ C.V. predictions for individual blocks (produced from geostatistical estimation techniques) may be reported as a series of block estimates with corresponding errors of estimation at specific confidence limit..

For example, six production areas for a given month may have the following individual values (90% confidence):

SECTION	PRODUCTION TONNAGE	CALORIFIC VALUE MEAN MJ/kg	ERROR VARIANCE $(MJ/kg)^2$
CM1	30 000	25.00	0.017
CM2	30 000	24.80	0.017
CM3	20 000	24.10	0.031
CM4	35 000	26.00	0.017
LW1	120 000	23.90	0.017
LW2	135 000	24.50	0.017
TOTAL/AVERAGE	370 000	24.49	0.018

This assumes the blocks have already been mined-out and the tonnages are accurately known. (This is not strictly true in practice, and the additional error associated with the tonnage estimates, namely for mined volumes and relative densities, could also be determined).

The tonnage-weighted average of 24.49 MJ/kg has an associated combined error variance, V_{Ti}^2, of 0.0030 $(MJ/kg)^2$, calculated as follows:

$$\text{Combined variance, } V_{Ti}^2 = \frac{0.018}{6} (MJ/kg)^2$$

$$= 0.0030 \ (MJ/kg)^2$$

The 90% confidence limits for this mean may then be calculated as follows (n = 6):

$$\textit{In situ} \text{ C.V. (90\% confidence)} = 24.49 \pm 2.02 \text{x} (0.0030)^{0.5} \text{ MJ/kg}$$

$$= 24.49 \pm 0.11 \text{ MJ/kg}$$

The "actual" ROM C.V. for the month is calculated as a tonnage-weighted average of some 24 daily results.

The daily results are themselves an average of two C.V. determinations (for a combined daily ROM coal sample) falling within 0.12 MJ/kg of each other, or $\pm$ 0.06 MJ/kg of their average (British analytical standard S1016-5). This range is known as the "within-laboratory tolerance" or the "repeatability". The variances of the daily average C.V. values can be calculated from the actual pairs of results. For this example the daily values are assumed to all have the same variance of 0.0025 $(MJ/kg)^2$ (or a standard deviation of 0.05 MJ/kg, for "n - 1" samples).

The combined variance, V_{Tr}^2, for the 24 daily values would then be 0.0001 $(MJ/kg)^2$, calculated as follows:

Combined variance, V_{Tr}^2 $= \frac{0.0025}{24} (MJ/kg)^2$

$= 0.0001 (MJ/kg)^2$

Assuming a normal distribution to the data, the monthly average of say 23.40 MJ/kg may now be reported in terms of confidence limits. The 90% confidence limits for this example would be estimated as follows (n = 24):

ROM C.V. (90% confidence) $= 23.40 \pm 1.71x(0.0001)^{0.5}$ MJ/kg

$= 23.40 \pm 0.02$ MJ/kg

The "accuracy" of the given laboratory has not been discussed, but two laboratories will generally return results of within 0.30 MJ/kg of each other for the same sample (British analytical standard S1016-5). This range is known as the "between-laboratory tolerance" or the "reproducibility". This "error" may be used in a similar manner to the above ROM C.V. calculation, but is not applied in this "one-mine, one-laboratory" example.

Contamination

The contamination calculated from the above results would be 4.45%, calculated as follows:

% Contamination $= \frac{(24.49 - 23.40)x100}{24.49} \%$

$= 4.45 \%$

The error variance, V_{cont}^2, for the above contamination value is 0.052 $\%^2$, estimated as follows (applying equation "1", Appendix 3.1):

Combined variance, V_{cont}^2 $= \frac{(0.0030 + 0.0001)x100^2}{24.49^2} \%^2$

$= 0.052 \%^2$

Assuming a normal distribution to the calculated contamination, the 90 % confidence limits may be estimated as follows (n = 6 for the most limiting variable, namely *in situ* C.V. predictions):

% Contamination (90% confidence)

$= 4.45 \pm 2.02x(0.052)^{0.5}\%$

$= 4.45 \pm 0.46$ %

<u>The measured contamination</u> may also be reported in terms of confidence limits.

The average contamination for a section is calculated from some 150 measurements, for which a variance may be readily calculated (assuming a normal distribution for the data).

<u>For a section</u> with a mean measured contamination level of 5.0% and sample variance of $6.25\%^2$ (n = 150), the variance, V_m^2, of the mean from the "actual" (unknown) mean is given by:

Variance for the mean, V_m^2 $= \frac{6.25\ \%^2}{150}$

$= 0.0417\ \%^2$

The 90% confidence limits for the estimation of the true section mean may then be reported as follows (n > 25):

% Contamination (90% confidence)

$= 5.0 \pm 1.64x(0.0417)^{0.5}$ %

$= 5.0 \pm 0.33$ %

The measured contamination and variance for each section may be combined to produce an <u>average figure for the whole mine</u> (completing a similar table and calculation to that illustrated for the *in situ* C.V., as above). Assuming the six production sections all have variances for their means of $0.0417\ \%^2$, the combined variance, V_c^2, for the whole mine would then be $0.00695\ \%^2$, calculated as follows:

Combined variance, V_c^2 $= \frac{0.0417\ \%^2}{6}$

$= 0.00695\ \%^2$

A typical mean contamination value for the whole mine may be 4.9 %. The 90% confidence limits for this mean could then be estimated as follows (n = 6):

Measured % contamination (90% confidence)

$= 4.9 \pm 2.02x(0.00695)^{0.5}$ %

$= 4.9 \pm 0.2$ %

The problems associated with determining the relative densities of coal and stone may add an additional degree of uncertainty to the measured contamination values.

3.3.2 Application of results

Existing grade control spreadsheets or programs can be readily adapted to include the calculation and reporting of confidence levels for the quoted estimates. At the very least, a report such as that in Figure 3.4 should include a summary of the possible ranges of selected estimates (at a given confidence limit). The "bottom line" of such a report could be reported as follows:

	CONTAMINATION		CALORIFIC VALUE	
	MEASURED %	CALCULATED %	PREDICTED MJ/kg	ROM MJ/kg
AVERAGE	4.9	4.4	24.49	23.40
RANGE (90% confidence)	4.7-5.1	4.0-4.9	24.38-24.60	23.38-23.42

Simple visual checking of the measured and calculated contamination ranges shows the two values to be statistically similar at the 90% confidence level (the ranges overlap). More sophisticated forms of hypothesis testing could also be applied (Koch and Link, 1970).

3.3.3 Conclusion

Combining variances to produce confidence limits for grade control estimates can prevent spending unnecessary time searching for reasons for "imagined" errors.

For example, the measured contamination level may be used to estimate a ROM, contaminated C.V. from the *in situ*, uncontaminated prediction (Appendix 1.1). Using the values from the above example, the ROM estimate would be 23.29 MJ/kg (assuming the contaminant has no heating value), calculated as follows:

$$\text{Estimated ROM C.V.} = \frac{(100 - 4.9)\times(24.49)}{100}\ \text{MJ/kg}$$

$$= 23.29\ \text{MJ/kg}$$

The combined variance for this estimation would be 0.0030 $(\text{MJ/kg})^2$, calculated as follows (Appendix 3.1, independent variables):

Combined variance, V^2

$$= V_{Ti}^2 + \frac{V_c^2}{100^2}$$

$$= 0.0030 + \frac{0.00695}{10\ 000}$$

$$= 0.0030\ (MJ/kg)^2$$

The estimated ROM C.V. could then be reported in terms of confidence limits as follows (n = 6 for the most limiting variable, namely *in situ* C.V. predictions):

Estimated ROM C.V. (90% confidence limits)

$$= 23.29 \pm 2.02x(0.0030)^{0.5}\ MJ/kg$$

$$= 23.29 \pm 0.11\ MJ/kg$$

Without the confidence limits, the estimated ROM C.V. of 23.29 MJ/kg may mistakenly have been thought to be significantly different to the actual result of 23.40 MJ/kg. However, it can be seen at the 90% confidence level that the ranges of the two values overlap, and therefore there is statistically no significant difference between the two values.

It may be asked how significant an estimation error of 0.11 MJ/kg actually is for ROM C.V.. A colliery that initially underestimated its contamination, may have ROM qualities running close to the lower-limit of contract specifications. In such a case, a change in C.V. of 0.10 MJ/kg, or even 0.05 MJ/kg, may result in the rejection of a coal consignment, or the payment of severe penalties. These costs could add up to millions of rands each year, and accurate predictions and reporting may help prevent such losses.

The most significant contributions to the above estimation errors have resulted from the *in situ* C.V. predictions (combined error variance of 0.0030 $(MJ/kg)^2$). Any reduction in contamination or grade control-related prediction errors would clearly require addressing this problem. The relative costs and prediction improvements associated with increased sampling may be modelled and quantified using geostatistical estimation techniques (section 2.4.4: Estimation accuracy and sample configurations). The results from such modelling will improve the objectivity of both decision-making and interpretations regarding sampling philosophy and estimates.

CONCLUDING REMARKS

"Geological control" encompasses a wide range of functions in various fields, including mineral economics, mineral evaluation, exploration, mining and management. This control begins well-before the first borehole is drilled on a prospect or target, and continues through to the end of a mining-venture's life.

The form these geological controls take tends to vary depending on the context in which they are applied. The emphasis during the early stages of planning and exploration is generally on making the right decisions, or "doing the right job". This emphasis shifts as one moves into the realm of mining, and "control" during these phases is often directed at ensuring "the job is done right" or correctly. This is generally achieved by way of a variety of grade control systems.

The success of many decisions or production-type controls in mining and exploration depend ultimately on the reliability of geological predictions and interpretations. Geologists must therefore ensure they provide the best-possible estimates and interpretations at all levels of the management-exploration-mining continuum.

The coal-related examples presented here illustrate the application of some statistical techniques towards improving geological estimation and reporting capabilities. The contamination control system provides a simple, but effective method for monitoring and potentially reducing contamination on a longwall colliery.

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APPENDIX 1.1

CONTAMINATION CALCULATIONS

CONTAMINATED QUALITY

ROM C.V. $= \dfrac{(100 - \%\text{cont.}) \times (\text{uncont. C.V.})}{100}$

ROM ASH $= \dfrac{(100 - \%\text{cont.}) \times (\text{uncont.Ash}) + (\%\text{cont.})(\text{cont.Ash})}{100}$

CONTAMINATION PERCENTAGE

% cont. $= \dfrac{(\text{Thickness of stone mined})}{(\text{Seam thickness})} \times \dfrac{(\text{RD of stone})}{(\text{RD of coal})} \times 100$

% cont. $= \dfrac{(\text{uncont. C.V.} - \text{ROM C.V.})}{\text{uncont. C.V.}} \times 100$

ABBREVIATIONS

cont.	= contamination
uncont.	= *in situ*, uncontaminated
C.V.	= calorific value (MJ/kg)
Ash	= ash content (%)
RD	= relative density
ROM	= run-of-mine

APPENDIX 2.1

CHANNEL SAMPLE DATA USED FOR GEOSTATISTICAL ANALYSIS

FLOOR ELEV. m
THICKNESS m
A.D. C.V. MJ/kg

	Y	X	FLOOR ELEV.	THICKNESS	A.D. C.V.
1	-6730	3119	1442.28	1.62	25.01
2	-6730	3229	1440.29	1.81	25.26
3	-6830	3119	1441.15	1.75	25.16
4	-6830	3229	1440.8	1.7	25.16
5	-6930	3119	1439.78	1.82	24.73
6	-6930	3229	1440.17	1.83	25.27
7	-7030	3119	1441.06	1.84	25.55
8	-7030	3269	1440.28	1.72	25.12
9	-7130	3119	1440.76	1.74	25.14
10	-7130	3269	1440.04	1.76	25
11	-7230	3119	1442.08	1.82	25.23
12	-7230	3269	1440.48	1.82	24.47
13	-7330	3119	1441.07	2.03	24.87
14	-7330	3269	1439.13	1.77	24.58
15	-7430	3119	1439.48	1.92	24.62
16	-7430	3269	1436.88	1.92	24.45
17	-6625	2579	1442.38	1.62	24.33
18	-6625	2679	1441.2	1.8	25.27
19	-6625	2779	1443.11	1.89	24.65
20	-6625	2879	1443.12	1.88	23.83
21	-6625	2979	1441.2	1.8	24.82
22	-6625	3079	1440.3	1.7	24.61
23	-6625	3179	1441.98	1.72	24.76
24	-6625	3299	1438.05	1.95	24.15
25	-6625	3399	1439.12	1.68	24.78
26	-6625	3499	1439.19	1.61	24.13
27	-6625	3599	1437.85	1.75	25.39
28	-6625	3699	1436.27	1.83	25.2
29	-6520	3699	1436.16	1.94	24.82
30	-6520	3599	1437.88	1.72	24.97
31	-6520	3499	1439.12	1.68	24.57
32	-6520	3399	1439.02	1.78	24.79
33	-6520	3299	1438	2	24.84
34	-6625	3799	1436.38	1.72	25.22
35	-6525	3799	1436.51	1.59	24.45
36	-6625	2319	1443.37	1.63	24.89
37	-6625	2419	1442.86	1.64	24.96
38	-6625	1969	1443.86	1.64	25.03
39	-6625	1819	1444.13	1.87	25.08
40	-6625	1669	1445.77	1.83	24.89
41	-6625	1519	1448.32	1.58	24.89
42	-6625	1369	1447.14	1.86	24.4
43	-6600	2114	1443.32	1.68	25.18
44	-6600	3114	1443.1	1.5	25.09
45	-6775	2519	1442.32	1.98	24.1
46	-6650	2519	1442.54	1.76	24
47	-7530	3119	1437.99	2.01	24.67
48	-7530	3269	1438.22	1.78	24.72
49	-7650	3119	1437.2	1.8	24.29
50	-7650	3269	1437.89	1.81	24.33
51	-7750	3119	1437.13	1.67	24.76
52	-7750	3269	1435.31	1.69	24.59
53	-6775	3374	1434.17	1.83	25.28
54	-6775	3739	1434.15	1.85	24.99
55	-6775	3559	1434.11	1.89	25.16
56	-6775	3659	1434.07	1.93	24.93
57	-6775	3769	1434.09	1.91	24.73
58	-7020	3014	1442.16	1.84	25.38
59	-6895	3014	1441.62	1.88	25.37
60	-7020	2909	1442.45	1.85	24.24
61	-6895	2909	1442.7	1.8	24.24
62	-6775	3019	1443.03	1.57	25
63	-6775	2919	1442.82	1.78	24.95
64	-6775	2819	1442.11	1.69	24.61
65	-6775	2719	1441.95	1.85	24.54
66	-6775	2619	1441.08	1.92	24.65
67	-6775	2519	1442.32	1.98	24.1
68	-7020	2709	1442.53	1.77	1E+31
69	-6625	1219	1442.18	1.92	24.48
70	-6625	1075	1443.48	1.82	24.16
71	-6625	919	1440.25	1.85	26.44
72	-6625	769	1441.2	2.1	25.69
73	-6625	569	1442.33	2.07	26.35
74	-6625	369	1443.92	1.78	25.84
75	-6625	169	1444.56	1.64	25.68
76	-7020	2509	1443.56	1.54	24.52
77	-6800	2139	1444.44	1.56	24.09
78	-7000	2139	1445.09	1.91	24.65
79	-7020	2429	1444.22	1.78	24
80	-6775	2319	1440.05	1.95	24.9
81	-7200	2139	1445.16	1.84	24.83
82	-7400	2139	1441.59	1.91	24.62
83	-6980	3564	1432.68	1.82	24.45
84	-7180	3564	1432.73	1.77	24.45
85	-7380	3564	1431.14	1.86	25.62
86	-7580	3564	1426.15	1.85	24.61
87	-6040	3389	1440	1.7	24.12
88	-6040	3189	1440.44	1.86	24.49
89	-6270	3304	1438.46	1.54	24.76
90	-6175	3304	1439.57	1.43	25.32

APPENDIX 2.1 (CONT.)

91	-6270	3199	1439.47	1.53	24.95
92	-6175	3199	1440.63	1.37	25.01
93	-6270	3099	1440.5	1.5	24.46
94	-6170	3099	1441.51	1.59	25.01
95	-6170	2999	1441.57	1.43	24.41
96	-6170	1259	1445.71	1.39	24.9
97	-6170	1459	1448.1	1.6	25.11
98	-6170	1659	1447.3	1.5	25.27
99	-6170	1859	1447.08	1.62	24.81
100	-6170	2059	1444.85	1.65	24.29
101	-6170	1059	1439.13	1.87	24.1
102	-6170	659	1441.8	1.7	24.15
103	-6170	459	1443.92	1.58	24.12
104	-6170	259	1440.77	1.73	25.22
105	-6170	59	1440.23	1.77	25.13
106	-6170	2259	1445.43	1.57	24.12
107	-6170	2459	1443.27	1.73	23.83
108	-6500	2414	1443.48	1.52	24.01
109	-6400	2814	1440.01	1.49	24.48
110	-6300	2814	1441.61	1.39	24.33
111	-6300	2714	1443.36	1.64	24.68
112	-6300	2614	1444.36	1.64	24.07
113	-6500	2714	1443.45	1.55	23.84
114	-6500	2614	1444.38	1.62	23.98
115	-6300	2414	1443.48	1.52	24.08
116	-6500	2314	1443.3	1.7	24.27
117	-6300	2314	1444.27	1.73	24.37
118	-6500	2214	1444.37	1.63	24.48
119	-6300	2214	1444.32	1.68	24.4
120	-6500	2109	1442.82	1.68	24.56
121	-6300	2109	1444.25	1.75	24.66
122	-6500	2009	1444.23	1.77	24.79
123	-6300	2009	1444.31	1.69	24.82
124	-6500	1909	1444.42	1.78	24.6
125	-6560	1909	1444.95	1.55	24.95
126	-6500	1809	1444.12	1.88	24.93
127	-6300	1809	1445.86	1.64	24.11
128	-6500	1709	1446.4	1.8	25.05
129	-6300	1709	1446.58	1.62	25.08
130	-6500	1609	1447.27	1.73	25.39
131	-6300	1609	1447.47	1.73	24.8
132	-6500	1509	1448.18	1.82	25.11
133	-6300	1509	1448.38	1.62	24.96
134	-6500	1409	1449.32	1.68	24.73
135	-6300	1409	1448.73	1.77	24.98
136	-6500	1309	1446.24	1.86	25.21
137	-6300	1309	1446.37	1.73	24.72
138	-6500	1209	1443.19	1.81	24.59
139	-6300	1209	1441.25	1.85	24.49
140	-6500	1109	1441.2	2.2	24.62
141	-6300	1109	1439	2	24.56
142	-6500	109	1437.59	2.21	24.96
143	-6300	109	1439.16	1.84	25.14
144	-6500	909	1440.02	1.98	24.94
145	-6300	909	1439.78	2.22	25
146	-6500	804	1441.33	1.97	25.29
147	-6300	804	1441.22	1.78	24.43
148	-6500	704	1442.05	1.95	25.62
149	-6300	704	1441.53	1.77	24.86
150	-6500	549	1443.12	1.88	25.16
151	-6300	549	1442.88	1.72	24.9
152	-6500	399	1445.1	1.9	25.25
153	-6300	399	1443.78	1.62	24.51
154	-6500	249	1444.38	1.62	24.78
155	-6300	249	1443.37	1.63	25.05
156	-6500	99	1446.06	1.64	25.13
157	-6300	99	1447.04	1.66	24.44
158	-6500	-51	1443.3	1.7	26.38
159	-6300	-51	1439.32	1.78	26.73
160	-6450	2924	1443.3	1.7	24.56
161	-6450	3014	1440.3	1.7	24.15
162	-6450	3114	1439.2	1.8	24.09
163	-6500	2924	1443.12	1.88	24.36
164	-6300	2924	1441.5	1.5	23.87
165	-6500	3014	1440.05	1.95	24.58
166	-6300	3014	1441.32	1.68	24.09
167	-6500	3114	1439.05	1.95	24.52
168	-6370	3114	1439.3	1.7	24.65
169	-6500	3214	1440.18	1.82	24.99
170	-6370	3214	1439.18	1.82	24.62
171	-6370	3314	1438.15	1.85	25.41
172	-6300	3719	1435.16	1.84	24.42

APPENDIX 2.2

SEMIVARIOGRAM RESULTS AND MODEL PARAMETERS

CALORIFIC VALUE

M O D E L — OMNIDIRECTIONAL

	Pairs	Avg Distance	Value
1	142	97.512	.095
2	581	186.402	.153
3	713	304.006	.178
4	904	418.395	.203
5	952	538.418	.213
6	965	659.762	.217
7	935	780.482	.218
8	837	900.189	.229
9	806	1019.248	.201
10	758	1138.382	.219
11	713	1258.857	.217

Model

Nugget : .010

Type	Sill	Range
Exponent.	.210	500.000

M O D E L — DIRECTION: 90° (east-west)

	Pairs	Avg Distance	Value
1	41	96.779	.083
2	111	178.911	.127
3	74	303.354	.177
4	57	423.691	.180
5	43	539.181	.194
6	36	658.688	.179
7	28	777.716	.193
8	23	895.875	.154
9	18	1020.038	.096
10	12	1138.197	.069
11	9	1255.189	.094

Model

Nugget : .010

Type	Sill	Range
Exponent.	.185	500.000

M O D E L — DIRECTION: 45° (northeast-southwest)

	Pairs	Avg Distance	Value
1	31	129.107	.130
2	207	233.791	.165
3	255	378.882	.199
4	252	523.315	.227
5	231	673.717	.214
6	177	818.872	.225
7	132	968.547	.187
8	99	1124.214	.146
9	51	1269.360	.148

Model

Nugget : .010

Type	Sill	Range
Exponent.	.220	550.000

M O D E L — DIRECTION: 0° (north-south)

	Pairs	Avg Distance	Value
1	90	98.599	.103
2	152	194.887	.180
3	326	357.251	.184
4	334	497.541	.214
5	404	629.665	.205
6	469	770.743	.221
7	433	909.772	.260
8	525	1047.385	.225
9	481	1189.500	.249

Model

Nugget : .010

Type	Sill	Range
Exponent.	.210	500.000

M O D E L — DIRECTION: 135° (southeast-northwest)

	Pairs	Avg Distance	Value
1	9	111.662	.086
2	183	204.062	.153
3	152	325.741	.183
4	133	448.870	.195
5	113	576.817	.235
6	91	710.609	.218
7	58	843.825	.250
8	46	969.601	.267
9	40	1104.877	.240
10	35	1243.714	.282

Model

Nugget : .010

Type	Sill	Range
Exponent.	.200	500.000

APPENDIX 2.2 (CONT.)

SEAM THICKNESS

MODEL — OMNIDIRECTIONAL

	Pairs	Avg Distance	Value
1	129	95.764	.012
2	492	175.915	.017
3	627	278.834	.021
4	806	383.781	.021
5	881	493.987	.021
6	897	604.676	.023
7	887	715.673	.023
8	825	824.387	.023
9	764	932.151	.024
10	762	1041.067	.023
11	741	1155.167	.025

Model

Type	Sill	Range
Nugget:		.005
Spherical	.017	350.000

MODEL — DIRECTION: 90° (east-west)

	Pairs	Avg Distance	Value
1	87	115.148	.015
2	153	214.540	.023
3	233	347.909	.024
4	215	488.648	.022
5	186	623.561	.020
6	194	765.367	.024
7	165	909.101	.025
8	123	1048.388	.026
9	109	1184.643	.037

Model

Type	Sill	Range
Nugget:		.005
Spherical	.019	280.000

MODEL — DIRECTION: 45° (northeast-southwest)

	Pairs	Avg Distance	Value
1	26	125.357	.013
2	205	229.215	.017
3	229	364.971	.022
4	268	504.880	.023
5	239	656.462	.024
6	188	797.914	.028
7	141	941.852	.029
8	99	1083.809	.024
9	70	1224.112	.027

Model

Type	Sill	Range
Nugget:		.005
Spherical	.019	550.000

MODEL — DIRECTION: 0° (north-south)

	Pairs	Avg Distance	Value
1	87	97.202	.012
2	138	191.044	.016
3	176	344.489	.019
4	102	483.122	.022
5	113	602.644	.024
6	139	748.504	.025
7	90	884.193	.020
8	96	1008.063	.019
9	121	1153.174	.022

Model

Type	Sill	Range
Nugget:		.008
Spherical	.016	650.000

MODEL — DIRECTION: 135° (southeast-northwest)

	Pairs	Avg Distance	Value
1	5	105.830	.017
2	134	184.537	.021
3	177	287.853	.021
4	216	401.680	.022
5	232	519.282	.021
6	224	633.437	.022
7	232	744.005	.025
8	200	856.941	.023
9	186	975.040	.021
10	174	1092.403	.025
11	158	1207.970	.023

Model

Type	Sill	Range
Nugget:		.005
Spherical	.017	230.000

APPENDIX 2.2 (CONT.)

SEAM ELEVATION

M O D E L

OMNIDIRECTIONAL

	Pairs	Avg Distance	Value
1	129	95.764	1.435
2	492	175.915	2.366
3	627	278.834	3.388
4	806	383.781	4.470
5	881	493.987	6.496
6	897	604.676	7.295
7	887	715.673	8.133
8	825	824.387	9.812
9	764	932.151	10.555
10	762	1041.067	11.835
11	741	1155.167	12.201

Model

Nugget : .250

Type	Sill	Range
Linear	10.000	900.000

M O D E L

DIRECTION: 90° (east-west)

	Pairs	Avg Distance	Value
1	64	107.559	.669
2	149	192.468	2.304
3	206	322.476	2.325
4	217	456.519	2.528
5	183	584.403	3.040
6	181	719.497	3.814
7	153	847.180	5.495
8	130	971.322	10.243
9	123	1103.607	9.394
10	94	1238.001	10.511

Model

Nugget : .000

Type	Sill	Range
Spherical	2.350	180.000

M O D E L

DIRECTION: 45° (northeast-southwest)

	Pairs	Avg Distance	Value
1	31	129.107	1.012
2	209	233.967	2.518
3	258	378.926	4.000
4	258	523.364	5.578
5	239	673.515	6.509
6	187	819.488	7.065
7	139	968.924	7.128
8	103	1125.121	6.651
9	52	1268.802	7.831

Model

Nugget : .000

Type	Sill	Range
Spherical	7.500	820.000

M O D E L

DIRECTION: 0° (north-south)

	Pairs	Avg Distance	Value
1	87	97.202	1.766
2	147	189.447	3.106
3	161	307.271	5.917
4	285	417.100	8.005
5	355	541.519	7.940
6	367	660.975	9.294
7	390	783.085	9.644
8	393	902.754	10.049
9	414	1018.104	10.918
10	426	1138.909	12.487
11	435	1260.252	12.955

Model

Nugget : .000

Type	Sill	Range
Spherical	8.600	500.000

M O D E L

DIRECTION: 135° (southeast-northwest)

	Pairs	Avg Distance	Value
1			
2	42	137.559	2.115
3	108	204.300	2.129
4	126	278.128	4.044
5	144	359.953	5.501
6	145	438.895	5.663
7	160	518.816	7.265
8	159	597.654	8.879
9	169	678.332	9.537
10	158	757.500	11.916
11	151	838.172	13.509
12	129	920.219	15.139
13	124	1000.322	14.676
14	122	1081.114	17.571
15	102	1155.315	18.966
16	114	1236.280	23.445

Model

Nugget : .000

Type	Sill	Range
Spherical	6.500	500.000

APPENDIX 2.3

ESTIMATION ERROR RESULTS

GRID X x Y	SAMPLING DENSITY /Ha	CALORIFIC VALUE BLOCK 200m x 200m (Block variance 0.173) (Within-block variance 0.047) KRIG.ERR. VARIANCE (MJ/kg)^2	95% CONFIDENCE ESTIMATION ERROR MJ/kg	% of 24.77	CALORIFIC VALUE BLOCK 200m x 20m (Block variance 0.185) (Within-block variance 0.035) KRIG.ERR. VARIANCE (MJ/kg)^2	95% CONFIDENCE ESTIMATION ERROR MJ/kg	% of 24.77	CALORIFIC VALUE BLOCK 100m x 100m (Block variance 0.190) (Within-block variance 0.030) KRIG.ERR. VARIANCE (MJ/kg)^2	95% CONFIDENCE ESTIMATION ERROR MJ/kg	% of 24.77	CALORIFIC VALUE BLOCK 10m x 10m (Block variance 0.208) (Within-block variance 0.012) KRIG.ERR. VARIANCE (MJ/kg)^2	95% CONFIDENCE ESTIMATION ERROR MJ/kg	% of 24.77
100x100	100.00	0.004	0.12	0.50	0.010	0.20	0.79	0.010	0.20	0.79	0.025	0.31	1.25
200x100	50.00	0.010	0.20	0.79	0.018	0.26	1.06	0.017	0.26	1.03	0.033	0.36	1.44
200x200	25.00	0.017	0.26	1.03	0.028	0.33	1.32	0.031	0.35	1.39	0.048	0.43	1.73
250x250	16.00	0.027	0.32	1.30	0.038	0.38	1.54	0.042	0.40	1.62	0.060	0.48	1.94
400x200	12.50	0.042	0.40	1.62	0.053	0.45	1.82	0.059	0.48	1.92	0.077	0.54	2.20
400x400	6.25	0.057	0.47	1.89	0.070	0.52	2.09	0.074	0.53	2.15	0.093	0.60	2.41
500x250	8.00	0.058	0.47	1.91	0.071	0.52	2.11	0.075	0.54	2.17	0.095	0.60	2.44
750x250	5.33	0.086	0.57	2.32	0.099	0.62	2.49	0.104	0.63	2.55	0.127	0.70	2.82
500x500	4.00	0.076	0.54	2.18	0.089	0.58	2.36	0.094	0.60	2.43	0.112	0.66	2.65
500x500I	4.00	0.085	0.57	2.31	0.093	0.60	2.41	0.103	0.63	2.54	0.122	0.68	2.76
600x600	2.78	0.093	0.60	2.41	0.106	0.64	2.58	0.111	0.65	2.64	0.129	0.70	2.84
1000x1000	1.00	0.144	0.74	3.00	0.158	0.78	3.15	0.163	0.79	3.19	0.181	0.83	3.37
MAX.		0.173	0.82	3.29	0.185	0.84	3.40	0.190	0.85	3.45	0.208	0.89	3.61

GRID X x Y	SAMPLING DENSITY /Ha	SEAM THICKNESS BLOCK 200m x 200m (Block variance 0.015) (Within-block variance 0.008) KRIG.ERR. VARIANCE (m)^2	95% CONFIDENCE ESTIMATION ERROR m	% of 1.76	SEAM THICKNESS BLOCK 200m x 20m (Block variance 0.016) (Within-block variance 0.007) KRIG.ERR. VARIANCE (m)^2	95% CONFIDENCE ESTIMATION ERROR m	% of 1.76	SEAM THICKNESS BLOCK 100m x 100m (Block variance 0.017) (Within-block variance 0.006) KRIG.ERR. VARIANCE (m)^2	95% CONFIDENCE ESTIMATION ERROR m	% of 1.76	SEAM THICKNESS BLOCK 10m x 10m (Block variance 0.018) (Within-block variance 0.005) KRIG.ERR. VARIANCE (m)^2	95% CONFIDENCE ESTIMATION ERROR m	% of 1.76
200x200	25.00	0.001	0.06	3.52	0.002	0.09	4.98	0.002	0.09	4.98	0.003	0.11	6.10
250x250	16.00	0.002	0.09	4.98	0.003	0.11	6.10	0.003	0.11	6.10	0.004	0.12	7.04
300x300	11.10	0.003	0.11	6.10	0.004	0.12	7.04	0.004	0.12	7.04	0.005	0.14	7.87
400x200I	12.50	0.003	0.11	6.10	0.004	0.12	7.04	0.004	0.12	7.04	0.005	0.14	7.87
500x250I	8.00	0.003	0.11	6.10	0.004	0.12	7.04	0.004	0.12	7.04	0.005	0.14	7.87
400x400	6.25	0.004	0.12	7.04	0.004	0.12	7.04	0.005	0.14	7.87	0.006	0.15	8.63
750x250I	5.33	0.004	0.12	7.04	0.004	0.12	7.04	0.005	0.14	7.87	0.006	0.15	8.63
500x500	4.00	0.005	0.14	7.87	0.005	0.14	7.87	0.006	0.15	8.63	0.007	0.16	9.32
500x500I	4.00	0.005	0.14	7.87	0.006	0.15	8.63	0.006	0.15	8.63	0.008	0.18	9.96
600x600	2.78	0.006	0.15	8.63	0.006	0.15	8.63	0.007	0.16	9.32	0.008	0.18	9.96
1000x1000	1.00	0.011	0.21	11.68	0.012	0.21	12.20	0.013	0.22	12.70	0.014	0.23	13.18
MAX.		0.015	0.24	13.64	0.016	0.25	14.09	0.017	0.26	14.52	0.018	0.26	14.94

GRID X x Y	SAMPLING DENSITY /Ha	SEAM ELEVATION BLOCK 200m x 200m (Block variance 6.870) (Within-block variance 1.130) KRIG.ERR. VARIANCE (m)^2	95% CONFIDENCE ESTIMATION ERROR m	% of 1441.60	SEAM ELEVATION BLOCK 100m x 100m (Block variance 7.432) (Within-block variance 0.568) KRIG.ERR. VARIANCE (m)^2	95% CONFIDENCE ESTIMATION ERROR m	% of 1441.60	SEAM ELEVATION BLOCK 10m x 10m (Block variance 7.943) (Within-block variance 0.057) KRIG.ERR. VARIANCE (m)^2	95% CONFIDENCE ESTIMATION ERROR m	% of 1441.60
200x200	25.00	0.248	0.98	0.07	0.512	1.40	0.10	0.919	1.88	0.13
400x200I	12.50	0.494	1.38	0.10	0.728	1.67	0.12	1.106	2.06	0.14
500x250I	8.00	0.915	1.87	0.13	1.325	2.26	0.16	1.718	2.57	0.18
750x250I	5.33	1.208	2.15	0.15	1.584	2.47	0.17	2.041	2.80	0.19
500x500	4.00	1.473	2.38	0.17	1.921	2.72	0.19	2.331	2.99	0.21
500x500I	4.00	1.765	2.60	0.18	2.226	2.92	0.20	2.703	3.22	0.22
1000x1000	1.00	4.505	4.16	0.29	5.045	4.40	0.31	5.548	4.62	0.32
MAX.		6.870	5.14	0.36	7.432	5.34	0.37	7.943	5.52	0.38

I = grid orientated according to anisotropy (and staggered in the case of 500x500 I)

APPENDIX 3.1

RULES FOR COMBINING VARIANCES

Rules for combining variances (Suite, 1965; Koch and Link, 1970; Hemala and Stewart, 1979), assuming a normal distribution to the given data:

1. For a product or sum of "n" variables, each with a variance V_n^2, the resulting variance (V_T^2) is given by:

   INDEPENDENT VARIABLES

   $$V_T^2 = V_1^2 + V_2^2 + ... + V_n^2 \quad \text{-------1}$$

   FULLY DEPENDANT VARIABLES

   $$V_T^2 = (V_1 + V_2 + ... + V_n)^2 \quad \text{-------2}$$

2. In the case of the result being a sum of "n" variables, the above calculation assumes all components are equally sized. Since this is seldom the case, the combined variance must be calculated using weighting coefficients as shown in "3" below.

3. For an average ($\bar{X}_T$) of a number of independent values (X_i, V_i^2), the resulting variance (V_T^2) is given by:

   $$V_T^2 = \frac{(N_1V_1^2 + N_2V_2^2 + ... + N_nV_n^2) + (N_1E_1^2 + N_2E_2^2 + ... + N_nE_n^2)}{N_1 + N_2 + ... + N_n} \quad \text{-------3}$$

   where $E_i^2 = (X_i - X_T)^2$

   N_i = size or weighting of a block (number of samples)

   and $$\bar{X}_T = \frac{N_1X_1 + N_2X_2 + ... + N_nX_n}{N_1 + N_2 + ... + N_n}$$

4. If a constant is added to each value (X_i), then the combined variance is unaffected.