

A REVIEW OF MINERAL EXPLORATION DRILLING

WITH PARTICULAR REFERENCE TO

SOUTHERN AFRICA

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ABSTRACT

The field of mineral exploration drilling is reviewed with particular reference to examples, and techniques practised, in Southern Africa. Drilling is the most definitive process in exploration and the most cost intensive. It is, therefore, imperative to insure that the maximum geological information available is obtained from a borehole to warrant the cost of drilling it. Methods and techniques of obtaining this information, at little additional expense, are described and reviewed. Non-core percussion and rotary drilled boreholes cost significantly less than diamond drill holes and, as a result, many more holes can be drilled for the money available. While the logging of cuttings is notoriously neglected in most exploration programmes, a great deal of information is available to the conscientious evaluator. Down-the-hole logging and sampling techniques improve the reliability of the borehole samples and provide rapid and inexpensive analyses and lithological data. Cored, or diamond drill boreholes, are the most versatile of the drilling methods available and provide the most reliable lithological and grade information. The handling, logging, sampling and storage of core and core data is discussed. For little additional expense, comprehensive and accurate borehole surveys can be conducted and oriented cores obtained from the borehole. The natural tendency of a borehole to deviate can be used to advantage with controlled drilling techniques. Multiple ore intersections are possible through controlled deflections from a master hole. Mathematical, statistical and simulation models are available to optimise borehole siting, spacing and grid dimensions.

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CHAPTER ONE

INTRODUCTION

1.1 SCOPE AND LIMITATIONS OF REVIEW

The aim of this dissertation is to review the field of exploration drilling techniques and practices, with particular reference to conditions in Southern Africa. It is presented as a document written by an exploration geologist and is intended for the use of exploration geologists. There is no intent to duplicate the standard reference on drilling by Cummings (1975), but rather to enlarge on it by concentrating on aspects pertaining to mineral exploration. As a result, technical considerations of interest to an engineer or driller are not included and the reader is referred to Cummings for this. Furthermore, owing to a limitation on space, certain highly specialised fields are touched on only briefly in this account. The largest of these is that of borehole geophysics. This is such a large topic that it warrants a dissertation on its own. Another such field is that of borehole logging which has so many ramifications pertaining to particular environments, that it falls outside the scope of this review. In these, and other cases, only introductory notes and a bibliography of the relevant literature is provided.

1.2 A BRIEF HISTORY OF EXPLORATION DRILLING IN SOUTHERN AFRICA

The first exploration borehole of any significant depth was drilled in 1870 for coal, near Pottsville, Pennsylvania. This was a steam driven diamond drill and penetrated to a depth of 228 m. Prior to this, the diamond drill had been used in blast hole drilling since about 1862 when Jean Rudolphe Leschot, a Swiss engineer living in Paris, conceived the idea of penetrating hard formations by using diamonds embedded in a rotating annular ring. (Cummings 1975).

Exploration diamond drilling in Southern Africa first made an impact in the Witwatersrand in the 1890's, when it became apparent that the auriferous Witwatersrand "banket" reefs extended to deeper levels than previously thought. The earliest recorded hole was started in 1889 on the Village Main Reef property and reached a final depth of 203 m. It intersected the South Reef at 158 m. and the core assayed 9 ozs. 12 dwts.

(298 g.) to the ton. At 177 m. it reached the Main Reef Leader and brought up samples that assayed 11 dwts. (17 g.). This was followed by the Rand Deep Levels borehole which intersected the Main Reef at 273 m., assaying at 13 ozs. 13 dwts. (425 g.). Soon after this, the Rand Victoria borehole reached the Main Reef at a staggering 729 m., followed by the Bezuiden-ville borehole in 1895 at a depth of 1136 m. In 1899, the "Turf Club Holes" intersected the Main Reef at depths between 1433 and 1494 m. (Cartwright 1967). The tremendous potential of the Witwatersrand was now becoming appreciated. After the success of the MacArthur-Forrest cyanide process for treating pyritic ore from below the water-table, core drilling started to boom.

These early holes were probably drilled by a Sullivan H steam powered machine using "A" size bits and, by 1902, holes had been drilled as far afield as Heidelberg and Klerksdorp. (Popplewell 1978). One of the earliest technical accounts of this drilling was by G.A. Denny. Entitled "Diamond Drilling for Gold and other minerals", it was published before his well-known "Deep-Level Mines of the Rand", by Crosby, Lockwood and Sons, London, 1902.

The next significant development in exploration drilling in Southern Africa was the discovery of the East Rand reef whilst drilling for coal! Further drilling saw the establishment of Geduld Mines in 1906. The reef did not outcrop on surface and only assays taken from the borehole cores of a sub-outcrop reef, 700 m. below surface, could be used for evaluating the property. (Popplewell 1978). This was later to become a major feature of Witwatersrand exploration.

Exploration drilling had now spread to coal and other minerals. Boreholes proved the continuity of the Platiniferous Merensky Reef of the Bushveld Complex, in depth, as they had done for the gold mines in the early days. In 1927, diamond drilling in the Copperbelt was started almost simultaneously by the E.J. Longyear Company and the Sullivan Machinery Company. These companies had been active in the United States since 1890 and 1880 respectively.

The application of geophysical techniques in exploration provided a new source of drilling targets in Southern Africa. 1930 saw the first results of these, when Rudolph Krahnemann indicated the continuity of the magnetic shale horizons of the Lower Witwatersrand succession beneath the cover

of the Transvaal dolomites, in the area which was later to become the West Wits Line. Drilling in 1932, and in the subsequent years, was based upon projections made from Krahmann's anomalies and 21 boreholes intersected payable reef. In doing so, the Carbon Leader and Ventersdorp Contact reefs, hitherto unknown on the Witwatersrand, were discovered. These phenomenal results, and the abandoning of the gold standard by South Africa in 1932, led to a renewed drilling boom and the discovery of the West Wits Line was soon followed by discoveries in the Klerksdorp area and the Orange Free State.

This new era of drilling brought about a resulting wave of technical change. In 1930, internal combustion engines began to replace the steam engines as the drillers' "power pack". By 1936, the "black carbon" or "carbonado" was replaced by "boarts" in the drill bits, entailing a significant price reduction. The use of these cheaper, smaller stones now made it possible, from the point of view of capital outlay, for a large number of crowns to be held on site. This in turn led to the manufacture of crowns in central workshops. Prior to this, crowns were "hand-set" on site by the driller or "setter". The impregnated bit was developed in 1935 and slowly became recognised for its longevity.

Core barrels were constantly being improved and tailored for local conditions. The Blair and Garret core barrels were designed specifically to improve core recovery in the soft, carbonaceous reef horizons of the Witwatersrand such as the Carbon Leader. Drilling methods and techniques became more orientated towards accuracy and core recovery as clients' demands for precision increased. The use of wedges and deflections to obtain further samples of the reef horizon became common practice during the development of the West Wits Line, Klerksdorp and O.F.S. goldfields. Similarly, the control of unwanted borehole deviation became necessary as holes progressed deeper into the Witwatersrand strata. The early calculations of Garret (1952) went some of the way to solving this problem.

A milestone was reached in 1938, when a diamond drill-hole (GMB.I. on Gerhardminnebron), passed the 3000 m. mark. Since then, boreholes have penetrated to 4500 m. in the Witwatersrand. South Africa led the way in deep diamond drilling techniques in the early days and still dominates the field today.

1.3 THE USE OF DRILLING IN MINERAL EXPLORATION

Drilling is the only means whereby qualitative and quantitative sub-surface geological information can be obtained short of actual mining development. In any exploration programme, some aspect of drilling is used at various stages throughout it. Prior to the selection of future areas for exploration, old records of drill holes are consulted and cores re-logged or re-evaluated if still available. During regional reconnaissance exploration, drilling is used to provide stratigraphic and lithological information from which target areas may be chosen. These are then investigated by more detailed drilling which may be based on geophysical and geochemical anomalies. These anomalies should be explained by interpreting cores recovered from the drill hole. During this phase it is the drilling programme which provides the three dimensional stratigraphic and structural solution to anomalies in the surface geology. From the fresh unweathered cores; mineral zoning, alteration phenomena, facies variations and geochemical features can provide information that enables the geologist to delineate the geometry and dimensions of a prospective ore body more accurately, or to decrease the size of a target area. This normally necessitates follow-up drilling when grade determinations from core becomes of prime importance. If an ore body is indicated, further drilling for evaluation purposes follows. This entails sufficient drilling to delineate the ore body fully and establish its economic viability. This is normally the last phase of exploration drilling as such, but drilling techniques are still used to block out ore reserves, provide bulk samples for laboratory testing, and for structural and stratigraphic information for mine layout and development planning. Throughout the life of the mine, exploration drilling is used to increase ore reserves and establish on-strike and down-dip extensions to the known ore bodies. From all this drilling information, the mine geologist can build up a comprehensive three-dimensional picture of the mine. The final stages of a mine's life are often typified by an urgent drilling programme to establish more ore reserves before the known reserves are mined out.

1.4 THE COST AND VALUE OF BOREHOLE INFORMATION

Until the recent energy crisis, the cost of drilling per metre had not increased greatly in the last 20 years. This was because of increased efficiency and technological improvements which enabled greater penetration

rates and less time wasted on hoisting and lowering rods. However, in 1978 and 1979, costs escalated dramatically after the sharp increase in fuel prices and it is now apparent that the value of drill hole information must be carefully weighed against the cost and drilling method used.

There is little doubt that diamond drill core provides the best sub-surface geological information available (if logged properly). This core-log includes, in the form of a description, lithological and mineralogical data and information pertaining to fracture patterns, mineralisation etc. The assay log should contain reference to hole size, core and sludge recovery and density of the sample, as well as individual assays with combined and weighted composite intervals. The driller's log should contain a record of the technological aspects, such as bit life, drilling fluid loss and depth and date of each core run. From all this, a complete record is available of the geology and drilling conditions. Furthermore, if a split-core is sampled, the remainder of the core should always be kept for future reference and a permanent record.

In comparison, the non-core rotary and percussion drilling methods do not provide as much detailed information but this is compensated for (to some extent) by significantly lower costs. A number of holes can be drilled for the cost of one diamond drill hole. The cuttings of these holes should be examined and sampled carefully and it is amazing how much information can be obtained if this is done diligently. Unfortunately, this aspect is generally neglected and valuable information is irretrievably lost. "In-situ" logging and sampling methods can be used in conjunction with chip sampling and logging, increasing the value and reliability of the information. With these methods now generally available and the increased scope offered by the cheaper, non-core drilling, the value of geological information in relation to cost-outlay is becoming more compatible. A combination of coring and non-coring methods can provide a good spread of holes, and enough detailed information from the cored holes to extrapolate in the non-cored holes at a reasonable cost.

It is now becoming essential to ensure that the maximum geological information is obtained from a drill hole to warrant the cost of drilling it. Thus, detailed surveying is necessary to know exactly where the borehole has traversed and which points, in a spatial sense, have been sampled. Furthermore, directional control of the borehole track is worth the slightly extra expense involved. Down-the-hole logging methods can

be considered for cored and non-cored holes to provide additional information over a larger sample area.

The additional expense of core orientation methods and optical down-the-hole methods to establish structural orientation should also be considered. All drill hole collars should be accurately surveyed and elevated to a datum. Drill hole collars should be plugged and casing left in the hole if there is the smallest chance that the hole might be deepened at some time. No expense should be spared in storing borehole core for future reference. If the cost of suitable core sheds is calculated against the value of the core in rands per metre, and the value of the information the core contains, it is negligible. It is an unfortunate fact that, all too often, drill core is disposed of, or left in the field, at the termination of a drilling programme.

Obtaining maximum information from a drill site often entails multiple intersections of an ore body from one site involving deflections or "arc-cutting" from a central hole. This is a particularly good technique where target areas are deep and where the collaring of holes would mean thousands of metres of extra drilling. Arc-cutting is also useful where more samples are required than would be furnished by a single intersection.

The value of information obtained from an exploration drilling programme is seldom appreciated until well after the programme has ceased. For this reason it is imperative that the maximum information available must be collected at a little extra expense during the drilling programme itself.

The techniques for gaining maximum geological information from drill holes form the context of the following chapters of this account. The final chapter looks at methods of establishing the optimum number of holes and their geometrical arrangements to provide optimum information.

CHAPTER TWO

NON-CORE DRILLING

2.1 PERCUSSION DRILLING

The compressed-air hammer drill or rock drill, normally used for conventional blast hole drilling, is a popular method in exploration for outlining shallow ore bodies by providing quick, inexpensive samples of finely broken rock chips in a stream of air or water. The drill rigs are light and manoeuvrable but connected to a heavy compressor. Drilling costs are low, progress is fast, and hard, abrasive rocks are penetrated more easily than by other drilling means.

In percussion drilling the rock is fragmented by repetitive impaction. The drill consists of a piston which gives a series of impacts to the drill rods, transmitting them in the form of shock waves travelling at approximately 5000 metres per second (Atlas Copco Manual), to the drill bit. This contains either buttons or insets of cemented tungsten carbide (see Fig. 1), and is rotated slowly to re-index the bit between impacts. In this way, a fresh rock surface is presented for each blow.



Fig. 1. Percussion drill bits. (a) "button" type, (b) "cross" type.

As the depth of the hole increases, additional drill rods are connected with sleeves to the rod controlling the bit. Increasing length and number of joints causes a loss of energy being transmitted to the drill bit. In the first joint, the loss is in the order of 8-10% but decreases markedly for each succeeding joint. The point is reached finally when all the energy is absorbed in the drill string and penetration ceases. Thus

this method of drilling has clearly defined depth limitations.

The wagon drill is the most common of the percussive drilling machines used in exploration. It is a relatively cheap, lightweight machine that can drill holes at most inclinations. The drilling capacity is limited to short holes approximately 30 metres deep and can only operate efficiently in dry ground above the water table. Compressed-air is forced down a small diameter hole in the centre of the drill steel to cool the drilling bit and free the cuttings from the drilled face. These cuttings are carried to the surface by the return air travelling up between the drill steel and the side-walls of the drill hole.

A more sophisticated version of the wagon drill is the "crawler" or "track" drill which is self propelled and can tow its own compressor behind it. These cost significantly more than the standard wagon drill but can drill larger diameter holes and to greater depths. In this case the depth is limited by the ability of the return air to carry the cuttings to the surface.

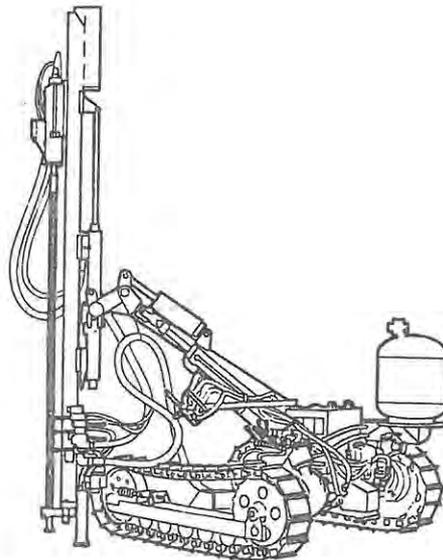


Fig. 2. Self propelled "crawler" or "track" drill.

A more efficient drilling method for deeper holes is the "down-the-hole hammer". In this case the pneumatic hammer is cylindrical in form and is located inside the drill hole immediately behind the bit. The piston strikes the bit directly, so that the pulses only travel a short distance

and do not suffer attenuation through the drill string. The drill string, complete with hammer, is rotated from outside the hole to re-index the bit. (McGregor 1967).

Down-the-hole hammer drills have many advantages over conventional percussion drills in exploration work. In the latter, the speed of the air travelling up the annulus between the drill string and the sides of the hole is limited by the small quantity of air that can be passed downwards through the hollow drill steel. However, in down-the-hole drills, the amount of air required to operate the hammer at the bottom of the hole, is greater. This additional volume of air is then available for cuttings removal and transport. The annulus between the drill string and the borehole wall is much narrower and as a result the higher air velocity up the annulus results in better sample recovery and less opportunity for salting. (Muhlhaus 1968).

Pneumatic drills in the mining industry are now superceded in efficiency by hydraulic rigs. These have even greater advantages for exploration drilling as the compressed air supply is no longer used for penetrative power but purely for drilling and returning the cuttings.

One of the oldest forms of the percussion drill is the "churn" or "cable-tool" drill. The cable-tool rig raises and lowers a heavy steel spud with a cutting edge. The cuttings are removed by a bailer. These drills are extensively used for water drilling and have been used for oil-well drilling in the past. They have generally been superceded by large rotary drilling equipment. Churn drills have some unique advantages for exploration work. Given time, a churn drill can obtain sample fragments by chopping its way through almost any kind of ground; unconsolidated, heterogenous, hard or completely fractured ground. Casing is advanced behind the drill and after each sample interval. Its widest application in exploration work has been in alluvial sampling where the alternation of hard and soft material in boulder beds renders conventional drilling and sampling very difficult. Churn drilling is restricted to vertical holes and is occasionally used for collaring and casing vertical diamond drill holes through difficult near-surface conditions. (Lissiman 1965). The results of an exploration programme for gold tellurides in Tertiary volcanics on Fiji using churn drilling is documented by Denholm (1965). A good correlation of sample results between development and drill hole data was established. The environment was unsuitable for diamond

drilling because of extremely poor core-recovery in the deeply weathered volcanics.

2.2 ROTARY DRILLING

Rotary drilling has now virtually replaced churn drilling in the need for deep, straight, large diameter exploration holes. The principal use of such holes is for geophysical and radiometric logging and is used widely in exploration for coal, lignite, uranium and phosphate. The drill rigs are heavy, normally mounted on trucks and have a limited ability to drill angled holes. Plain rotary drilling, using a drag bit, is the fastest and most economical method of drilling where it can be employed. A drag bit usually has two or three wings faced with tungsten carbide and is rotated whilst in contact with the rock surface. Thrust is required to make the bit penetrate and rotation shears the rock particles in front of the cutting face. (McGregor 1967).

The cuttings are removed from the hole by suspension in the circulating medium which can be air, vacuum, mud or water. Rock-type and mineralogy can be identified to a fair extent from these cuttings. Dry cuttings are preferred for sampling purposes but as this is not always possible, wet cuttings can be collected in settling tanks, cyclones, vibrating screens or filters. Short, but very expensive core-runs can be obtained by using a special core-bit in place of the standard roller-type rock bit or "fish-tail" drag bit. Wire-line coring can be done with heavy oil-field rotary equipment but it cannot normally be done with the equipment designed for shallower and smaller diameter exploratory holes. (Peters 1978).

"Down-the-hole" rotary equipment utilises a fluid or air driven motor located near the bottom of the drill string. This prevents flexing and whip-lashing of the drill string, provides faster penetration rates and more efficient cleaning of the fresh rock face. Because the rods do not turn they can be fitted with a deflecting "sub" which may be orientated in a chosen direction to make a hole deviate as much as 15° per 100 m. from its original path.

Rotary drilling for large diameter and deep holes, such as those used in the oil industry, employ a rock roller bit which incorporates three or more cones set with teeth or buttons, each capable of rotating independently. As the drill rods are rotated from above, the teeth or buttons engage with the rock surface and each cone is caused to rotate in the

reverse direction to the rods. The drill hole is made by thrusting the teeth into the rock so that the drill must be capable of tremendous down pressure in the range between 200-1200 kg/cm. The hole size varies from 117mm-458mm. (Peters 1978).

2.2.1 Applications of rotary drilling techniques in exploration

The application of rotary oil-field drilling equipment to mineral exploration has been advocated by Eyde (1974). In a deep drilling programme looking for the extension of the Magma vein-fault, Superior, Arizona, two holes were drilled with such equipment and the results compared to those of conventional diamond drilling in the area. The obvious advantage of the former was the considerably faster drilling rate with a penetration of 1570 m. in 21 days. The result was very straight, vertical holes from which cores were obtained in selected horizons. Down-the-hole oil-field logging techniques and dip-metre reading provided information in non-cored zones. The main advantage in this particular case was the speed of the entire exploration programme which saved the company a considerable amount of money in pending option payments. By completing exploration holes in the shortest possible time many of the costs (such as land payments, staff salaries and expenses, office and core storage rentals), associated with a deep drilling project can be eliminated. Much of the time lag between the exploration and production phases at many mineral deposits results from the slow diamond drilling rates. Exploration takes longer than it should, adding excessive amounts of time and expense to the project. This has the effect of reducing the payout and the discounted cash flow of the project during production. (Eyde 1974).

The ability of normal rotary equipment to drill fast, straight, vertical holes is utilised in Witwatersrand gold exploration to penetrate the cavernous Transvaal dolomite succession overlying much of the Witwatersrand quartzites. The stratigraphy of the dolomite succession is well established and cased holes are necessary to prevent drilling fluid loss and loss of the borehole itself in caverns. Rotary drilling, including the cost of casing, is cheaper, faster and straighter than conventional diamond drilling in this operation. Diamond drilling is continued into the quartzites from the base of the rotary hole beneath the dolomites.

The Tsumeb Corporation, drilling in the dolomites of the Otavi Mountainland in South West Africa, use rotary drilled pilot holes in much the same way. Here it was found that the diamond drill holes deflected

strongly in a direction normal to the dip of the dolomite and interbedded chert layers. The exploration targets were steeply dipping tabular bodies at considerable depth and were difficult to intersect with the deviating diamond drill holes. To counteract this, a large diameter rotary drill was used to pilot the holes to a point close to the target. This resulted in relatively straight holes and a number of controlled diamond drill deflections were drilled from the bottom of this hole a short distance to the target.

2.2.2 Reverse-circulation rotary drilling

In conventional rotary drilling the flushing medium (air, water, foam, mud etc.) travels down the inside of the drill string, lubricates the drilling bit, and then returns to the surface in the annulus between the drill string and the borehole wall, carrying the cuttings with it. The main disadvantage of this system is that the sample can become contaminated from contact with the borehole wall. Irregularities in the borehole wall cause loss of sample material, settling out of heavy particles and inter-mixing of heavy and finer particles from different samples. To counteract these disadvantages, the reverse-circulation method was developed.

In reverse-circulation drilling, the fluid is pumped down between the drill string and borehole wall, returning up inside the drill string and carrying the cuttings with it. This is known as "single-tube" reverse-circulation drilling and was the earliest attempt at solving the sampling problem. This technique was impractical for drilling through relatively unconsolidated material, and even in more solid ground there is a danger that the media would pick up material from the sides of the holes before reaching the cutting head. This would result in a contaminated sample. Drilling fluid can still be lost in porous ground conditions.

The improved version, the "dual wall" method was evolved to counteract these deficiencies. In this method the fluid follows a course down the space between an inner and outer pipe (i.e. the "dual walls"), mixes with the cuttings at the bit head and passes as a slurry or solid fluid suspension through the centre of the drill string. This system is highly suitable for use in alluvial ground as it ensures that high constant recovery is achieved with little or no contamination. This is due to the fluid passing through the annular space within the drill pipes prior to mixing with the cuttings. (Mining Magazine Nov. 1969).

Reverse-circulation rotary drilling is gaining widespread acceptance as a reliable sampling method. This is particularly true in difficult and unconsolidated ground conditions. Irvine et al (1974) compare the results of sampling from a number of drilling methods in the evaluation of lignite resources in Saskatchewan. These were: (i) normal rotary drilling, (ii) reverse-circulation drilling with dual-wall pipe, (iii) continuous rotary core drilling, and (iv) side-wall coring.

Samples from these various drilling methods were compared to the values obtained from the side wall of a nearby open mine face and in all cases, the dual-wall reverse-circulation method gave the highest correlation and best recovery.

In another documented example, Sagi (1977), of the Iron Ore Company of Canada, reports that the dual-tube reverse-circulation method obtained the most representative samples of a soft, friable iron ore deposit. The following drilling methods had been tried in the past: (i) standard diamond drilling, (ii) chop and drive drilling, (iii) standard rotary drilling, (iv) mud drilling, (v) coring with fuel oil, (vi) triple-tube wire-line coring, and (vii) the dual-tube reverse-circulation rotary drilling. Due to the unconsolidated nature of the iron ore, contamination from the side walls, loss of sample material, low core recovery, poor hole stability and high consumption of drilling bits typified all the other methods used.

The most representative samples were obtained from the reverse-circulation drilling methods. In a follow-up programme, the chemical analysis of these samples was compared with the actual shipping samples and the iron and silica levels were found to be within the accepted tolerance level of one per cent. The system used is described in detail by Sagi (1977). It consisted basically of a rotary dual-tube, reverse-circulation drilling rig using polymer mud as a flushing medium and a mechanical sample splitter.

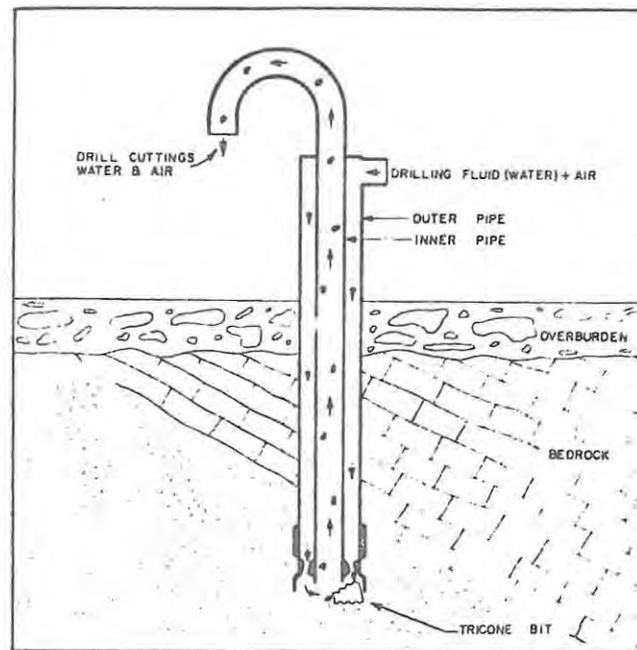


Fig. 3. Schematic outline of the Reverse-Circulation Process (after Sagi 1977).

Reverse-circulation drilling using air as a circulating medium is known as "vacuum drilling". A venturi effect is created behind the face of a specially designed tungsten carbide bit which "airlifts" the sample through the inner tube and flushes it to the surface. The sample is collected in plastic bags via a cyclone. This is the basis of the "Wallis Aircore Sampling System" (Hopley et al 1978). The vacuum drill is ideal for holes of up to 200 metres and for use in arid regions where sufficient quantities of water are not available. In water-logged formations, the vacuum principle is not applicable, but the equipment can still be operated as a conventional rotary drill with mud or water circulation. (Lissiman 1965).

2.2.3 Auger drilling

Auger drilling with continuous flight augers and tungsten carbide drag, or finger bits, is a fast and economical method of obtaining samples from soft formations, - free of boulders, and able to stand without caving. As there is no fluid circulation, fissures and cracks do not affect progress or sample reliability. For testing beach sands, alluvial deposits without boulder beds, unindurated bauxite, mine dumps and other deposits of similar unconsolidated nature, auger drills are extremely popular. (Lissiman 1965).

When using an auger, the hole is advanced in stages equivalent to the auger length and the auger is then withdrawn from the hole allowing recovery of

the disturbed soil from the flights of the auger. Since the hole is continually cleaned as it is drilled, each auger full of soil represents a more or less accurate, although highly disturbed sample, from a given length of hole. As the continuous flight auger drills into the earth, additional auger sections are added until the required depth is reached. The flights of the auger act as a screw conveyor bringing cuttings to the top of the hole. Samples can be collected from these cuttings or from the material adhering to the cutter head which is then frequently pulled out of the hole. Small diameter auger holes range from 75 mm. to 355 mm. in diameter. The most commonly used size is probably 100 mm. (Nowlan 1978).

2.2.4 Combination of rotary and percussion methods

The OD method of drilling through difficult overburden conditions involves a combination of rotary and percussion methods. The equipment consists basically of an outer casing tube with a ring of cemented carbide at the lower end. The casing tube encloses an inner drill string made up of standard drill steels with a cross-bit. The casing tubes and the inner drill strings are of the same length and are joined by coupling sleeves independent of one another. The whole system is connected to the rock drill by a special shank adapter, which transfers both impact force and rotary force to the string of casing tubes and to the string of extension steels. (Atlas Copco Manual).

The advantage of this operation is that the casing is advanced immediately behind the bit. This prevents caving of the borehole wall and thus drilling is possible through totally unconsolidated overburden.

2.2.5 The ODEX overburden drilling method

This method is similar to the OD method described above in that it allows casing of the borehole to proceed concurrently with drilling. During drilling, a reamer on the ODEX bit swings out and reams a hole larger than the outer diameter of the casing tube. As drilling proceeds, the casing is driven, without rotation, into the hole, following closely behind the drilling bits. On completion of drilling, the direction of drilling is reversed briefly to swing the reamer to its position of minimum diameter, and the rod string may then be withdrawn from the casing. (Nowlan 1978).

2.3 SAMPLING AND LOGGING PROCEDURES IN NON-CORED BOREHOLES

2.3.1 Sampling wet drill cuttings

The sludge from diamond, churn, rotary and percussion drilling is caught in various devices, ranging from simple overflow buckets hung beneath percussion drill collars to elaborate multiple sludge tanks where all cuttings are carefully settled out of the drilling water. The cleared water from the sample is allowed to settle enough to siphon off for recirculation, and the sludge is either air dried or dried on steam tables.

As the quantity of water used in some drilling methods is very high, a series of interconnecting troughs is used. These trap most of the material by settling, with the overflow repeating the process in the next trough. Many variations of this system have been used in sludge collecting and some of these are demonstrated in Peele (1941, pages 10-40), Moehlman (1945) and Cummings (1975, p.278). One such method is the McDonald sludge collector which consists of a trough into which the sludge is pumped. Pipes, with valves, are set into the trough at three different levels. Clear water is drawn from the top valve, mixed sludge and water from the middle valve, and the sludge sample from the lower valve. This method requires that a number of these troughs are kept at the drill site as the sludge from the earlier samples must be allowed to settle while other samples are being taken.

In many instances it is not necessary to retain all the material pumped out of a borehole collar in a sample interval. This involves a tremendous amount of liquid and a number of settling containers. To overcome this, a representative sample can be collected from a continuous stream of sludge by a mechanical sample collector.

The sample wheel, or Thompson sludge cutter, is shaped like a small Pelton wheel and is activated by the flow of sludge from the drill hole. This takes a fractional sample and discharges it into a sample bag of fine nylon that allows very few of the cuttings to escape. (See Fig. 4).

This system was found to be unsatisfactory by Sagi (1977) when used with the dual-tube reverse-circulation method. It used high pressure, high volume air, with three to five gallons of water a minute and the wheel revolved so rapidly that a large proportion of the sample was thrown out of the retaining cup before it could run down to the bottom outlet of

the cup. In this case, the Drilco Sample Splitter (see Fig. 5), was preferable.

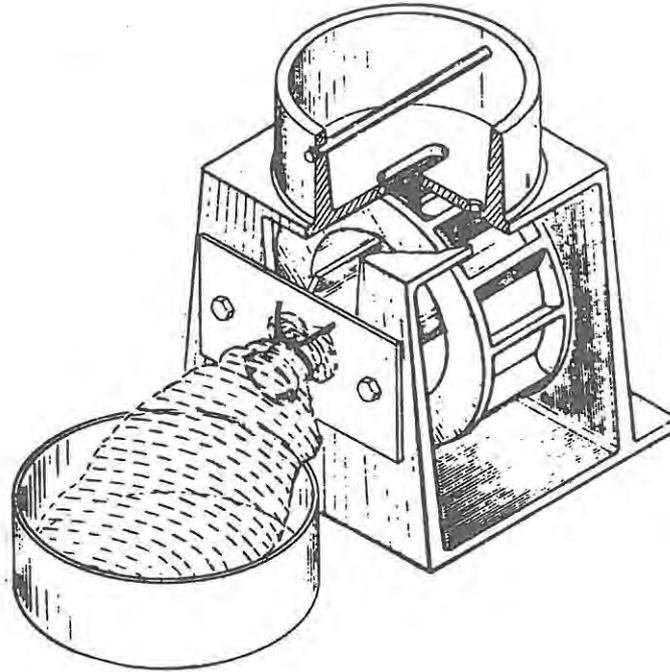


Fig. 4. The Thompson Sludge Cutter (after Cummings 1975).

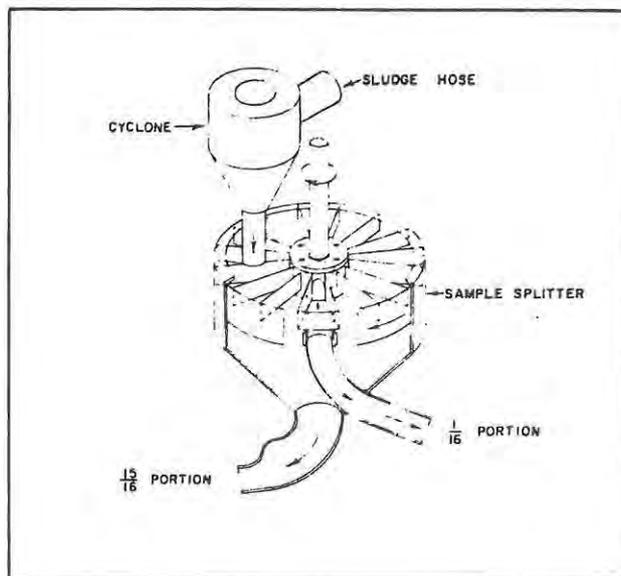


Fig. 5. The Drilco Sample Splitter (after Sagi 1977).

This is a horizontal wheel splitter which turns at a constant speed on a vertical axis. This wheel is divided into 16 equal portions. One of these divisions is designed to accept the sample while the remainder of the cuttings are discarded.

The sample obtained from any one of these types of sample splitters is in the form of a slurry. The water can be removed by bagging the sample in a fine cloth bag and allowing it to drain and dry slowly. Prior to this, it can be thickened in centrifuges or cyclones. The disadvantage of the latter is that the possibility of contamination increases with the number of times that the sample is handled mechanically, unless the equipment is very thoroughly flushed between samples. Settling rates of sample material in troughs or settling tanks can be increased by adding a flocculant such as "Seperan AP-30", which causes total flocculation of suspended matter in about 30 minutes. If, for some reason, samples have to be dried rapidly, they can be dried over a fire, but great care must be taken to ensure that the sample does not get too hot and cause oxidation of the material.

When drilling with a liquid circulating medium, allowance is made for "sample lag", i.e. the time taken for the cuttings to reach the drill collar. At a depth of 100 metres, the lag might be as much as half an hour and by the time the cuttings are sampled the drill may have penetrated 3 or 4 metres into a critical zone. A dangerous bias can occur in the sampling with this sample lag. The cuttings that arrive in a sample at the drill hole collar are composed of minerals with a wide range in densities and, therefore, with a range of travel time from the bottom of the hole. This is the most difficult form of contamination to detect. If very accurate samples are required through a critical zone, this problem can only be overcome by halting the drilling after each sample interval and flushing the drill hole thoroughly before drilling proceeds.

2.3.2 Sampling dry drill cuttings

The problems associated with obtaining a representative sample from percussion or rotary drilling using air as a circulatory medium depend on the method used. Mixing of sample material of different densities is a common cause of contamination and thorough flushing of the hole between each sample is necessary. Unlike the wet drilling techniques previously discussed, there is virtually no sample lag when air is used. A short flushing period after each sample interval should, therefore, insure

that contamination is reduced to a minimum. However, some of the material may come from the walls of the drill hole; some may accumulate in open fissures and vugs until it becomes incorporated in a later sample. Weighing the total volume of each sample and comparing the result against a calculated expected volume of the material, will soon establish if material has been lost or gained.

During drilling, a continuous stream of dry cuttings and dust is expelled from the borehole. In conventional rotary and percussion drilling this is collected from around the collar of the borehole. In reverse-circulation systems, it is collected from the inner tube. This stream of cuttings and dust is passed through a small cyclone which separates the solid material from the air. The solid material passes out of the base of the cyclone and is collected in various containers and in larger or smaller amounts depending upon the length of borehole sample required. While drilling in non-critical strata, this stream of cuttings from the cyclone is often laid out in small piles on the ground, each pile representing a certain borehole meterage. This is then logged and grab samples are taken when required. However, in the critical zone, the samples are carefully collected from the cyclone and immediately bagged to prevent loss or contamination of the sample. Lissiman (1965) records a method where these samples are continuously transferred into clear plastic tubes or cylinders of about the same diameter as the borehole. This forms a very accurate, visible and continuous granular "core" record from which changes in lithology and strata can be observed while drilling.

During sample collection, great care must be taken to prevent contamination or loss of sample material. After each critical sample interval, drilling should be stopped and the hole thoroughly flushed out until little or no dust or chips are recovered. If dust is allowed to dissipate and is not collected, the result will reflect higher than representative values from the sample interval due to the loss of material. It is inevitable that some dust will pass through the cyclone, but an efficient collection system will reduce this to a workable minimum level.

Sample splitting is usually necessary from dry drill cuttings to reduce the volume of material in each sample interval. A mechanical sampler can be used to remove a fixed proportion of sample material from a stream of dry cuttings in much the same manner as that already described for

the wet sludge samples in Figs. 4 and 5 on page 17. Large samples can be reduced by passing the cuttings through a sample splitter in which equal portions are separated, or in which a small portion is accepted and a larger portion is rejected. The most common type of splitter for dry material is the Jones riffle sampler, a container with channels discharging in two directions. By mounting several splitters in a "stairlike" arrangement, a series of reductions can be made in one pass.

Where a mechanical splitter is not available the old but effective method of "coning and quartering" can be successfully used to reduce a sample. The sample cuttings are mixed and shovelled into a conical pile on a clean, smooth surface. The pile is flattened into a disc and then separated into quarters with the edge of a shovel. Alternate quarters are accepted or rejected. The process is repeated with the accepted quarters until the sample is reduced to a suitable size. (Peters 1978).

2.3.3 Logging non-core drill cuttings

The drill cuttings are geologically logged during, or after, the collection and sampling process. When an important intersection is expected, the geologist will take small grab samples almost continuously from the stream of drill cuttings and pan them for heavy minerals or examine them closely for indicator minerals. This might determine at which point more accurate or detailed sampling techniques should be employed through a forthcoming "critical zone".

After the hole has been completed, the detailed geological logging of the sludge, or chips, can be undertaken. Again, this involves close scrutiny of the sample material through a hand lens or binocular microscope. A magnet (for magnetic susceptibility), and a scintillometer for radiometric identification are routinely used in certain environments. Chips should be logged "wet" versus "dry" to compare differences as some features are only shown up by one or the other of these conditions. The chips are carefully logged for lithological and mineralogical changes; structural aspects such as schistosity, foliation, mylonite or fault filling; and, most important, - mineralisation. The sample can be panned or jigged (to concentrate the heavy minerals), sieved or electromagnetically separated, if desired. Elementary chemical tests are carried out for identification purposes. The object of this is a coordinated comparison of chips from hole to hole and the surface outcrop position.

The information gained is carefully recorded on a log sheet. This should be constructed graphically in a similar manner to that of a diamond-drill log. A permanent record of the drill hole can be preserved by glueing rock chips or panned concentrates of heavy minerals to a "chip board" or "sludge board". Alternatively, samples can be preserved in long, clear plastic bags ("sausages"), or laid out in troughs (half piping or corrugated iron) as a continuous record from the borehole.

The degree of thoroughness of logging, recording and storing the data depends to a great extent on the "end use" of the data. If assay results only are required, as is often the case in large low grade deposits, thoroughness in logging becomes of secondary importance to accurate sample collection. However, if attaining the maximum possible geological information is the aim of the borehole, then logging and sampling become a very important and skilled exercise.

2.4 "DOWN-THE-HOLE" LOGGING AND SAMPLING TECHNIQUES

"Down-the-hole" techniques are used for many aspects of borehole logging. These may be categorised as follows:

- i. Optical Methods.
- ii. Physical Methods.
- iii. Geophysical Methods. (a) Physical Property Logging.
(b) Remote Sensor Logging.
- iv. Nuclear Techniques.
- v. Borehole Geochemical Logging.

As this is such a large field, the following account will, of necessity, be brief. The reader is referred to the relevant literature for details.

Down-the-hole techniques were developed in the oil-well industry where they have been extensively used since the 1930's for delineating and evaluating oil producing formations. The minerals exploration industry has been relatively slow in recognising the value of these methods but now there are many systems adapted for use in "slim-line" exploration holes. The main advantage to the minerals industry is that significant geological information can be obtained from inexpensive non-cored boreholes at a fraction of the cost of diamond drilling. This results in more boreholes per exploration budget and a subsequently wider ground coverage. With the rapidly increasing cost of diamond drilling, down-the-

hole techniques and non-cored boreholes will find application in many situations previously dominated by diamond drilling. Furthermore, in order to obtain the maximum geological information available from the strata drilled, down-the-hole techniques should also be applied to certain diamond drilling programmes. This data will increase the reliability of core assays and provide a third dimension to the sample area, i.e. that area-of-influence adjacent to the borehole.

2.4.1 Down-the-hole optical methods

Various optical and photographic methods of borehole logging have been developed. This includes a down-the-hole television unit. One of the simplest and most versatile of these units is an optical borehole periscope described by Krebs (1967). This is diagrammatically represented in Figs. 6 and 7.

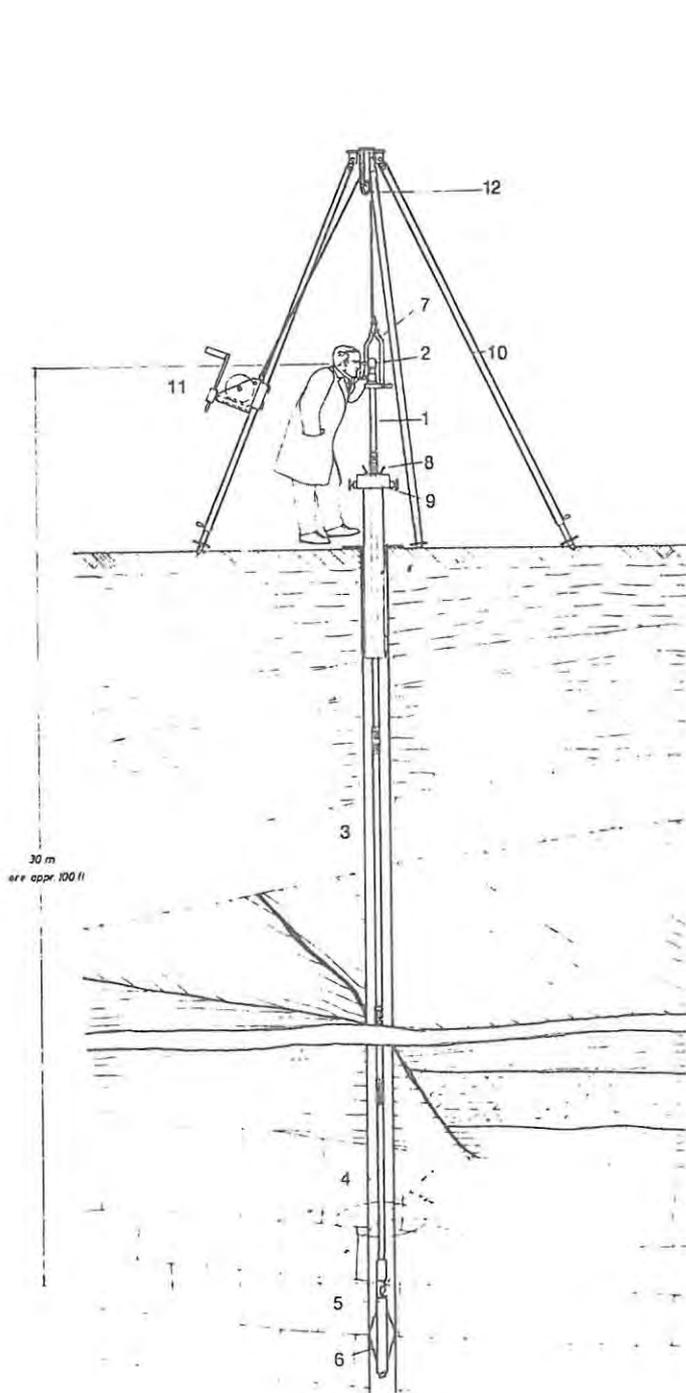


Fig. 6. Method of surveying with a borehole periscope.

The numbered parts are:

- | | |
|------------------------|----------------------|
| 1. Ocular Tube | 9. Suspension Collar |
| 2. Eyepiece Attachment | 10. Tripod |
| 3. Extension Tube | 11. Hand Winch |
| 4. Objective Tube | 12. Sheave |
| 5. Baffle Plate | |
| 6. Centering Springs | |
| 7. Bail | |
| 8. Baffle Plate | |

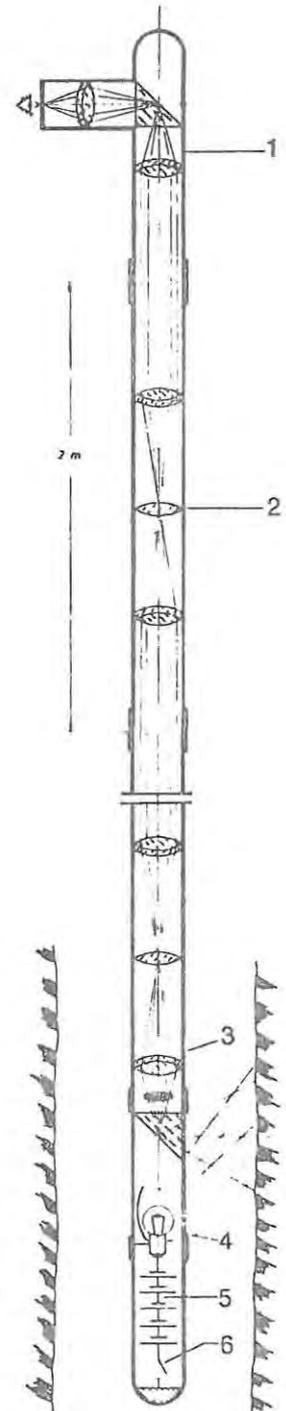


Fig. 7. Diagrammatic representation of the periscope.

(after Krebs 1967).

This instrument allows immediate investigation of the walls of the borehole and colour photographic records can be taken of any feature of interest in the hole. By means of a built-in orientating device, the strata and three-dimensional position of any geological plane can be measured and the dip and strike of bedding (layers, strata), established immediately by two subsequent scale readings. The diameter of the periscope is 56mm. and it has a depth limitation of 34m. While its main application is in engineering geology, it can be successfully applied to shallow exploration holes for "in situ" measurement of physical characteristics.

A refinement of the borehole periscope is the drill hole television camera. With this instrument, a television probe with outside diameter of 63mm. is lowered into the drill hole over a depth-measuring wheel. The probe is mechanically rotated in a helical motion and the television picture transmitted to a television screen through a cable. A compass pendulum within the probe enables simultaneous readings of the inclination and azimuth of the borehole. (Eastman Instrument Catalogue). The short distance between camera lens and drill-wall results in good definition of textural characteristics, such as crystal size, shape and porosity, and of mineralised zones, bedding joints, cleavages and open fractures. At any stage a permanent photographic (35mm.) or video-tape record can be made.

Another version of the drill hole television camera is an acoustic television viewer. This takes an oriented acoustic picture of the borehole wall in the form of a continuous log. This picture shows fracturing in detail and is not adversely affected by normal homogenous borehole fluids, a shortcoming of the optical television viewer. (Zemanek et al 1970).

Down-the-hole acoustic (or seismic, sonic, elastic wave) methods have found wide application in oil well investigations. A number of these different logging devices are in use and are described by Dyke et al (1975).

Essentially, the method involves reflecting acoustic signals from the anomalies in the rock surrounding the borehole. This reflected sound energy is converted to electrical signals. These are "unscrambled" and a three-dimensional representation of the structure becomes available. What this is really achieving is a mapping of the acoustic interfaces

of the surrounding rock. This mapping may be interpreted to indicate fault structures and mineralised zones. One such probe that is designed for use in AX size holes or larger is described by Brenden et al (1970). This is sensitive enough to detect echoes from anomalies which are at a distance of 7,5 m. from the axis of the diamond drill hole and is capable of a target resolution of approximately 20 cm. It has been tested in horizontal and vertical exploration holes to a depth of 500 m.

2.4.2 Down-the-hole physical methods

The "side-wall" sampling method is used down-the-hole to obtain samples of strata for geochemical analysis. The main application of this method is in the re-evaluation of old exploration boreholes or water boreholes where the core, chippings and/or assay results are not available. Another application is in zones of incomplete core and sludge recovery, or when recovered samples are believed to be contaminated.

The technique involves lowering a specially designed probe down the hole to below the required depth. This is slowly raised and a small scraping of the side wall is funnelled into a sample container. The method is not in common use as it is restricted to soft strata only.

The Caliper Log is used to correct the response of other logging devices. This provides accurate and precise diameter measurement of the borehole. This is an absolute requirement for quantitative analysis by nuclear logs. (See later notes.)

2.4.3 Down-the-hole geophysical methods

There are two principal applications of geophysical down-the-hole logging, i.e. physical property logging and detection logging (remote sensing). The physical properties commonly measured are: resistivity, induced polarisation (IP), magnetic susceptibility, gamma radiation (spectral and total count), density, seismic velocity and conductivity. These are particularly valuable in the direct evaluation of minerals intersected (e.g. coal seams), the lithology of the hole, faulting and fracturing, anomaly explanation and elimination or minimising of coring. Detection logs, on the other hand, are employed for direct detection of features lateral to, or beyond, the drill hole. Logs in this category include: resistivity, self-potential, IP/resistivity and self-potential in combination, magnetic vector, *mise-à-la-masse* and electromagnetic logging. (Cummings and Given 1973).

The application of these various methods to exploration drilling will be described briefly. This topic is dealt with in greater detail by Dyck (1975), Threadgold (1970), McCracken (1976), Baltosser and Lawrence (1970), Anderson (1974), Evans (1970), Pickett (1970), and Scott and Tibbets (1974). Of these, Dyck gives a comprehensive review of the literature on borehole geophysical methods. He deals with each method in turn and reviews its application in exploration. Similarly, Scott and Tibbets review the same topic but deal with each mineral commodity in turn, describing what logging techniques should be used. This work also contains a comprehensive bibliography for foreign languages as well as English.

One of the commonest applied techniques in borehole logging is that of resistivity and self potential. The basic idea of this (electrical) logging is to determine the exact depth below the surface and the nature and fluid content of the various horizons traversed by the borehole. The resistivity log measures the electrical resistance (in ohms), of a volume of unit length and cross section. A simplified circuit for measuring resistivity is shown in Fig. 8.

A variety of factors influence the magnitude of the resistivity recorded. Rock formations are capable of transmitting an electrical current only by means of the absorbed water which they contain. The exception to this being massive sulphide bodies and graphitic horizons. Compact rocks with little interstitial moisture exhibit high resistivities while unconsolidated and semi-consolidated formations (clays, shales, marls, sands, sandstones, etc.) contain much interstitial water and possesses low resistivities. The magnitude of the resistivity recorded on the log also reflects the thickness of the beds, the diameter of the electrode, the diameter of the borehole and the resistivity of the drilling mud.

The self-potential log (SP) records the spontaneous electrical potential profile that exists between contrasting strata traversed by the borehole. The presence of the mud filled borehole causes currents to flow from zones of high electrical potential (shales), into permeable zones of lower potential (such as permeable sands). During their passage in the borehole these currents cause changes in the level of the electrical potential of the borehole mud which are measured between two electrodes. One of these electrodes is lowered into the hole while another is grounded in a shallow hole at the surface.

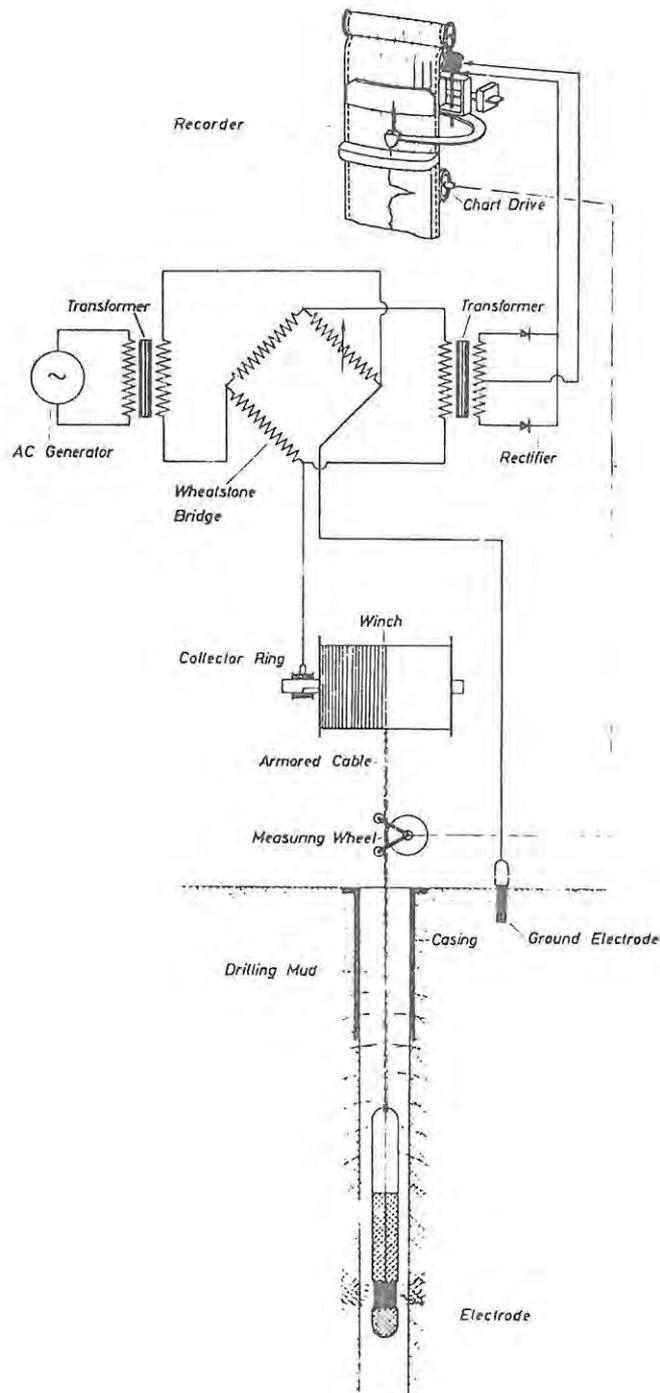


Fig. 8. Simplified circuit for measuring Resistivity.
(Eastman Brochure).

Fig. 9 illustrates the schematic circuit and equipment-configuration. In practice the potential measured is closely related to the porosity or

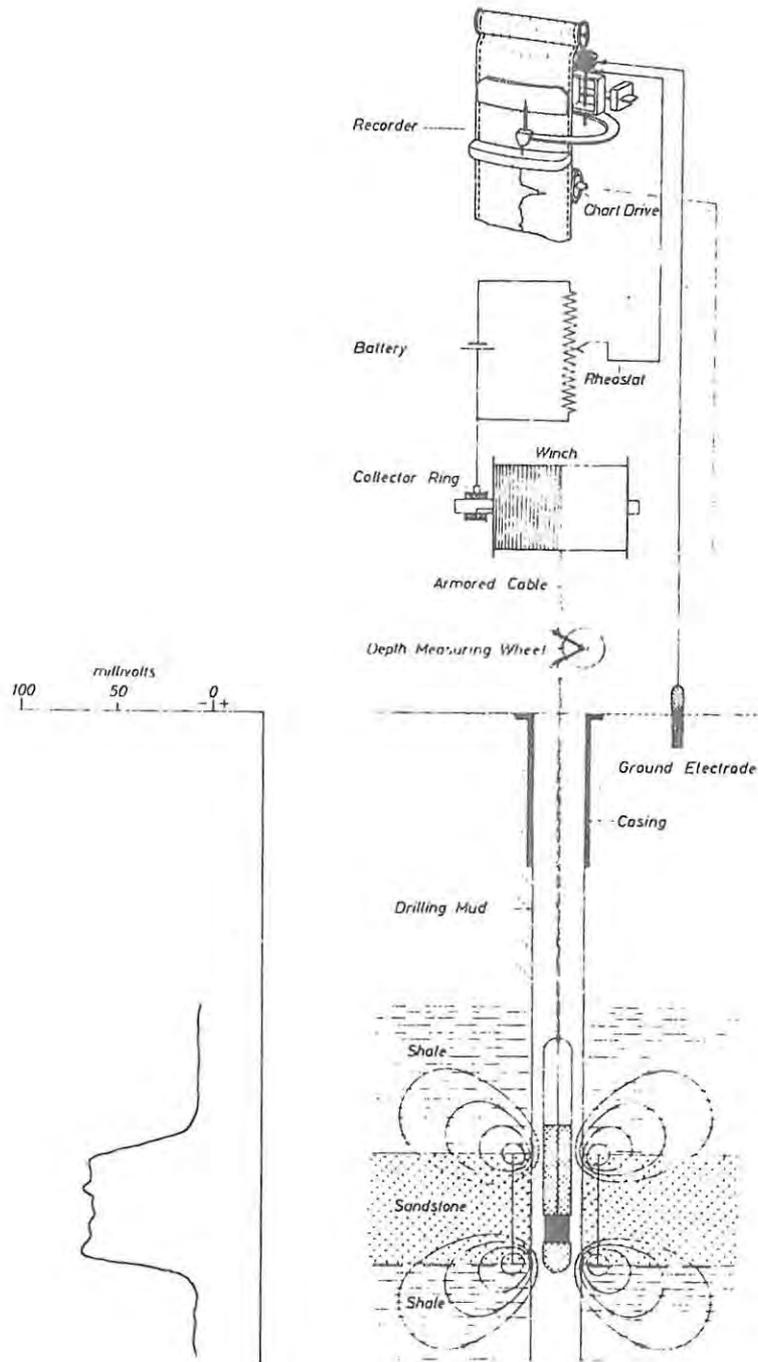


Fig. 9. Schematic circuit for recording SP logs with picture of borehole currents which cause SP behaviour. (Eastman Brochure).

permeability of the beds opposite the electrode. The curve indicates high permeability by curving to the left and a lesser degree of permeability by curving to the right.

It is common for these two logs (resistivity and self-potential) to be read in conjunction with one another. Coal and petroleum usually have a higher resistivity than their associated strata and can be successfully located in exploration well-logs. Other minerals, such as graphite, magnetite, pyrite, the arsenites, tellurites, many sulphides and a few oxides can also be detected in boreholes because of their extremely low resistivity. (Eastman catalogue). Further applications in uranium and coal exploration are discussed in section 2.4.5.

At present there is no basic theory for predicting the type of potential profile in any given sequence; however, potential profiles can be correlated between boreholes penetrating similar geological sequences. The simplicity of the measurement makes it a very attractive exploration tool in development areas where the geology is well defined. (Threadgold 1970).

The Induced Polarisation Log (IP) records the anomalous storage of electrical energy in certain minerals (sulphides) from an externally applied source. Basically the method involves pulsing an electrical current between two probes and measuring the resulting potential generated with a third probe situated next to the required geological formation in the borehole. The primary object is to measure the decay curve of the stored energy as it falls off between pulses. In disseminated sulphide deposits, each individual sulphide mineral will store such a charge and thus makes this a very attractive exploration tool for such deposits.

The position of an ore zone or IP anomaly in the side of the borehole can be detected by the IP azimuth procedure. Here, the source current electrodes are placed at equal distances on opposing sides of the hole (see Fig. 10). The direction of polarisation will be a result of the position of the anomalous feature, thus enabling its location to be determined. (Cummings and Given 1973).

In the applied potential or mise-à-la-masse, technique the range of drill hole information can be extended by placing a current electrode in or near an ore body and mapping the response pattern on the surface or in

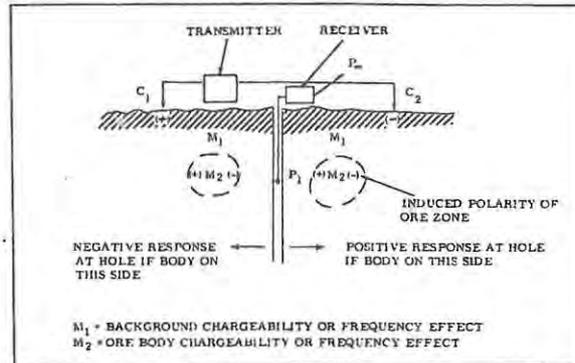


Fig. 10. Schematic section, IP azimuth procedure.
(after Cummings and Given 1973).

another drill hole. This method only applies where the drill hole has intersected mineralisation which "channels" the applied current. The resulting distortion of the potential pattern is diagnostic of the persistence of the ore zone. The method can either be employed by contacting a zone at the surface and measuring the potential at the surface and down-the-hole, or by contacting the zone downhole and measuring potential at the surface.

Electromagnetic (EM) logging is a complicated procedure and generally avoided in choice of logging techniques. Drinkrow R.I. in McCracken's 1976 volume provides a comprehensive review of the various EM down-the-hole techniques, and evaluates them against the invariable blanketing effect of the surrounding medium. Dyck (1975) reviews the relevant literature and suggests, in the majority of cases, EM techniques have been successfully applied to exploration boreholes. Unfortunately, this review presents the positive aspects only.

The objective of EM techniques is the detection of heavily disseminated/stringered or massive sulphides lateral to the hole. The configuration of transmitter and receiver depends on the geometry of the expected target. For instance, in "downhole Turam" techniques for flat lying conductors adjacent to a vertical hole, the transmitter loop is placed concentrically about the hole, with the receiver coil down-the-hole. (Cummings and Given 1973). Some of the various configurations used are summarised in Fig. 11.

Bosschart (1979 - personal communication) claims that the principal disadvantage of down-hole EM methods is that extremely large anomalies are generated from minor mineralisation. Thus, the effect of "noise" inside the hole successfully blankets any appreciable anomaly. Down-hole EM methods are expensive, time consuming and give little directional sense to any anomaly.

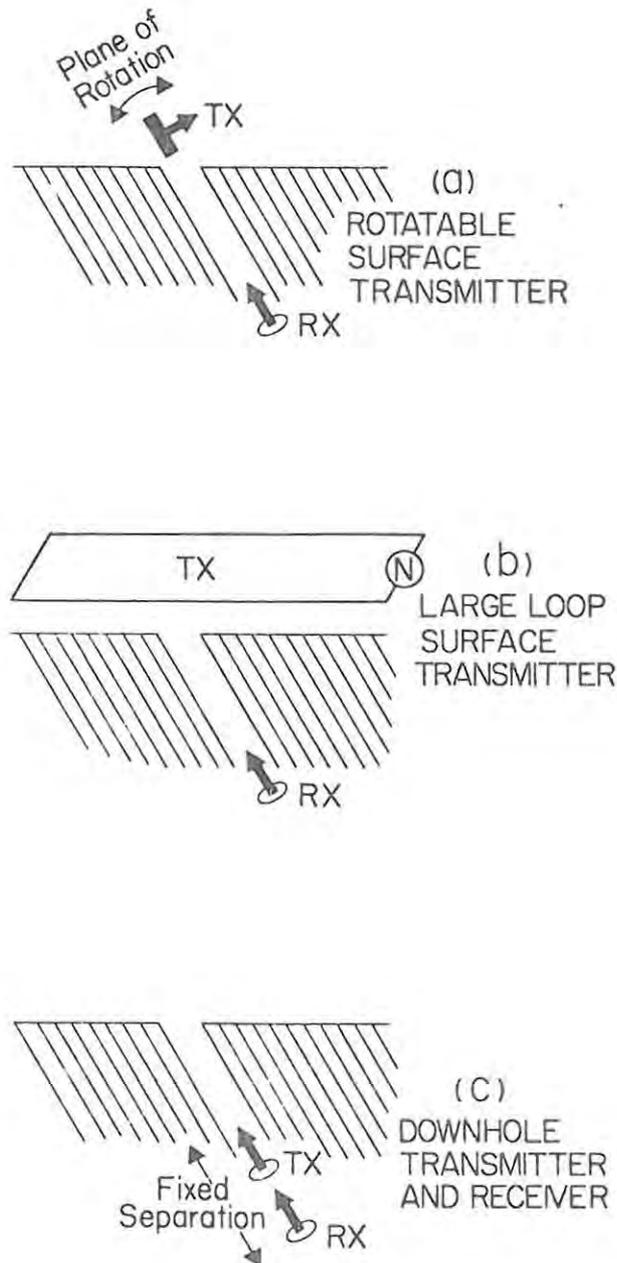


Fig. 11. Coil configurations used in EM surveying of boreholes. (after Dyke 1975).

Borehole gravimetry techniques are used mainly for elucidating surface gravity anomalies. Other applications are: density determinations, interpretation of seismic data and borehole formation geology. The advantage of borehole gravimetry in the interpretation of surface gravity surveys is that precise Bouguer corrections are possible using borehole gravity measurements. Alternative density devices are available but are particularly sensitive to borehole conditions such as fracturing, drilling fluid invasion into the borehole wall, and are ineffective in cased boreholes. The borehole gravity meter is less sensitive to these conditions and can be run in cased boreholes, giving bulk density estimates for very large rock volumes. (Dyke 1975).

Down-the-hole magnetic susceptibility measurements are not common. This is because it is a lot simpler to measure magnetic susceptibility in core or cuttings. Equipment is available to permit continuous logging of drill core by passing a sensor over the samples without removing them from the core box. (World Mining August 1979). The measurement of electromagnetic induction (EM) also determines the magnetic parameters of the formations traversed by the borehole and is used more commonly than the magnetometer. However, slim-line borehole magnetometers have been extensively used in Scandinavia for iron ore prospecting. The correlation between magnetic susceptibility and magnetite content has proved to be a useful grade control tool for "in situ" bulk measurement in iron ore mines in Finland. The larger volume "sampled" by down-the-hole measurements are preferable to the volume of core usually used in the laboratory. (Dyck 1975).

In prospecting for base metal deposits, other than iron, the success of magnetic methods in boreholes depends upon the presence of pyrrhotite. As pyrrhotite is a variable commodity within base metal deposits its magnetic expression has little application.

2.4.4 Nuclear techniques for down-the-hole logging

Various nuclear logging techniques have been used in exploration for petroleum, coal, potash and uranium. A recent development, that of neutron-activation analysis, has extended this field into base and precious metal exploration.

Nuclear borehole logging techniques may be considered to be either passive or active. In the former, the natural radiation in the hole is measured

by an appropriate detector. In the latter, both a radio-active source and a detector are placed in the borehole. The radiation which reaches the detector from the source is modified by the physical properties of the rock and the radiation detected can be translated into a measure of rock density, moisture content etc.

Killeen (1975) suggests that 90% of the natural gamma rays come from a 23cm. radius of the detector. This increases the volume of rock being sampled down-hole by an order of magnitude compared to core samples and results in a more representative sample at a third of the price of core samples.

The passive systems of borehole logging include the Gamma (γ) log and the gamma-ray spectral log. The gamma log is a measurement of the intensity of the total natural radiation of rocks, emitted by the Uranium and Thorium decay series and by Potassium -40. Killeen, quoting Dodd (1974), states that in 1973, drilling in the United States for uranium exploration amounted to about 5 million metres and most of that was logged using the natural gamma log. The gamma log can be used in air-filled, oil-based muds, foam or cased holes and has thus been of great use in the oil industry.

The gamma-ray spectral log is a refinement of the gamma log and can differentiate between the thorium, uranium and potassium components of natural radiation by utilising a gamma-ray spectrometer. Killeen (1975) suggests that this method, apart from its obvious application to uranium and potash exploration has good potential for use in base metal exploration. He cites as examples the relationship between Th/U ratios in granitic rocks; that of radioelements and hydrothermal mineralisation and alteration; the distribution of Th, U and K in porphyry copper deposits; the possibility of measuring significant changes in Th and U distribution as haloes around mineral deposits; and the association of uranium with pyrite, sphalerite, galena and molybdenite.

The active logging systems involve the use of a source of radiation. The gamma-gamma log is a measurement of the density of the rock in the borehole wall, obtained by detecting gamma rays from a radioactive source after they have undergone scattering in the region between the source and the detector. Density information is used in several ways: (i) to obtain tonnage factors to compute reserves; (ii) to indicate the competence

of rocks, which affects mining methods and costs; (iii) to determine porosity; and (iv) to determine interstitial formational water which modifies the response of nuclear logs. (Dodd and Eschliman 1972).

The selective gamma-gamma log works on the same principle as the gamma-ray spectral log except that a gamma source is used instead of natural gamma rays. Gamma ray distribution at low energies is influenced by the chemical composition and heavy element concentration of the rock. It has potential for use in base metal exploration owing to this property and has been used to evaluate coal, Zn, Pb, Fe, Hg, Cu, and Mn, content. (Killeen 1975).

The basic principle of the neutron log is that neutrons emitted by a source are slowed down and scattered by collisions with atomic nuclei. The process continues until the neutron is in thermal equilibrium with its surroundings and is captured by the nucleus of an atom. Capture of the neutron is usually accompanied by the emission of high energy gamma rays ("prompt" rays), which is the principle of the neutron-gamma log. The detected response in the log is proportional to the hydrogen content of the rock (usually in the form of water in the pores), and is therefore a measure of porosity. The neutron log provides qualitative lithological information similar to that obtained from the resistivity log. Shale, mudstone and clay commonly have a greater porosity and contain more water than sandstone. The neutron log, therefore, indicates a sandstone by an increased count-rate much as the resistance log indicates a sandstone by an increased resistance. (Dodd and Eschliman 1972).

2.4.5 Various applications of down-the-hole geophysical and nuclear techniques in mineral exploration

In the exploration for sandstone-type uranium deposits, the principle down-the-hole tools used are a combination of the gamma, resistivity and self potential logs. The resistivity log is used mainly to define lithological contacts and for correlating rock units between boreholes. Because of the simplicity of its equipment, the single point resistance log has become the standard electric log for this task. In combination with the gamma and S.P. logs, it is very useful for detailed stratigraphical correlation of thin units, between boreholes. See Fig. 12. (Dodd and Eschliman 1972). In sandstone uranium prospecting the resistivity log finds its widest application in delineating the extent of porous sand bodies (potential uranium hosts) amongst impermeable strata in the bore-

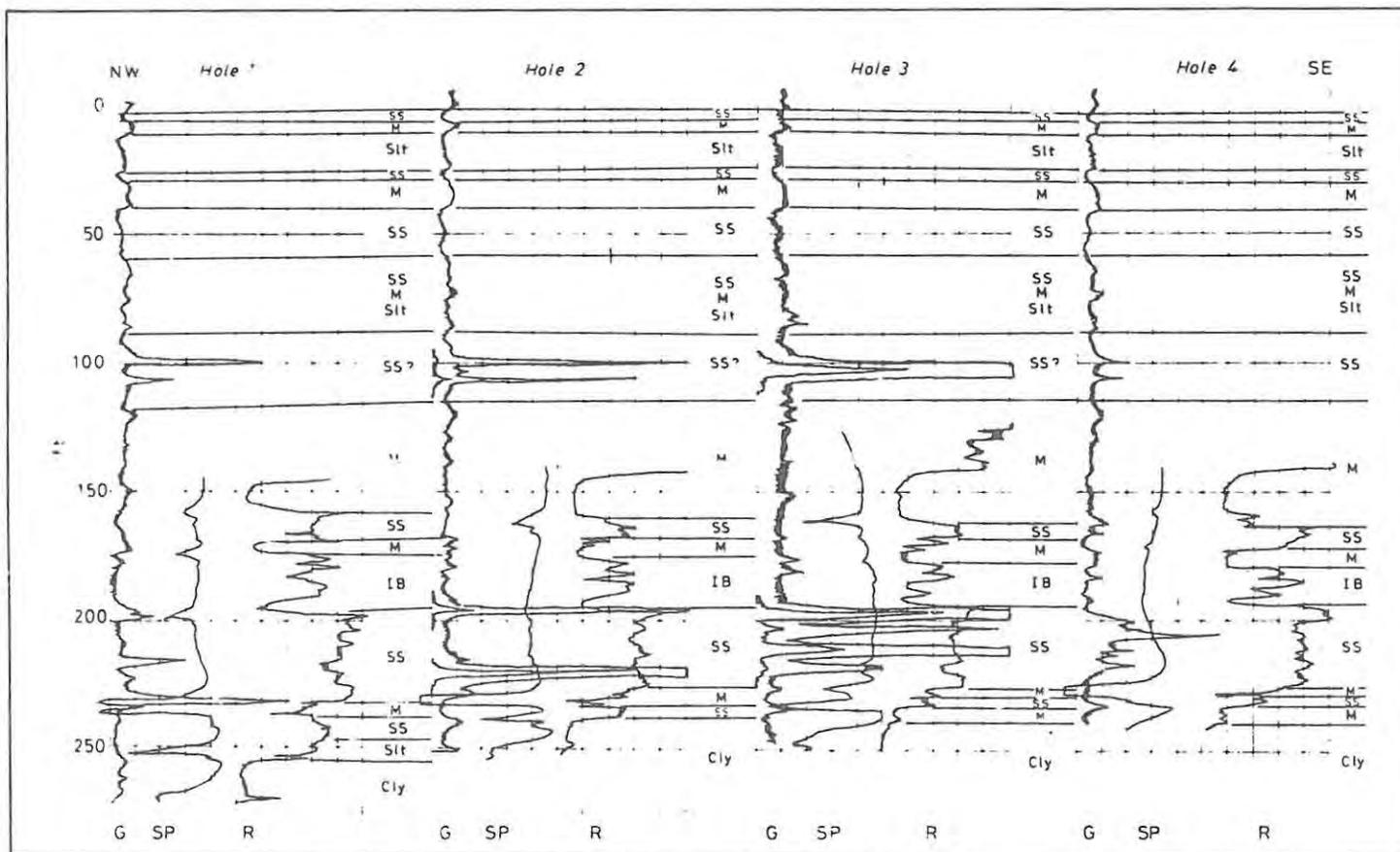


Fig. 12. Identification and correlation of lithological units by use of gamma-ray, resistance and SP logs (after Dodd and Eschliman 1972).

holes. Furthermore, formational changes in the borehole are often defined more clearly in the resistivity log than in the chip samples from the borehole.

The S.P. log is used mainly in uranium exploration to confirm the lithology and correlations derived from the resistance log. Dodd and Eschliman (1972) suggest that S.P. effects may be used to advantage to establish redox potentials in the environments of roll fronts. However, self-potential "highs" in the log are often associated with sulphide minerals, which in turn concentrate in the same roll front environment as the uranium.

Radiometric logs, and especially the natural gamma log, are the most widely used of all down-the-hole techniques in mineral exploration. In addition to detecting and accurately measuring uranium mineralisation, the radiometric log can provide alteration data through analysis of certain gamma anomalies together with the lithological and resistivity logs. Sensitive gamma equipment can also be used in stratigraphic correlation. (Davis 1973). Apart from these obvious applications, the logs may give significant lithological information. Various minerals may contain small amounts of radioactive material. Apatite, for instance, often contains traces of uranium and thus phosphate rock is usually radioactive. Carbonaceous rocks usually have high background radioactivity due to the "fixing" of uranium by carbon. Argillaceous minerals have surface-absorbed ions which tend to be preferentially replaced by radioactive minerals. For this reason shales usually show a higher radioactive background than do sandstones and limestones. Some minerals, e.g. sulphide minerals, anhydrite, coal and salt, characteristically have low radioactivity. (Taube 1978).

The gamma logging units now in use provide an extremely accurate means of measuring the grade of uranium in the hole. The count within a sphere of influence of radius 23 cm. is averaged, consequently offering a larger sample volume than that of a cored hole. One disadvantage of the method is the inability to determine uranium disequilibrium. In unoxidised deposits, where the degree of disequilibrium is often relative to the ore body, a few strategically placed core holes are necessary to make the grade adjustments. In oxidised or weathered deposits the gamma results are much more erratic and should not be relied upon for reserve calculation (Davis 1973).

A down-the-hole system for measuring uranium directly, and thereby circumventing the disequilibrium problem, is described by Steinmann et al (1976). This is a "Californian-252-based" logging system which employs a delayed neutron activation (DNAA) technique and can determine concentrations of 0.01% U_3O_8 . Two other methods for the direct detection of uranium have also been developed. These are: 1. Pulsed neutron log with the detection of prompt fission neutrons. 2. Pulsed neutron log with the detection of delayed fission neutrons. These have the disadvantage of requiring a pulsed borehole neutron generator, whereas in the DNAA technique, the steady-state neutron source is used. (Czubek and Loskiewicz 1976).

Although most uranium exploration has relied on the gamma, resistivity and SP logs, Davis (1973) points out that other techniques have potential application for determining such features as permeability, sulphide and carbon content. These methods include IP, neutron-neutron, density, temperature, "long" and "short" normal resistivity and caliper logging.

Taube (1978) describes in detail the results of an exploration programme (in the Woodcutters, Rum Jungle Area, N.T. Australia), for uranium and base metals using resistivity, SP and gamma logging methods. He concludes that from the geophysical viewpoint, surface IP, EM and SP anomalies were explained, or at least more accurately delineated, by the down-hole probing. Rotary-percussion drilling combined with geophysical probing proved to be an inexpensive method of testing some geophysical anomalies. From the geological viewpoint the logs provided an objective means of differentiating between gradational rock types and added parameters not otherwise available for description of rock units.

In the exploration for coal deposits, down-the-hole techniques can provide valuable analytical and lithological information at a low cost and in a very short period of time. Traditional coal evaluation techniques involve coring the coal seams and doing a laboratory analysis of the core for carbon, ash and moisture. Two major difficulties are encountered with this method. The first is keeping the core in its original condition, so that the analysis of the data reflects the true condition of the coal. The second problem occurs when core recovery is poor or whenever the coal is drilled into by non-coring techniques before the commencement of coring. Both of these problems can lead to a poor evaluation of the reserves. A combination of core and chip analysis plus down-the-hole methods would

ideally be the most accurate and representative. However, as coal is a relatively low priced commodity strataformly underlying large tracts of land, evaluation by diamond drilling becomes very expensive. It is for this reason, therefore, that down-the-hole techniques and the use of non-cored boreholes have gained such wide acceptance in the coal exploration industry. Furthermore, the trend of the multinational oil companies towards diversification into other energy resources has made their technical expertise, developed in well logging, available to coal exploration.

Various logging techniques and combinations of different logs are used for coal exploration. The most commonly used are: density, neutron, conductivity, gamma-ray and caliper logs. Some operators make use of acoustic responses. The responses of these various logs to common lithologies can be seen in Fig. 13.

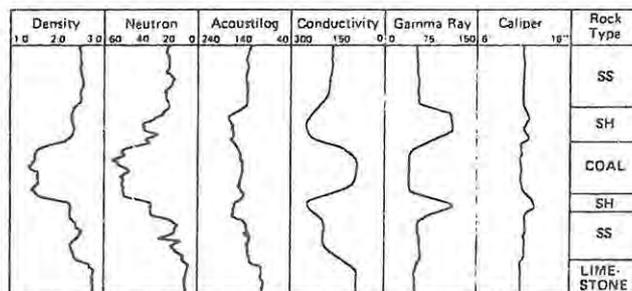


Fig. 13. Generalised response of logs in various lithologies (after Kowalski and Holter 1976).

Reeves (1978) claims that the most popular logging combination is a gamma-ray, density and caliper log. When plotted together as a "coal lithology log" (see Fig. 14), this allows rapid identification of coal seams, sandstones and shales. The density log records the apparent density of the rock formations penetrated by the borehole. The gamma-ray log records the natural radiation of the various formations while the caliper log measures the borehole diameter. This gives an indication of the zones that have been eroded during the drilling process.

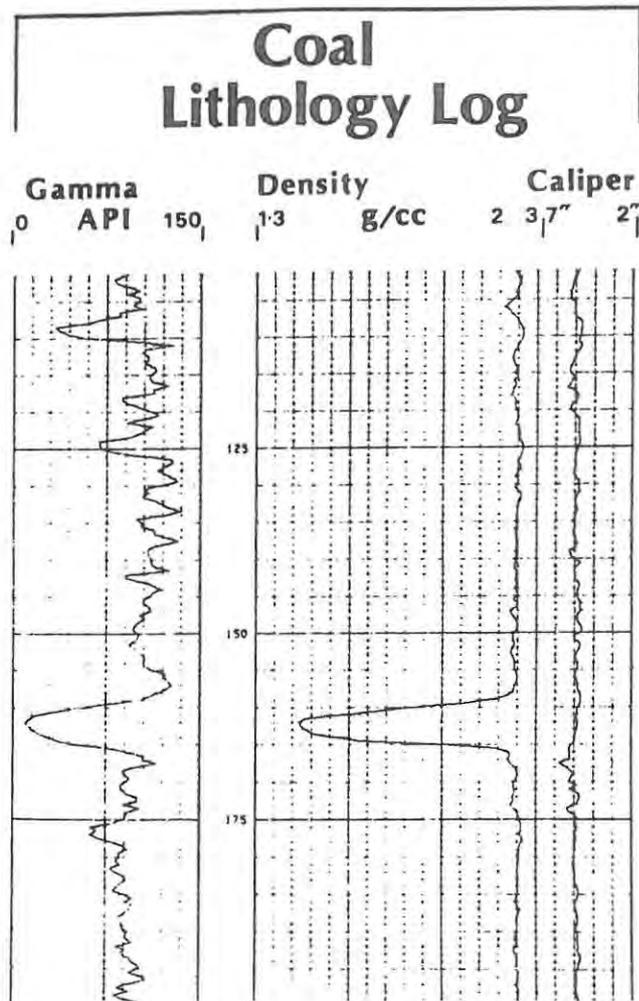


Fig. 14. The Coal Lithology Log (after Reeves 1978).

A second combination of "short spacing" density and caliper logs, run on an expanded scale over the actual coal seams, gives an accurate interpretation of bed thickness and waste partings. Finally, a combination of "long spacing" density and the gamma ray log (reversed), presented on a detailed scale forms the basis of measurement for ash content. (See Fig. 15). With modern equipment now available, all of this information can be obtained with one run of the logging device at speeds of approximately 10 m. per minute. Over the coal seam itself, this is reduced to 2 m. per minute.

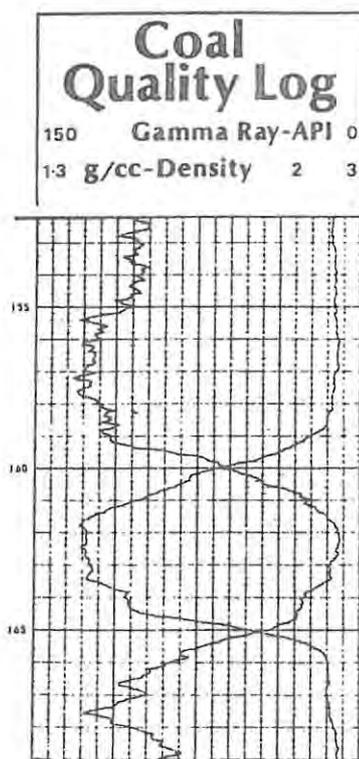


Fig. 15. Coal Quality Log (after Reeves 1978).

Kowalski and Holter (1976) describe how this information can be synthesised and interpreted through the use of a computer. Furthermore, they claim to have developed an interpretative method which determines the elastic properties of hanging wall and foot wall strata to assist in future mine design.

The following mineral types can also be determined by logging methods: potash, salt, gypsum and anhydrite deposits can be adequately analysed by a combination of gamma-ray, neutron and density logs. Gamma radiation effects have been found to be useful for making approximate grade assays in determinations for phosphate rock. Density techniques (with modified sondes for the high density values) are suitable for iron ore grade determinations and, in the right circumstances, for bauxite. Simple neutron activation methods are applicable for fluorspar and apatite evaluation. (Reeves 1976).

2.4.6 Down-the-hole geochemical methods

Borehole geochemical logging (or "in situ" multi-element analysis) is an extension of the nuclear logging techniques previously described. This is a relatively new research field which has great potential for the minerals

industry. The most comprehensive description of nuclear analysis techniques is to be found in McCracken et al (1976). These nuclear methods permit the evaluation of a much larger and therefore, more representative sample than is allowed by conventional coring techniques. One such nuclear method is the prompt gamma-ray analysis technique. This is actually a neutron-gamma logging technique incorporating a spectrometer.

In this method, the neutrons emitted by the source are captured by nuclei (present in the surrounding matrix), which emit "prompt" gamma rays. These gamma rays are characteristic of the elements that produce them. They are detected by the sonde (or probe), in the borehole which passes signals to a multichannel analyser system. Here the signals are sorted out and appear as a spectrum. The type and concentration of the elements detected can be interpreted from this spectrum.

Nargolwalla (1977) describes the "Metalog" system (a "prompt" gamma ray analysis technique) developed by Scintrex. This system was developed for simultaneous down-hole analytical determinations of nickel, iron and silica in a lateritic ore-body; or seam determination and sulphur content in a coal deposit. After extensive field tests the degree of accuracy of the system was compared to geochemical analyses of core samples. Certain standard errors were detected but were consistent enough to be compensated for. The advantage of this system is that it is a self-contained, vehicle mounted and computerised analysis technique. It provides instant information, in hard copy form on site, as well as on tape for later computer processing for grade and tonnage. (Mining Journal 1976 Sept., p.230). Scintrex claims that the Metalog assays correlate better with chemical analyses of 66 cm. bulk samples than with core samples. This demonstrates the applicability of the method for bulk sampling and grade control. (Nargowalla 1977).

Neutron activation logs have been used to differentiate aluminium-containing shale bands in the sedimentary Pilbara iron ores at Tom Price, Western Australia. (Wylie 1976). Aluminium detection can also be used to determine zones of low ash content (i.e. low aluminium) in coal seams.

Borehole analysis for copper and nickel using Gamma-Ray Resonance Scattering is described by Sowerby (1976). This technique involves the use of a radioactive gamma-ray source to bombard the rock, which then emits rays of the same energy if conditions are right for nuclear resonance.

These conditions require that the radioisotope source decays via the excited state of the element to be analysed. For example, if the element to be analysed is nickel then a radioactive ^{60}Co source may be used. This radioisotope decays by beta and gamma-ray emission via excited states of ^{60}Ni . This source requirement puts a limitation on the number of elements that can be determined with the technique. These are: Cu, Ni, Cr, Hg, W, Ti, Li, V, Ge, As, Dd, Cs, and Tl. In his summary, Sowerby (1976) suggests that the technique is better suited to borehole core and drill cuttings analysis than in situ borehole logging. A severe limitation in borehole logging is that the equipment can only be used in holes of > 14 cms. diameter. Killeen (1975) suggests that this technique may prove more favourable for specific elements, such as Ni or Cu, than neutron activation methods.

X-ray Fluorescence analysis techniques are commonly used in the minerals industry for laboratory determination of minerals. This method has now been modified for use as a borehole probe for in situ element analysis. Basically, the technique uses an isotopic source of gamma rays to irradiate the borehole wall which in turn emits x-rays with energies characteristic of the elements in the rock.

One of the earliest x-ray fluorescent borehole probes brought into commercial production was by Ekco Instruments Ltd. of Britain. This instrument was developed for assaying tin and was extensively tested at the Wheal Jane Tin Mine in Cornwall. The probe will operate in AX or EX boreholes under wet or dry conditions and can tolerate substantial unevenness in the borehole walls. It has a digital electronic read-out and indicates tin concentrations of 0,1 per cent and upwards.

The measuring time for each position of the probe is half a minute. The results of the field test at Wheal Jane correlate closely to chemical analyses of cores from the same borehole. (World Mining Jan. 1971).

Two drill hole probe assemblies, designed and constructed for the United States Bureau of Mines, were field tested in 1969 and 1970 in lead mines in Missouri. These had a practical detection limit of 0,5 weight-per cent lead and correlated closely with core samples from the same hole. The only source of error was found to be an elongated smear of lead on the walls of a borehole after drilling through a particularly high grade zone. The borehole probe was unable to detect the difference between this and in situ lead. (Marr and Campbell 1972).

One advantage of the technique is that it is applicable to a wide variety of elements such as: Ti, Cr, Mn, Fe, Ni, Cu, Zn, Pb, Mo, Zr and W. (Mining Journal Nov. 27 1970 p.483.) Other advantages are that the sensitivity is high and the technique of XRF is well documented. (Killeen 1975). In order to be an effective tool for mining applications, the instrument must be sensitive to ore concentrations which are significantly below the concentrations economically feasible to process. The X-ray drill hole probe could be used as an exploratory tool for ores that are found in localised concentrations greater than about one per cent; such as lead, zinc, copper, tin, mercury, and molybdenum. One additional limitation is the decreased penetration of the X-rays from elements of low atomic number. For elements near or below iron on the periodic table, the presence of water in the drill hole would seriously impair any attempt at analysis. (Marr and Campbell 1972). It is therefore, only a "surface analysis" technique for those elements below X 40 in the periodic table. It has been suggested that its main application may be in development drilling to locate ore-grade cutoff. (Killeen 1975).

CHAPTER THREE

CORE, OR DIAMOND, DRILLING

3.1 THEORY, OPERATION AND TECHNIQUE

"Diamond drilling is the most versatile of all methods, and it is designed specifically for mineral exploration." (Peters 1978). The diamond drill can operate at any angle from surface or underground mine workings and the depth to which it can drill is limited only by the capacity of its power pack. The rigs are relatively light and manoeuvrable and can be transported through very difficult terrain. They are often vehicle mounted or mounted on skids and can be winched into position utilising the rigs' own rods-hoisting mechanism. Diamond drill rigs can be airlifted by helicopter, or mounted on barges for underwater drilling.

The main disadvantage of diamond drilling is that the process is slow and expensive. Shallow diamond holes are anything from three to four times the cost of non-core holes per metre. The cost of diamond drilling increases rapidly with depth and with larger diameter holes.

The aim of diamond drilling is to obtain an intact core of rock strata for geological investigation and evaluation. For reasons of cost, these boreholes usually have as small a diameter as is practical under the circumstances and are thus sometimes referred to as "slimline holes".

Diamond drilling equipment used in South Africa can be categorised as:

- i) The A.B.C. system. A standardised system adopted initially by America (U.S.A.), Britain and Canada, hence A.B.C., and now also adopted by Australian and South African manufacturers.
- ii) "T" or thinwall system. A system developed in Sweden and Continental Europe using thinwall bits and corebarrels with a range of related drill rods.
- iii) Wireline system. A system developed as a proprietary design by the Longyear Company. (Beukes et al 1978).

3.1.1 Theoretical aspects of rock penetration by diamond drilling

A cylindrical bit, set with diamonds, is rotated at a relatively high speed whilst being steadily thrust forward. This hole is, therefore, produced by a form of coarse grinding. In this operation, an annulus only is removed, the rock cylinder (core), which is left within the annulus passes up into the core barrel. (McGregor 1967).

The actual cutting action of the diamond has been the subject of various research programmes with a view to improving diamond bit design. Two fields of thought exist to explain the theory behind this cutting action. Adamson (1977) reviews the merits of these theoretical aspects, colloquially calling them the "ploughing" and "stress relaxation" theories. The concept behind the ploughing theory is that of a single diamond making an elastic impression into the rock face when subjected to a load. When the diamond is moved (i.e. during rotation), the rock is broken by a shearing action. One of the main advocates of this theory is Montgomery (1971), who uses the analogy of agricultural ploughing to describe the cutting action of the diamond; i.e. the plough weight represents the bit force, the pull of the tractor represents the torque (or rotational) force and ploughshares represent the diamonds.

The theory of "stress relaxation" is advocated mainly by Spink (1972) who envisages the propagation of microcracks behind the moving diamond. This is caused by the material behind the diamond being stretched during plastic deformation and fracturing because of tensile stress. Adamson (1977), supports the concept advocated by Schlössin (1969) who considers that the cracks form beneath the diamond. These cracks are more the result of the vertical load on the diamond than the shear stress of the torque force. Marx (1973), observing the effect of diamond tip scratch tests on glass, reports two types of cracks: those formed in front of a tool as shear cracks and those formed behind the tool in the tensional area. The formation of these cracks is more in line with the shearing concept than that of stress relaxation. However, it appears that the most significant feature is that the diamond is continuously traversed across a previously fractured rock face, thus destroying the rock material in the process.

3.1.2 Conventional diamond drilling technique

The conventional diamond drilling technique uses the A.B.C. category of equipment and continues to be the most widely used type of equipment in South Africa.

The drill is a complete unit in itself, consisting of a heavy steel frame on which is mounted the "power-pack" (usually a commercial diesel motor). This supplies the power to the hoisting drum and drill head.

The drill head utilises hydraulic cylinders to power and control the axial feeding movement of the drill rods. These are rotated by means of a mechanical drive from the drill transmission through connecting gearing. In all but the shallowest of holes, the drill rod string weight exceeds the desired thrust of the drill-bit. Thus, the control of the axial feed by the hydraulic cylinders is very important and prevents damage to the bit and rods.

In most light and medium duty machines the hoisting drum forms an integral part of the drill. However, in heavy duty machines, the hoisting apparatus forms a separate unit. The drum operates a wire cable running through a sheave wheel at the top of a tripod and thence downwards to its attachment with the drill rod string. In heavy duty machines, a travelling sheave is used with two or more falls of rope between a multiple crown and travelling sheave assembly. Furthermore, the deeper the hole, the more important it is to raise and lower rods in greater incremental lengths, thus minimising hoisting time. Light duty machines operate satisfactorily with derricks providing 6 metre "pulls". On the other hand, heavy duty drills operate with derricks with up to 30 metre "pulls". The hoist drum must have the capacity for sufficient rope storage. For example, at a depth of 4000 metres, a BX rod string weighs approximately 36 tonnes. The rope is commonly 20 mm. in diameter. In a multisheave assembly using 6 falls of rope, about 200 metres of this rope are reeled for a clear 30 metre pull. (Beukes et al 1978).

The rods, couplings and core barrels are robust and can take a lot of abuse without failure. A purely South African size of BX rods and couplings have had to be developed for the Witwatersrand gold exploration holes of 2000 metres plus. This is to obtain the correct weight to strength ratio in the rods needed for drilling these deep holes.

The core barrel is attached to the lower end of the rods. It holds the core while drilling and in bringing the core to the surface. Single tube core barrels are generally used with non-coring bits or where core

recovery and ground conditions are good. The double-tube core barrel (see Fig. 16), is more commonly used in broken or friable formations where washing or grinding of the core may seriously affect the value and the amount of the core sample. Drilling is continued until the capacity of the core barrel is reached. This capacity varies considerably but is usually in the order of 3 metres. The drill rods and core barrel are then hoisted up to the top of the tripod, held by a chuck, and the rods unscrewed and stacked. When the core barrel reaches the surface it is disconnected, the bit and reaming shell removed, followed by the inner barrel or core tube. The core is then taken out of the tube and placed in core boxes.

The type of drill string and associated core barrels referred to above have to be hoisted from the hole for each length of core recovered and, at depth, this is a very time consuming operation.

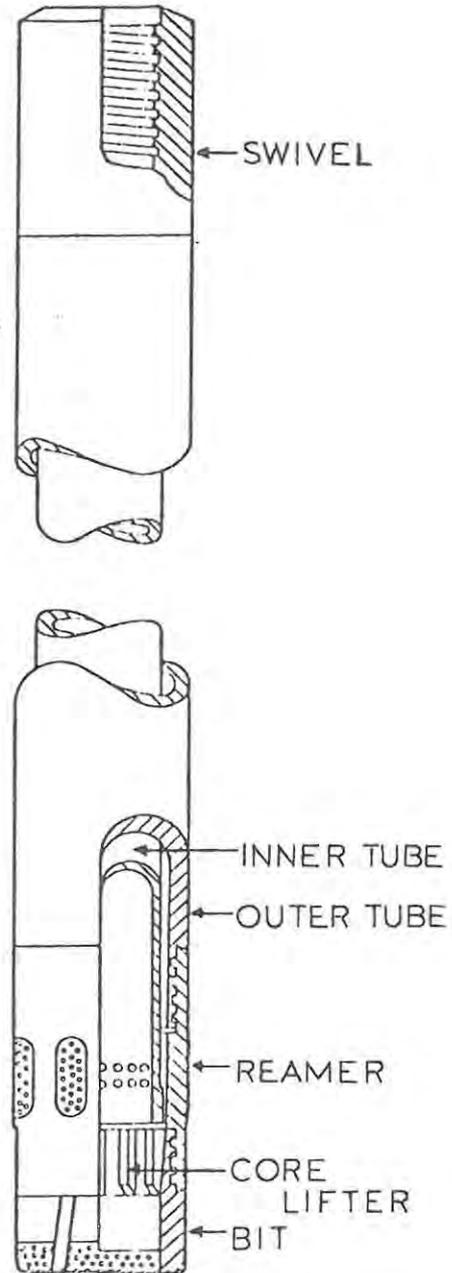


Fig. 16. Diagrammatic section of double-tube core barrel (after Beukes 1978).

3.1.3 "T" or thinwall system

This system, developed in Sweden and Continental Europe, is generally lighter in weight than the conventional equipment. Aluminium rods, thinwalled core barrels and thinwalled bits are cheaper due to the decreased number of diamonds in the narrow kerf. A greater degree of mechanisation in rod handling is possible due to the lightweight equipment. Vibration in the drillstring is substantially reduced and good core recovery is possible. This equipment is not suitable for deep boreholes and is limited to fairly shallow holes. (Beukes 1978).

3.1.4 Wireline system

The oil industry has made extensive use of the wireline system of drilling for recovering core samples or "spot cores". This has been facilitated by the large diameter drill rods in use in the oil industry, which allow an inner core barrel to be lowered to the bit. This principle was applied to small diameter diamond drill holes by the Longyear Corporation to reduce the amount of rod handling by requiring rod pulling only when a bit change is necessary.

The core-laden inner tube of the core barrel is retrieved by means of a small diameter wire cable (the wireline), and hoisted through the centre of the rods without removing them from the hole. In down holes this is aided by gravity, but in horizontal or upwardly inclined holes, the wireline cable (with its overshot assembly) and core barrel are pumped into place by the circulating fluid. The wireline system has the advantage over conventional techniques of reducing time spent on raising and lowering of rods, thus facilitating longer periods of coring per shift. Since the bit remains at the bottom of the hole during successive core-runs, less "caving" occurs in the hole. This alleviates drilling of the "cave", a major cause of diamond-bit damage and short life. Longer core runs and higher core recovery is, therefore, possible.

The initial costs of wireline drilling are higher than conventional drilling as a string of drill rods is required for each hole diameter drilled. Furthermore, the service life of wireline drill rods is considerably shorter than that of conventional rods and the core barrels and bits are costlier. Despite these initial higher costs, the total cost per drilled metre is, in most cases, lower in wireline drilling than conventional drilling. This is due to the considerably higher drilling output which

can be achieved per shift and total drilling time is shortened. For example, the time taken for a full cycle of withdrawing the complete drill string, emptying the core barrel, changing the bit and running back to the bottom of the hole from a depth of 600 metres takes approximately 2 hours in conventional drilling. This compares to 20 to 30 minutes for withdrawing and lowering a wireline inner core barrel for the same depth.

Wireline drilling is economically advantageous in difficult ground conditions where core blockages occur and the core barrel can be retrieved after every metre if necessary, thus economically avoiding core loss. In addition, the Longyear Q system corebarrel incorporates valves in the corebarrel which shut off the water circulation when the inner tube becomes blocked. This is indicated to the driller by a jump in pressure on the water pressure gauge. He can then take remedial action.

Another advantage of the wireline system is that down-the-hole equipment for borehole surveying can be lowered via the wireline hoist without removing the rods from the hole. This means surveys can be conducted quicker, more frequently, and with less inconvenience to the driller. In a similar way, a specially designed retractable wedge for directional drilling, can be lowered and retrieved via the wireline hoist. (R.Bovim - personal communication).

The higher costs of the wireline equipment dictates that the system only becomes economic at a certain depth. For shallower holes, where rod handling is minimal, conventional drilling is more economical. Wireline drilling becomes progressively more economical with deeper holes as rod handling (in conventional drilling), increases proportionally with depth. Bit life is the major limitation and has to be long enough to warrant the use of wireline. Drilling hard quartzite formations where an advance of only 3 metres per bit is obtained, defeats the object of wireline drilling. This relies on long core runs per bit to be economically effective.

Wireline rods are thin and not as rigid as conventional rods and will deviate appreciably more than the latter. They are expensive and have a shorter drilling life. This is particularly so in some abrasive sediments such as karoo sandstones. As the rods are not as strong as conventional rods there is a depth limitation to which wireline can be used (\pm 3000 m.). However, even at this depth, conventional rods may

be added to the top of the drill string and to retrieve the inner tube, these rods must be pulled up in the normal way to expose the wireline rods.

In North America and Australia, the wireline system is used almost exclusively. It should become more widely used in South Africa as increasing labour costs account for higher proportions of the total drilling cost. This will have the effect of outweighing the higher initial cost of the equipment with higher productivity.

3.1.5 Reverse-circulation diamond drilling

This process is also known as the "Con-core", "Continuous Drilling Process" or "Counterflush method". It is a version of the dual-tube reverse-circulation rotary drilling method described previously, but uses a coring bit to obtain drill cores as well as cuttings from the bottom of the hole. This allows for the continuous recovery of the core through the centre of the rod string by means of the return flow of the flushing medium. The core is broken into 12 cm. sections by a core breaker in the inner tube and at the normal pumping rate the core delivery velocity is approximately 190 metres per second. Core and cuttings are collected at surface and together form the sample. Five per cent of the drilling fluid returns slowly up the hole annulus which provides hole lubrication and assures hole stability. (Cummings 1975).

A conventional tri-cone rock bit can be used through formations where core is not required. Cuttings are returned at the rate of 250 m./second when using air. This high return rate makes it practical to suspend drilling momentarily and bag complete samples at designated intervals. (Wilderman 1973).

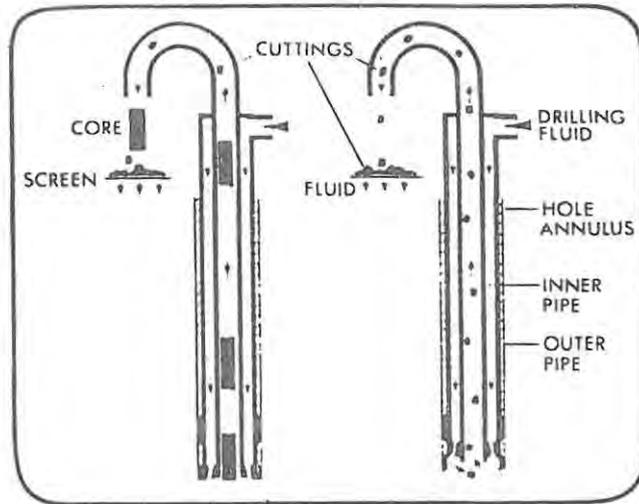


Fig. 17. The reverse-circulation diamond drilling process (after Wilderman 1973).

Billiton Corporation have equipped a pontoon, the "Kontiki Mekanik I", with a "counterflush" core-drill for use in seabed sampling in Indonesia for alluvial tin. This incorporates a double tube rotary system that drills and cases the borehole simultaneously. The advantage of this, in underwater drilling, is that clean water can be used for flushing, instead of the conventional mud. A continuous sample is obtained which is reported to be fully suitable for quantitative core analysis. (Mining Journal, July 27th 1979).

Hopley (1978) records that a world record for core drilling production has been achieved by this system. During an eight-hour shift, a Boart Drilling crew from South Africa achieved a 118,9 m. advance (NQ size) at the Cleveland Potash Mine in Yorkshire, England. He reports that many holes in excess of 1000 metres have been completed along accurately controlled paths using reverse-circulation drilling. This control requires repeated and regular borehole surveys with rapid calculation of survey results on site by computer, otherwise correction to the hole direction is too late.

3.1.6 Core recovery

As the objective of diamond drilling in exploration is to obtain core samples, reasonable precautions should be taken to ensure as complete a core recovery as possible. Longyear, in 1937 regarded the following factors as being detrimental to good core recovery but still partially controllable by the driller:

- i) small core diameter.
- ii) vibration, or chattering, of the drill rods.
- iii) excess drilling speed.
- iv) erosion by circulating water.
- v) grinding of the core by running with a blocked bit.
- vi) dropping core from the bit which, if not picked up before the next run, must be chopped up and washed from the hole as sludge.

To counteract these and to ensure a reasonably high core recovery, as large a sized core should be drilled as is economically justified. Rod vibration should be reduced by: maintaining the equipment in good condition, increasing rotation speed, changing bit pressures or experimenting with different drilling fluids or muds, such as a soap emulsion. Damaged, or worn core barrels and rods are also responsible for a large degree of vibration. (Cummings 1975).

Moehlman (1945), suggests that any competent driller can increase core recovery, in most situations, by carefully adjusting the speed of rotation and forward feed to give the smoothest penetration with least vibration of the rods. He also recommends that when drilling in hard rock, low rotation speed and high forward pressure should be used. In soft rock, core recovery is improved with high rotation speeds and light forward pressure. The amount of water pumped into the hole during the drill run should be only enough to prevent burning of the bit, so that there should not be excess eroding of the core. At the end of the run, if pulling the core is difficult, the drill should be dry-blocked by running the drill without water for a few minutes. This forms a dry mud pack below the core barrel and prevents core fragments from falling out.

The double-tube core barrel increases core recovery by preventing the water that descends through the rods from eroding any of the core. The water is unable to pass through the inner core tube but is circulated through the space between the inner and outer tubes. This is known as

the "M" type double core barrel and is illustrated in Fig. 18. Some double tube core barrels make use of stationary inner tubes, connected to the outer tube by ball bearings. These improve core recovery by preventing friction or twisting of the core.

Under certain conditions core recovery is improved with wireline drilling. Here, the corebarrel and rods are of similar diameter which results in a better stabilised rod string due to smaller clearance in the hole. This results in less vibration and higher core recovery. In addition, the Longyear Q system corebarrel incorporates valves in the corebarrel which shut off the water circulation when the inner tube becomes blocked. This is indicated to the driller by a jump in pressure on the water pressure gauge. (Beukes 1978).

Reverse-circulation diamond drilling tends to float the core to the top of the inner double tube to prevent wedging and to give longer runs with better core recovery. (Cummings 1975). This, in effect, reduces the pressure on the core in the barrel but is no good for soft or friable cores which may be subjected to erosion inside the core barrel.

Knight (1971) reports that during the evaluation drilling programme of the Panguna Porphyry Copper orebody in Bougainville Island, one of the main problems was to obtain a reliable figure for core recovery.

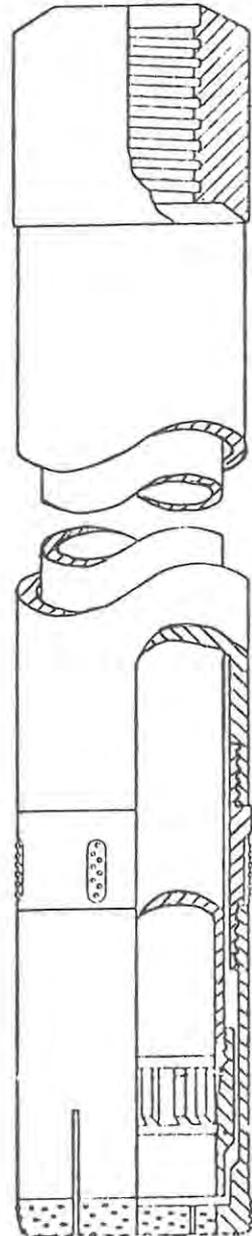


Fig. 18. WM series double-tube core barrel (after Beukes 1978).

Due to a high degree of fracturing, the core arrived at surface in a highly fragmented state which made direct measurement of core length an impossibility. No answer was found to this until core barrels with split inner-tubes were introduced. From then onwards, core recovery improved radically to an acceptable 94,3 per cent. (Knight 1971). The split inner-tube has the advantage that the core is not further damaged or disturbed when it is removed from the core barrel. Further innovations include clear plastic inner-tubes and spring loaded inner-tubes for specialised coring. (Nowlan 1978).

Core recovery in soft formations, liable to erosion by the drilling fluid, can be improved by using bottom discharge bits. These bits are specially designed so that the drilling fluid is passed through the wall of the bit via the kerf thereby eliminating contact with the drill core. This method is popular in Witwatersrand exploration for obtaining good core recovery in the soft, friable, carbonaceous reef horizons such as the Carbon Leader and Vaal Reef.

Another method of obtaining good core recovery under extremely friable conditions is described by Morgans (1970). He advocates the use of a special drilling mud and modified core barrel. The latter has large fluid clearances and an inner tube considerably larger in internal diameter than the core that is cut. Heavy mud is used and the core passes up into the inner tube encased in a coating of mud. When the barrel is full, the mud covered core is slid out and preserved in this state until it is cut open in the laboratory. The method is slow and tedious for the operators, but good core recovery is possible. The core cut by this method is rather small in comparison with hole size, but it is effective in coping with special conditions.

Compressed-air circulation, instead of drilling fluid, was used at Tynagh in Ireland to improve core recovery. Here, the core recovery in soft and poorly consolidated ore materials (such as the residual "mud" ores at Tynagh), was very low with conventional methods. Using compressed-air circulation, excellent recoveries were achieved. (Schultz 1971).

Coal is a notoriously difficult material to core, being extremely friable, prone to washing away and scrubbing. Normally, thin walled bottom discharge bits and triple tube core barrels improve recovery to some extent. Shaw (1976) reports on the use of oilfield-type wireline equip-

ment by the British National Coal Board to improve recovery. Due to the significantly larger diameter core produced, core recovery is improved to a "very acceptable" level. Svendsen (1976) states that there has been a very marked trend towards the larger diameter cores in coal exploration. Currently the HQ (63,5 mm. diameter) and PQ (84,96 mm. diameter core) oil-field tools are being used in the United Kingdom as well as in Canada, Poland, Australia and the United States. The British Coal Board adopted this practise after the successful application of oil-field core-barrels in drilling for tar sands in the Athabasca basin of northeast Alberta. Here the core recoveries were so successful (compared to slim-line diamond drilling), that almost all subsequent exploration was undertaken with oilfield type rigs and equipment. (Shaw 1976).

3.1.7 Pitfalls for the geologist

The responsibility for good core recovery is largely in the hands of the driller. He is often paid an incentive in the form of a bonus requiring him to meet a certain percentage core recovery. Alternatively, a percentage core recovery figure may be included in the written contract between the contractor and client. If this figure is not met, the hole may be repeated at the contractor's expense. However, the geologist in charge of a drilling programme should be aware of certain practices and pitfalls in the drilling operation. Core recovery can be severely retarded by an operator drilling too fast in order to meet a "footage bonus" figure. Similarly, overdrilling a run causes grinding of the core, and unfortunately, this normally occurs on mineralised contacts. Excess pressure exerted on the drilling bit causes an increased tendency towards deflection and thus the target is not intersected at the required position, or is missed altogether.

It is an unfortunate fact that the portion of the core required by the geologist for detailed sampling in the ore zone, is usually the softest formation encountered by the drill. This is the case in: mineralised shears, faults or fracture zones; drilling for coal seams where the hardness contrast between the coal and enclosing sediments is high; drilling for strataform sulphide deposits; drilling through oxidised portions of an orebody; and, in an extreme case, in drilling for a gold bearing "carbon reef" in the very hard Witwatersrand quartzites. In cases such as these, additional precautions should be taken when drilling through the critical zone. Double, or triple tube core barrels (previously

described), and thin-walled bits should be supplemented for the standard bit shortly before the anticipated ore horizon. Garret (1952) describes how this was achieved during the exploratory drilling for the Carbon Leader in the West Wits Line. As the stratigraphy was not well established in those days, it was difficult to anticipate the reef intersection and it was often intersected using conventional coring. However, two deflections using a specialised core barrel (the Garret core barrel), and thinwalled bits, were drilled once the exact position of the reef was known. It was then possible to obtain very good core recovery in the deflections to compensate for poor core recovery in the pilot hole.

3.2 TREATMENT OF DRILL CORES AND SLUDGES

3.2.1 Core handling and storage

As it is removed from the core barrel, the core consists of one or more cylindrical pieces of rock. In exceptionally solid ground it may be in a single piece the length of the core barrel, but more often consists of a number of smaller sections and broken fragments. The core is removed from the core barrel by unscrewing the bit and reaming shell. It is then allowed to slide gently out. If it sticks, the sides of the barrel should be tapped softly with a wooden hammer handle so as not to damage the core barrel.

The core is placed in a core tray which has longitudinal compartments of the correct size to accommodate it. In South Africa, core is commonly placed in the core tray from left to right in the first compartment, with increasing depth to the right. This is repeated in the second groove so that the core may be "read" in the same way as a book is read from the top of the page to the bottom. Other methods are sometimes used but are not common in Southern Africa and should be discouraged as they only lead to confusion.

The first core out of the core barrel represents the last core produced in that drill run. Thus, when placing the core in a core box, sufficient space for the run must be left and the core allowed to slide from the barrel starting from the right to the left. This will ensure that the meterage increases progressively from left to right with increasing depth. (See Fig. 19).

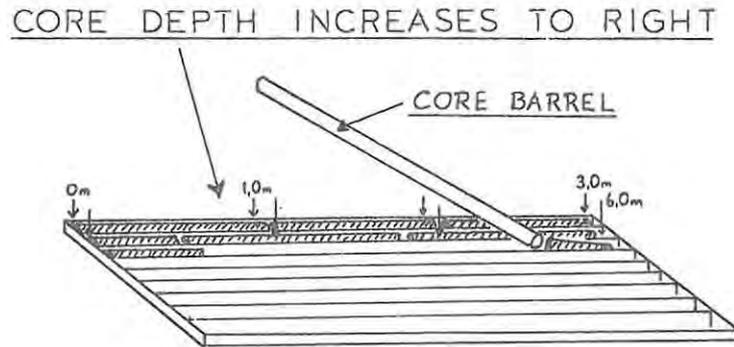


Fig. 19. Method of placing core in a core tray.

When extracting core from a wireline core barrel in which the head has been removed, the first core out of the barrel represents the first core produced. (Cummings 1975). Thus, the laying-out is straightforward and can progress from left to right.

Each "run" is divided from the next by a small block of wood or similar material. On this, the depth of the borehole and borehole number, or identification mark, are displayed. Paint, or indelible ink should be used for this to avoid fading and loss of the information.

The depth of the borehole is ascertained by keeping track of the number of drill rods (which should be of standard length), and correcting for the distance between the collar of the hole and the top end of the last rod. Where core is lost or ground up, care should be taken to have the core in the box represent the exact location in the hole. Thus, the gaps in the core should be filled with round sticks of the same diameter as the core and the amount of core missing recorded on them.

In order to preserve a continuous sample of the core for a permanent record and at the same time obtain an assay of it, each section of core for which an assay is required should be split longitudinally; half of it is preserved and the other half divided for sampling. Numerous mechanical devices are available commercially to split the core. For short jobs it

can be split with a chisel while placed in a length of angle iron. For critical intersections where accuracy is of the utmost importance (such as the sampling of precious metals), the core should be split longitudinally on a specially designed diamond saw.

It is not necessary to split all of the core and often only the mineralised sections are split and assayed. In many cases it is not feasible to split the core and the entire core is bagged and sent for assay. In the evaluation of large, low grade occurrences such as the porphyry copper deposits, literally thousands of metres of core are drilled and all have much the same geological and mineralogical features. In this case to cut and preserve half the core would be very time consuming and expensive. Therefore, it is sufficient to log the core accurately and assay whole core samples. Half of the sample "pulp" is retained for future reference. In such cases it is desirable to maintain a photographic record of the core, which can be stored and preserved at very little cost.

Once core has been sampled, the remaining portions of the core should be carefully stored in a suitable place. The value of the core is not only reflected in the cost per metre of drilling it, but in addition, geological information could exist that may not have been apparent to the geologist who logged it initially. There are many instances on record where old core has been re-logged and new economic minerals discovered which changed the economic possibilities of the deposit. The cost of storing the core is a very small fraction of the cost of obtaining it.

Core trays are made of wood (predominantly), metal, plastic, or cardboard. Each has advantages and disadvantages in terms of cost and durability. The use of poor quality core trays is false economy and will lead to eventual loss of the core. Sheets of corrugated iron, reinforced at their ends by angle iron are cheap and durable if stored properly, i.e. above ground and out of the rain.

Adequate core sheds are expensive but essential for storage. Central core sheds may house many thousands of metres of core and should be designed with a view to optimum space utilisation. The core trays are placed one upon the other in rows of racks within the core shed. Sufficient space must be allocated between racks to enable the core trays to be removed and examined. This space usually accounts for about 50 per cent of the available space in the core sheds.

Improved utilisation of space in core sheds can be accomplished at little additional cost. The space allocated for removing the trays from the racks can be reduced by fitting the racks with heavy duty coaster wheels. They can then be stored next to each other, manhandled into position and easily removed when required. A more efficient method utilises rail-track and wheels. The wheels may be situated at the base of the racks and run directly on the rails, or may be suspended from aerial rails in the core shed. Another space saving method involves free standing racks. These are hoisted from their position in storage by an overhead winch and placed in a convenient area for tray removal when required.

3.2.2 Core logging

Geological core logging begins at the drill site, where instant decisions are made. More thorough core logging, at this stage, may result in expensive, useless drilling beyond the ore zone, or premature hole abandonment. Drill-site logging is not normally recorded but is a simple and quick examination of the wetted core.

At a more convenient time the core is logged in detail. This is done ideally before the core is split and sampled, but since some features are shown up better on a broken surface, it may also be examined after splitting. The first operation in core logging is to ascertain the areas where core recovery is very poor or core loss occurs. These should be recorded carefully and reconciled with the depth of the hole. The cause of the core loss should be investigated fully as it may give some clue as to faulting, cavities or other such features. If faulting is suspected, the suspect zones should be correlated with likely faults on surface or other boreholes. The core sequence is then examined for major lithological changes. These are roughly categorised, examined, and logged in detail. To quote from Peters (1978), "How much detail? As in geological mapping, the potential information is almost without limit and time is a constraint. Unlike geologic mapping, some of the information - half of the core - is kept so that it can be retrieved for more detail. Logging is therefore most often done quickly and for the main objective of the moment. The main objective of the drilling is to find or outline an orebody, not to log core!"

Although this is basically true, the value of the information available must be weighed against the value of the time taken to log the core. If this time is not taken at an early stage, in all probability the core

will not be examined in detail again, and thus the potential information is lost. If the core is not stored in a satisfactory manner, the information is irretrievably lost. Furthermore, structural information, such as fracture spacing, orientation and filling material should be recorded in any event as it will be lost when the core is split for sampling.

The geologist's log varies from a "foot by foot" description written in a notebook, to a tabulated standard format for consistency and filing, or to computer forms and punch cards for storage on magnetic tape. The standard tabulated format for logging is the most commonly used system in South Africa. An example of this is shown in Fig. 20. The lithological changes are symbolised in a geological section on the left, to a standard scale. This is followed by a description of the core. In most core logging this description includes the: colour, fabric, texture, diagnostic mineralogy, type and degree of alteration, state of preservation of the core and nature of the contacts between rock types. A column is normally allocated for recording structural measurements such as dip of bedding, faults, fractures, foliation and lineations. This is in the form of "apparent dip" and will later have to be converted to "true dip" after computations taking into account the direction and inclination of the borehole. Other essential information recorded on this log pertains to mineralisation and alteration, structural and sedimentological discontinuities and percentage core recovery. In the case of the latter, special attention must be given to intervals of lost or broken core as these may represent fault zones or zones of mineralisation.

Check lists, or reminder sheets are commonly used by geologists in logging. These ensure that all pertinent observations have been made and recorded. On large jobs, where secretarial help can be utilised, portable dictating machines in conjunction with check lists can be used to good advantage. Large-scaled coloured photography of the drill core is a useful addition to the log.

In addition to the purely geological information on rock type and mineralogy, logging should also include observations that might be of use in mine and plant design. The competency of the hanging wall of an ore zone and the effect of possible dilution of the ore zone from it should be recorded. Lithological and structural features such as incompetent bedding planes in shales and slates, or strongly foliated rocks such as

BOREHOLE NO. _____ REPORT FOR WEEK ENDING _____
 FARM _____ REPORT NO. _____ PAGE _____

DATE	CHANGE IN STRATA		GEOLOG. SECTION	GEOLOGICAL DESCRIPTION OF CORE	APPT. DIP	
	FT.	INS.				
7. 5. 70:	770	0		Soft strongly broken dark grey shale w. highly irreg black streaks & patches & N grey streaks & patches.	<p><u>PRETORIA SERIES - Time Ball Hill Stage:</u> Above 787': Mostly very strongly broken & steeply tilted very dark grey to black calc. lam. shale, inter-bedded w. grey to black fine calcareous & talcose rock, shony, broken & N. black shale streaks & patches All contacts steep & often highly irreg. At 787' - 804' 6": Mostly finely x-litine, very dark to dull greenish <u>Intrusive rock</u> w. shaley Ken <u>Intrusive rock</u> w. indist. shaly streaks & patches at top, some shale streaks & patches at top. Downward inclusions of <u>Intrusive rock</u> at 800' more uniform, dark green grey - 802' & subal. coarse, spotted <u>Intrusive</u> broken & finely veined. At 804' 6" - 811': Mostly strongly broken black shale as above 787'. Highly mylonitized w. fol chcs & inclusions of <u>igneous rock</u> near top & downward w. pale grey calc. & talcose streaks & to a irregular patches. At 811' - 826": (See foot note, page 3) Grey to pale, mostly fine, rarely subal. coarse granular, fairly soft, highly calc. & in some soft & rather talcose lam in parts. Rare black shale streaks & patches. Fine to coarse, rather talcose rock. Fine to coarse, rather talcose rock. Fine to coarse, rather talcose rock. Fine to coarse, rather talcose rock. At 826' - 834' 6": Black shale, steeply tilted & broken & downward mostly highly mylonitized w. irreg. small to larger, patches & fragments of <u>Intrusive rock</u>. At 834' 6" - 844' 6": Finer greenish-grey <u>Intrusive rock</u> with steep contacts. <u>Intrusive rock</u> & downward w. patches of calc. rock & chert, occ. to several inches. Thicker band of fine <u>Intrusive rock</u> at 834' 35' 6". At 844' 6" - 840': Finer dark greenish-grey, broad unbroken <u>Intrusive</u> talc.</p>	
	772	0				
	773	6				
	775	6				
	778	6				
	780	0				
	781	6				
	786	0				
	787	0				
	790	0				
10. 5. 70:	800	0				
	802	0				
	804	6				
	806	0				
	810	0				
	811	0				
	820	0				
	824	0				
	826	0				
	828	0				
10. 5. 70:	830	0				
	834	0				
	835	6				
	838	0				
	840	0				
	841	6				
844	6					
840	0					

Very steep tilting in fault zone.

Fig. 20. Example of a standard geological log.

sericite-chlorite schist, are important features to record. Such features may become responsible for extremely costly hanging wall support systems during the mining operation, seriously affecting the economics of recovering the ore. Detailed geotechnical observations and determinations of rock strength do not normally fall within the field of the exploration geologist, but rather that of the rock mechanic. However, the exploration geologist can be of vital use to the engineering department by recording faithfully any of the features mentioned above. Furthermore, the behaviour of the drill and its response to different ground conditions can provide information useful in mine and plant design. The "drillability" of the various formations traversed gives a good indication of the hardness of the strata. This information is useful in planning shaft sinking and development, and rock crushing equipment in the mill. Most of this information can be obtained from a combination of the geologist's and driller's logs.

Wilderman (1973) describes a system (his "Field Determinator"), for automatically recording various field parameters for use in mill and plant design. He proposes that a close relationship exists between the drillability factor and the "work index" as required in mill operation. From the automatically recorded log, a means of predicting the "work index" throughout the deposit will permit detailed planning or selective mining decisions to be made at a very early stage.

The unit records such items as weight on the bit, penetration, rotary speed, torque and circulating pressure. Interpretation of one or more of these values can provide a variety of geological information, including formation changes, fault and fracture zones, changes in drillability and an indication of the presence or potential for ground water. Operating information, such as bit life and bit footage, connections, tripping and bit changes, together with general mechanical performance, is also obtained. Chart forms are mounted on a drum carrier and rotated against a bank of pens. Reliability, with no human bias, is assured by the use of these automatic monitoring recorders. (Wilderman 1973).

3.2.3 Core sampling

When core recovery is good, whole core samples are the most accurate method of obtaining a representative sample. However, the core is normally split and half of it retained and half of it divided into convenient lengths for sampling. The split core, unless carefully sawn, seldom

represents an exact and constant length to volume ratio, and the representative nature of individual samples can be adversely affected. All core samples required for assay should therefore be sawn. The half-borehole core may be sawn again so that a 90° segment of the complete core is sampled and the remaining 270° retained, or vice versa. Kock and Link (1970 p.281), demonstrate statistically that in an accurately sawn core section, little additional reliability is gained by assaying the whole core rather than one half of it. Unfortunately, lithology is often more easily interpreted from broken, rather than sawn, core.

The length of core that should be taken to represent each sample interval depends upon the nature of the rock, mineralisation and the "end use" of the data. The tendency is to sample at regular intervals to facilitate mathematical treatment of the data. This practice should be avoided as it leads to a serious dilution of potential information which may be vital in proper ore reserve estimation, mine development and grade control. Instead, the length of each sample should be dictated by natural geological boundaries within the core. If a sample incorporates two different rock types in a core, it is not representative of either of them. Therefore, sample intervals should be matched to lithological or mineralogical changes in the core, irrespective of the fact that this may result in small samples or a great variability in sample lengths. Maximum and minimum sample lengths should be established to facilitate handling and to provide sufficient material for analysis. Competent mathematical handling of the data and the calculation of composite sample values, overcomes any objections to this treatment.

A notable exception to the above method of sample selection occurs in the evaluation sampling (by boreholes) of large, low-grade orebodies where many thousands of samples are required. Here, the geological controls to the mineralisation must first be firmly established before any departure from this sampling method is used. Once this is achieved, cores may be sampled at regular intervals providing that some control boreholes are sampled in the correct, albeit laborious, manner.

Bonsall *et al* (1968) of Falconbridge Nickel Mines, describes a method of core sampling that they have termed "fillet-cutting". Here a continuous fillet of constant cross section is cut (or, more correctly, shaved) from the cylindrical surface of the core for the appropriate sample lengths. The apparatus employs a "tapered, abrasive-set, cutting wheel, the conical

cutting surface of which projects slightly into a rectangular core-guiding trough set parallel to the axis of rotation. The depth of fillet, and thus the amount of sample per unit length can be regulated by adjusting the core guides." (Bonsall 1968). The advantage of this method is that the material is reasonably representative of the core, being a continuous mechanical cut, and it is ready for analysis without crushing or reduction in size. However, the smaller bulk of the sample collected by fillet-cutting yields slightly less precise results when compared to half or whole core analyses. In spite of this, fillet-cutting represents a fairly accurate, cheap, rapid and convenient method of core sampling. The method is capable of leaving undamaged, relatively large portions of the core for future examination, check sampling by further fillet-cutting, and rock mechanic strength testing. (Bonsall 1968).

Samples of split borehole core are usually bagged and labelled and sent to a laboratory for assaying. This involves considerable handling of the sample material before it reaches its destination. In the laboratory it is ground, split and pulverised so that a representative portion of the sample can be tested. This operation requires further handling and the chance of contamination, or mixing of labels, is increased with each mechanical process. To overcome this there have been many attempts to devise a method of instantaneous analysis at the borehole site. Down-the-hole geochemical methods overcome this problem to a certain extent (see Chapter 2.4.5), but do not provide as reliable an analysis as that of a core sample.

In uranium prospecting, core is scanned with a hand-held scintillometer in the field, but is subject to a large amount of background radiation. In an attempt to improve and quantify the method, a system has been developed for the mechanically continuous and step wise, scanning of rock drill core as it passes through a gamma-ray spectrometer. The apparatus is mounted on a 1 x 1 m. worktable and accommodates 3 to 4 cm. diameter core as it passes two opposing 2 inch diameter by 3 inch thick Na I (Tl) detectors. The resulting gamma ray spectre is recorded with a multi-channel analyser and is computer processed. From this, scale diagrams of individual radioelement contents and Th/U ratios in the core can be obtained. (Lovborg et al 1972).

Portable X-ray fluorescent spectrometers were first introduced in 1965 for the in situ analysis of the Ti, Cr, Mn, Fe, Ni, Cu, Zn, Pb, Mo, Sa

and Ba in the field. Although these were initially designed for use on outcrops and working mine faces, they have been found to give more accurate results on drill cores and chippings as well as on powered ore samples. (Gallagher 1970). The portable instrument (known as the PLF analyser), operates on similar principles to those used in conventional X-ray fluorescence. Essentially, this is the response of an element to irradiation with α , β , γ or X-radiation, resulting in the emission of secondary or fluorescent X-rays characteristic of that element (c.f. Chapter 2.4.5 p.34 on "down-the-hole" applications).

Gallagher (1970) reports that the use of the PLF analyser avoids splitting and destruction of the drill core and, for the elements previously mentioned, the degree of accuracy is sufficient for rapid semi-quantitative analysis of solid core at the drill site. The best results are usually obtained on unbroken core in which the core minerals are fine grained and fairly uniform in distribution and, at the same time, difficult to identify without recourse to chemical analysis. A major application of the PLF analyser has been for the rapid qualitative analysis of some 61 000 m. of drill core in Cornwall, leading to more accurate definition of sampling intervals prior to conventional tin analysis. (Gallagher 1970).

Graham et al (1975) describe a semi-portable system for the direct scanning of moving drill core by X-ray fluorescent spectrography. The spectrograph is a portable unit and can be transported, with a portable generator, in a half-ton truck. It can be set up and in operation in a tent within three hours of arriving at the drill site. The spectrograph scans the exposed bottom surface of the drill core, passing above it in a split conveyor, at a constant speed of 610 mm. per minute. A single channel wave-length dispersive spectrograph records continuously the concentration of any chosen element (heavier than titanium), by measuring secondary X-radiation excited in the surface of the core. The non-destructive nature of the method permits successive core spans for different elements, and retention of the core for later physical or metallurgical testing. The equipment operates at a speed of approximately 100 metres per eight-hour shift, at a cost of less than U.S. \$1-00 per metre (1975).

3.2.4 Sludge, and combined core/sludge sampling

In many instances, where core recovery is poor, sludge collection provides the only means of obtaining a sample. McKinstry (1948) states: "Many operators are inclined to disregard sludge because in their experience its assays have not been consistent with those of the core. But failure to obtain a check between the core and the sludge usually indicates faulty technique in collection of sludge samples. Discrepancies may be caused by: (i) incomplete washing of the hole between runs, (ii) loss of drilling water, (iii) either salting or dilution from material higher in the hole, (iv) overflow of fines from the sludge boxes, and, (v) adherence of metallic particles to greased rods." He provides details of measures that can be taken to prevent these factors from occurring. (McKinstry 1948 pp.90-91).

The methods of collecting sludge have been described previously on page 16 in connection with non-core drilling. The technique for diamond drilling is basically the same, with the most important aspect being the collection of all the sludge that represents one "run" of the core. This is because sludge samples cannot be divided into smaller portions than represented by one "run" (or core-barrel), of core.

When combining core and sludge assays, the combined weights of the two forms of sample must correspond to the calculated weight of material that should have come from the hole. If the sample exceeds the expected weight, or is less than it, it represents a gain or loss of sample respectively and is not representative. The usual practice is to analyse the core and sludge from each sample interval separately, and then to combine the assays in proportion to their respective core or sludge recoveries. Various elaborate methods of determining the proportions of core and sludge recovery accurately and hence the methods of combining the assays proportionally, are given by McKinstry (1948), Moehlmann (1945), Cummings (1975), and in a good summary by Pfleider (1972). The degree of reliability of the resulting data can be statistically tested. Koch and Link (1970) provide an example of this based on sludge and core assays from diamond drill holes in the Chuquicamata Copper Mine in Chile. They suggest that you can combine the assays according to the formulas:

$$\bar{w} = \frac{k_1 \bar{w}_c + k_2 \bar{w}_s}{k_1 + k_2}$$

and

$$s_w^2 = \left(\frac{k_1}{k_1 + k_2} \right)^2 \frac{s_c^2}{n} + \left(\frac{k_2}{k_1 + k_2} \right)^2 \frac{s_s^2}{n} + 2 \frac{k_1 k_2}{(k_1 + k_2)^2} \frac{s_c s_s}{n} r_{cs}$$

where k_1 and k_2 are percentage recoveries of core and sludge, r is the correlation co-efficient, and the subscripts c and s stand for core and sludge.

3.2.5 Borehole sample-interval corrections

A sampling borehole, either by intention or by accident, does not always penetrate normal to the strata or plane of the orebody. As a result, the distance intersected between the bounding walls of the orebody is always greater than its true width. For sampling purposes these drilled widths must be corrected by applying simple trigonometry to obtain the true sample width from the apparent sample width. Storrar (1977 pp. 84-91) gives full details of various trigonometrical applications to various problems associated with this. However, the commonest problem requires the determination of the true width of a lode or reef from the apparent intersected width. This can be calculated by applying the following formula:

$$TW = AW (\sin a \times \cos b - \cos a \times \sin b \times \cos c)$$

where TW = true width

AW = apparent width of borehole intercept

a = dip of drill-hole

b = dip of formation

c = angle between direction of dip and direction of hole

(Berkman and Ryall 1976).

A graphical solution to the above problem is also possible and is given by Peele (1945 pp. 9-68) and Berkman and Ryall (1976 p.259).

If the true width of the mineralised intersection is not calculated by one of the methods referred to above, serious overevaluation of tonnage

and underevaluation of grade will occur. Problems can arise in exploration when a hidden geophysical or geochemical target is intersected by boreholes and the attitude of its dip and strike is not known. However, for sampling purposes the apparent dip of the formation can be measured from the contacts in the core. The attitude of bedding in a stratabound deposit, or streaking in a banded vein or lode deposit, are useful parameters to measure. From this information, the true thickness can be obtained by a graphical plot and sample results adjusted accordingly.

3.2.6 Recording borehole data

There are numerous methods for recording borehole data and each exploration company has devised a method to suit its individual requirements. Assay data is usually recorded on a log sheet similar to that used for the geological log. This data should be recorded on the same vertical scale as the borehole section in the geologist's log, for ease of comparison. Failing this, the geological and assay data should both be recorded on one sheet of paper so that sample intervals can be readily matched with the geological description of the core. This material can be filed in the normal way or reduced on microfilm for convenient storage.

A number of computerised methods have been described for storing and retrieving borehole data. In 1969, the U.S. Geological Survey developed a process for storing and retrieving borehole data on coal-bearing rocks in south-western Pennsylvania, by computer. (Kent 1969). A similar scheme was devised to handle large volumes of borehole data from the Carboniferous strata in the Midland Valley of Scotland. (Gover et al 1971). This was followed by an attempt to computerise all the borehole data associated with drift geology (thickness and composition of drift), in the Edinburgh area. (Rhind and Sissons 1971).

In 1972, Blanchet and Godwin reported on the GEOLÓG system which was devised for recording geological field observations, primarily from drill holes, in porphyry type and other mineral deposits. The data compiled on GEOLÓG includes the principal zone interval (i.e. cap, supergene, hypogene, fresh rock, fault etc.), rock type, fractures, alteration mineral assemblages and ore mineral assemblages. The information to be recorded is keypunched directly and retrieved readily from the computer, while rapid computerised interpretations are made of structure and ore reserves. This is because assay data and drill hole location specifications are

written on GEOLOG but are stored on another format ASSAYLOG. Consequently it is possible to merge and correlate ASSAYLOG and GEOLOG data in different ways.

Another such system, COREMAP, is described by Ekström et al (1975). This is a computerised data system for recording and processing data from drill cores and boreholes. With COREMAP it is possible to record general, geological, chemical, joint and borehole deviation data in standard formats. It can also plot the deviations of boreholes, data locations on chosen levels, and vertical sections parallel to the x- or y- axes. All boreholes running through a defined slice of the rock can be projected on a given plane.

3.3 BOREHOLE SURVEYS

Diamond, percussion and rotary holes all tend to deviate from their course. In exploration drilling this is particularly noticeable with diamond holes as they are usually drilled to much greater depths than the other types. Furthermore, the diamond hole is a "slimline" system, i.e. very small rod diameter versus drill string length, compared to the greater diameter of rotary holes, for example, in deep oil exploration. Cumming (1975 p.329), discussing the flexibility of a drill string of 150 metres or more in length states: "If a string of rods could be laid on the ground and raised high enough at the centre without buckling, it might readily assume a curve approximately a semi-circle, and yet could be rotated quite freely, even in the close confines of a drill hole." The causes of hole deviation and remedies to prevent or control it are discussed in a subsequent section.

Directional borehole surveys are essential to determine the degree of deviation of exploration boreholes and thus facilitate the accurate correlation of geological information. Its greatest use is to locate accurately the three dimensional position of borehole samples obtained. A directional survey report on the course of any borehole is an important tool available at even the most remote locations at a fraction of the cost of drilling the hole. The survey shows the inclination from the vertical, direction, and horizontal and vertical location of each survey station measured. Any other points on the course of the borehole can be

extrapolated from these mathematically, graphically or with the use of calculator or computer programmes.

The first survey instruments were fairly crude, either acid bottles or single shot instruments, making use of a compass and plumb-bob with a mechanical timing device. Today magnetic or gyroscopic instruments are available, capable of taking several hundred survey measurements as they are lowered or raised in a borehole. These measurements are recorded on microfilm.

The early instruments are described in detail by Cumming (1975 pp.330-368), Peel (1941 pp.9-64 to 9-68) and will not be described here. Today, there are many makes of borehole surveying instruments, all of which differ slightly only in design principles. Those most commonly used utilise a magnetic compass. Basically, these units are made up of a battery section, a timing control, a camera section and the angle-unit section, containing the compass and plumb-bob. All the components are assembled and stacked in a protective case of non-magnetic material for lowering into the hole. The unit is entirely self-contained and requires no external power. In deep hole surveying, special film is required for the borehole camera. "Hot hole" film can withstand temperatures of up to 300°F. Non-photographic, mechanical survey instruments can withstand higher temperatures of up to 400°F.

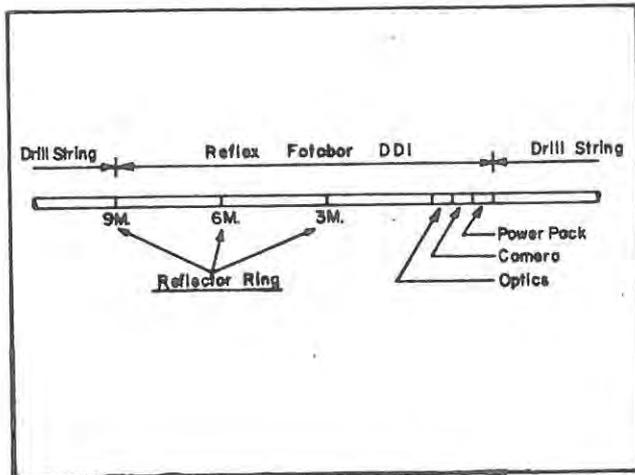
The instruments are manufactured employing either a single shot camera, which takes only one exposure per horizon, or a multi-shot camera. The latter makes a continuous recording of both the direction and inclination of the borehole at a large number of "survey points" in a single run of the instrument in the hole. The data is recorded photographically on a strip of microfilm and is interpreted through a special projector.

The magnetic instruments are, however, subject to faulty azimuth readings due to local magnetic fields caused by the presence of: magnetite, very basic geological formations, pyrrhotite in massive sulphides, or by attractions from drill rods and casings. Instruments using a gyroscope compass instead of the traditional magnetic compass have been developed to overcome this problem and are capable of highly accurate multi-shot surveys. These have the distinct advantage of being able to operate in cased boreholes and within the drill string of wireline rods. In this latter application, the instrument can be lowered and raised on the wire-

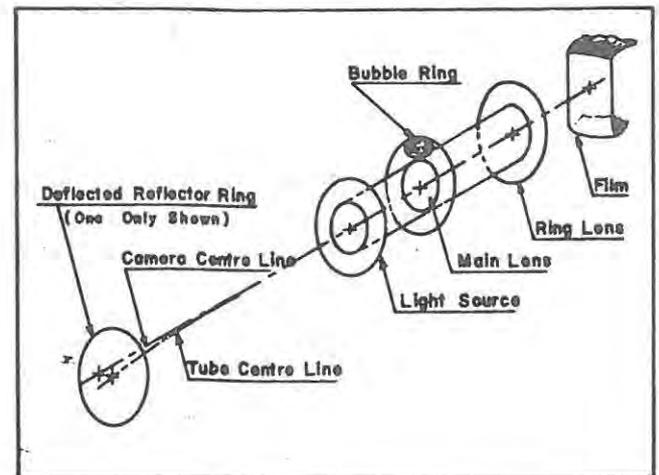
line hoist without removing the rods from the hole. This represents a considerable saving in time.

The gyroscopic instruments are highly priced and hence not a viable proposition for routine exploration drilling purposes. However, in certain instances where magnetic interferences are suspected, they offer a reliable means of obtaining very accurate survey information. They are complicated instruments and are usually operated only by the manufacturer's trained technicians. (Hopley et al 1978). Prior to surveying with a gyroscopic instrument it is tested in a special test stand in which the borehole conditions are simulated. This enables the technician to adjust the gyro accordingly. The instrument with gyro, angle-unit and camera is oriented at the surface prior to lowering into the borehole. For this purpose the entire gyro system is fixed in a certain position in relation to its outer barrel axis. The instrument is aligned with a reference point, usually geographic north, and the compass card plus pointer set to the same point. The survey is then conducted and, after being removed from the hole, the instrument is again oriented to its original direction to detect the amount of drift of the gyro from geographic north. The survey film is developed and interpreted through a special projector in much the same manner as for the multi-shot magnetic instrument. A survey with the gyro, including the orientation at the start and at the end, is performed relatively quickly. A 400 metre borehole, for instance, can be surveyed in about one hour with survey readings taken at 10 metre intervals. (Krebs 1976).

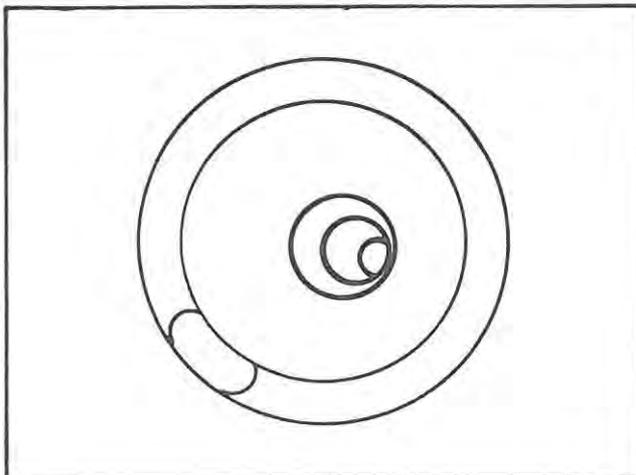
Another magnetic survey instrument is manufactured by Craelius and is known as the Reflex-Fotobar. It consists of a nine metre probe (in three metre sections), containing a minute camera aligned with three reflector rings set at three, six and nine metres from it. (See Fig. 21). As the probe becomes bent by the deviation in inclination and azimuth of the hole, the photographs taken by the camera show the changes in position of the three rings in relation to each other. The film is projected onto a viewing screen from which the angular deviations of the three rings are measured. The unit is undisturbed by magnetic interferences, or casing and can be used inside wireline rods.



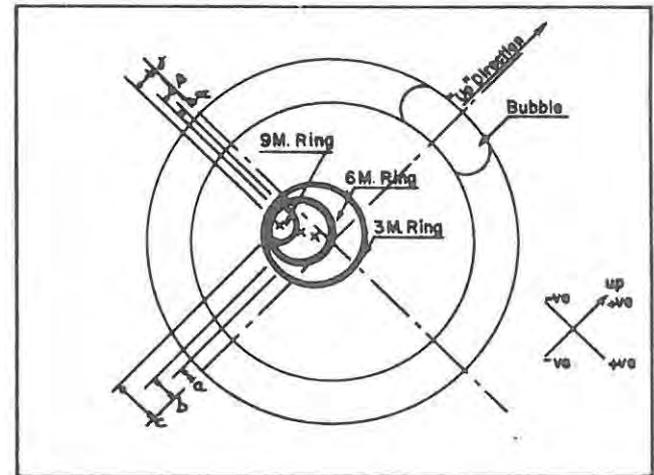
A. Assembly



B. Optics



C. Diagram of a typical survey photograph



D. Measurements taken from each frame

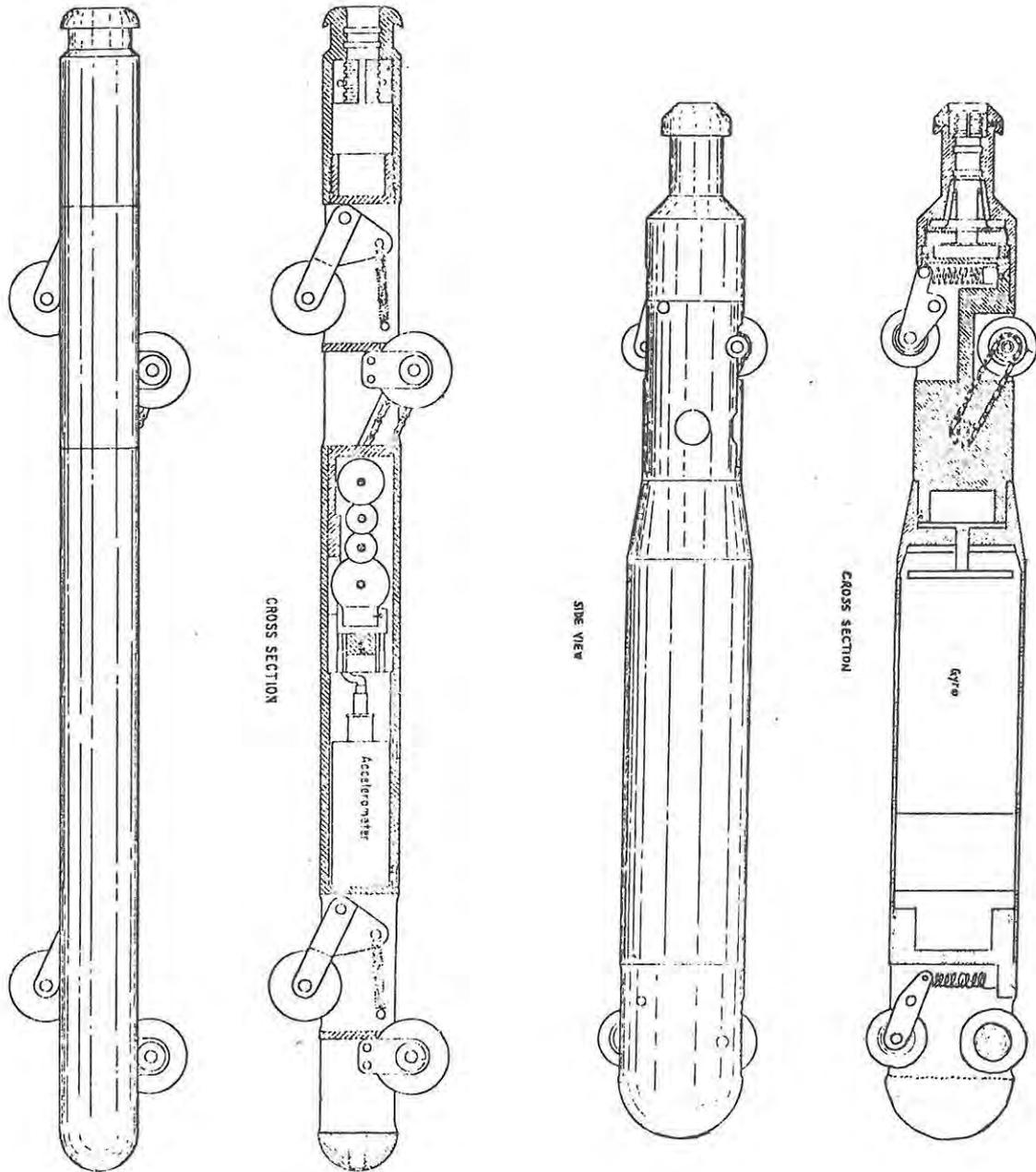
Fig. 21. Operation of the Reflex-Fotobar (after Speakman and Plummer 1978).

Speakman and Plummer (1978) report on the very successful use of a Reflex-Fotobar at Sherrit Gordon's Ruttan Mine. This method was chosen for detailed surveys of a large number of exploratory and evaluation holes because it suited the following qualifying parameters:

- i) It is not affected by the magnetic base-metal orebody.
- ii) Simple to maintain and operate.
- iii) Good mechanical reliability and availability.
- iv) Accurate to within three metres on a 300 metre hole.
- v) Low operating cost.
- vi) The instrument can be purchased rather than rented.

Magnetic survey instruments could not be used because of (i) above. The cost and complexity of operating gyro devices, together with the requirement that they be operated by the manufacturer's technician, eliminated them in favour of the Reflex-Fotobar. Speakman and Plummer state that the accuracy obtained was sufficient for their requirements and in holes which were surveyed more than once, the differences were minimal. They had several instances where it was possible to check the accuracy of the survey. One of the first holes to be surveyed was subsequently intersected in mine development. Its underground survey position was found to differ by only 7 cm. from that of the borehole survey. However, Hopley et al (1978) report that it suffers two serious disadvantages. Firstly, it is extremely long and cumbersome to handle, and secondly, but of greater importance, errors of measurement made at any station, are carried forward and compounded with the next reading. The advantage of a magnetic compass system, compared to this, is that each reading is independent of the previous one and hence errors cannot be accumulated.

Tesch (1976) reports, at length, on tests conducted by the U.S. Bureau of Mines on an "inclinometer" borehole probe. This system operates within grooved casing inserted into the hole. Two probes are used to obtain the data necessary to calculate the hole position versus depth. One of the probes contain a gyroscope, and the other contains a biaxial force-balance servoaccelerometer. Guide wheels on the probes run in longitudinal grooves in the inside walls of the casing. (See Fig. 22). Data from transducers in the probes is automatically punched on paper tape as the probes move up or down the hole at approximately 10 metres per minute. This data is later used as input to a computer programme which calculates the hole position versus depth.



A. Inclinator probe

B. Gyroscopic probe

Fig. 22. Automatic borehole recording devices used by Tesch (1976).

A variation on the method described above, known as a "Portable Borehole Deflectometer", is described by Lauden (1977). This instrument is an articulated probe consisting of two fixed-length arms each one metre long, connected by a deflection angle transducer of high resolution. It is designed for use in smooth walled holes, pipes, drill casings, or in grooved casings between 50 and 75 mm. in diameter. The measured deflection is the relative displacement of three fixed wheels on the probe, from a straight line. (See Fig.23). The transducer is a strain-gauged cantilever and readings are taken on a standard strain-gauge bridge with a resolution of one micro-strain. Each reading is subtracted from the undeflected reading of the instrument to give the deflected angle between the two arms at that point. Measurements are taken at metre intervals on the way in and out of the hole. The readings are recorded electronically. As these represent incremental offsets, they have to be tabulated and summarised to obtain the deflection along the length of a borehole. The instrument is uncomplicated and robust and has the advantage of a readout unit outside the borehole.

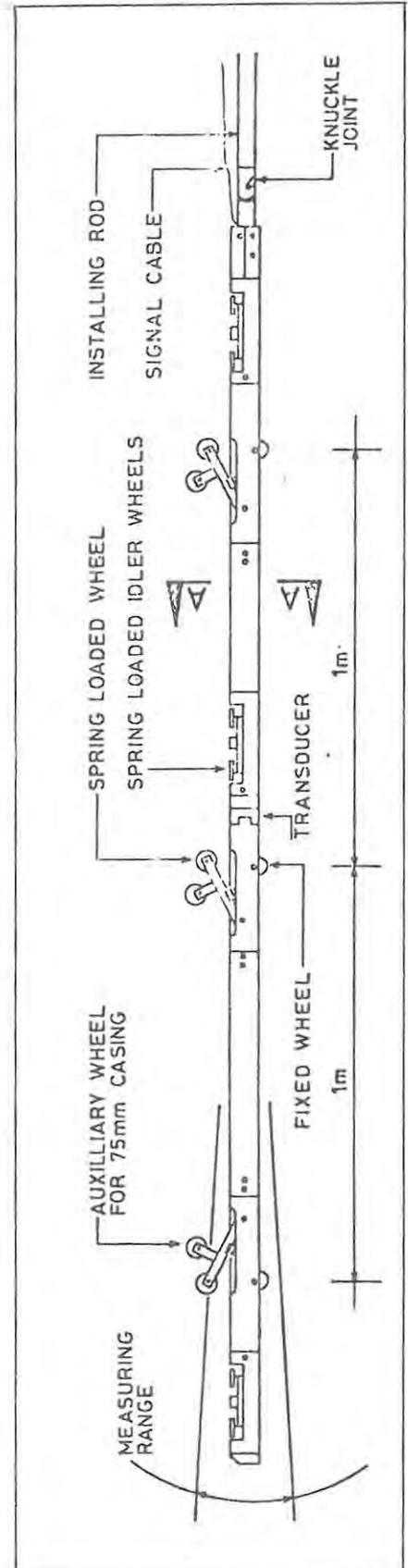


Fig. 23. Borehole Deflectometer (after Lauden 1977).

Of all the instruments mentioned, the most practical for exploration purposes remain the single or multishot magnetic (or gyroscopic) surveying instruments. These have the distinct advantage over the inclinometers and deflectometers of not being incremental, i.e. each reading is individual and not dependent on the reading before it. Thus, errors are not compounded and one point errors are easily noticed. The advantage of the inclinometers and deflectometers is the fact that they have a continuous readout on surface, allowing instant plotting or automatic recording of deviation data. However, this facility has now become available with electronically recording magnetic survey instruments. This data is computerised and the latter automatically provides plans and sections of the borehole track. (B.P.B. Instruments Brochure). However, this equipment is complex, expensive and needs the care of qualified electronics experts. Pritchard-Davies (1971) see the future of such continuous deviation surveys as belonging to a combined package service that can evaluate holes on completion, e.g. hole survey, temperature log, magnetic log, radiometric log etc.

3.4 ORIENTATION (STRATOMETRIC) SURVEYS

The determination of the attitude of planar features such as bedding, faults and fractures from borehole core forms an important aspect of any exploration or evaluation drilling programme. This is possible using various orientation survey techniques ("stratometric surveys") for producing specially marked and oriented cores. This has fulfilled a need that has been apparent since Mead (1921) wrote ".....since the core cannot be removed from the hole without losing its orientation relative to horizontal direction, the strike and dip of the beds cannot be determined by observation on core from one drill hole." Zimmer (1963) provides an excellent summary of the history and early attempts of core orientation. He also describes the prototype of a very popular method in use today, that of scratching the core, which will be described more fully in a succeeding paragraph. His method incorporates, with the orienting device, an acid etch tube for simultaneous inclination surveying. Once out of the hole, the scratch on the core is aligned with a scratch on the acid tube and they are both placed in a two circle goniometer. The goniometer is rotated until the acid tube etch is in a horizontal position and then aligned for the azimuth reading on the horizontal base scale. The

azimuth is assumed to be that of the borehole collar direction. The core is then considered to be in its correct orientation and dip and strike can be measured.

There are a couple of well field-tested methods currently in use in South Africa for obtaining oriented cores. One of these utilises the following method: a line is mechanically scratched, or scribed, in the rock across the bottom of the borehole and, simultaneously, a photograph is taken by a borehole survey instrument recording the position of the scribed line in relation to the compass dial. The photograph also records any deviation of the borehole from the vertical position, as represented by a plumb line. Drilling is then resumed and the marked piece of core recovered.

The main disadvantage of this method is that caving of the hole prevents the scribing tool from marking the bottom of the hole clearly with the scratched line. This problem is overcome by the Christensen-Hugel method which scribes lines continuously along the length of the core as it is cut. Three grooves are cut by triangular, hardened "knives" in the core as it enters the core barrel. The orientation of these grooves in relation to the survey compass is simultaneously recorded on film by a multishot borehole camera. Through a common reference "log sheet", the film, grooves and core depth are correlated for interpretation purposes.

A similar version to this (developed by Eastman), utilises a single groove and single shot survey equipment. The photographic recording of the survey data and groove orientation is taken before the core is broken from its base and pulled to surface.

In all the methods mentioned, the core is interpreted in a mechanical orienting device known as a Goniometer, or Corex Reader. (See Fig. 23). The marked core is clamped in the Goniometer and rotated in both horizontal and vertical planes until it is aligned in accordance with the borehole survey data. It is then rotated about its own axis until the reference groove is aligned with the bearing recorded on the survey photograph. The core is now considered to be in an orientation similar to that in which it occurred in situ, and structural measurements (such as dip and strike), are recorded from it.

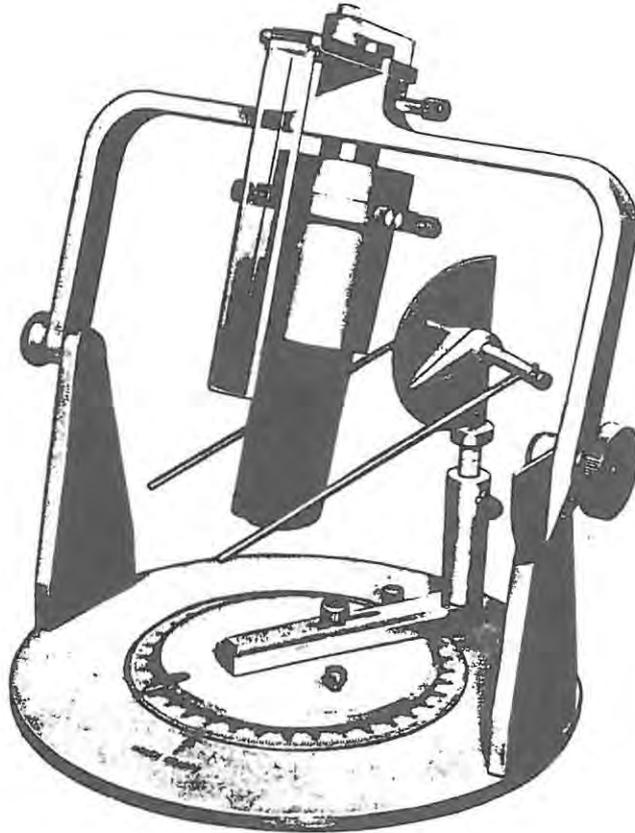


Fig. 23. An Eastman Corex Reader

The Christensen-Hugel and Eastman methods are used with conventional NX and EX rods. However, the last couple of rods and the core barrel should be made of non-magnetic material. The non-magnetic core barrel can be raised and lowered inside the rods in a wireline system. This represents an efficient and rapid method as the conventional wireline core barrel can be replaced with the barrel designed for the oriented core when necessary, without having to raise and lower the drill string.

The measurement of fracture orientations in non-cored boreholes is fundamental to oil exploration. This is described by Zamenek (1970) who uses a down-the-hole borehole televiewer (cf. Chapter 2.4.1, p.24) for this purpose. Babcock (1978) reports on a method of measuring the overbreak of a borehole encountered where steeply dipping subsurface fractures intersect the hole. The position and intensity of the overbreak is measured by the caliper portion of a 4-arm dipmeter lowered into the hole. These recordings, Babcock suggests, constitute a useable tool for measuring

subsurface fracture orientations. There is no record of the application of these methods to the minerals exploration industry as yet.

3.4.1 Applications of stratometric surveys in exploration

One of the earliest recorded applications of stratometric surveying in South Africa was during the exploration drilling of the Klerksdorp goldfield by the Western Reefs Exploration and Development Company. The auriferous horizons in this area are highly faulted and dip in a number of opposing directions, necessitating closely spaced evaluation boreholes. After the introduction of stratometric surveying techniques, the number of holes required to elucidate the structure of the area was significantly reduced. (Mining Journal, 18 June 1954).

Determination of the attitude of dip and strike of bedded units in borehole core is the principal application of stratometric surveying. However, for structural analyses, the attitude of foliation, cleavage, joints, contacts, veins, shears etc. can be determined. The method is very useful for determining the direction of dip of a fault, and hence its direction of throw. In sedimentological studies, transport direction can be ascertained from a stratometric survey of selected portions of oriented borehole core. Cross-bedding directions are of value in defining and predicting the direction of transport and thus, disposition, of fluvial sands in channel deposits. Pebble orientation can similarly be used. From this data, channel direction and, hence, trends can be delineated for follow-up drilling. This is particularly applicable in the Witwatersrand where recent drilling programmes are concerned with delineating higher grade channels (pay shoots) in low tenor reefs. This is also applicable to the Black Reef and Lower Witwatersrand reef horizons which are becoming increasingly more attractive propositions with the rising gold price. (\$504/oz. 27-12-79.) The fluvial-channel environment is also host to uranium mineralisation in the Beaufort Group of the Karroo Supergroup. Stratometric surveys and the subsequent early delineation of channel trends would prevent unnecessary blind exploration drilling. In oil exploration, the elucidation of sub-surface fracture directions is of great use in predicting the porosity and flow direction of fluids (oil, water, brine, etc.), through the strata. The trends occupied by reservoir horizons are also obtainable from oriented core.

In igneous and metamorphic terrains, mineralisation is invariably oriented in relation to linear tectonic features. Micro-features in drill core usually reflect the trend of macro-tectonic features in the area. The features in core thus provide a clue to the structural configuration of the sub-surface geology. It follows from this that if oriented cores of sub-surface structures are obtained, the structural history will be defined more accurately. From the direction of lineations, boudins, etc. observed in oriented cores, fairly accurate trend predictions can be made of sub-surface ore bodies. Petrofabric analyses of thin sections from oriented cores are a tool with a great potential in exploration for this type of mineralisation.

During the exploration and evaluation drilling of lode deposits, it is normally difficult to correlate mineralised intersections between boreholes. This is due to the erratic nature of the mineralisation and the fact that multiple lode intersections are often encountered in the drill holes. The use of oriented cores should facilitate more accurate correlation between holes compared to the "random choice" correlation of unoriented cores. Fracture trend analysis is important in "porphyry" deposit evaluation and oriented cores should delineate these trends at an early stage.

The above mentioned examples are a few of the many applications of stratometric surveys in mineral exploration drilling. As drilling costs continue to rise it is essential that the maximum possible geological information is gained from all boreholes. For little additional expense, a comprehensive borehole survey programme and the collection of oriented cores on key horizons, should become standard practice. In the not too distant future it is hoped that all core obtained from a diamond drill hole will be automatically oriented during the drilling process. This would represent the most significant advance in borehole technology since the development of multi-shot directional surveys.

3.5 BOREHOLE DEVIATIONS AND DIRECTIONAL DRILLING

3.5.1 Deviation

In this account the term "deviation" is used only to signify unwarranted wandering of the course of the borehole. The term "deflection" refers to a planned change in direction and is also used to signify additional target intersections drilled out of a pilot hole.

Cummings (1975 p.329) lists the forces that tend to cause deviation in drill holes as follows:

- A1. Drill rods which are worn and hence much smaller than the drill hole size.
- A2. Drill rotation that, even in homogeneous formations, tends to produce a clockwise spiral in the hole.
- A3. Stratified, schistose or gneissic formations through which the drill hole passes at an angle. If the hole is nearly at a right angle to the stratification, it tends to become more normal to it. However, as the angle between the hole and stratification lessens, at some critical angle it tends to follow these planes. The walls of veins, faults and dykes often produce the same results.
- A4. Formations consisting of alternating hard and soft layers tend to force the hole towards the softer material.
- A5. In very flat holes, the crown tends to float or flatten out. Deviation may also be lateral as well as vertical.
- A6. Factors, such as type of crown, rate of feed, crown load and circulating fluid supply, all have some influence on hole deviation.
- A7. The presence of crevices or vugs in the formation.

The controls which can be applied in order to prevent deviation or to promote deflection include the following:

- B1. Rotational speeds, rate of feed, crown pressures and circulating fluid supply, all of which can be varied as required by conditions.
- B2. Special diamond crowns such as pilot crowns, and crowns with various face contours are used.
- B3. Among special devices used interchangeably for straightening or deflecting drill holes are whip-stocks, plugs and wedging devices.
- B4. Long and full diameter (that is, not worn), core barrels.
- B5. Full diameter rods, with good connections, for full length of hole.

- B6. Free-cutting and correctly set diamond crowns.
- B7. Stabilisers or guide rods.

(Modified after Cummings by Williams 1978).

The worst cause of deviation, and the one that is most difficult to control is A3 above (deviating so as to approach at a direction of 90° to the bedding, or to parallel it when close to the bedding). This is due to the differing drillabilities of harder and softer strata encountered during drilling. Very often, the bulk of the strata is relatively soft but contains layers of harder, resistant material, such as chert rich horizons in dolomite. The reason for this deviation is summarised in Figs. 24 and 25 below.

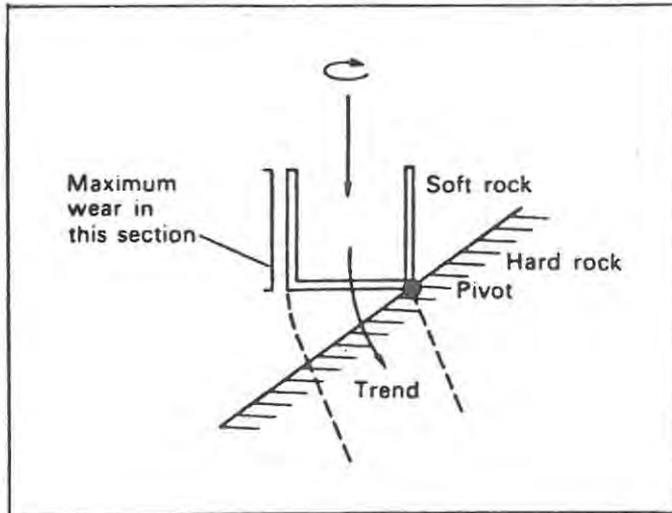


Fig. 24. Deflection occurring through pivotal movement in soft rock at a soft-to-hard interface.

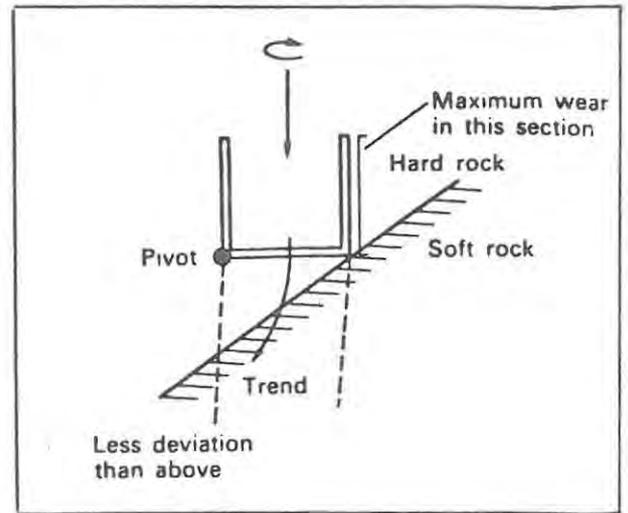


Fig. 25. At a hard-to-soft interface, pivotal movement tries to take place but is restrained by the harder rock.

(after Pritchard-Davies 1971).

From this it can be seen that the tendency to deviate into the bedding is due to the rapid wearing of the soft rock when pivoting on a soft-to-hard transition. On the opposite hard-to-soft interface, the pivoting would tend to prevent this. Pritchard-Davies (1971) suggests that the only way to overcome this would be to increase pressure on the lagging side of the bit relative to a decreased loading on the other leading side as shown in Fig. 26. However, this is impractical with modern rotary diamond drilling equipment.

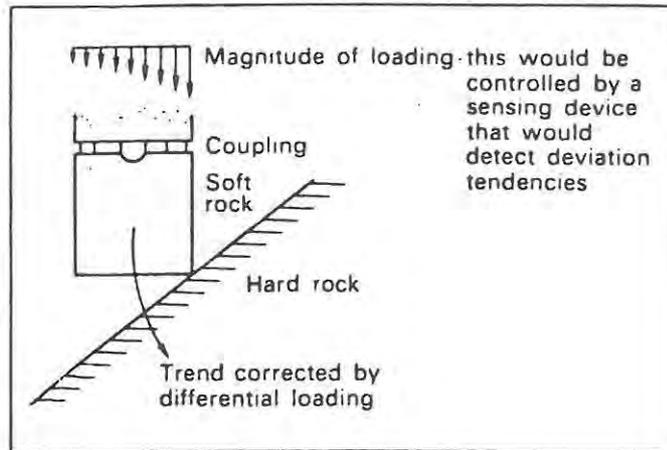


Fig. 26. Illustrating how a corrective loading would restrain deviation. Such a device has not yet been invented. (after Pritchard-Davies 1971).

He also suggests that the use of different bit designs would remedy this situation to a limited extent. By modifying the kerf profile it would be possible to increase or decrease the tendency to deviate into the strata. For example, Fig. 27.

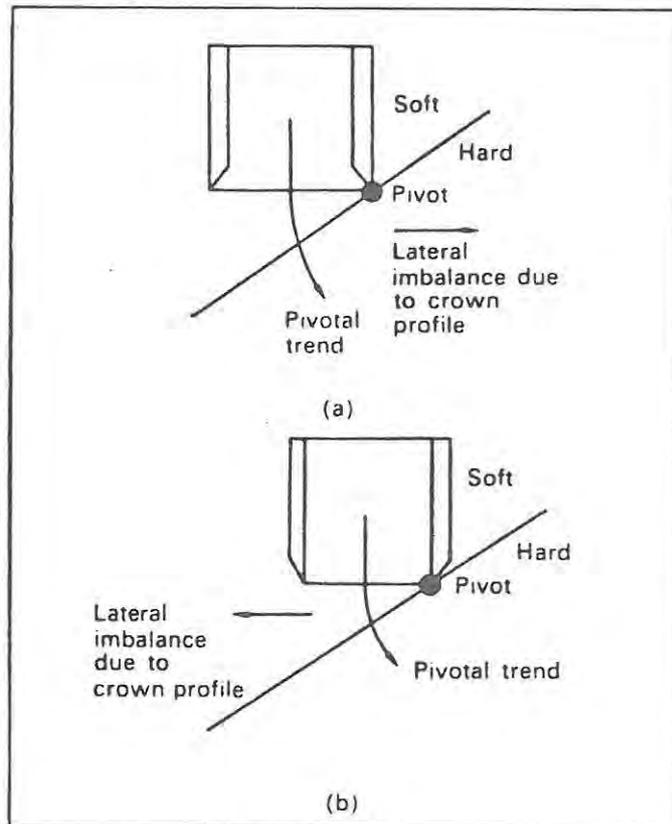


Fig. 27. Bit face profiles (a) to increase and (b) to decrease deviation into the beds.

Deviation may be remedied by wedging and arc cutting, if detected by borehole surveys in time. However, this is often ineffectual and only deflects the borehole path for a short distance before it returns to its deviation trend. (See Fig. 28). The geologist can alleviate the problem to some extent by predicting the effect of the stratification on the course of the borehole before choosing a drill site. In this way the hole should be planned to intersect the stratification at right angles and thus keep a fairly constant course. (Borehole 2 in Fig. 29). Alternatively, the hole could be started vertically, but allowance made for it to deviate slightly and, if planned correctly, towards the target. (For example, Borehole 3 in Fig. 29).

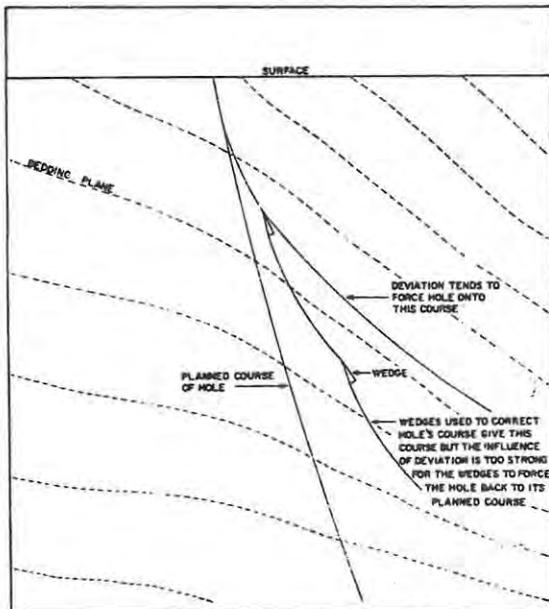


Fig. 28. Cross section illustrating the failure of deflection methods to restore a borehole to its planned course. (Hughes 1965).

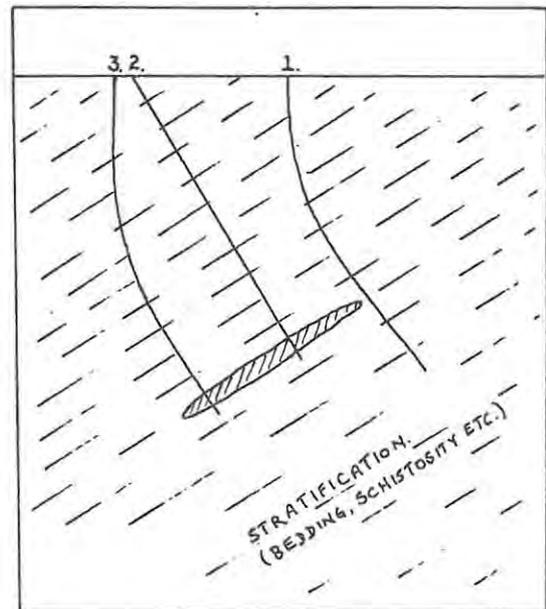


Fig. 29. Predicting the course of a borehole and making an allowance for deviation.

The force of gravity plays a greater part in deviation than is generally suspected. The mass in this instance is the rodstring and the gravitational pull is downwards through the centre of gravity of the rodstring. In flat holes this becomes especially noticeable where the pull of gravity on the rodstring will cause the bit to deflect upwards.

Garret (1952) attempted to solve the problem of borehole deviation mathematically. He pointed out that the thrust on a B size bit, when drilling

hard quartzite, must be of the order of 2000-3000 pounds (910-1360 kg) and that the rod and core barrel string is incapable of supporting this load. As a result, it buckles and can only operate as a series of bends where it touches the sides of the hole in wavelengths of ± 3 metres. Wear marks can be seen on a core barrel string to substantiate this. Moreover, examination of these wear marks indicates that the wear at any given point is always on the same spot. This indicates that the rod string does not flex lying on one side of the hole, but rotates in a sinuous axis. From this, and assuming a clearance of 2,9 mm between corebarrel and borehole, Garret calculates that a deviation of $0,02^{\circ}$ is possible. Even if this figure is doubled, and the angle of deviation increased to $0,06^{\circ}$, it is still far from sufficient to explain the excessive deviations that have been recorded.

This work was followed-up by Liebenberg (1971) while trying to explain severe deviations at the Klein Aub Copper Mine in South West Africa. Vertical exploration boreholes drilled from surface deviated so severely that they tended to flatten out and assume a near horizontal course before reaching their target. The average deviation arc, measured in section, had a radius of 372 m.

While Garret applied his calculation to buckles in the drill string, Liebenberg suggests that the corebarrel itself buckles, and being closer to the bit, it would cause greater deviation. Applying Garret's formula, he calculated the deviation arcs that would result from corebarrel buckling at various distances behind the bit: e.g.

<u>Distance behind the bit</u>	<u>Deviation Arc</u>
1 ft. (,30 m)	128 ft. (39 m)
2 ft. (,61 m)	511 ft. (156 m)
3 ft. (,91 m)	1149 ft. (350 m)
4 ft. (1,11 m)	2043 ft. (622 m)
5 ft. (1,52 m)	3191 ft. (973 m)

In an attempt to apply this data to his problem, he measured the diameter of a worn BX corebarrel and found that excessive wear occurred at a point 1,04 m behind the bit face. This correlated sufficiently with the calculated data which suggested that the corebarrel would buckle at 1,02 m to obtain the radius of 372 m in the deviations.

From this, Leibenberg concluded that boreholes deviate because corebarrels are not rigid enough to transmit the thrust normally required by a bit to penetrate effectively. The closer the point of flexing to the bit face, the greater the deviation. However, given a corebarrel-bit combination that will not flex under the thrust required for effective penetration, an almost perfectly straight hole can be drilled.

One way of reducing the buckling effect of the corebarrel is to reinforce it or to build it up with numerous spots or strips of hard metal to bring the outer diameter of the corebarrel as close as possible to hole diameter. A commercially available "guide tube" has been produced by Sandvik to increase the rigidity of the corebarrel and first drill rods. This is made of hardened steel and a star shaped cross section with an external diameter slightly less than that of the drill hole. The drill steel can rotate inside the tube, independently of the latter, thus reducing external wear on the tube. (Drilling News Spring 1979, p.35). This item of equipment appears to fulfil the requirement of preventing undue deviation and increases the chance of drilling a straight hole.

Worden (1978) describes the problems involved in minimising deflections in pilot holes for raise boring equipment. He concludes that bit selection, bit weight and rotary speeds are of fundamental importance to prevent deviation. However, he also suggests that if the bottom of the hole is not properly cleaned, excess weight is required for penetration, and therefore, deviation results. Thus, a control on the hydraulics to ensure proper down-the-hole cleaning also becomes necessary.

3.5.2 Controlled or directional drilling

In controlled or directional drilling the factors that cause deviation are used to advantage to guide the hole towards a target. Directional control of drilling is not usually required in shallow exploration holes. However, when a discovery is made or extensions to ore bodies are sought, borehole paths become increasingly important. In the case where deep drilling near known mines or deposits occur, the targets are often well defined while the collaring positions may be limited. Boreholes should be drilled with a view to obtaining maximum geological information for the cost budgeted. In this way, an underground borehole may be drilled from deep mineworkings and guided towards a prospective target, improving the knowledge of the immediate mine surrounds at the same time. Various

examples and applications of controlled drilling programmes are presented in Section 3.5.3 following. Before these are discussed, the theory and techniques behind controlled drilling should be evaluated. The first and most important aspect of controlled drilling is to keep the borehole on course.

"The secret of controlled drilling lies in controlling the Bit Weight and R.P.M." (Montgomery 1971). Bit weight is the weight to be applied to the bit to overcome the resistance of the rock and to keep the bit impressed to a uniform depth of cut whilst being rotated. It is made up of rod weight plus "top" weight minus "up" weight. Top weight is the force provided to penetrate the rock while drilling and is added to the weight of the rods on the bit. "Up" weight (or "hold off" weight) is applied when the effective rod weight is in excess of the total required bit weight. (Montgomery 1971). These parameters must be strictly controlled to ensure that excessive bit weight does not occur. If excessive bit weight does occur, then the drill string and corebarrel will tend to buckle and cause excessive deviation. Therefore, in controlled drilling, it should be ensured that only the part of rod weight that is required as bit weight acts on the bit. Excessive rod weight must be taken up and recording devices on the top and bottom hydraulic cylinders should continually monitor this parameter. Continuous recorders should also be employed to monitor rotational speed, rate of advance, water flow and pressure as well as bit weight. This would simplify the controlled drilling technique and most of the decisions required for changing the controls can be related to certain levels on the machine gauges. (Pritchard-Davies 1971). Prior to the introduction of such recording devices, changes were made at the drillers' discretion and were based on experience and "feel". This method sufficed for the less complicated drilling assignments but with the increased need for accuracy in controlled drilling, some standardisation and automatic recording of drilling parameters should be kept. The logs provided by such recorders can be correlated with the borehole survey log and the cause of excessive deviation in the latter will usually be attributable to excessive bit pressure, and recorded on the automatic recording log. In this way, the geologist in charge of a drilling programme can monitor the driller's actions and ensure that excessive bit pressures are not employed, thus decreasing the chance of deviation.

The other aspect of controlled drilling is the use of borehole deflections. This term incorporates two common practices which are: 1. The controlled deflection of a "master" (pilot) hole by various means to cause it to change direction from the course on which it was heading; or 2. The controlled drilling of one or more "secondary holes" (deflections) from a single "master" (pilot) hole.

The use of wedges in the hole to make a controlled change of direction is by far the most commonly used deflection technique. A wedge normally has a maximum deflection, or correction, of $1^{\circ}30'$ and in order to obtain a greater angle, many wedges must be used. Wedges can be used to re-align a drill hole that has deviated from its intended direction. They can also be used intentionally to deflect a drill hole from its original course in order to:

- i) Make an intersection of a suspected ore occurrence;
- ii) Make additional intersections of an ore occurrence;
- iii) By-pass rods, core barrels, drill crowns or other tools which may have become stuck in the drill hole;
- iv) Counteract the tendency to deviate in response to dipping stratification (schistosity, bedding, etc.).

The widest application of wedging is, however, to deflect secondary drill holes over wide areas from a master drill hole, resulting in multiple intersections as widely spaced as may be desired from one master hole.

There are many wedging devices and methods available. Some of them are of a permanent nature and once inserted in the drill hole, remain there. Others are of a recoverable type. Of the permanent type, the Hall-Rowe wedge (See Fig. 30), is the most universally used. It has the advantage of not only deflecting a drill hole, but deflecting it in a definite direction in relation to the previous direction of inclination. Unfortunately, this type of wedge requires several trips in and out of the hole before the completion of the deflection. These trips are necessary for locating, surveying, orienting and re-surveying the wedge in order to align it correctly. This fact makes it an expensive and time consuming procedure.

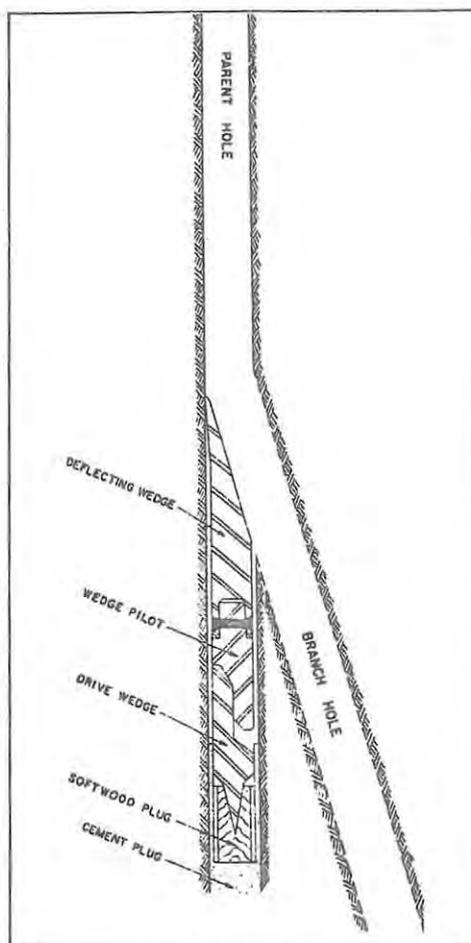


Fig. 30. Application of the Hall-Rowe wedge to start a branch hole. (Hughes 1965).

Retractable or retrievable wedges have been designed for use with wireline equipment. These, and the accompanying surveying instruments, are designed to work inside the wireline rods and are raised and lowered by the wireline hoist. This represents a significant saving on time and expense. The disadvantage of the system is that, after surveying, the wedge has to be correctly oriented to the required direction by turning the drill string by hand, at the collar of the borehole. Due to the flexibility of the drill string, this is a time consuming trial-and-error procedure. A further disadvantage is that the deflection can only be undertaken at the bottom of the existing hole as the wedge is required to be "locked" in position against this face. Thus, secondary deflections from nearer the collar are not possible. (Williams 1978).

The expense and obvious disadvantages of wedging have increased the necessity to find alternative methods of directional control. The

emphasis has concentrated lately on harnessing the forces that cause deviation and directing them to provide useful deflections. For example, if an operator increases the bit pressure on a deviating hole, he will cause it to deviate more strongly in that direction. This is used to advantage in making a flat borehole climb more steeply or to deflect secondary holes rapidly away from the master hole. Liebenberg (1971) suggests that pressures of 10 000 lbs. (4545 kg) cause drastic deflections and a change in direction of up to 30° per 30 m can be obtained using AWY rods. Under these conditions a non-coring "bull-nose" bit is substituted for the normal diamond crown. Pressures of this magnitude require modified drilling machines, or ones in first class mechanical condition. Hydraulic machines should be fitted with pressure gauges on the top and bottom cylinder, pressure relief valves and flow regulators, so that the load on the bit can be determined at any moment by reference to a graph. Chuck speeds as high as 2000 R.P.M. are used with rate of penetration of up to 15 cm. per minute.

The inherent flexibility of the core barrel can similarly be used to advantage. In the preceding section (3.5.1), it was pointed out that sharper deflections were caused by core barrels buckling closer to the bit. Thus, given a core barrel that can be made to flex at a required distance behind the bit face, a circular arc of required radius can be cut. This can be made possible by mechanically skimming, and thus weakening, the core barrel at the designated point behind the bit. However, if such precision is not required or deemed necessary, an old and worn core barrel will have a similar effect. (Liebenberg 1971).

Hopley (1978) foresees a future for directional control in the miniaturisation of steering tools and in-hole turbo-drill units suitable for "slim-line" exploratory holes. The oil industry currently use these in deflecting their larger diameter rotary drilled holes. Eventually it is hoped that electronically controlled steering units and continuous surveying apparatus might enable boreholes to be piloted in much the same way as a guided missile is directed in the air.

3.5.3 Various examples and applications of controlled drilling techniques in mineral exploration

One of the best documented case histories of controlled diamond drilling is by Liebenberg (1971) and has already been referred to

extensively in the previous sections. Liebenberg, working at the Klein Aub Mine in South West Africa, was faced with a Board resolution "calling for a higher rate of exploratory drilling, more ore per foot drilled and a reduction in cost per foot." To achieve this he embarked on a programme of controlled drilling with accurately controlled master holes and multiple deflections from these master holes. The results of this are summarised in Fig. 31. From this it can be seen that multiple ore intersections were achieved using a few master holes and a considerable saving in footage resulted. His methods of obtaining the sharp deflections have been discussed and involve using weakened core barrels, non-core bits, and high bit pressures.

Tsumeb Corporation, drilling in the Otavi Mountainland of South West Africa, utilise a large diameter rotary drilled master hole from which diamond drill holes are deflected towards the target. The object of the large diameter rotary pilot hole is to prevent deviation (cf. section 2.2.1, p. 11) in an environment where diamond drill holes deflect strongly in response to the bedding. Once in the vicinity of the target, multiple short diamond drill deflections are used to intersect and evaluate it.

Long inclined boreholes drilled from underground mine workings have become popular in the last couple of years. These are used essentially to prove further ore reserves along the strike or down dip of a tabular stratabound ore deposit. The obvious advantage of drilling from underground is a saving on long vertical boreholes from surface, where mine workings and prospective target are at a considerable depth. An inclined hole is normally drilled from a development drive in the footwall of the ore horizon and maintains a course below it. From this "master" hole, multiple secondary deflections are drilled to intersect the reef horizon.

One of the earliest applications of this method was by Boart Drilling in the Western Deep Levels Gold Mine in the West Wits Line. An inclined borehole was drilled from a point nearly $2\frac{1}{2}$ km underground for a length of 1237 m to intersect the Ventersdorp Contact Reef down-dip. The borehole was collared at an inclination of minus 34° and through wedging and directional control was allowed to curve gradually upwards until intersecting the reef horizon at an inclination of plus 15° . (Mining Journal, December 17 1976). Since then other holes have been drilled on the mine with similar success, and on the neighbouring Doornfontein Gold Mine and on gold mines in the Free State.

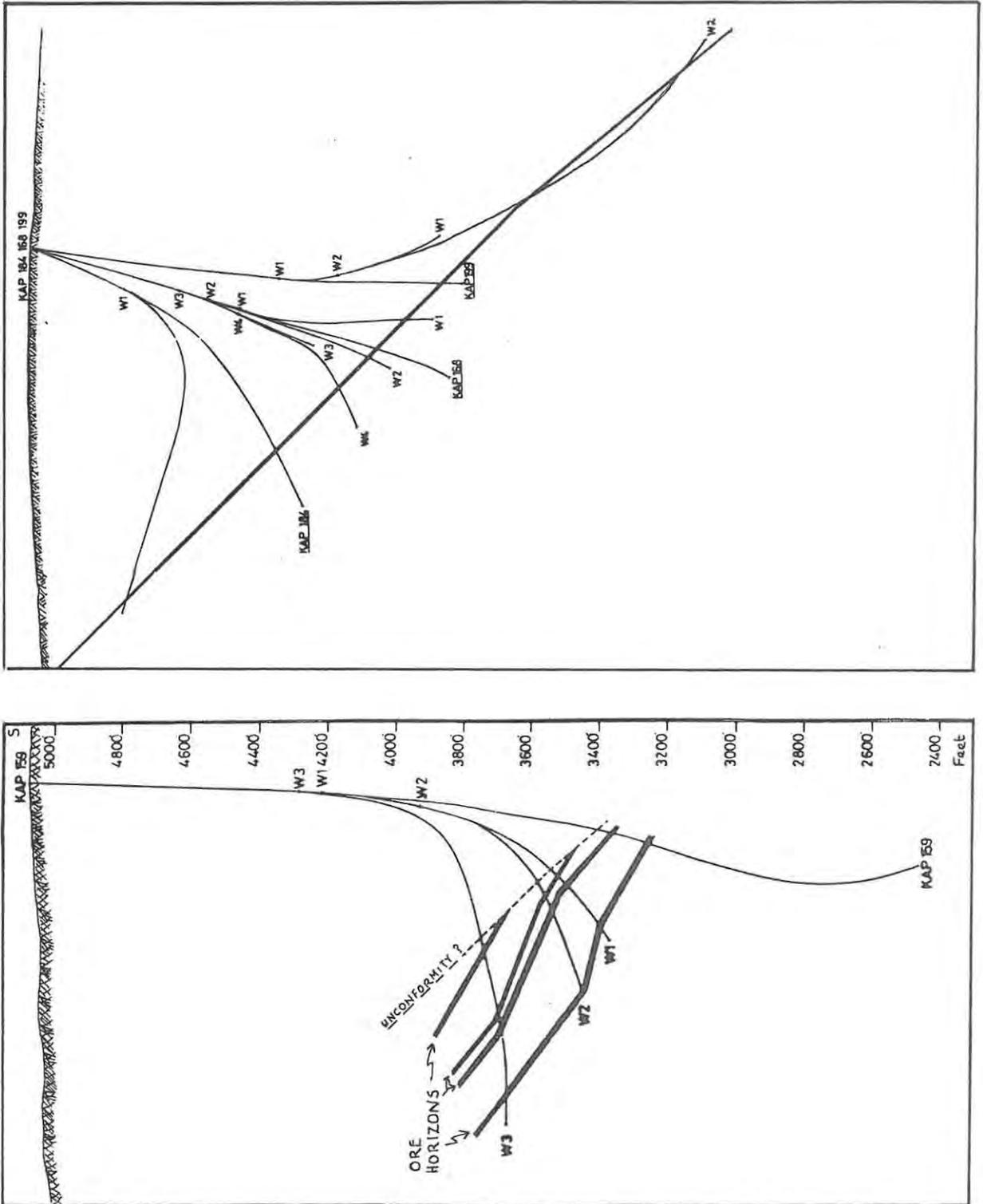


Fig. 31. Controlled drilling at the Klein Aub Copper Mine as seen in section (after Liebenberg 1971).

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SECTION B.H.'s. 184, 168, 199.

The greatest advantage of the inclined underground borehole method is that while deflections drilled from the bottom of a surface hole would be limited to a radius of less than 100 m from the master hole, any number of deflections can be drilled from an inclined borehole at any point. (See Fig. 32). This not only gives a far wider spread of sampling information but also provides detailed structural and geological information about dykes, faults, stratigraphy etc. in the reef zone and its immediate footwall.

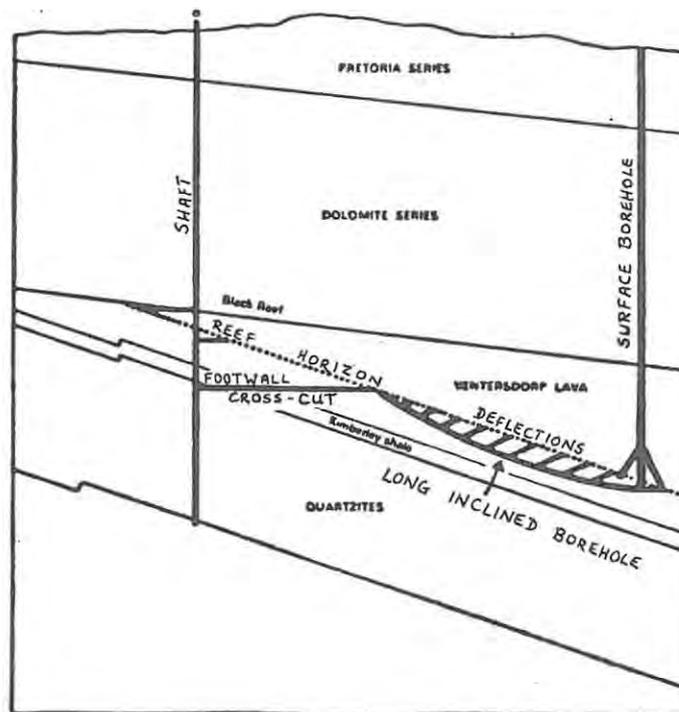


Fig. 32. Section illustrating the greater spread of available deflections from a long inclined underground borehole when compared to a borehole drilled from surface. (S.A. M/EJ. Jan. 1977, p.39).

While the early holes drilled on Western Deep Levels relied heavily on wedging to maintain the desired course, a hole drilled on Doornfontein Mine by Gold Fields Cementation Company was achieved primarily through controlled drilling. As secondary deflections are drilled in a retreating manner, the prime object of the pilot hole is to intersect the reef as far away from the collar as possible. In order to do this a close control on drilling operations was of the utmost importance. As a means of control a graph recorder to record spindle speed and top and bottom

cylinder pressures was fitted to the machine. Control of pressure on the bit was extremely critical as too high a pressure would result in buckling of the core barrel and increasing upward deflection. The hole was surveyed using an Eastern Multi-shot borehole camera at approximately 55 m. intervals. The surveys showed that at approximately 632 m. the inclination of the hole was minus 11° indicating a deviation of 29° through this distance. This was considered too excessive and to counter the upward trend, reinforced core barrels were used with low rotational speeds and light bit pressure. The core barrels were reinforced with steel ribs along their longitudinal axes and used to a depth of 946 m. At this depth the hole had only deviated another 6° in 314 m. compared to 29° in 632 m. in the early stages. Thus, the technique of controlled drilling was highly successful in guiding the hole to a position from which the desired reef intersections could be obtained. (Daniel and Van Tonder 1977).

A variation of a long inclined borehole was drilled at the Boulby Mine, Yorkshire for Cleveland Potash Ltd. by Boart Drilling of South Africa. This differed from the Witwatersrand holes in that it utilised reverse-circulation drilling. Furthermore, deflections from the master hole were made progressively into the potash seam as the hole was deepened instead of on a retreat system on completion of the master hole. (See Fig. 33). The information gained from the progressive core intersections indicated the trend of the dip of the seam and this helped in the accurate planning of future intersection points. The reverse-circulation system with its continuous core recovery meant that the mine geologists knew at any time what beds were being traversed by the borehole. (World Mining September 1977).

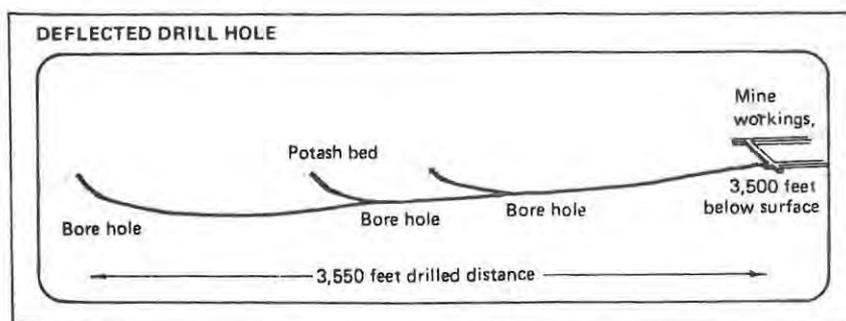


Fig. 33. Long inclined underground borehole drilled at Cleveland Potash Mine. (World Mining, Sept. 1977).

Williams (1978) states that deflection control and techniques were developed to within very close limits during this operation. No deflecting devices were used in the holes apart from drill rod stabilisers for certain of the operations. Otherwise, careful matching of the bit pressure and rotational speed were maintained.

CHAPTER FOUR

OPTIMISING EXPLORATION DRILLING

Exploration drilling is a sequential process, even though several drills are being used simultaneously. It involves a sequence of decisions relating not only to the drilling itself, but also to the problem of obtaining as much information about the mineralisation as possible for the least cost. During the actual drilling, decisions are required before selecting each new drill site, such as the borehole spacing and the geometrical configuration of the exploration grid. The value of the first few boreholes in an exploration project is usually relatively easy to define. At this point the desired information is of a "yes" or "no" sort. Should the programme be terminated at this point or extended? If the latter is decided on, the value of the information from each succeeding borehole becomes more difficult to define. Are these additional boreholes justified or not? The old adage about the boy and his donkey is true to exploration drilling. There is no point in whipping the donkey when it is dead, but while a glimmer of life remains, it may still get to its feet. In exploration drilling the biggest decisions are required to define when a programme is "dead".

Once ore has been encountered and the economic evaluation of an ore-body is proceeding, each borehole retains major significance. If a mine should result, each poorly planned or incompletely utilised borehole is an unnecessary expense. Even though there are many boreholes in the pattern to share the burden, the information from each hole controls a large future outlay of capital. An exploration man cannot afford to walk away from a borehole until he has obtained all the data and applied all the imagination that he can muster to justify the hole in terms of cost and value.

Optimising drilling requires the correct decisions to be made with respect to the following circumstances:

- i) Exploration borehole grids and sequence of drilling.
- ii) Dimensions and spacing in target area follow-up grids.

- iii) Borehole spacing and number of holes required in the evaluation of a potential orebody.
- iv) When is the optimum time to stop drilling?

Each borehole programme must be evaluated with regard to its particular conditions in the light of the decisions mentioned above. No two programmes are alike and, while experience gained in an earlier programme can be useful, a fresh set of decisions must be formulated. In the following sections, certain mathematical and statistical models relating to these problems will be reviewed.

4.1 EXPLORATION GRIDS AND SEQUENCE OF DRILLING

In exploration drilling the data "end-use" dictates, to a large extent, the pattern and sequence of drilling. In reconnaissance, boreholes are likely to be isolated and drilled to test an hypothesis. This might be formulated on the basis of the area representing the correct environment for ore deposition. It might also be drilled to investigate a stratigraphic sequence or to probe beneath an unconformity or thrust fault. The site of the borehole is often dictated by access, availability of water and topography. Even though reconnaissance boreholes are located for geological orientation rather than for target investigation, they are still part of a sequence in drilling and should be planned as such. Thus, reconnaissance boreholes should be sited in such a way that subsequent drilling, if required, should complete the gaps in a grid, or fence, of boreholes.

Davis (1973), in discussing exploration drilling for sandstone-type uranium occurrences, suggests the following sequence: A preliminary drilling programme is undertaken to assess the regional favourability of an area - essentially the definition and general location of possible host rocks. The drill holes are widely spaced (10 to 15 km) and penetrate to sufficient depths to pass through the likely formations of interest. After the existence of favourable rock types has been determined, boreholes spaced 2 to 5 km apart are drilled to search for, and define, portions of the area which have been affected by the passage of mineralising solutions. A combination of favourable host rocks and secondary alteration, together with geochemical and geophysical indications, and possibly some mineralisation, is followed up by "trend" or "district" exploration. Fences (or relatively closely spaced holes of 100 to 200 m

apart on lines), at intervals of 1 to 2 km across the general area of the trend provide rapid delineation of favourable target areas.

The discovery of the Elmwood Zinc Deposit, Tennessee, U.S.A. (Callahan 1977) was to prove that randomness also has scope in exploration drilling. In contrast to the ordered and sequential grids and fences advocated by Davis, Callahan embarked on a random drilling programme to cover an area of potential mineralisation. The area was selected on geological and stratigraphic grounds as being favourable for occurrences of blind zinc deposits of the Mississippi Valley type. Furthermore, an oil exploration hole in the area had intersected traces of zinc mineralisation. In order to determine the optimum spacing between boreholes, he simulated a random borehole "walk" over a map showing the distribution of mineralisation in the Tri-State District. This indicated that holes spaced 5 to 6 miles apart would stand a fair chance of penetrating ore in a similar environment. Thus a drilling programme was initiated and boreholes drilled on a "random walk" basis (See Fig. 34). The 79th borehole penetrated 8 m of ore assaying at 16,5% zinc, at a point 38 miles northeast of the initial hole.

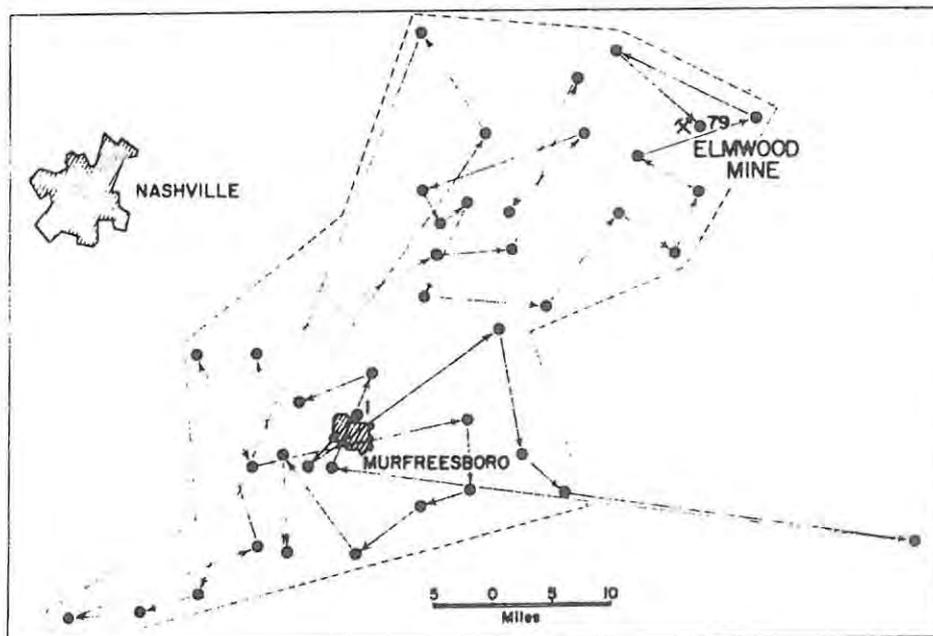


Fig. 34. Callahan's random walk drilling sequence leading to eventual success in the 79th hole (Callahan 1977).

In exploration for blind ore bodies, such as Mississippi Valley types, there are a number of mathematical and statistical approaches that can

be used to guide drilling. However, basic geological premises must first be used in deciding favourable areas where there is the greatest probability of intersecting ore. Economic factors, such as the probable or expected size of a potential ore body, and its grade, must be considered before embarking on an expensive drilling programme. The choice of grid spacing is then related to the size of the anticipated target. If the target is circular, it will always be found, provided that the grid spacing is smaller than $r\sqrt{2}$ where r is the target radius. For larger grid spacings the probability of detection is demonstrated graphically in Fig. 35.

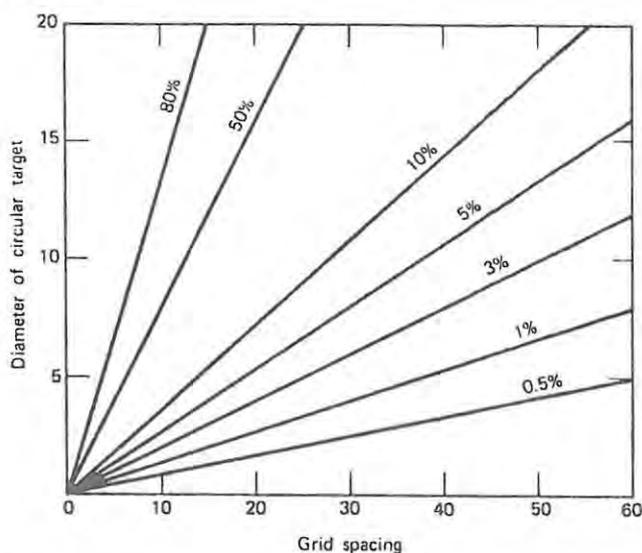


Fig. 35. Probabilities of detection of circular targets with grid spacing of various sizes (after Koch and Link 1970-1971).

These odds are further reduced if the target is elliptical. This is demonstrated by Koch and Link (1970-1971) on p.212. These authors describe in detail the statistical chances of hitting one or more targets of varying sizes by grid drilling.

The following generalisations from McCammon (1977) summarise the probability of target intersection in either parallel line or continuous grid types of search:

- i) The probability of intersecting a hidden target is proportional to the ratio of the greatest dimension of the target to the minimum line spacing of the search pattern.
- ii) The shape of the hidden target does not greatly affect the probability of intersection when the largest dimension of the target is small

relative to the minimum spacing of the search pattern.

- iii) The geometry of the search pattern becomes more critical as the largest dimension of the target approaches the minimum line spacing of the grid. When the largest dimension is less than the minimum line spacing, the probability of intersection is greater for parallel-line search than for an equivalent square-grid type search. The opposite is true when the largest dimension exceeds the minimum line spacing.
- iv) The probability of intersection of an elliptical target for a rectangular grid can be approximated by considering the limiting cases of a line and a circle for a parallel-line and square-grid type of search, respectively.
- v) Nonorthogonal grids do not greatly affect the probability of intersection, provided target orientation is unknown.

The conclusions of McCammon (1977) stated above, provide a general framework for the explorationist in deciding upon the most suitable drilling grid or pattern in the search for hidden targets. In his paper he provides numerous graphs from which the probabilities of intersection can be read for combinations of the above factors.

An increasingly popular approach to borehole planning in exploration is to devise a mathematical simulation model that can be tested in a series of computer runs until an optimum pattern is found. In the simulation approach certain natural variables - the number of ore bodies and their size, shape, orientation, location and value - are assumed. Controllable variables such as borehole locations, azimuths, inclinations, lengths and costs are then assigned. Alternative assumptions about the natural variables and alternative patterns for the controllable variables are considered in all of their most probable combinations until the most cost-effective layout is determined. (Peters 1978).

One such study (Malmqvist 1976), was carried out to determine the optimum factors required in exploration for deep-seated sulphide ore bodies. A mathematical model of an ore field containing deep-seated sulphide bodies was constructed. One set of parameters described the geometry, size distribution and intensity of the ore bodies, while another set described the geometry and intensity of the boreholes. The first set of parameters was based on the distribution and shape of known orebodies in the central

part of the Skellefte field of Northern Sweden. Each simulation experiment consisted of a number of drilling programmes. Each of these consisted of a certain number of boreholes located on a regular grid. For each new drilling programme, a new random set of ore bodies were constructed. The results of the simulation experiments are shown in Fig. 36. This represents the number of deep-seated ore bodies (> 4Mtons) intersected by the boreholes. Malmqvist concludes that in order to be more realistic, this simulation model should be extended to include more geological parameters and the possibility of simulating geophysical down-the-hole measurements.

Experiment	Size distribution	Ore intensity	Number of drillholes	Maximum depth of drillhole	Average quantity struck ore (Mton)	Average number of hits
1	S	15 Mton	40	750	1.0 ± 0.22	0.15
		25 Mton			1.8 ± 0.34	0.28
		35 Mton			2.6 ± 0.39	0.40
2	S	24 pieces	40	750	1.2 ± 0.27	0.19
		36 pieces			1.8 ± 0.38	0.30
		48 pieces			2.4 ± 0.47	0.40
3	S	25 Mton	21	750	0.6 ± 0.19	0.10
			30		1.2 ± 0.25	0.19
			40		1.8 ± 0.38	0.28
4	S	25 Mton	40	300	0.3 ± 0.09	0.05
				500	0.9 ± 0.23	0.15
				700	1.6 ± 0.33	0.25
				900	2.2 ± 0.37	0.35
5	M	20 Mton	40	750	0.3 ± 0.15	0.04
		25 Mton			0.4 ± 0.18	0.06
		30 Mton			0.5 ± 0.22	0.08
6	M	24 pieces	40	750	0.1 ± 0.12	0.04
		36 pieces			0.5 ± 0.16	0.07
		48 pieces			0.9 ± 0.25	0.11

S = Skellefte Field distribution
M = Modified Skellefte Field distribution

Fig. 36. Results of simulated drilling programmes for deep seated ore deposits (after Malmqvist 1976).

A similar study (Koch, Schuenemeyer and Link 1974), was devised to guide the layout of underground mine workings, drill stations, and drill holes that were normally established in order to search for tabular ore bodies in a Coeur d'Alene lead-silver mine. Programmed for a digital computer, the model contained both stochastic (natural) and deterministic (controllable), variables. The former included the number of ore shoots

and their sizes, orientations, locations and values. The deterministic variables included locations and cost of working; drill-station locations and costs; and drill-hole azimuths, inclinations, lengths and deflections. Through manipulation of the model and the natural variables, the controllable variables are tested until an optimum cost-effective combination is achieved. According to the authors, "...the process is a sequential one of intelligent trials". If, for instance, a particular layout of working and drill holes gives high probabilities of discovering whatever is there, a cheaper layout might be tried. Koch et al also suggest that their simulation model can be modified for almost any situation. As an example they modified the model to investigate whether or not a uranium-bearing bed prospected by surface boreholes was evidently underdrilled, overdrilled or suitably drilled. "The weakness...[of the model] ...is that the user must be prepared to invest a fair amount of time in familiarising himself with the operation of the model. While the user's manual is, we hope, clear, nonetheless it contains 138 pages of technical details". (Koch, Schuenemeyer and Link 1974).

A novel approach to exploration by boreholes is presented by Singer and Drew (1976) who calculate the "area of influence" of exploratory boreholes. They suggest that each exploratory borehole, whether successful or not, has some area around it where resource targets could not have existed without having been found; that is, each hole sweeps out part of the total region available for exploration. Thus, as holes are drilled, areas are condemned ("physically exhausted") and the area remaining to be searched diminishes. Furthermore, a successful borehole also reduces the quantity of undiscovered resources in the region. The manner in which the region is exhausted during its exploration history is referred to as the "physical exhaustion sequence". This provides information about the number and sizes of deposits yet to be found. At any point in the sequence, the part of the search area which has been exhausted is completely determined by the area of influence of the holes and the arrangement of the holes drilled to that point. The area of influence of a single hole is determined by the size and shape of the resource target sought. Thus, the area of influence about a drill hole is equal to the radius of the expected target (for circular targets). A more complicated relationship is developed for elliptical or other target shapes. However, once the areas of influence about the boreholes are calculated, the data is then used to generate a map demonstrating where targets should occur. An example in which 14

exploratory holes, plus several holes outside the map area, have been drilled in search for a hypothetical target is presented in Fig. 37. Contour lines in this figure show how the completeness of search varies across the region. Points within the 1.0 contour line are totally explored with respect to the target. Points on the 0.5 contour lines are 50% exhausted and so on. (Singer and Drew 1976).

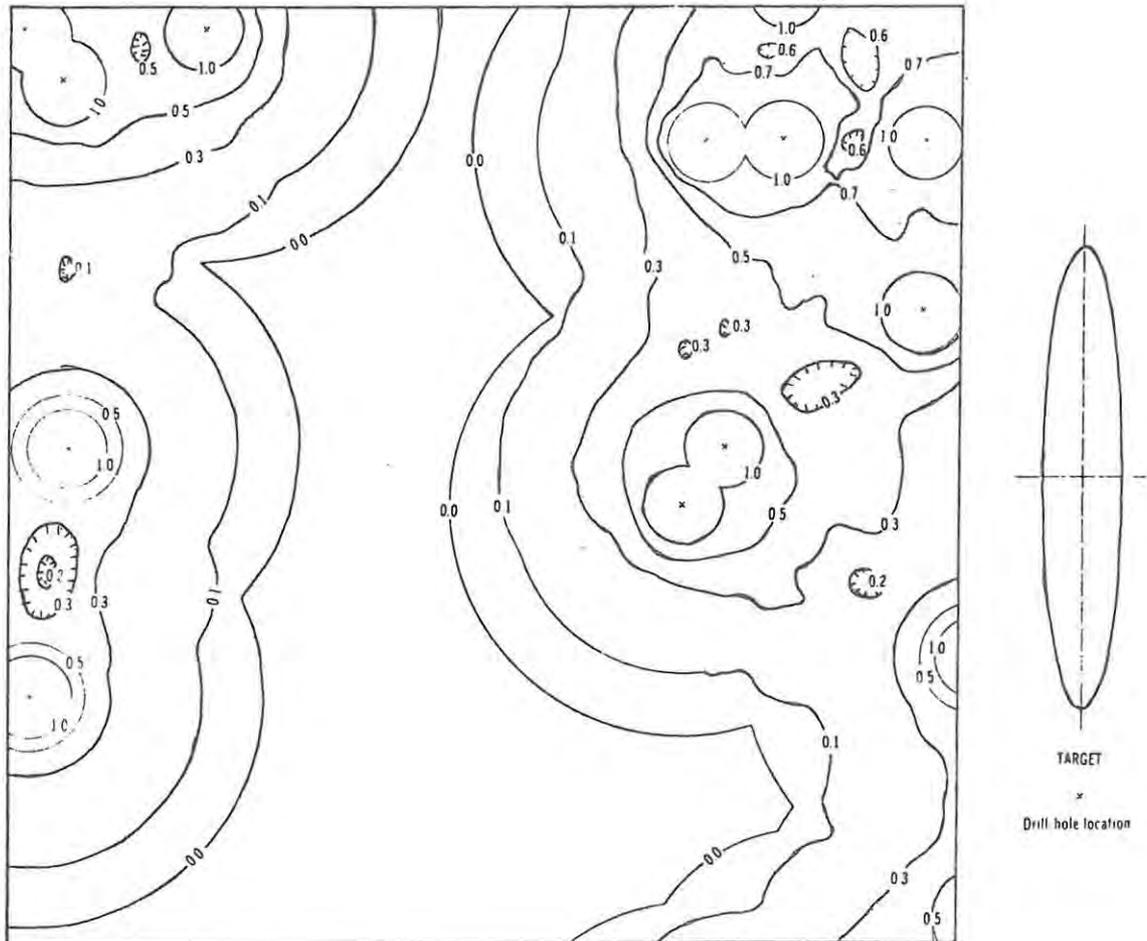


Fig. 37. The area of influence of exploratory boreholes and the degree of "physical exhaustion" of the area for the size of target shown on the right. (after Singer and Drew 1976).

Maps generated by this method offer a means of determining how well any area within a region has been explored. They also demonstrate the effectiveness of borehole spacing and show when an area is in danger of being overdrilled. They could well be used to determine spacing between boreholes in Callahan's (1977) "random walk" technique of ground coverage.

The ultimate in grid drilling, as proposed by Griffiths and Drew (1966 in Koch and Link 1970-1971), is the complete coverage of a country by a

borehole grid. They suggest that "...given a continental area like the U.S.A. of 3000 by 1000 miles in size and a spacing of say 20 miles, the grid would require some 7500 wells; supposing each well was drilled to 1500 feet at say \$250000 each, then the cost of the programme would be some $\$1875 \times 10^6$. Any target exceeding 28 miles in extent would be found by this grid with probability one, and such large targets yield a return of several billions of dollars so that the programme appears commercially feasible." They believe that this programme would find five major oil fields in the United States to add to the 20 already known, and that the value of the new fields would be about \$5 billion, well above the \$1875 billion cost of drilling. In addition, they expect that other valuable natural resources would be found. (Koch and Link 1970-1971, p.331).

The use of statistical decision models is advocated by DeGeoffroy and Wignall (1970) in the selection of target areas for follow-up drilling. Due to the widespread number of targets generated by geochemical, and especially geophysical prospecting techniques, the use of statistical decision functions in mineral exploration is becoming more widespread. Firstly, a regression function is formulated on the basis of information on known deposits in a control area. A Bayesian procedure is then introduced to calculate the success probability and expected dollar payoff at each location of the study area. The targets showing simultaneously high probabilities of success and high regression estimates are selected for drilling. (DeGeoffroy and Wignall 1970).

Peters (1978) summarises the effects of a statistical drilling grid or pattern by likening them to the data screens that a geologist uses in locating a target area. If the mesh is too coarse, most of the ore bodies (the information), will fall through; if the mesh is too fine, the cost of a series of target-area investigations will exceed the value of the ultimate discovery.

4.2 TARGET FOLLOW-UP GRIDS AND DEPOSIT EVALUATION

The selection of targets for detailed follow-up by a drilling programme falls outside the scope of this review. These decisions are based on geological, geochemical, geophysical, historic and economic parameters. It is the task of the geologist conducting the follow-up drilling programme to justify the expense involved in his choice of target area. The tendency

to drill a hole prematurely, without due consideration of all the factors involved, should be carefully weighed against the tendency to "overkill" a target with various geochemical, geophysical and other techniques before drilling it.

Once a target has been selected, follow-up drilling normally proceeds in a sequential manner, following on from where the exploration or reconnaissance drilling stopped. Optimising the follow-up drilling programme is a complicated procedure as it is dependent on so many economic variables. Basically, the object must be to drill sufficient holes to delineate an ore deposit (if one exists), so that a decision can be taken on whether to proceed with the following stage of deposit evaluation. To obtain the optimum benefit from these two separate programmes they should be planned to coincide with each other and should not be treated as separate entities. In other words, the drilling pattern used for follow-up drilling must be able to be modified for deposit evaluation drilling without duplicating work in any sphere. Ideally, the first programme should be on a fairly widely spaced grid and the evaluation programme drilled on an infill basis. In practice this is seldom achieved as the follow-up boreholes are normally sited only with a view to investigating the cause of anomalies related to possible targets. This results in a compromise grid when the evaluation grid is selected to fit in with the randomly spaced target investigation boreholes, without due consideration to selecting the optimum grid spacing for the latter.

The decisions leading to the stage of mine development is similarly a sequential process. After each drilling phase, the knowledge gained from that borehole, or programme, must be considered in the light of the investment, in cash, made towards the drilling. As more money is invested in the programme, the economic uncertainty of the project becomes progressively reduced. The principle factor is, therefore, to decide at which stage of drilling has an acceptable level of uncertainty been reached. Thus, investment in more drilling must be economically justified, i.e. the marginal benefits must exceed the marginal costs.

Once a mineral occurrence is discovered, the geologist in charge of the drilling programme has the option of abandoning the project or implementing a delineation drilling programme. If he follows the latter course, he will be faced with a set of four decisions at the termination of the programme (See Fig. 38). This sequential delineation process will

continue until the expenditure on drilling has reduced the uncertainty about mine development to an acceptable level. (Mackenzie 1979).

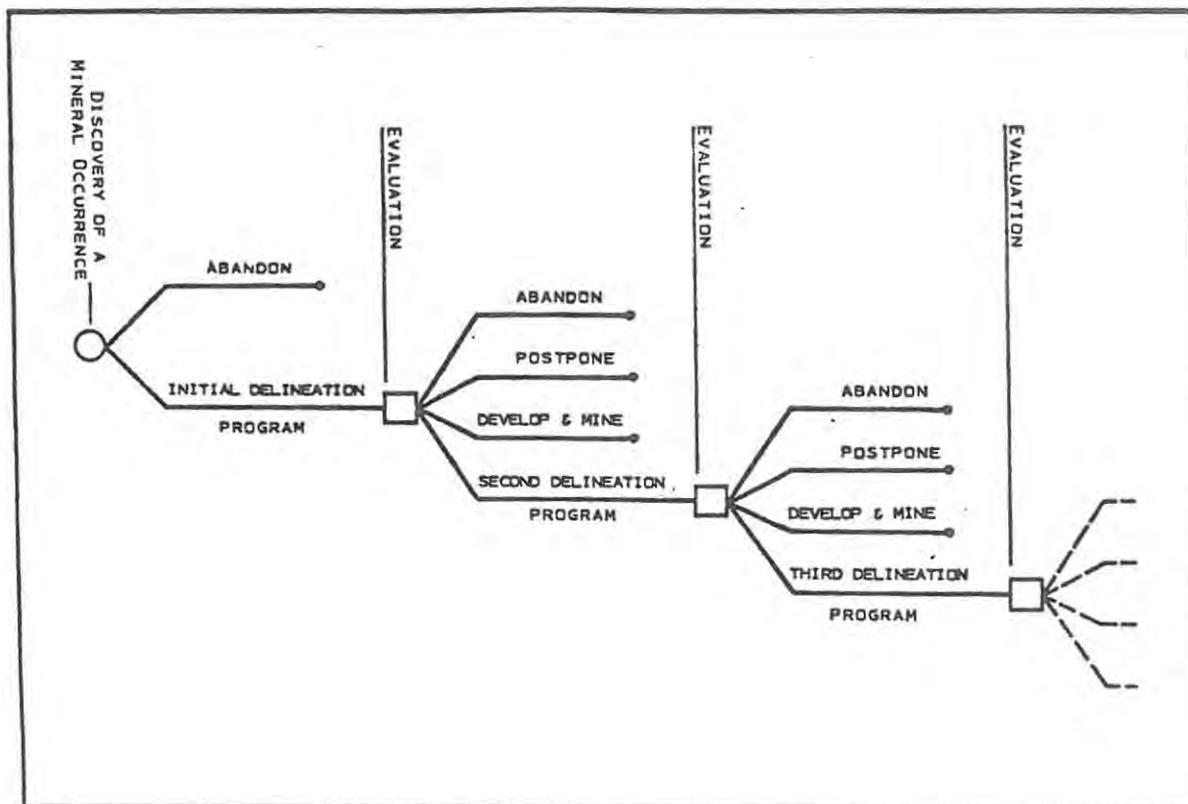


Fig.38. The Sequential Drilling Decision Process (after Mackenzie 1979).

Geological uncertainties are reduced and grade and tonnage estimates are improved as delineation drilling information accumulates. This, in turn, reduces the economic uncertainty associated with the evaluation of mine development. However, the benefits of additional information in reducing economic uncertainty are subject to diminishing returns. As borehole information accumulates, the reliability of the profitability estimate improves, but at a decreasing rate. If the drilling is continued beyond an optimum point, the marginal costs will outweigh the marginal benefits achieved. Mackenzie (1979) demonstrates how this optimum level can be achieved using statistical concepts. The reliability of information already gained is a function of the degree of variability of the deposit characteristics which in turn is estimated by the standard deviation of the sample frequency distribution. The standard deviation estimate is used to assess the reliability of the mean value estimate which is measured by the standard error of the mean. This defines the possible

distribution of actual mean values about the mean value estimate and is used to define confidence limits. These limits reflect the reliability of sampling and can thus be used to establish at what point the optimum value of data has been obtained.

Mackenzie (1979) also describes geostatistical concepts using the Monte Carlo simulation approach to "...relate quantifiable geological parameters to expected profitability and economic risk criteria, in order to optimise investment in delineation." From his work it is also clear that any consideration of drilling beyond a critical point will not only reduce expected profitability but will also reduce the insurable lower limit profitability and thus increase economic risk.

Much the same concept is advocated by Brooker (1975 and 1976) who uses the semi-variogram to measure the effectiveness of a drilling programme. Brooker (1975), estimates the value of the additional information gained from halving the spacing between boreholes in an evaluation grid. He demonstrates that with the use of advanced estimation procedures and geostatistics, sufficient tonnage and grade information can be obtained at a stage prior to normal infill drilling and the use of conventional elementary estimation techniques.

To determine the size and spacing of an effective drilling grid over an indicated ore body, statistical methods can also be used. Imai and Itho (1971) describe the procedure, with the use of examples, to establish an effective drilling grid for representative ore reserve estimation. In their method, they establish the coefficient of variation (cv) as a measure of the accuracy of ore reserve estimation. This is the ratio between the expected, or estimated, ore reserve and the standard deviation. Thus, a gradual reduction of the borehole spacing is co-ordinated with a step-by-step measurement of accuracy. Once the coefficient of variation is reduced to an acceptable level, the optimum grid dimensions have been realised.

Another approach to this problem is the determination of the correlation of sample values between two adjacent boreholes in a grid. In the evaluation drilling of the Sar Cheshmeh Porphyry Copper Deposit in Iran, infill drilling was undertaken only if a +50% variation existed in the grade between two adjacent holes on a 100 m grid. (Selection Trust Limited, 1970). Kuhn and Graham (1972) take this one step further by

applying a correlation analysis to the data of adjacent drill holes. This method is based on the measurement of the linear correlation between the spatial distribution of the two sets of data. The statistical significance of the correlation coefficient at 95% confidence level is the main criterion. Assays from the boreholes are averaged into equal composite intervals of convenient length. If two sets of composite borehole data produce a significant correlation, it is assumed that the areas of influence of each of the holes overlap and further drilling along the line between the holes is not needed. The use of this correlation method eliminates unnecessary drilling. It provides means of determining the "area of influence" which can be assigned to a borehole and, coupled with the confidence interval of the mean, provides areal dimensions for application of grade estimation. (Kuhn and Graham 1972).

In a detailed report of 45 pages, Hewlett (1965) describes how a limited number of preliminary exploration borehole assays from a mineral deposit can be used to design grid spacing that is both economic and efficient for subsequent evaluation drilling of that deposit. He applies assay data from 50 known low-grade copper deposits to establish a relationship between grade estimates and borehole grids. Thus, the relationships between the desired precision of the estimate of grade of ore and drilling cost are used to determine the economic borehole spacing. The statistical concept of precision in estimating the grade of the core is used to determine the efficient drill-hole spacing.

The use of geostatistics on sample and borehole data has been applied to determine the optimum number of borehole deflections that should be drilled during Witwatersrand gold exploration (Lemmer and Howard 1979). In deep exploration drilling (± 4000 m) the cost of a surface borehole is in the order of R400 000. A long deflection, costing R20 000, may intersect the reef at a distance of about 9 m from the master borehole, while a short deflection, costing R5 000, may intersect the reef about 23 cm from the borehole. In some instances, between 10 and 20 intersections have been obtained, through deflections, of a reef horizon from one surface hole. This is naturally very expensive and Lemmer and Howard set about investigating how the number and pattern of these deflections will influence the confidence with which tonnage and grade forecasts can be made. In order to do this, they selected a mined out area for which detailed assay data was available and constructed variograms of this

data. The average grade of a 100 m square block centred on an exploration borehole could, therefore, be determined for the area. The results of the study indicated that an additional four deflections (costing R50 000) from the master hole would radically improve the level of confidence of the data. However, the improvement in information and confidence gained by additional deflections thereafter, is minimal and not justified by the expense of the additional deflections.

CONCLUSIONS

While diamond drilling is by far the most expensive of the mineral exploration drilling techniques, it does provide the most comprehensive and reliable data, if interpreted correctly. The reliability of this data can be improved considerably by the additional expense of directional and stratometric surveys. Full utilisation of the borehole "meterage" is possible through the use of controlled drilling techniques. This optimises the length of the borehole in relation to the value of the available information. For little additional expense it can be insured that the maximum available information is obtained from a borehole, prior to the geologist's walking away from it.

Non-cored boreholes cost less than cored boreholes but the information they provide is not as detailed or reliable as that of the latter. However, with conscientious logging, data handling, and the intelligent use of advanced down-the-hole techniques, considerable information is available. The advantage of the cheaper, non-cored holes over the cored holes, is that a number of the former can be drilled for the cost of one of the latter. This serves to increase the spread, or ground coverage, of the drilling survey, often outweighing the benefits obtained by a diamond drill hole.

The most cost effective drilling programme uses, when feasible, a combination of cored and non-cored holes, down-the-hole surveys and chemical analyses of core and cuttings. The spacing between holes in a borehole grid can be determined by statistical or geostatistical applications of the borehole data. Similarly, the optimum number of holes drilled in an evaluation programme can be determined through the use of fundamental economics and statistics.

The original intention of this review was to cover the entire field of exploration drilling in as much detail as space and time permitted. However, it soon became apparent to the writer that this is a considerably larger field than was initially appreciated. As a result, most of the topics described could be expanded to the extent of forming a dissertation, or review, in their own right. The most obvious of these is that of down-the-hole logging and sampling techniques. Borehole logging itself

is an extremely diverse field. A review of the applications of the various logging parameters required for specific genetic types of ore occurrence would be very useful to the exploration geologist. The contents of Chapter 4, "Optimising Exploration Drilling", should be expanded to cover the full economic aspects of drilling, with the aim of providing the maximum information, at an acceptable level of confidence, for the minimum of expense.

It is sincerely hoped that the contents of "A Review of Mineral Exploration Drilling" has introduced the subject in a broad context and might stimulate further documentation of this fascinating, and academically neglected, field.

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