5 *Natural Resource Management*

Technological and Economic Evolution of Diamond Wire Use in Granite or Similar Stone Quarries

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ABSTRACT: Cutting with diamond wire saws in hard stones, such as granites, gneisses, syenites and diorites, taken together with the dynamic splittmg system, using a detonating cord, has become more widespread because of the operating versatility of the cutting machinery and because of the good productive results. It is obvious that there is an advantage in marketing blocks which have finished-off faces, with no wastes in the squaring phase, especially when high-value dimensional stones are concerned. The excavation of channels within the good rock and the realization of artificial "easy ways" (splitting surfaces) could economically justify the higher cost of diamond wire technology in comparison with explosive splitting. The present paper, according to recent surveys carried out in many stone quarries in Piedmont and Sardinia, supplies new data in order to make a technical-economic comparison between different quarrying systems.

1 INTRODUCTION

The wide and heterogeneous family of dimension stones has in common the fact that they must be obtained in the form of commercial blocks, which means sound blocks of parallelepiped shape usually weighing 5 to 15 t, suitable for further processing and for the production of slabs or other architectural items.

The mining stage of the dimension stone industry conforms to one of the two general strategies quoted below:

A. large volumes of rock, usually in the range of thousands of m^3, are removed by primary cuts from the rock body and, subsequently, stepwise, divided into commercial blocks; unsuitable material is discarded at this stage;

B. commercial blocks are directly cut from the rock body; this method is mostly applied to very regular and homogeneous deposits and especially to underground operations.

Ample cut or split surfaces have to be produced at a reasonable cost and without impairment of the rock soundness.

2 TECHNIQUES EMPLOYED IN ITALIAN QUARRIES

Italian production of commercial blocks (nearly 5 Mt/y of marbles, more than 1 Mt/y of granites and about 1.5 Mt/y of other stones) is mainly the result of three well-established technologies: diamond wire sawing, chain cutting (for *soft* rocks only) and detonating cord splitting.

In *hard* (granites and similar) dimension stone exploitation, the most economical production system, at least for primary cuts, is still splitting by explosives, which can be considered an extreme application of controlled blasting concepts and precision drilling techniques. Very thin linear charges, represented by strands of detonating cord, are placed in parallel, closely spaced holes, stemmed with a suitable shock-absorbing material like water, and simultaneously detonated by a master cord. Fracture is due to tensile stresses in the rock inter-hole bridges, and excess energy from the blast provides the required small displacement of the separated mass (Mancini et al., 1996a). In well-conducted operations, the half casts of the holes are perfectly observed and no extra cracks occur. Productivity is high and İs dictated by the drilling system (Mancini et al., 1994). Although there have been numerous research studies carried out by a lot of laboratories in order to improve the directional splitting action of explosive charges, the most commonly used technique İs still the conventional, firmly established detonating cord splitting method (an estimated 12000 km of cord is consumed yearly for this application in Italy).

Examples of the application of this method are given İn Figure 1.

Figure 1. Example of a typical hard stone quarry exploited with dynamic splitting method

The other two common methods rely on stone machining by microtools, that is to say, tools acting with a very small depth of the elementary cut. Both belong, in principle, to the continuous saw family: cutting tools are attached to a flexible, continuous support, whose motion is unidirectional. The chain cutting method will not, however, be treated in this context, being employed exclusively for marble and, generally speaking, for *soft* rock extraction.

Diamond wire sawing has been, since the end of the seventies, the most widespread technology for marble cutting: in fact, more than 90% of marble quarries use it systematically in cutting operations.

However, problems related to the abrasive properties of *hard* rocks have made its use with them very difficult. Consequently, only at the beginning of the nineties were there the first convincing, though not regular, applications. Today, in Piedmont extractive basins, at least 30% of *hard* rock quarries use diamond wire in several exploitation steps.

Only *plasticized* or *rubberized* wires are used, but the latter is coming to be preferred in quarrying activities. Beads are always *impregnated* with synthetic diamonds; their diameter is 10-11 mm and there are 35-43 beads per meter of wire.

The main features and performances of the diamond wire sawing method are summarised in Table 1. As the wire is rather expensive (over 100 US$/m, and up to 150 US$/m for special purpose types), it is important to get good performances, which are usually defined by two parameters: productivity (m^2/h) and wire service life (m^2/m). The latter parameter is only loosely defined, because the wire can be reconditioned many times before discarding (up to 5 times, usually 1-2 times for hard rocks); reconditioning İs a high-skill, labour-intensive task, entailing wire disassembling, reusable bead selection, discarded bead replacement and wire reassembling. Judgement of what is a reasonable, economically efficient, reconditioning rate depends on local circumstances. Diamond wire is often used to "death" in *hard* stone quarrying operations, that İs, until the cutting tools are completely worn out, just because bead life is more or less equal to that of the support.

Table 1 Main features of the diamond wire sawing method

Features of the diamond wire saw	
Tool material	Diamonds
Tools geometry	Random
Depth of trie elementary cut	10 ÷ 30 μm, vairable
Width of the cut	≅ 1 cm
Flushing	Water (≅ 50 l/min)
Tools speed	20 ÷ 40 m/s
Flexible support type	Steel cable
Length	>100 m
Guidance	Pulleys
Beads type	Impregnated (hard rocks)
Beads, number/m	35 ÷ 40
Wire cosi, USS/m	140 ÷ 150
Productivity, m/h	1 ÷ 4
Service life, nr'm	8 ÷ 10 (hard rocks)

3 ECONOMIC COMPARISON OF CUTTING TECHNOLOGIES

Basic conditions for the success of a new technology are its simplicity and cheapness İn comparison with other already-used methods. Of course, there are other important aspects which should be considered in the choice of the cutting method: first of all its safety, its flexibility and adaptability according to rock body characteristics (Mancini et al, 1995).

Not only should low production costs be taken into account, but also the method's *recovery;* that is, how much of the quarried stone will be really exploitable in further processing. For instance, a technology with higher costs, but which is more accurate, could turn out to be cheaper when high-value materials are concerned.(Agus et al., 1990)

A schematic comparative analysis of costs connected with primary cutting of a gneiss bench in an Alpine basin is reported here. Quarry-front cutting is the first phase of the production cycle in a dimension stone quarry and it involves an expenditure of energy, machinery and human resources which are variable according to the

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technology used. The main purpose is always to get a cut without cracks in the bench or in the rock body, m a short time and with low costs. The data given here refer to studies carried out recently in many quarries (Zoppo, 1997 & Bergamasco, 1990), and the aim is to supply an order of magnitude of unit costs (US$/m^3) and information about the incidence of the main productive factors.

For the case represented in Figure 2, a comparison is made between the dynamic splitting method and diamond wire sawing; the volume of the primary block, as shown, is 512 m^3. In order to separate the bench, three cuts are required, assuming that there is not an easy way, while a vertical side has already been opened.
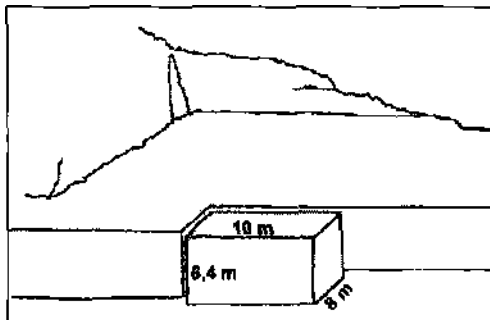

Figure 2. Bench scheme.

The main cost headings taken into account are: machinery depreciation, manpower, service, wear and energy costs. In Table 2, quarrying activities and items are given with the cost headings according to the cutting method.

Table 2. Main cost headings.

	Dynamic Splitting	Diamond wire saw
Machinery depreciation*	-Pneumatic drill** (small diameter 30-45 mm)	-Drill (bigger diameter); -Diamond wire saw
Manpower	-Drill placing; -Drilling; -Hole charging, -Priming -Blasting	-Dniling; -Closing wire loop inside holes; -Placing wire saw; -Cutting supervising
Service and wear	-Explosive; -Drilling items	-Diamond wire, -Drilling items; -Wire saw service
Energy	-Compressed air	-Electrical energy

* Only machinery used during cutting operations is considered Other machines, such as wheel loaders and excavators, are not taken into account because they are needed independently of the cutting method.

** Pneumatic drilling is considered here. It has lower investment costs but also a lower drilling speed compared to a hydraulic drill; hence, the global costs are nearly the same

The unit costs expected for bench cutting by dynamic splitting come to about 5.9 US$/m^3, while in the case of diamond wire sawing a cost of 18 US$/m^3 is obtained. The per cent incidence of different cost headings is illustrated in Figure 3
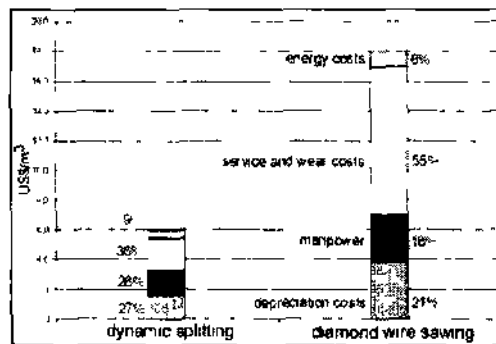

Figure 3 Per cent incidence of cost headings (US$/m^3)

As mentioned above, the choice of a specific technology should also take into account the commercial block price. Many operators agree on the point that explosive cutting causes a 7-10% waste of material due to cracks which may occur near the splitting surface, while only a 2-2.5% waste is produced by diamond wire sawing.

The real unit production costs of the two compared methods are plotted as a function of the unit price of the blocks m Figure 4, where a spoil of 7% is taken into account for the dynamic splitting method, while this is 2.5% for die diamond wire saw.

The data from the investigation show that primary cutting by dynamic splitting is about 60% cheaper than "pure" diamond wire sawing. However, at least part of the cost difference can be balanced by the better quality of the blocks obtained and by less production of waste. This last point represents a benefit whose importance grows as the commercial value of the product grows and, hence, while İn low value stones splitting by explosive is the lower cost option, in high-value materials the wire option is preferable, as shown in Figure 4.

It should be considered that the drilling and blasting method for *hard* stone primary cutting will hardly provide great improvements, mainly because the productive cycle does not allow deep automation, thus keeping high labour costs. Moreover, even though mitigable, problems related to noise, vibrations and unexpected fly rock still

impose restrictions on the method, especially close to towns or roads.

As far as diamond wire sawing is concerned, the wire itself is the biggest part of the total production cost, but substantial progress is still possible. For instance, research is being carried out on the optimal bead matrix for different rock types, the rubber coat quality to avoid metal cable wear, etc. Further efforts should be made to improve the safety conditions of diamond wire use; accidental wire breakages are still very dangerous because of wire lash and the throwing of beads or other metallic parts (Berry et al., 2000).

Finally, the global reliability and service performances of diamond wire could still be improved in abrasive stones.
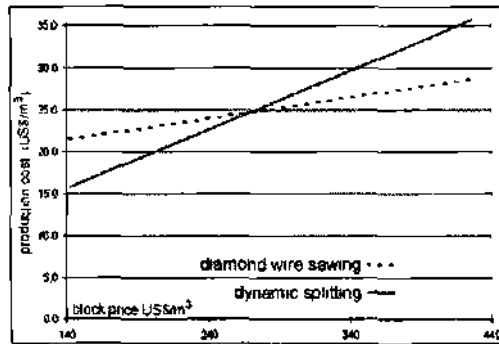

Figure 4 Variation of costs of different methods according to block price

4 EXPECTED IMPROVEMENTS

As detonating cord splitting is a simple and mature technique, some progress is expected only on the drilling side of the production process: more productive drills and, above all, more accurate guidance. As to diamond wire sawing, the main problem to be solved is the increase in service life with abrasive stones. Diamond is harder than any material and hence it can, in principle, cut anything, but cutting diamonds are held in place on beads by a metallic matrix, susceptible to wear and still perfectible as to binding power.

Wire saw beads fall into two basic categories: *electrodeposed* (a single layer of coarse diamonds is fastened to the bead by electrolytically deposed metal) and *impregnated* (a fine diamond/metal sintered layer is applied to the bead). The former type, more productive, cheaper but less durable, is commonly used with marbles, the latter, slower in action but longer lasting, with *hard* rocks.

By analysing the diamond grit recovered by chemical dissolution from the cuttings, it has been found that usually up to 40% is made up of diamonds not much smaller than original size, which means that a lot of unexploited diamonds are simply lost due to imperfect binding or to premature wear of the matrix. Indeed, manufacturers are concentrating on this point. Some success has been achieved: purposely made impregnation bead wires have already attained service lives of 10 to 12 m^2/m in gneisses. In *hard* stone cutting, diamond wire sawing is not yet completely competitive with detonating cord, however, as pointed out above, advances m wire performance are rapid and a lot of grämte quarrymen already use diamond wire sawing at least in part of the production process (Figure 5).

Figure 5 Side cut with diamond wire in a Sertzzo quarry

In practice, diamond wire is nearly always used with explosives (Figure 6). It is used especially to create *channels* in the rock mass or to make ample side cuts. Other cuts which definitely isolate the bench are obtained by the explosive method. At the moment, the application of mixed diamond wire-explosive technology allows better recovery in comparison with simple dynamic splitting, with a small cost increase. Therefore, it is necessary to pursue optimisation of this method in order to take advantage both of diamond wire's *precision* and possibility for automation, and the versatility of explosives. Where high-value dimension stones are concerned (e.g., Beola), sometimes the cut to create

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a smooth artificial easy way is also made with diamond wire in order to get higher slab recovery dunng further processing It has been noticed that blocks squared by explosives "lose" up to 10 crn on each side dunng the sawing process, producing considerable waste which, apart being a lost profit, represents an extra cost in dumping Moreover, dunng the sawing process, there is a spoil of abrasive gram and a remarkable waste of time at the beginning and at the end of slab cuts
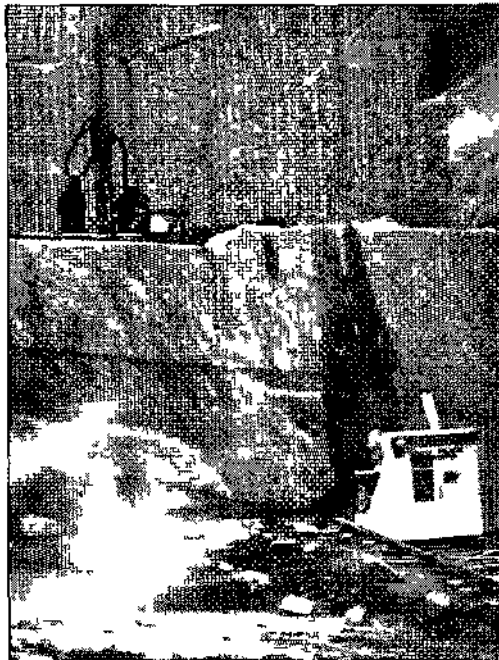

Figure 6 Application of mixed technology dynıı _ ı associated to diamond wire sawing ('Lusema ' stone)

In Table 4 below, some data referring to side cuts made by diamond wire in different *hard* stone quarnes are reported

Table 4 Side cuts in different *hard* stones

	Granite (Sardinia)	Gneiss (Lusema)	Sen220
Bead type	impregnated	impregnated	impregnated
Bead/metre	40	40	40
Bead $ (mm)	ll	ll	10
Wirelenglh	65	65	35
Drilling ϕ (mm)	900	900	800
Wire speed (m/s)	24	23	24
Cut area (ra²)	164	115	50
Productivity (m /h)	16	1 7	2
Service life (nT/m)	7 5	10	11

5 CONCLUSIONS

An evaluation of me ments of an exploitation method must take into due considération me ore recovery aspect (Del Greco et al, 199^) An exception could be represented by the case of unlimited reserves of low value material located where environmental damage from exploitation can be considered negligible, but such conditions simply no longer exist

Even in the case of dimension stone exploitation, the only mining sector presently important in Italy, good deposits are uncommon or require costly development work, and must be fully exploited, hence, the best exploitation method does not always coincide with the cheapest excavation method

Investments per unit of in-situ resource can be very large the example can be given of a gneiss quarry, where a 500-m-long, 25-m²-cross-section tunnel, costing around US$ 500000, had to be dnven and each cubic meter wasted of in principle marketable rock will consume, on the one hand, a share of this amount and will reduce, on the othei hand, the useful life of the expensive infrastructure

Therefore, in the dimension stone sector, more refined and, mtnnsically, more costly techniques are gaining importance

Table 3 Qualitative companson between main quarrying techniques used in Alpine *hard* stone quarries

Comparison elements	Detonating cord technique	Diamond wire technique	Detonating cord + diamond wire
Cut accuracy	Low	High	High
Cut productivity	High 7 lOmVh	Average 1-4 m Vh	High 10m²/h
Energy consumption	Low	Average low	Average low
Capital cost (mech equipment)	Low	Average	Average
Tool consumption	Low	High	High
Environmental impact	High	Low	Average
Recovery on primary blocks	92%	98%	9 5%
Possibility of mechanisation process	Low	Average	Average low
Working conditions (safety)	Low	Average	Average*
Water consumption	Low	Average	Average
Influence of ore body shape and structure	Low	High	Average

* Block cut obtained by wire can be dangerous when working rack bodies with sloping easy splitting ways (danger of block displacement dunng cutting), in this case (quite common indeed) splitting by explosives is prepared for the back cut

Figure 7 An Alpine granite quarry. Side cuts are made by diamond wire, while horizontal and front cutting are made by dynamic splitting
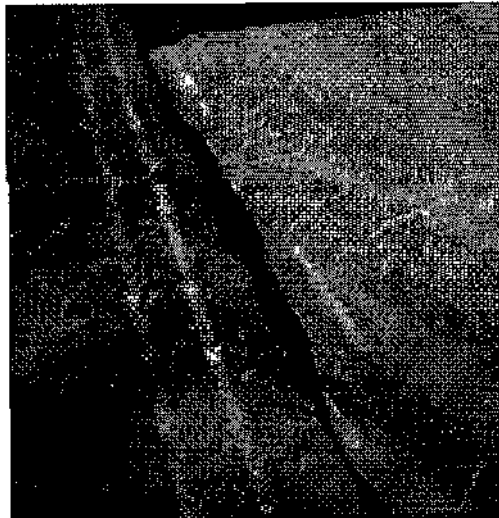
Figure 8 On the left an uneven surface cut by explosive; half casts of the holes are well visible ("Verde" gneiss)
On the right a smooth surface created by diamond wire saw

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Application of Pit Optimisation Algorithms Beyond Open Pit Limits

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ABSTRACT: Pit limit optimisations are used extensively m open pit mine planning to determine the ultimate pit limits and open pit mining sequences. Various standard techniques for the analysis of pit limit optimisation results have been developed and accepted by the mining industry today. This paper presents two relatively new techniques employing pit limit optimisation algorithms beyond the definition of open pit limits: (1) optimisation of waste dump limits and (2) definition of optimum mining sequences through blending pit sequences from multiple optimisation runs.

I INTRODUCTION

Pit limit optimisations are integral part of open pit mine planning today combined with the other mine planning tools such as pit design generators, production schedulers and cut-off grade optimiserai Pit optimisation algorithms in various implementation forms are the only planning tools that can produce feasible optimum pit development geometries automatically utilising the given geology, grade, slope and economic information.

Pit optimisations can be used at almost every stage of a project, from exploration program definitions to the preparation of feasibility studies, and finally evaluation of development options İn an operating open pit mine. Although pit optimisations are used widely in open pit mine planning, the use is rather limited İn context to the determination of ultimate pit limits and a pit development sequence only. It is common that sensitivity analyses for variations of individual input parameters are also included in the analysis of the optimisation results for the selection of ultimate pit limits.

In current long-term planning practice, waste dumps, the largest surface structures in open pit mining, are usually designed manually without assistance of any computer tool for optimisation and sizing. Various rules of thumb are used through a trial and error approach for the calculation of volumes and minimisation of haulage and other related costs.

The importance of mining sequence definition is also usually not evident in long-term open pit planning procedures. Usually a mining sequence is derived from a simple selection of pit shells based on optimum pit limits parameterised by the variation of a single input parameter. The performance of the obtained mining sequence with respect to the production constraints is generally not questioned prior to the detailed production scheduling stage of a project.

Extending the use of pit limit optimisation algorithms in long-term mine planning, two techniques are presented in this paper. The optimisation of waste dump limits utilising standard pit optimisation algorithms will be discussed in the next section. The optimisation process provides the optimum waste dump limits that minimises the dumping costs for given cost, distance, area and topographic surface variables.

In the Üıird section of the paper, a technique based on blending pit shell sequences from multiple optimisation runs will be introduced to achieve mining sequences that production constraints can be varied through time. This technique brings some degree of dynamism into the pit optimisations where the input parameters cannot normally be changed dynamically in the process.

Both techniques to be introduced for dump optimisation and mine sequencing were successfully applied recently in the development of open pit mining projects in Australia. The waste dump optimisations were used in three open pit gold mines to provide guidance in the mine designs. Syerston Nickel-Cobalt Project will be presented as a case study for the application of mining sequence definition technique in the fourth section of the paper.

2 WASTE DUMP OPTIMISATION

For large open pit mines, the haulage costs may constitute almost half of the mining costs. With reduced mining costs, lower grades and the added costs benefits of bulk mining, high stripping ratios ranging from 5:1 to 10:1 are common in surface mining today. This means that waste mining can make as much as up to 40% of the total mining costs. With the environmental issues and associated additional cost, waste dump design becomes an important task in today's open pit mining.

As established by Bohnet and Kunze (I990), important factors in the design of waste dumps are:
- Pit location and size through time
- Waste rock volumes by time and source
. Topography and property boundaries
- Existing drainage routes
- Reclamation requirements
- Foundation conditions
- Material handling equipment

Most of the design factors mentioned above can be quantified by assigning a cost factor which varies by surface topography and location. The ultimate objective of a dump design would normally be to minimise die total dumping cost, including haulage and other dump area related costs.

It is common practice that the CAD programs used for open pit designs are also employed to generate waste dump designs. No other computer tool or method was known until recently to assist, or most importantly, to improve the waste dump design process. Dincer (1997) introduced the application of a waste dump optimisation process in a case study. A custom computer program was developed to create a dump cost model and Whittle Four-D pit optimisation program was used in the case study, to optimise the waste dump limits.

2.1 *Dump v Pit Optimisation*

The dump optimisation problem can be described as a mirror image of the pit optimisation problem vertically. The slope constraints in dump optimisation are defined by using a set of structural arcs as in the case of pit optimisation. The slopes defined by the structural arcs are simplified in the form of cones in Figure 1 : an inverted removal cone for a pit and a dumping cone for a waste dump. In order to mine and ore block at the base of an open pit, the associated blocks within the removal cone should be mined first. In the case of a dump, the block within the dumping cone should be dumped first to be able to dump a block at the top of the cone.

The general procedure used for the optimisation of waste dumps are provided in Figure 2. The procedure is similar to that of pit optimisation but

the dump cost model is created through a computer program outside the modelling package. The area codes generated in the planning package can be used to divide the topographic surface into different cost areas. By using the dump area codes as the equivalent of ore types in a pit optimisation, it becomes possible to report and analyse the dump volumes and costs by different dump areas.
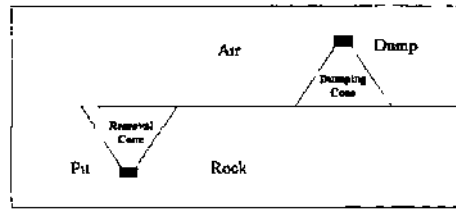

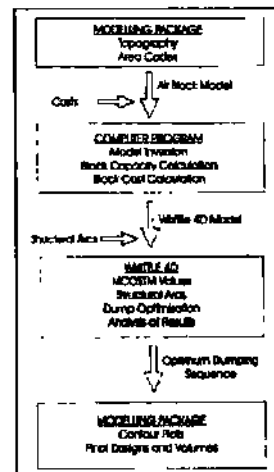Figure I Removal v dumping cones

Figure 2. Flowchart for dump optimisation process.

The block model input to the optimisation is inverted around the horizontal plane so that the original air blocks transform into solid blocks. Conversely, the solid blocks in the original model becomes air blocks since they do not have any effect in the dump optimisation process. After model inversion, the slope angle constraints are defined by the creation of structural arcs in exactly the same way as for pit optimisation. Additional arcs (or slope constraints) can also be defined to represent surface structures, such as roads and drainage routes that will affect the waste dump limits.

2.2 *Dump Cost Model*

Haulage and area costs are the two main categories of costs that can be used to construct a model for

dump optimisation purposes. The haulage cost is calculated for each block in the model depending on the block's location with respect to pit exit and dump access points. In the case of multiple pit ramp exits, the pit exit providing the lowest haulage cost can be selected for the calculation of haulage costs. Since the haulage cost depends on the vertical displacement as well as the total distance travelled, it is divided into horizontal and vertical components. The operating cost for the haulage equipment is also required in *the* calculation of the haulage *costs.*

The area costs apply to the blocks on the topographic surface. They can be allocated either as a direct area cost or lump sum cost assigned to a single block linked to other blocks in the area. The direct area cost is allocated on the basis of the unit area and can be used for such items as land acquisition, clearing and rehabilitation costs. In the lump sum cost assignment, the total cost would be incurred fully in order to access any of the blocks in the specified area. This method can be used to allocate the cost of diverting a drainage route or shifting a surface structure such as a road.

2.3 *Dump Optimisation ami Results*

After the calculation of the dumping costs and the available dump volume for each block, it İs necessary to transform these variables into a form that can be used by the pit optimisation process. The open pit economic variables in the calculation of net block values are substituted in the dump optimisation model as follows:

. Dumping cost in dump optimisation replaces the mining cost in pit optimisation. Processing cost becomes redundant in dump optimisation since all the costs are represented in the dumping costs.

. The dumping capacity (block volume) İn dump optimisation replaces the product (metal or mineral) in pit optimisation.

• Product price in dump optimisation becomes a factor applied on the dump volumes to generate net block values used in the optimisations. The magnitude of revenue factors to be applied in the dump optimisations depends on the magnitude of the cost values stored in the dump model.

With fee application of a range of revenue factors, the resultant dump increments from the optimisations are ordered from the best, having the lowest dumping costs, to the worst, having the highest cost. As well as the determination of an optimum dumping strategy, the original case study (Dincer, 1997) showed that the dump optimisations can also be used for the evaluation of options for the placement of major surface structures. The optimum dumping cost curves such as shown in Figures 3 and 4 can be generated for the evaluation of mine design options for major structures.
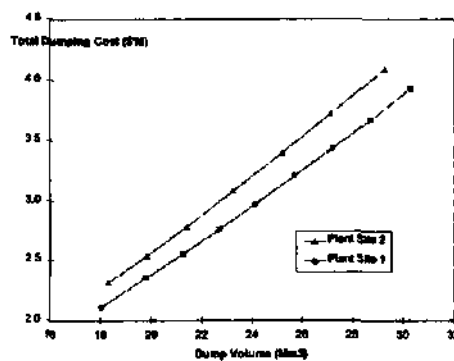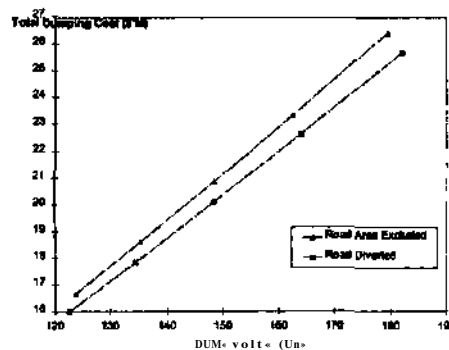
Figure 3. Comparison of plant site options.

Figure 4. Comparison road diversion options.

3 MINING SEQUENCE DEFINITION

In current long-term open pit planning practice, mining sequence definition is usually based on the pit shell selections from a family of nested optimum pit shells valid for one set of technical and economical parameters. This approach might be valid for relatively simple deposits with short mine life but probably will not provide the optimum mining sequence in the case of the following:

• Deposits containing multiple elements with revenues factored by product categories;

• Deposits with significant lateral extent and multiple mining areas;

• Massive relatively uniform grade deposits with pit economics depending on the surface geometries and slight variations m grade; and

• Long-term projects requiring the inclusion of risk factors, market limitations and other corporate objectives.

3.1 *Pit Limit Parameterisation*

The process of obtaining the family of nested pit shells for a range of parameters through pit limit

optimisations is called "pit limit parameterisation". The pit optimisation process, and consequently the parameterisation process, is static so that the parameters can vary İn the calculation of the block values within the optimisation model but they cannot be changed dynamically through time. Repeated runs are required to determine the optimum pit limits for a range of parameters that can be used for sensitivity analysis purposes and definition of a mining sequence for incremental mine development.

The previous work for the parameterisation of pit limits and development of a mining sequence can be summarised as follows:

- Lerchs and Grossmann (1965) highlighted the complexities in defining intermediate pit contours and suggested the parametric analysis of the optimum pit shells to determine an optimum digging partem to achieve the final pit limits.

- Bongarcon and Maréchal (1976) assumed a constant cut-off grade and used a parameter (X) defined by the ratio of mining cost to unit price of the metal to parameterise the open pit limits.

- Whittle (1988) produced a pit parameterisation program (Four-D) based on a parameter defined by the ratio of the product price to the mining cost (l/A). This parameter was utilised in the optimisation such a way that the resultant pit shells were basically parameterised by price.

Besides the techniques involving parameterisation of open pit limits, there are also some other approaches to determine the optimum mining sequences (and in part the production schedules). These approaches can be summarised as dynamic programming techniques (Wright 1989, Dowd and Onur 1992), heuristic search methods (Wang and Sevim 1992) and artificial neural network method (Tolwinski and Underwood 1992).

In the case of a single element or product, simple parameterisation of pit limits and other approaches would probably be sufficient to determine an optimum mining sequence. Even in die single element case, depending on the type of the deposit and the grade distribution, the varying cut-off grades and metal prices may require further analysis of the optimum pit shells. The mining sequence to be adopted may also be affected by the factors associated with die production constraints and risk such as confidence levels on die resources. Palma (1997) provided such a case in which several mining sequences were studied for the same deposit. The selected sequence from the study was one of the sequences (not the original price parameterised sequence) that would satisfy the corporate risk management policy.

3.2 *Mining Sequence and Production Schedule*

Prior to preparation of the detailed production schedules, definition of the mining sequence is a critical stage of a project's development since it combines geometry, volume, tonnage, grade, time and economic dimensions for a project as follows:

- *"Geometries"* in the form of pit shells partly addressing mining practicality and accessibility issues

  "Quantities" reported within the geometries (bench volumes, tonnages and grades)

- *"Economic"* evaluation of the quantities based on cost and revenue factors

  "Dependency" of geometries and mining *"order"* of quantities

- inclusion of *"time dimension"* in the preliminary schedules and option evaluations

As schematically shown in Figure 5, me mining sequence would constrain the production scheduling process by defining die bench quantities and dependencies as main input to the schedules. The production scheduling process does not usually have the geometrical concept and the dependency relationships defined by the pit slopes and access considérations used in die generation of the mining sequences. As the production schedule is mainly driven by the input data, this will in turn will have a fundamental effect on the mine and mill production rates, cut-off grades, ore quality and stockpiling strategies. If the mining sequence does not account for me production schedule constraints, then major alterations to the mining sequence (pit stage designs) are often required to improve and optimise the resultant production schedules.

3.3 *Blending Optimum Pit Mining Sequences*

As die complexity of tile mineral deposit and scheduling process increases, it is important that more attention should be paid to the mining sequence definition process. The proposed mining sequence definition methodology can be summarised as follows:

- Define a set of pit optimisation runs that will investigate the critical factors and areas for the definition of the mining sequence;

- Combine and examine the families of the nested pit shells from the set of pit optimisations for:
  o The change in physical quantities for defined mining areas and/or ore types,
  o The schedule objectives, blending and likely stockpile build up requirements,
  o The variation in operating costs and cash flows,
  o The variation in any other constraint or schedule objective that would affect the Mining Sequence,

  Select individual pit shells from the pit optimisation runs that suit the constraints and criteria for each option;

- Rationalise the pit shell surfaces to create a Wended mining sequence; and

• Prepare a preliminary production schedule to verify the sequence with the inclusion of the time dimension.

In this method, the pit shells obtained from pit optimisations are treated simply as shapes that are analysed and manipulated to obtain a practical mining sequence that will maximise the project cash flow within production and corporate constraints. In addition to the definition of the optimum mining sequence for the project, further advantages and contribution of the proposed methodology might be summarised as follows:

• Definition of the ultimate pit limits can be carried out dynamically taking into account the product specifications, blending requirements and variation in input parameters.

• Earlier analysis and development of the pit development strategy with various options and preliminary schedules save time and cost in the development of the project.

• Problem areas and periods can also be identified and various measures can be taken to solve the production problems in the mining sequence.

• As in real mining practice, the mill feed would be physically controllable in the source defined by the mining geometries rather than trying to deduct meanings from the behaviour of a scheduling tool.

• A comprehensive understanding of the mineralisation provided in terms of contribution of different ore types, geology and areas.
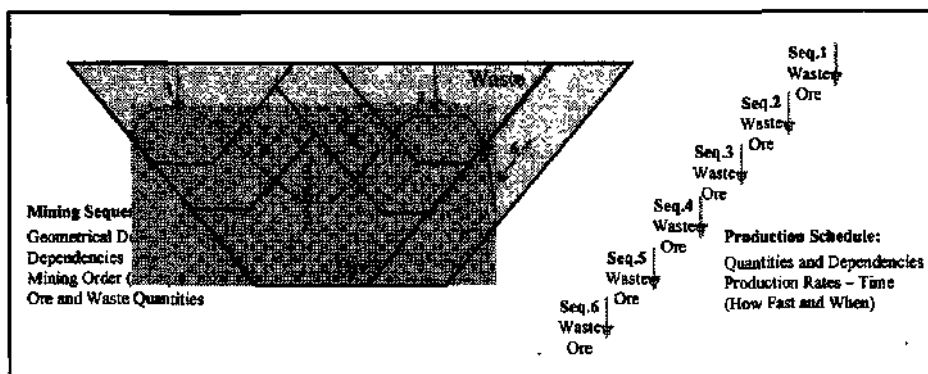

Figure 5. Comparison of mine sequencing and production scheduling processes

4 CASE STUDY- SYERSTON NICKEL-COBALT PROJECT

Syerston Nickel-Cobalt Project (Syerston) is located 400km west of Sydney in central New South Wales, Australia (Figure 6). The Syerston mineralisation is a limonitic nickel-cobalt latérite containing a resource of 100 million tonnes at 1.06% Nickel equivalent. The relatively compact resource at Syerston, covering an area of some 2 kilometres by 3 kilometres, is suitable for low-cost open pit mining.

The Syerston processing plant has been designed at a nominal capacity of 2.0 million tonnes per annum autoclave feed following a ramp up period of two years. The capacity in terms of metal production is 20,000 tonnes nickel and 5,000 tonnes cobalt (platinum by-product). The required mining rates per annum for a sustainable mill feed rate of 2.0 million tonnes vary between 6-10 million tonnes (ore and waste).

4.1 Syerston Feasibility Study

llie feasibility study schedule for Syerston was based on a pit shell sequence selected directly from a Whittle Four-X optimisation run. A linear programming tool was used to schedule the quantities calculated within the optimum pit shells.
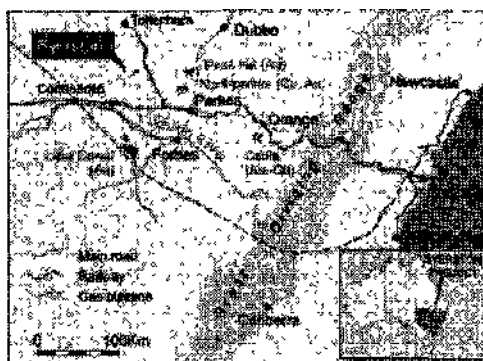

Figure 6. Syerston project location.

The operation has initially been planned for a 20 year mine life in high grade ore (+1.0% Nickel equivalent). The total operating life is expected to be in excess of 35 years including treatment of the low grade ore mined and also that rehandled from the stockpiles built during high grade operation.

The distribution of metal production during 20 years of high grade ore treatment İs shown in Figure 7. The total metal production varies between 21,000 and 27,000 tonnes in the initial 10 years of the high grade operation with an average nickel to cobalt production ratio of 4.6:1.0. The production starts to decline slowly after 10-12 years of the operation down to 15,000 tonnes of total metal at the end of the 20 years with the treatment of gradually lower grade ore. Figure 8 shows the amount of stockpile re-handling during the first 20 year's of operation as a percentage of mill feed. As seen in the figure, the stockpile re-handling can comprise up to 30% of the mill feed in some years with the overall average ratio of approximately 20%.
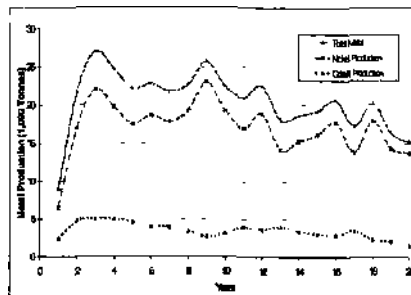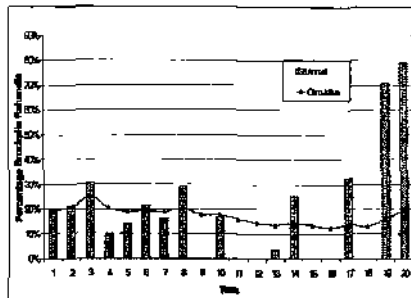

Figure 7. Syerston feasibility study metal production schedule.

After the review of the Conventional Schedule and the quantities in the pit limits, possible areas
Figure 8. Syerston feasibility study production schedule stockpile «handling.

4.2 *Redefinition of Syerston Mining Sequence*

After the review of the feasibility study results, the possible areas for improvement in the Syerston production schedules were recognised as follows (Dincerand Peters, 2001):
- Definition of relatively larger/continuous mining areas for the improvement of the mining widths and access ramp configurations;
- Decrease in high grade ore stockpile movements with mining larger areas which would provide more flexibility with ore and waste mining rates; and
- Increase in metal production in the early years of the operation by concentrating on areas with relatively high nickel and low cobalt grades.

To achieve these objectives, further pit optimisation runs were planned for systematic analysis of the optimum pit sequences. The base case pit optimisation using only high grade ore at study nickel and cobalt prices indicated a high rate of cobalt production in the early years. This was not desirable as the marketing analysis indicated that the total world production of Cobalt is approximately 35000 tonnes per year and the Syerston Study should target 5 000 tonnes per year. Four additional optimisation ruas were completed for a range of nickel/cobalt price ratios resulting in five different optimum pit shell sequences. As expected, the cobalt production was decreased in the early years of the sequences obtained from optimisations with higher nickel/cobalt price ratios.

The pit shell sequences obtained from each optimisation run were analysed together for the following indicators in the given order of importance:
- Nickel/cobalt production ratio of more than four (20,000 tonnes of nickel and less than 5,000 tonnes of cobalt) in the initial 3-5 years of the operation. The ratio is normally lower in the pit shell sequences for later years.
. While achieving the nickel and cobalt production limitations, maximise the early cash flow to provide early return from the operation as much as possible {maximum NPV)
- To achieve the above, the smaller pit shells for the mining sequence were selected from the higher nickel/cobalt price ratio optimisation. The price ratio was gradually decreased with the selection of the larger shells so that the ultimate pit limits from the base case optimisation were at the study nickel and cobalt prices.

4.3 *Syerston Case Study Results*

After the analysis of the mixed list of optimum pit shell sequences, two final mining sequences were blended each composed of 7-9 pit shells selected from different optimisations. The pit shells in the blended mining sequences were rationalised to provide a new family of pit shell surfaces. The metal production charts for the preliminary

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production schedules based on Mining Sequences 1 and 2 are provided as a percentage of the initial feasibility study schedule in Figure 9.
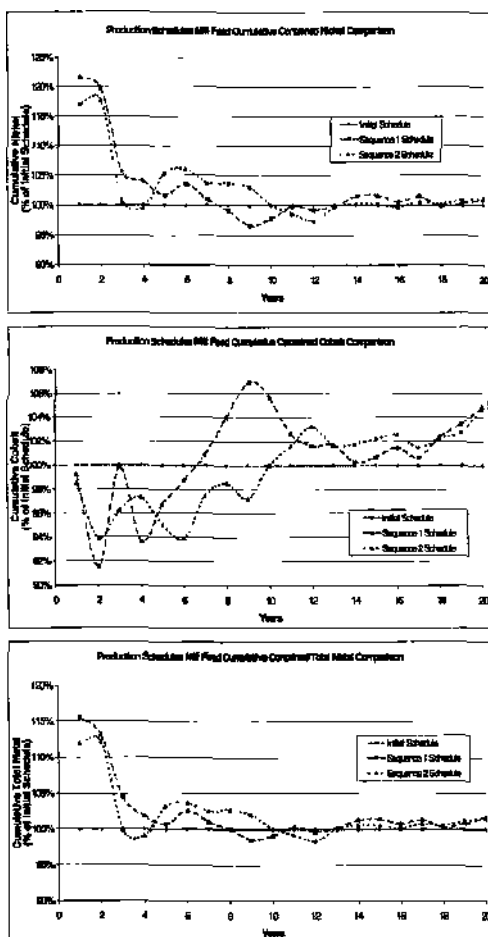

Figure 9. Production schedule cases- Comparison of metal production.

Comparison of the charts above shows the following improvements with the development of the blended mining sequences:
- The cumulative nickel production is increased by approximately 15-20% in the initial two years of the operation and 4-6% in the first six years.
. The cumulative cobalt production is decreased by 6-8% in the in the initial two years of the operation and 1-6% in the first six years.
. The cumulative total metal production is up by 12% in the initial two years and 3-4% in the first six years.

- The cumulative total metal production is very similar for all schedule cases after 10 years of operation. The metal production in the Mining Sequence 1 and 2 schedules are slightly higher at the end of the 20 years due to treatment of high grade ore only. No high grade ore stockpiles are allowed in the generation of Mining Sequence 1 and 2 schedules.

5 CONCLUSION

Two applications of pit optimisation algorithms beyond the definition of open pit limits have been discussed in this paper. Further to the standard application of pit optimisation algorithms, these applications significantly improve the main long-term planning tasks of dump designs and detailed mining sequence definition.

The dump optimisation process introduced in the second section of the paper is a useful planning tool especially where:
» The topographic surface is variable;
- The operation size is large with a high stripping ratio;
- There are multiple dump areas and these areas depend on the location of other surface structures; and
. The costs are variable between the dump sites due to differences in clearing, reclamation and acquisition requirements, and
In addition to the direct assistance to the dump design process with the definition of the minimum cost dump boundaries, dump optimisation can also be effectively used for mine site design purposes. The site options for major surface structures can be analysed with respect to die dumping costs using the optimisation results. The options for pit ramp exit positions can be evaluated iteratively to reduce the waste haulage costs.

As shown in the third and fourth sections of the paper, mining sequence definition İs a critical stage of project development combining the geometrical definitions from pit optimisations and the time dimension from production schedules. With the proposed methodology, the production constraints and other factors that cannot be quantified İn the pit optimisation models can be addressed in the generation of the mining sequences.

The application of the proposed methodology in the Syerston Nickel Cobalt Project was able to show practical mining sequences that can facilitate the control of different production rates for multiple elements and management of stockpiles. Preliminary production schedules of mining sequences showed that favourable results are

achievable compared to a production schedule generated by using a single optimum pit shell sequence and a linear programming tool.

It İs considered that the proposed mining sequence definition technique is applicable and bring significant benefits to the projects where there are conflicting and competing constraints, including nickel latente, iron ore, polymetallic, base metal and mineral sand deposits.

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Present State of Hard Coal Underground Mining in the Czech Republic

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ABSTRACT: This paper describes the development of mining in the Czech Republic. Special attention is paid to development in recent years characterized by mining activity damping; i.e., reduction in mining activities. The organization of underground coal mine damping and die experience with damping in the most significant hard coal deposit in the Czech Republic - the Ostrava-Karvinä Region - is described.

1 INTRODUCTION

The development of human society is integrally bound with the use of raw materials. In the historical Czech countries in Central Europe, the beginnings of mining are connected with the mining of precious metals and ores İn the Middle Ages. The rapid development of industry at die end of the 19ᵗ century and in the 20ᵗʰ century affected coal mining and the development of mining technologies in underground as well as opencut mining. After World War II, intensive mining of uranium began. Significant disruption in Czech mining occurred after the year 1989, when, in association with the transformation to a market economy, the restructuring of the whole national economy took place. The decrease in demand for products of the mining industry as well as difficulties with the sale of these products on the home market led to the liquidation of ore mining, limitation of uranium mining with the prospect of uranium mine activity finishing in the short term, and also a decrease in underground as well as opencut coal mining.

These changes also affected the largest domestic hard coal deposit - the Ostrava-Karvinâ Region. The Ostrava-Karvina Region was the greatest producer of hard coal mined underground İn the past and remains the most important deposit although production has significantly dropped. The greatest output in the history of this region was 24.8 million tons, which was achieved in the year 1979. Then production started to drop, the only exception being in the year 1987, when output compared to the preceding year increased, to the level of 22.8 million tons. Since that time, production has been constantly decreasing, to the current level of approximately 13 million tons.

2 DAMPING OF UNDERGROUND COAL MINING

In the years 1945-1989, in the former Czechoslovakia 4 723 million tons of coal was mined, of which 1 278 million tons was hard coal and 3 495 million tons was brown coal and lignite. Since the year 1987, the process of significant deoüne in annual coal output volumes has continued. In the year 1989, 112.1 million tons of hard and brown coal was mined, while in the year 1999, production dropped to 59.3 million tons of hard and brown coal. The decrease in hard coal production was smaller than the decrease in the brown coal output.

The restructuring of the mining industry took place with mining damping and the rise of new mining companies oriented towards profit and working in a competitive environment. Mining damping was, in essence, the process of adaptation of previously oversized mining capacities to new economic conditions. This process was caused by a whole series of factors: the decreasing importance of heavy industry; a decrease in economic efficiency; the opportunity to obtain all raw materials by means of importing; an emphasis on efficiency; the end of subsidiary and redistribution processes in the national economy; and the increasing importance of the ecology.

The reason for mining limitation and mines closing in the Ostrava-Karvina Region since the beginning of the 90s has been the economic conditions, especially the long-term negative economic effects and problems with coal sales. Mines characterized by long-term mining activity, mining at great depths, the necessity of air

conditioning or shortening of the working time underground were closed. The damping of underground mines in the Ostrava-Karvina Region began at fhe end of the year 1991, when mining finished and the liquidation of the first mine was commenced. To date, mining has finished in 7 mines.

3 ORGANIZATION OF UNDERGROUND COAL MINE DAMPING

Mining damping, with respect to significant economic, social, safety, ecological, technical aspects, etc., has to be uniformly organized and controlled.

The organization of underground mine damping is as follows: the existing state and outlook of a mine in question is considered from the viewpoint of economic results, ensuring sales of its production, ecological loads resulting from its activity, the safety of mining activity, and from the viewpoint of social, technical and other factors. On the basis of analysis, it is decided whether the coal deposit is suitable for further mining and processing of the production. In this way, the pressure is developed to leave such coal deposit parts which do not comply with the criteria for the regions considered. After this comes fhe completion of mining activity and the liquidation or securing of a mine or its non-effective part.

In these cases then, the application of the organization for inclusion in the program of coal and ore mining damping is worked out. The application must be documented by a variant solution of the liquidation or securing of a mine. The basic variants include the following:

- liquidation of a mine with mining completion at the commencement of damping;
- liquidation of a mine with additional mining of reserves after the commencement of damping;
- securing of a mine;
- a combination of the methods above.

After the selection of the optimal variant and its approval by the Ministry for Industry and Trade, the mine is included in the relevant decree of the Czech Republic government. In this way, a substantial part of the costs for damping can be covered by state budget subsidy.

The condition for commencement of damping with the participation of the state budget m the form of subsidy is the working out of the Technical and Social Project of a Mine Liquidation or Securing, which is the document worked out according to the unified methodology. The mine damping then includes the following:

- technical liquidation or securing of a mine or surface;

- elimination of the mining activity consequences;
- socially-health costs connected with damping.

Liquidation is the activity which includes the completion of mining works and cancellation of the property of the enterprise underground as well as on the surface. The securing of a mine, on fhe other hand, is the procedure of the finishing or limitation of mining works and temporary securing of the enterprise property for the possibility of later renewal of mining

The costs for damping should preferably be settled from:

- the reserve created according to mining law,
- the means of the mining enterprise,
- the yields of mining damping.

The greatest share of costs, however, is contributed by the state budget - in the Ostrava-Karvina Region it accounts for approximately 90 % However, there is no legal claim on this subsidy and its size remains uncertain. For this reason, the damping organization has to take into account ihe economics of the process

4 EXPERIENCE WITH DAMPING OF UNDERGROUND COAL MINES

The beginning of the 90s, with mining damping, took *the* industry of underground coal mining and the mimng of other raw materials by surprise. After the first hurried mining closures, the problems started to be solved methodically, and the empirical experience of the substantial variability of forms and ways of underground coal mine damping began to be compiled and evaluated. From existing knowledge, two marginal damping variants can be considered.

- The go-ahead variant. This İs characterized by die short time from the announcement of damping to the finish of mining. The reserves prepared for mining are not exploited, the liquidation works m a mine are carried out very quickly, the clearance of only some machines is carried out, and, for ecological reasons, the quick sale or elimination of surface objects takes place. The shortest time from the announcement of mining damping to the finish of mimng finishing was only 3 months. This variant shortens the operation time of costly power-consuming devices (compressors, ventilator fans, pumps). On the other hand, it can be the source of social problems related to job losses for a great number of employees, and the solution of these social problems is very costly.

- The step-by-step variant. This İs characterized by a longer time from the announcement of damping to the finish of mining. The coal

reserves prepared for exploitation are used, the large depletion of mine workings and clearance of machines are carried out, single areas of a mine are gradually liquidated or secured, the mine workings are used for alternative purposes (e.g., for depositing wastes), and the surface objects are gradually eliminated. The longest interval between the announcement of damping and finishing was 18 months in one mine and 39 months in the another. The advantage of this variant is that the solution of social problems is simpler and less costly, and substantially higher damping yields are obtained.

Should the time period between the announcement of damping and the fimsh of mining be evaluated from the viewpoint of technical-organizational parameters, the following can be stated:

- damping leads to the selection of relatively good conditions of mining (greater thickness of seams, longer average length of working faces);
- from the viewpoint of organization and management, in the time period of damping the trends are the same as in the preceding evaluated time period of 10 years - a decrease in certain parameters (e.g., average daily outputs of 1 working face, average daily areas worked out) in the previous time period is followed by a decrease in the damping period, while growth in given parameters in the previous decade is followed by growth in the damping period. In the case of a shorter damping time period, however, the negative changes are more conspicuous - decreases in the indicator values by tens of percentage points compared with the previous decade. This leads to the assumption that the cause of this is the quick method of damping, i.e., the go-ahead variant.

For evaluation of the time period between the announcement of damping and finish of mining, from the viewpoint of economic parameter development, the development of three liquidated mmes was analyzed where this period was three, six and fourteen months. The analysis showed that the shorter the time period is between the announcement of damping and the finish of mining, the worse the economic results compared with the previous time period. The results of analyses proved the positive effect of extending the time period between the announcement of damping and the fimsh of mining. In the case of unavoidable decrease in yields due to the decrease in mining, this longer time period gives greater opportunity not only to decrease costs dependent on the output volume (variable costs), but also to decrease costs of fixed character.

The experience from the mines closing also showed that the subsidy from the state budget was lower than expected by the mines. In this case, it was necessary to use the resources of mining enterprises which - as mentioned above - are created by:

- reserves created according to mining law: it is a question of strictly purposeful reserves for the removal of mine damage, maintenance and reclamation. With respect to their amount and possibility *of use, they* can cover only a *very* small part of the costs for mining activity damping.
- the financial means of the mining enterprises themselves: if a mine is closed for economic reasons, it means that the effect on its *economy* is a negative one. The expending of its own financial means on mining activity damping will deepen this loss and the enterprise will be in a worse economic situation than it would be if it continued its mining activity.
- the yields of the mining damping: theoretically, it is possible to obtain a number of yields İn the course of the damping of a mine - revenues from sales of coal mined after the announcement of damping, revenues from sales of metal waste arising in the course of technical liquidation of a mine, revenues from sales of machines, equipment and materials, revenues from rental of property determined for liquidation, revenues from sales of building material arising in the course of technical liquidation of surface objects, etc. However, experience of damping program realization shows:
- the sale of mining machines and equipment outside the Ostrava-Karvmä Region is rare because there are only a small number of potential customers;
- the transfer of mining machines and equipment in the framework of the district represents a problem due to different mining and geological conditions;
- certain benefits follow from the removal and renewal of steel arch support, but the use of this support at greater depths is not optimal from the viewpoint of its load-bearing capacity;
- the costs of retrieving metal waste from underground are higher than the market price, which is why the clearance of machines for scrap is done only for ecological reasons;
- furthei benefits can follow from depositing waste in underground spaces; however, ecological problems can occur there again, and these activities cannot prolong the period of mine liquidation;
- the costs of liquidation of above-ground objects are higher than the value of the usable material obtained.

It is evident then that the only important benefit in the mining activity damping ume period, which can decrease the necessity for state subsidy or improve the economics of the mine being damped, is

the revenues from sales of coal which is mined in the time period after the announcement of damping.

For this reason, the go-ahead damping variant was rejected, and the time period between the announcement of damping and the finish of mining was extended. As a result of this, the following was made possible:

- exploitation of all coal reserves prepared for mining;
- increase in the degree of tangible investment property depreciation used for a mine activity;
- the carrying out of liquidation and clearance works in a mine in the time period of the decrease in mining because these works would have to be done for safety or ecological reasons anyway;
- less costly solution of social problems (retirement of miners, requalification, new employment, lower severance pay)
- decrease in variable and fixed costs of the mining enterprise.

The extending of the mining time period, although with a decreasing trend, means the prolongation of the mine service life, the gradual solution of social and property questions, a decrease in the requirement for state budget subsidy, limited drawing of social allowances, and the maintaining of social conciliation.

5 CONCLUSIONS

The entrepreneurial process in the mining industry starts with a deposit survey and a build-up in mining and preparation capacities, and goes on with exploitation of the deposit itself, ending with mining damping, mine liquidation and the elimination of mining activity consequences. Underground and opencut coal mining in the Czech Republic was concentrated on the first three stages of this process until the end of the 80s, from the viewpoint of both theoretical investigation and practical activity.

The experience of the past 10 years with regard to the finishing of mining and liquidation of mines has shown that the last stage of the mining activity also requires the generalization of knowledge and theoretical investigation.

In the conditions of underground mining in the Czech Republic, especially in the Ostrava-Karvinâ Region, the following can now be observed:

- the aim of additional mining of coal reserves prepared,
- decrease in fixed costs already in the time period of mining decrease, i.e., immediately after the commencement of damping,
- creation of hypotheses for quick completion of liquidation works underground after the finish of mining,
- understanding of coal reserves as a non-renewable source of natural wealth,
- research into ways of obtaining the greatest use of this wealth.

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Dvofacek, J et at., 2000. Nâvrh opfl'mâlnf délky 1. etapy utlumu taïebni Eınnostı hlubinnych uhelnych dolù {Project of Optimal Duration of ihe I^{st} Stage of Mining Activity Damping of Underground Coal Mines), Grant GACR 105/99/00800, VSB-TUO, November.

Economic-Environmental Description of the Exploitation Method by Horizontal Slices and Ore pass in Italian Surface Crushed Stone Quarries

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ABSTRACT: The considerable spread of the method of exploitation by descending slices in large crushed stone quarries (limestone for cement and the lime industry in general) in Italy has often involved the adoption, by some companies, of the well-known system of hauling by vertical raise and extracting by horizontal drift. This choice İs influenced by a number of different factors: the possibility of reducing traffic on ramps, of reducing the diffusion of dust in me environment resulting from haulage by dumper, and the possibility of avoiding the dumping of blasted rock in open channels or along quarry benches. The bonng of the raise and the tunnel, which can represent an additional cost in quarry exploitation, has nevertheless been proven to be a good investment for the reduction of operation costs. This is because of increases in crew productivity, reductions in energy and equipment consumption and, above all, improvement in the safety and efficiency of the quarry. The final, but not insignificant, advantage İs that the first results of environmental recovery are immediately visible, with progressive exploitation from top to bottom and simultaneous rehabilitation of exploited benches. In the concluding remarks, on the basis of some meaningful examples, indications of the feasibility "scale" of the method are given for major and long-term planned extraction sites.

1 INTRODUCTION

Italian cement production, m spite of a lack of national energy resources, has for 25 years ranked highly in Europe. During the second half of the seventies, Italy was in fact the most important manufacturer, providing nearly 18% of the total production in western Europe and more man 5% of world production. In 1999, Italian production amounted to 37.300 Mt. The huge per capita consumption (up to 800 kg/y) recorded during the nineties gave rise to strong protests led by "environmentalist parties" against the overuse of cement in the country. As a matter of fact, apart from the large availability of raw materials for cement production, Italy has quite an irregular orography, with a high population density, which often involves constructive models requiring large quantities of concrete. Hence, improvements in both environmental aspects and production cost reduction are urgentiy needed in order to reduce administrative problems in the opening of new quarries and to maintain the current level of production.

The examples given below describe the exploitation method currently used in important Italian quarries and are representative of different situations in the country: Üiey are intended to provide information about the current organization of quarrying and transportation systems used to feed large plants, while having competitive costs.
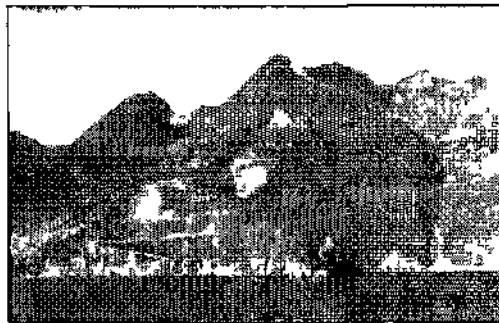

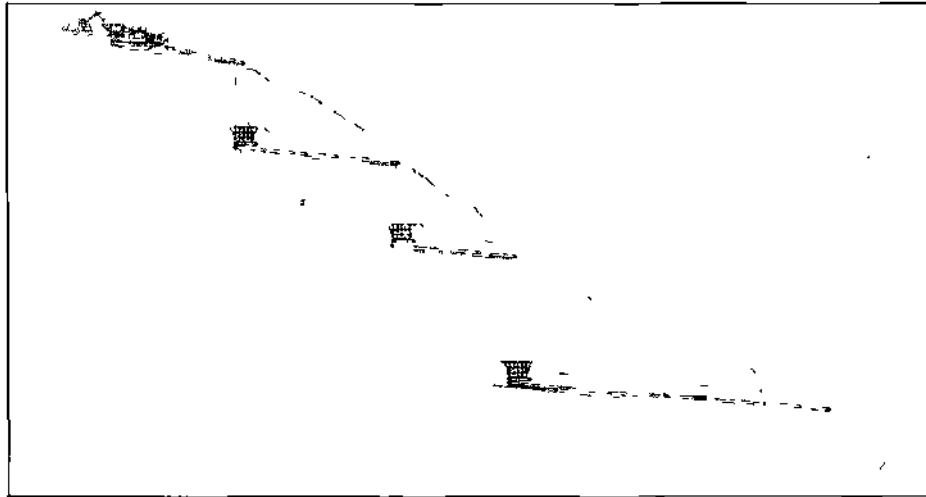Figure I General view of the quarry site (case A)

Figure 2 Scheme of the conveying system employed in case A.

2 CASE A

The section of the mine presently worked (see Figure 1) has been in production for one and a half years. The previously exploited section was worked by a different method, consisting of blasting by high benches (15 20 m), dozing to collecting troughs, rehandling by wheel loaders, and hauling to the crushing plant. A subsequent working method, in use until exploitation of the old section was terminated, was by ripping-dozing, with haulage by wheel loader directly to the plant.

The start of the present exploitation by horizontal descending slices (4 6 m thick), haulage by wheel loader to a mobile crushing plant, and subsequent transportation to the factory by an ore pass belt conveyor.system, entailed important development works:

- driving the ore passes (raises) and tunnels, representing the haulage way to the secondary crushing installation. Ore passes were excavated by raise borer;
- building access roads;
- installing an 18 kV power line;
- installing marl transportation facilities;
- installing the mine-factory transportation system.

A scheme-of the conveying system from the exploitation workings to the mine stockage site is shown in Figure 2.   •

Presently, the mine covers an area of 65,000 m², between the levels 930 m and 850 m a.s.l.. The exploitable volume is 1,100,000 m3. Future extensions are foreseen down to 640 m a.s.l., for an exploitable volume of 19,000,000 m³.

During the exploitation of the slices, a rock curtain is left in place, as shown in Figure 3.
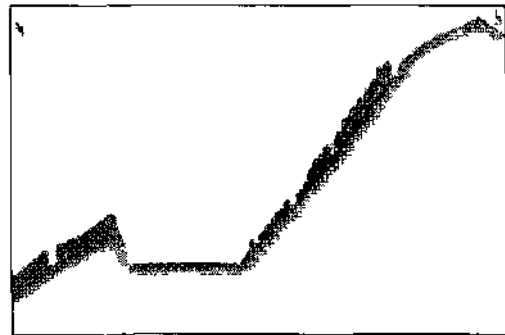

Figure 3. Rock curtain, to shield exploitation works (case A).

The average slope of the abandonment profile will be 36°, and the profile will be modelled by benches 4 m high and 4 m wide. The operational steps of the exploitation are:

1. Stripping the overburden, making the area to be exploited clear.
2. Drilling a square mesh, 3.5 x 3.5 m, of vertical holes 4 m deep, with a diameter of 70 to 80 mm, moderately charging the holes (less than 200 g/m³ of explosive) and blasting, just to loosen the rock.
3. Removing the loosened rock by ripper and backhoe excavator.
4. Hauling the rock to the primary crushing mobile plant.

Rehabilitation of the abandoned wall takes place as soon as the 4-m thick slice is removed, as shown in Figure 4; hence, excavation and rehabilitation are İn progress simultaneously during the exploitation.
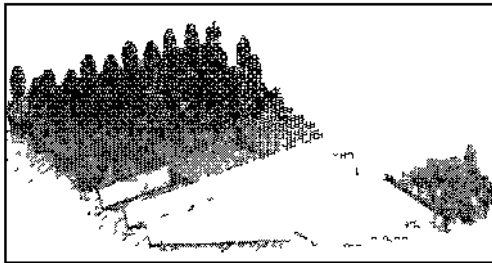
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Figure 4. Simultaneous progress of excavation works (case A)

The workplace contains 9 people, of which 6 are engaged İn exploitation and primary crushing, and 3 are employed in crushed rock transportation, secondary crushing and stockage.

As for the fleet, the main items are:
- 1 Tamrock CHA 660 drilling machine;
- 1 CAT 345 BL excavator and 2 Perlini DP 255 dumpers for production work;
- 1 CAT 330 BL excavator and 1 CAT 966 G wheel loader for environmental rehabilitation work.

The overall production is 3,550,000 t/y, constituting 70% of the factory's demand. As for the exploitation costs, İn total they approximate 2 US$/t (50% drilling and blasting, 45% mucking and hauling, 5% crushing and conveying to the stockpile).

3 CASEB

The cement factory has a nominal output of approximately 1,000,000 t of clinker per year (from 1,700,000 t of raw mix). The supply of raw material is provided by 3 quarries: two of these produce limestone and are located at 21 km and 0.5 km from the factory (the latter has been undergoing environmental rehabilitation since 1999, hence producing a very small output); the other produces clay, and is located 11 km from the factory.

Limestone represents 78% of the raw mix, and clay 22%. Hence, the main suppler of the factory is the first limestone quarry, producing around 800,000 t/y.

The original layout of this quarry comprised a mobile crusher, fed with run-of-mine rock taken from the muckpile with a hydraulic shovel and transferred to the crusher by a mobile belt system. The crushed material was then hauled by means of a conveyor belt installed at the surface to an ore pass, then by a conveyor installed in a tunnel to the facilities of the second quarry, and finally on to the quarry itself.

Later, on the basis of detailed studies and experience gained with similar plants, the design was revised so as to emphasize the importance of the environmental problem and make use of up-to-date technologies. The mobile crusher was replaced by a fixed model, and the quarry-factory link was established with a system of interconnected conveyors installed m tunnels and ore passes. Due to the location of the quarry with respect to the factory, haulage of the crushed material by truck is not possible, owing to the excessive distance and the unfavourable layout of the road.

The quarry floor is located at a height of approximately 650'm above the level of the old quarry, where the secondary crushing system is installed, and at a horizontal distance of 2000 m. The maximum size of the material obtained from primary crushing is 300 mm. The conveyor system linking primary crushing to secondary crushing consists of 3 ore passes and 3 tunnels hosting the conveyors, with a maximum slope of 17%. Tunnel-driving work started m 1993, and was completed in January 1998. The plant went into operation in June 1996. In Figure 5, a scheme of the system is shown.
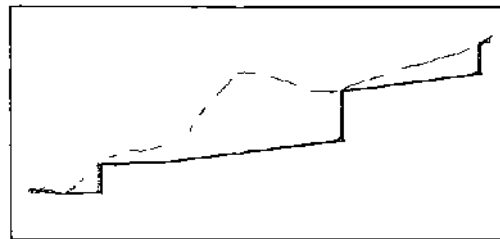

Figure 5. Scheme of the system employed in case B

Some details of the system and performance data are provided below.

The primary crusher directly feeds the first (upper) ore pass; a dust-collecting system İs installed at the feeding point. From the bottom of the first ore pass (vertical), the crushed material is transferred to the conveyor belt by an apron feeder. The conveyor runs in a tunnel and feeds the second ore pass (vertical), from which the material is transferred to the second belt, running partly underground, partly at the surface. The second belt feeds the third ore pass. A second dust collecting plant is installed at this feeding point.

The third ore pass is considerably larger than the first and second, serving as a surge bin as well as ore transportation facility, and has a capacity of 7,800 m^3 (11,800 t) of crushed limestone, which is a 2-day supply for the plant, working at full rate.

At the bottom of the third ore pass, in the third tunnel, an apron feeder feeds the third belt at the rate of 400 t/h, which represents the capacity of the secondary crusher, which is fed directly by the third belt

The secondary crusher is installed in the old quarry, and is an impact crusher with a maximum capacity of 500 t/h. It is driven by a 600-kW motor, with an inlet opening size of 2000 mm x 650 mm. The crushed product is conveyed to the factory.

The three ore passes connecting the conveying tunnels were driven by raise borer in order to obtain smooth and regular wall surfaces and to reduce construction time. The relevant geometrical data are given in Table 1.

Table 1. Geometrical data for ore passes pertaining to case B.

	Diameter (m) ·	Height (m)	Volume (m³)
Ore pass A	3.70	124 00	1,330
Ore pass B	3.70	202 05	2,170
Ore pass C	10.00	112 02	7,850

The tunnels have a cross section of 20 m , which provides ample room for inspection and maintenance. They were driven by drilling and blasting. The relevant geometrical data are given in Table 2.

Table 2. Geometrical data for tunnels pertaining to case B.

	Length <m)	Gradient (m)	Slope (%)
Tunnel A	718	75.60	12
Tunnel B	930	97.81	12
	304	8.64	3
Tunnel C	169	4.50	3

The exploitation method presently adopted by the quarry involves horizontal descending slices, leaving a rock curtain to hide the quarry work from the surrounding sites, thus minimising the visual impact, which is important in an area valued for its natural beauty.

Excavation is by drilling and blasting; the bench height varies from 10 m to 20 m (max). The explosives employed are ANFO (main charge) and Slurry (toe), with a powder factor of 260g/m³. Oversize blocks are crushed by a hydraulic breaker. Mucking and transportation to the primary crusher are carried out by means of a wheel loader and 24-m³ dumpers.

The primary crusher is a double rotor impact crusher, with a capacity of 800 t/h.

The quarry machinery fleet consists of:
- 3 heavy drilling machines, with diameters of 115, 105 and 89 mm;
- 1 hydraulic breaker installed on a backhoe excavator;
- 1 wheel loader with 5.4-m³ bucket;
- 5 dumpers with 2.4 m³ capacity;

- 1 crawler tractor for small works and site rehabilitation;
- 2 4wd vehicles and one minibus for the personnel.

The quarry is worked 1 shift per day (from 7.00 to 15.00), 5 days per week, by a crew of 9 workers.

A conventional transportation system would require a 30-km journey and 17 trucks, and could not compete with the system adopted.

4 CASEC

The cement factory rated capacity is around 1,600,000 t/y of clinker (2,560,000 t/y of raw mix), and the supply comes from 3 quarries. Two of these produce limestone and are located 12 km from the factory, while the other produces schist and is located 0.5 km from the factory.

Limestone represents 77% of the raw material needed, and schist (clay) 23%.

The total production of the two limestone quarries is 2,000,000 t/y. They went into operation at different times: the first in the sixties, and the second in the nineties.
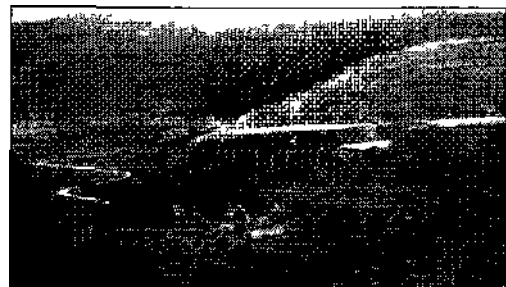

Figure 6. General view of the quarry site (case C).

The original method of exploitation was quite conventional, by high benches (15 m) starting from the toe of the mountainside. The exploited rock İs a grey Jurassic limestone, fine grained, in beds dipping against the slope. The stratigraphie succession is locally upturned, and the Triassic dolomia (not exploitable) rests above the Jurassic beds, representing the upper boundary of the pay rock.

At first, the transportation of the limestone to the factory was carried out by truck, which caused serious traffic problems. Later, in the seventies, a tunnel 6 km long and still in use, connecting the quarry to the factory, was driven, equipped with a conveyor belt, and a mobile (non-self-propelled) crushing plant was installed at the quarry. Meanwhile, the production of the factory increased (from 300,000 to 800,000 t/y of clinker),

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Figure 7. Conveyor belt in the tunnel connecting the quarry to the factory (case C).

Exploitation by descending slices started ahead of the completion of the exploitation of the old quarry in order to exploit the large reserves (>10 Mm) located on the same side of the valley, just above the village there.

The thickness of the pay was more or less the same as that in the old quarry (approximately 200 m), but the old method, consisting of successive exploitation of long benches, and dozing down the blasted rock from the upper benches to the quarry yard, was no longer practical, due to, among other things, the proximity of the inhabited area.

The exploitation by horizontal slices, starting from the top, minimised the circulation of machinery {access roads are needed, in principle, only for machine repair) and preparation work (one slice, covering a surface of approximately 6 hectares and being 15 m thick, gives 1 year of supply), provided that the transportation system was prepared in advance.

Transportation by ore pass provided a suitable solution (transportation by simply dumping the blasted rock in an inclined channel, employed for a small part of the upper levels of the quarry, proved to be unsuitable because of dust diffusion rain, snow collection and other problems).

An inclined (65°) raise, 145 m long (with a diameter of 4.5 m) was driven by drilling and blasting, employing several adits from the mountainside as intermediate attack points, in order to connect the top level to an underground crushing plant (see Figure 8).

The crushing plant is installed in a camera having a cross section of 110 m^2, and performs the primary and secondary, stages of crushing at a rate of 800 t/h; the crushed rock Is then hauled by the belt system to the factory. The excavation is by drilling and blasting with large diameter holes (115 mm), wide blasting mesh (4m x 4m) and low powder factor (80 g/t of ANFO explosive). Holes are detonated by cord, with interposed detonating relays to reduce vibrations. The preparation of a slice for the production stage consists simply in opening a trench reaching the ore pass collar. Mucking and transportation of the blasted rock to the ore pass is effected by wheel loaders.

The fleet consists of:
- 2 rotary drills;
- 1 hydraulic breaker for oversize block reduction;
- 3-wheel loader, with bucket capacity of 6, 8 and 10 m3;
- 1 excavator and 1 small wheel loader for preparation works and road maintenance.

A simple access road (1 km) is required, while the previously employed long benches method needed more access roads, which had to be maintained and adapted.

The ore pass has not become clogged to date. In any case, a stockpile of 10 production shifts is maintained at the factory (100,0001).

The work force was previously 18 people, which has been ceduced to 15 (the number of operators of wheel loaders and drills has increased, while the personnel required for road-building and maintenance has been reduced) and productivity has increased from 45 t/m.s. to 70 t/m.s.

Part of quarry where the transportation by dumping the blasted rock in an inclined channel was employed
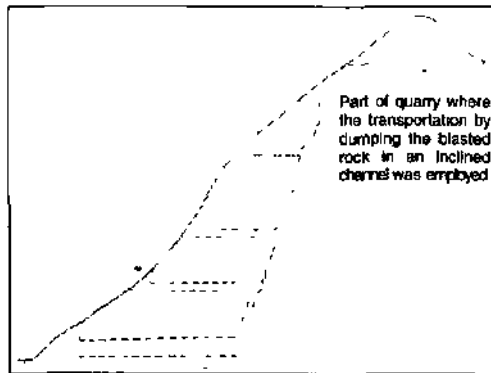
Figure 8. Scheme of the system employed in case C

Figure 9. Preparation of a slice for production stage (case C).

Cost reduction with respect to the previous method is estimated to be 25%. the previous long benches method utilized belt haulage; in the seventies, when transport to the factory was done by truck, the cost was much higher.

5 CONCLUSIONS

The exploitation method of horizontal descending slices with ore passes, crushing plants and tunnels representing the haulage way to the secondary crushing installation has several advantages:

Safety: quarry work is safer
- nobody has to work under the quarry front;
- there are bigger spaces for manoeuvring machinery;
- there are no falls of blasted rock in open channels or along the quarry benches;
- loading is simpler and faster.

Environment: environmental rehabilitation is very quick
- the progressive exploitation from top to bottom allows the simultaneous rehabilitation of exploited benches;
- the results are immediately visible;
- only a small part of the front is quarried.

Economy: exploitation costs are smaller
- rehabilitation of roads is not required;
- dozers are not needed and wheel loader work is more efficient and more flexible;
- the blasted rock is hauled in a simpler way;
- it is possible to improve the crusher efficiency by pre-screening the material;
- less manpower is needed.

In order to illustrate the technological progress and improvement of the methods of exploitation in Italian limestone quarries, some performance data are provided in Table 3.

The last datum refers to one situation and does not represent the national average. However, the result obtained is a great success for the cement industry in terms of technical-economic aspects and safety. It is also interesting to observe that this was achieved by an open pit exploitation method using large machines and underground development for efficient haulage. Such a method requires considerable investment, but it provides great results in terms of safety,"productivity and environmental aspects.

Table 3. Crew productivity in Italian quarries of limestone for cemenl in different periods,

	crew productivity	
1960	< 10	Underground exploitation by sub levé I caving
1970	<= 150	Open pit exploitation by vertical slices from bottom to top
1990	> 500	Open pit exploitation by descending slices, hauling by vertical raise and extracting by horizontal drift

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Evaluation of Single-Slice and Twin-Face Operations of Çayırhan Lignite Seams

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ABSTRACT: In recent years, Turkey has experienced very high demand for energy. The most reliable way of dealing with this problem is to build thermal power plants utilizing lignite coal as a source of energy. However, at this point, two other problems are encountered. First, those of Turkey's near-surface coal reserves that can be mined economically are being exhausted quite rapidly. Second, production of coal by underground methods is expensive due to the technology applied. Consequently, underground mining is becoming a burden on Turkish society. The two factors that prevent coal from being produced at low cost are the lack of advanced technology and the use of outdated mining methods. In order to overcome these obstacles, it is necessary to invest capital and establish modern mines. In this paper, the B and C fields at Çayırhan underground mine are compared in terms of mining methods and production efficiencies. In the C field, the coal is extracted in a single seam, whereas in the B field, the coal extraction is performed in a double seam. The annual production of Çayırhan mine is 4 million tons.

I INTRODUCTION

In 1996, Turkish Coal Enterprises (TKI) awarded a contract for the development of part of the Çayırhan deposit to a joint venture between Park Holding and the German longwall firm, SaarTech. Today, Park Teknik, as the joint venture is known, is at the forefront of Turkey's private sector producers, with output for the next 15 years scheduled for 3 Mt/year of power station fuel to be used in Çayırhan thermal power plant.

Eighty-five percent of Park Teknik is owned by Park Holding and 15% is owned by SaarTech, with the two companies taking responsibility for different aspects of the project's development. Park Holding looks after the local administrative aspects, while SaarTech provides technical management input as well as specifying and supplying equipment.

Modern equipment brought from Germany was installed underground by Turkish and German technicians. The mining method applied lias been the retreating longwall method.

Thé machinery and equipment brought from Germany is some of the most developed technology in the world in underground coal mining and is designed by Turkish and German technicians.

In the initial stage of the project, a total of 200 million DM of capital investment was made and a total length of 5000 m of roadways was driven in one year. This is considered a relatively short time in underground mining. After the installation of the face equipment was completed, a total of 2 Mt of coal was produced in the B Field in 1997.

After the investments for the C Field were finalized, during the second stage of the project, coal production was started in November 1999. Production was then increased to 4 Mt/year. As a result of the modern technology used and the high standards of safety established, efficiency and production increased in the mine.

2 PRODUCTION

Çayırhan lies in the Beypazarı lignite basin, about 100 km northwest of Ankara. It contains a number of separate fields, all of which host two principal lignite seams. The overburden thickness of between 150 and 200 m throughout the basin mean that open pit mining could not be used. A third seam some 130 m below the others also contains resources, but it is considered too deep to exploit. Across the basin, seams 1 and 2 vary in thickness from 1.7 to 2.0 m. They are separated by an interburden with a thickness of 1.3-2.0 m in the western part of the area and 0.5-0.7 m on the eastern side. The in-situ resources of these seams total 390 Mt, of which 236 Mt are considered to be mineable. While the thickness of the intermediate layer between the two seams is 1.3-2 m in the western part of die field, it is

0.5-0.7m on the eastern side.

Park Teknik's mining operations have been carried out in the B and C fields, which were developed as completely separate operations. The two fields host a reserve of 35.4 Mt, in which the lignite is of better quality than the average for the basin. The band of interburden is typically over 1.0 m in the B field (in the west) and less than this İn C (in the east), a variation that has had major implications for the mining layout of both fields. Hence, the two seams in the west of the field are worked separately, whereas in the east of the field the two seams together with the interburden are mined as a single seam.

The method of production applied in both fields is the longwall retreat mining method. In order to reduce the need for gateroad drivage, to enable more efficient use of reserves and to reduce the risk of spontaneous combustion of the coal, panel maingates were planned for use as the tailgate of the following panel.

2.1 B Field

There exist a total of 13 panels in the layout of this field. The initial development effort was focused here, with the aim of proving the capabilities of the twin-face system İn practice. A typical panel layout in the B field is seam 1, 1.6-m thick, separated from the 1.9-m-thick seam 2 by around 1.6 m of interburden waste (Figure 1). The mining sequence involves the upper face leading the lower one by around 25-35 m, a distance to ensure reasonable pressure distribution on both faces. The face inclination of the currently worked panel B-10 is 6 and the face length is 220 m.
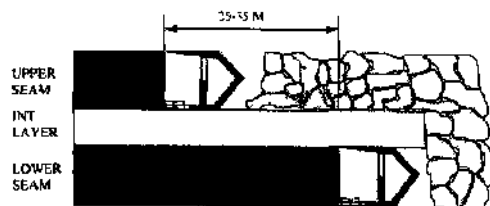

Figure 1 Production method in B field.

Production works in the B field are carried out with 148 workers. There was continuous production activity in Panel B-10 in the year 2000 and 1,651,000 tons of coal was obtained during this period. The monthly production levels are given in Figure 2.

The face equipment, especially the conveyors, nearly finishing the third panel after the contract was signed, required an excessive amount of welding and other kinds of maintenance. Therefore, in the year

2000, there was a remarkable increase in the number of breakdowns in mechanization. Figure 3 shows the proportion of breakdown events and their sources in the B field during working time as a whole in 2000.

Figure 2 Monthly production in B field in 2000
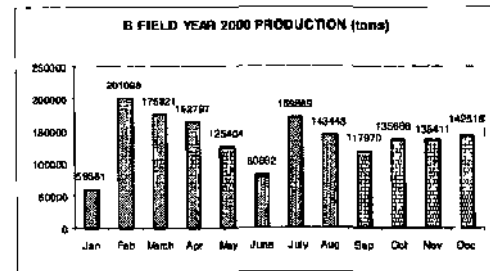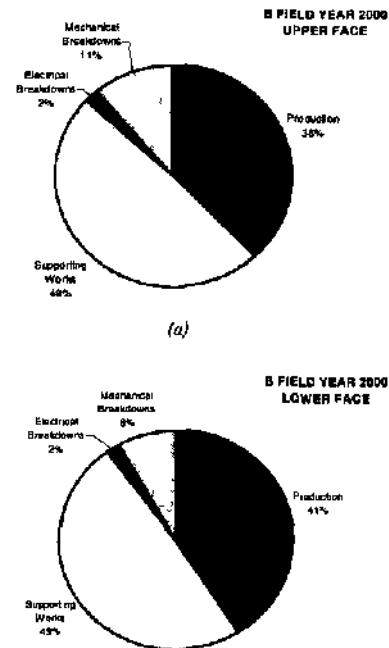

Figure 3 Distribution of breakdowns in B Field

12 C Field

Since the interburden layer is around 0.5-0.7m İn the C field, both of the seams and the interburden were combined as if they were a single seam and a single face was projected. The machinery and equipment were chosen accordingly. The face height, up to 5m, is as great as any longwall elsewhere in the world,

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and the panel inclination is about 30° against the retreating direction.

Panel lengths of 1700 m were chosen and the size of the face was 220 m, the same length as in the B field.

Production works in the C field are carried out with 106 workers, which is considerably smaller than the number of workers in the B field.

An Eİckhoff SL500, a double-drum shearer, is utilized in the faces of the C field. Each of the drums is driven by an electric motor of 500 kW. The total useful power is 1148 kW and the cutting head operating voltage is 3300 volts. The dimensions of the drums are 2300 mm in diameter and 900 mm İn width. The rotation speed is 23 rpm. The drums are globoid-type drums containing 76 cutting ends. The machine İs 14.27 meters long and has a maximum cutting height of 5.09 meters. The maximum hauling speed of the machine equipped with *a* Saartrack hauling system is 10.07 m/min.(F_m,,, = 445 kN). It weighs 66.5 tons without the drums.

A double-chain-cype AFC, manufactured by DBT, is used in the face. The motor power of this is 2 x 400 kW. The motor power of the transmission conveyors is 250 kW. Both conveyors have a carrying capacity of 2000 t/h. The face conveyor works at a speed of 1.11 m/s and the transmission conveyor works at a speed of 1.32 m/s, with a chain size of 34 x 126 millimeters. The tensioning units of both conveyors are Perinfalk-made PW 800s. Pouring from the face conveyor onto the transmission conveyor is carried out through a MRAR 35 side discharging chute. A SK 11 11 crusher with a power of 250 kW is mounted on the transmission conveyor.

A total of 127 25/50L advancing shields produced by SaarTech are utilized as face support (Figure 4). They are *of the* double-leg type. The height of these shields when opened is 5 m, while the closed height is 2.5 m. Each shield is 1.7 m m width and has a weight of 25.6 t. The setting force is 3400 kN and the sliding force is 4100 kN.

Three special STBS 27/50L shields are used for supporting each face end. Their open height is 5 m, closed height is 2.7 m, and their width is 1.75 m. The total weight is 38.3 tons. These face-end shields are of the four-leg type.

For both types of advancing shield supports, 16 functional Multimatik P-D, multi-channel hose connections to the roof control blocks are used.

In the year 2000, 2,275,000 tons of coal was produced from the C-12 panel. The production details are shown in Figure 5. The performance of the C field for the year 2000 was very high. The breakdown figures in the C field are given in Figure 6. The sum of the mechanical and electrical breakdowns was 10% of the total time.
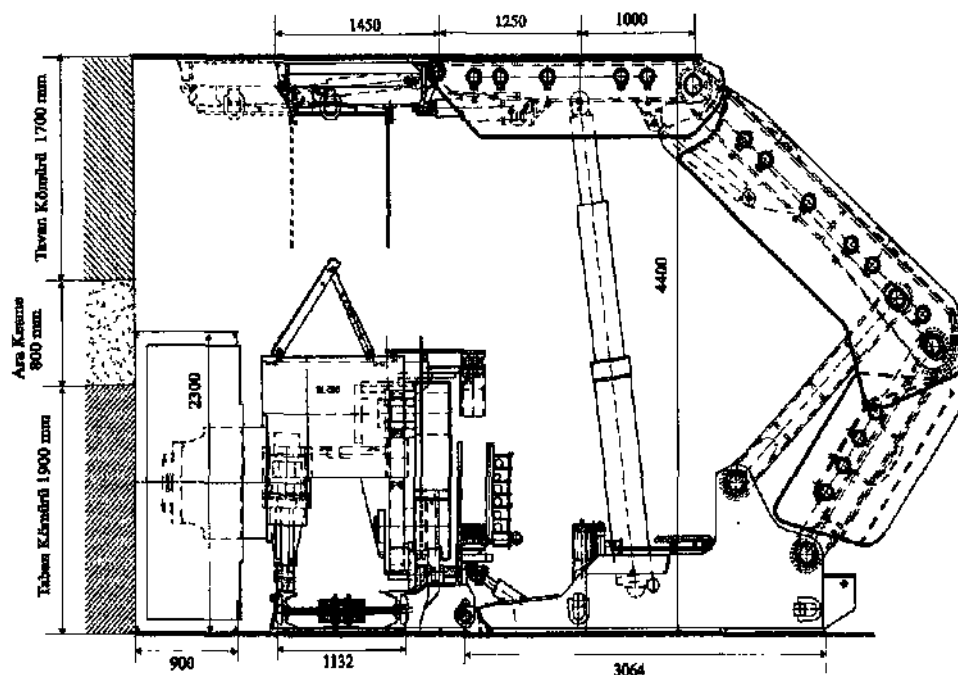

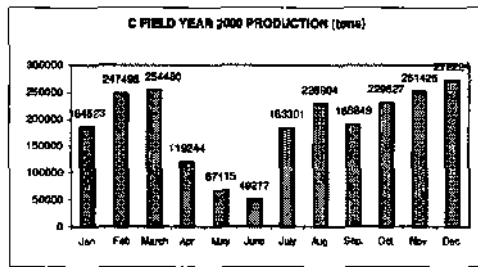Figure 4. Cross-section view of wall equipment tn C field
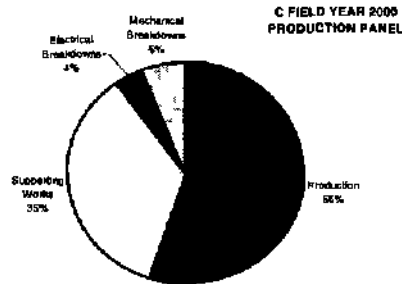
Figure 5. Production in C field in year 2000

Figure 6 Field C breakdown distribution

3 TRANSPORTATION

3.1 *Coal Transportatiom*

For coal transportation, belt conveyors 1200 mm in width are used. The strap used is a single-layer type covered with nitryl; it has a strength of 1250 N/mm. In all belt facilities, there are stretching and storing units. These facilities are controlled by pneumatic cylinders. All the drums in the belt facilities are covered with rubber. The driving and return end systems were obtained from Germany while the additional parts were produced in Turkey according to DLN standards.

In the transportation of coal in the B and C fields, 6 conveyor belts are used. Their technical specifications are given in Table l. The conveyor in the B field has a speed of 2.5 m/s and capacity of 1500 t/hr. The driving force varies from 270 kW to 750 kW. The conveyor m Field C has a speed of 3 m/s and a capacity of 2000 t/hr. The driving force varies from 220 kW to 750 kW.

Coal poured into silos with capacity of 200 t in the south portal of the B field and 300 t in the south portal of the C field is automatically loaded onto trucks and carried to the delivery point.

3.2 *Material and Man Transportatiom*

There are equipped endless wire systems operating a monorail and coolie-car driven and controlled by the B-C mine entrance to bring workers to their work place in a short time and in safety The monorail equipment is made by Scharf and the coolie-car systems are made by the Walter Becker firm. The monorail can transport weights of 3 t, 5 t, 10 t, 20 t and the rope has a diameter of 18mm. In Field B, transportation of the face equipment is by monorail in the gateroads and by mechanical hoist on the side of the face. In the coolie-car transportation system, conveyence cars of 100 kN and 220 kN can be used. In Field C, the transportation of face equipment is by coolie-car in the gateroads and by hydraulic hoist on the side of the face. The hydraulic unit of the hydraulic hoist is the E III type. The other hoists are of the GT 10000 type. The material transported by hydraulic hoist is moved on the rail system, which is located on the floor in the face. For both the coolie-car system and hydraulic hoists, ropes of 26 mm are used.

Table 1 Technical specifications of belt conveyors

(a) Field B

	Motor Power (kW)	Length (m)	Capacttv (t/hour)	Speed (m/s)
Belt-0	110	150	1300	2 5
Belt -1	2x110	650	1300	2.5
Belt -2	3x110	1200	1300	2 5
Belt -3	2x110	850	1300	2 5
Belt -4	2x110	650	1300	2.5
Belt -5	2x110	in preparation	400	2 5

(b) Field C

	Motor Power (kW)	Length (m)	Capacity (t/fiour)	Speed (m/s)
Belt -0	2x110	170	2070	3
Belt-1	3x250	570	2070	3
Belt -2	3x110	1200	2070	3
Belt -3	2x110	1100	2070	3
Belt-4	2x110	600	2070	3
Belt-5	2x110	in preparation	400	3

4 MINE-WIDE MONITORING CENTER

All the machines and equipment used İn the mine are fully automatic. There is also a push-talk phone system to provide communication between underground and the monitoring center.

This system, which is set up at the surface control center, ensures that all of the operating and non-operating equipment are monitored and their electrical currents are recorded. The monitoring and

control system was purchased from the Walter Becker company.

Again, from the monitoring center, CH4 and CO emissions and air quantities in the mine are continuously monitored.

5 CONCRETE PACKING

Since there is tendency for spontaneous combustion to occur in the Çayırhan lignite mine, in order to increase production by avoiding pillars left between panels and to use gateroads twice, as a maingate and later as a tailgate, thus to working adjacent panels with no pillars left in between, a ribside packing system containing concrete and power station fly ash is used as part of the gateroad support. The maingate, which is to be used as the tailgate of the adjacent panel, is held by a concrete block of 3-m width and excavated at coal height taken into the face. The concrete facilities are German patented and there are 4 surface silos in the B field. Each of these silos has a capacity of 100 t and they are utilized separately as cement, ash and 2 mixing silos. Electrofilter ash from the thermal power plant is mixed with cement at 30% in order to form a binding agent. In the C field, there are two mixing silos with a capacity of 100 t. After preparation of the dry mixture in the B field, it is sent to the C field by truck. Dry material in the silo is sent to an underground intermediate silo through pipes by means of compressed air. In this intermediate silo, it is mixed with water and sent to the region where the concrete packing will be built using a concrete pump.

6 COMPRESSED AIR

In the C field, compressed air required for hand tools, concrete units, tensioning stations of belt conveyors, waste water pumps operated by air, cooling of the main drive units, mechanical hoists, pick hammers, etc. is provided by compressors.

Thus, in the C field northern mine entrance, 4 compressors manufactured by Kaeser (see Table 2) and 3 MDH 90 compressors made by Hiross are installed.

Table 2. Technical Specifications of compressors in Field C.

	C FIELD
Manufacturer	Kaeser
Model	FS440
Weight (kg)	6300
Operating Pressure (bar)	75
Capacity (nvVsaat)	2000

In Field B, air is provided through OAL compressors.

7 VENTILATION

For the main ventilation gates of the mine, 2 suction-type fans are used in both the C and B fields (Table 3). Both have a flow rate of 40 m /s. The motor power of the one in the B field is 90 kW and that of the one in the C field is 75 kW. The airways in the mine have ventilation doors which are operated by compressed air.

While they are being driven, the sections of new roadways are ventilated by booster fans of 400 m^3/min capacity with dust depressants of 200 mVmm.

Table 3. Technical properties of fans used in ventilation

	B FIELD	C FIELD
Manufacturer	Davidson	Korfmann
Flow Rate (m^3/sn)	40	40
Motor Power (kW)	90	75

8 TECHNICAL COMPARISON OF EFFICIENCIES AND PRODUCTION PERFORMANCES

Details of the numbers of workers and their jobs are given in Table 4. The monthly production and worked days for 2000 are shown in Table 5. By using data from these figures, monthly efficiencies for the year 2000 were calculated; these are shown in Figure 7.

As can be seen from Figure 7, the monthly underground efficiencies in 2000 were high except for the months of April, May and June. In the C field, very high efficiency values were reached when they are compared with those for the B field. Generally, the decrease in the number of workers and the higher production due to the use of modern technology and extraction of coal in a single seam makes the C field more productive than the B field. The results proved that the use of modern technology increases efficiency to high levels.

In Figure 8, overall managerial efficiencies with respect to total production and total staff number are shown by month. The overall managerial efficiency for 2000 was 19.3 t/shift If we consider the average European efficiency, which is 6 t/shift, the efficiency of the mine becomes more meaningful. A comparison of the efficiencies of Park Teknik. Endsdorf Colliery and Deutsche Stein Kohle (DSK) is given in Figure 9.
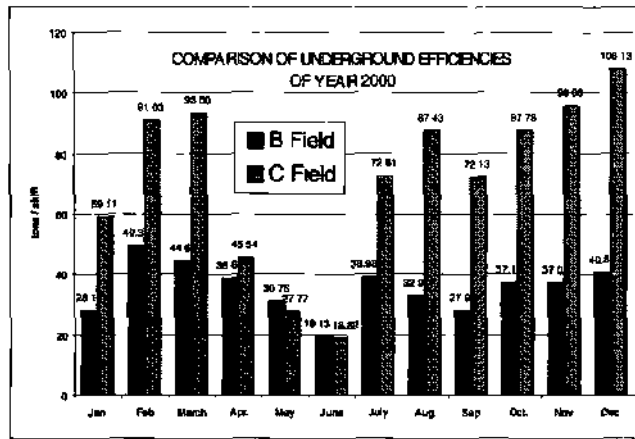
COMPARISON OF UNDERGROUND EFFICIENCIES OF YEAR 2000

- B Field
- C Field

Figure 7 Underground efficiencies in 2000

Table 4 Number of workers and their jobs in B and C Fields

	Shift 1		Shift 2		Shift 3		Daily	Total
	B Field	C Field	B Field	C Field	B Field	C Field	B Field	C Field
PRODUCTION								
Shearer Operator* Cable Controller	5	3	6	3	6	3	17	9
Shield operator	4	3	5	2	6	3	15	8
Face End Supports + Concrete	4	4	8	4	8	5	20	13
Return End. Supports + Concrete	4	4	8	4	8	4	20	12
Shield advancing team	4	4	4	4	3	5	11	13
Face cleaning	1	0	2	0	2	0	5	0
Monorail	4	4	4	3	4	3	12	10
A, F, C Dnver	2	1	2	1	2	1	6	3
MECHANICS								
Intermediate Concrete Silo	3	3	3	4	3	3	9	10
Shearer maintenance	4	3	3	3	3	3	1 0	9
Hydraulic shield maintenance	5	3	4	3	4	3	13	9
ELECTRICITY								
Face 1 Shearer ((B field)	1	0	t	0	1	0	3	0
Face 2 Shearer (B field)	0	0	1	0	0	0	1	0
Face SL500 (C field)	0	2	0	2	0	2	0	6
Face End	1	0	1	0	1	0	3	0
Concrete	1	1	1	2	1	I	3	4
TOTAL	43	35	53	35	52	36	148	106

Table 5 Monthly work days and production m 2000

Months	Number of work days		Production (tons)	
(Year 2000)	B Field	C Field	B Field	C Field
January	15	31	59381 00	184523 00
February	29	27	201098 00	247498 00
March	28	27	175821 00	254480 00
April	30	26	162797 00	119244 00
May	29	24	125404 00	67115 00
June	30	26	80682 00	49277 00
July	31	25	169889 00	183301 00
August	31	26	143443 00	228904 00
September	30	26	117970 00	18884900
October	26	26	135666 00	229827 00
November	26	26	13541100	251425 00
December	25	25	142518 00	272224 00

9 CONCLUSIONS

From a general point of view, it should be noted that single slicing is superior to twin-face production in three aspects.

I) When compared to twin faces, some operations underground have automatically been discarded İn single slicing, for example, the removal of supports and stowing material between upper and lower faces becomes unnecessary. In addition, the drawbacks of excessive closure both in main- and tailgates, observed especially between the upper and lower faces, are almost eliminated. In other words, less labor and lower production cost figures are achieved.

2) The immediate roof overlying the Çayırhan lignites mainly consists of strong claystone, whereas the interburden composition is a relatively weak claystone-marl layer. This may cause problems in roof support in the gateroads of the lower face. Consequently, the advance rate and coal production are higher with single slicing than with twin faces.

3) It is clear that the quantity of equipment used in single slicing is low and all the mining operations are concentrated at a single face. Therefore, there is significantly better supervision in such panels.

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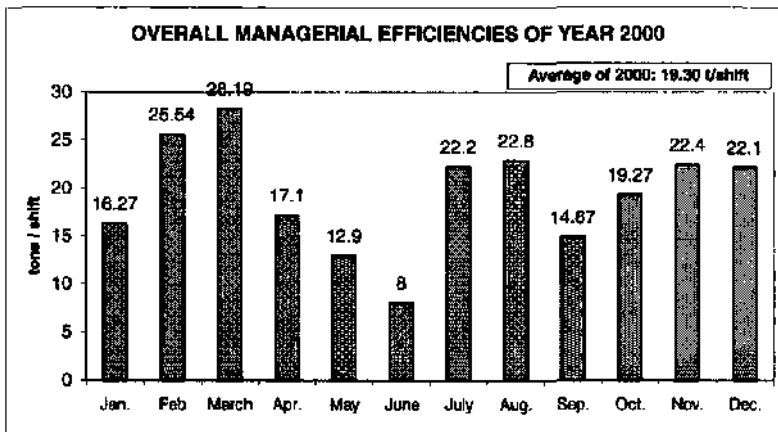Walker, S., 1999. Çayırhan challenge. *World Mining Equipment,* issue of December, pp. 12-13.

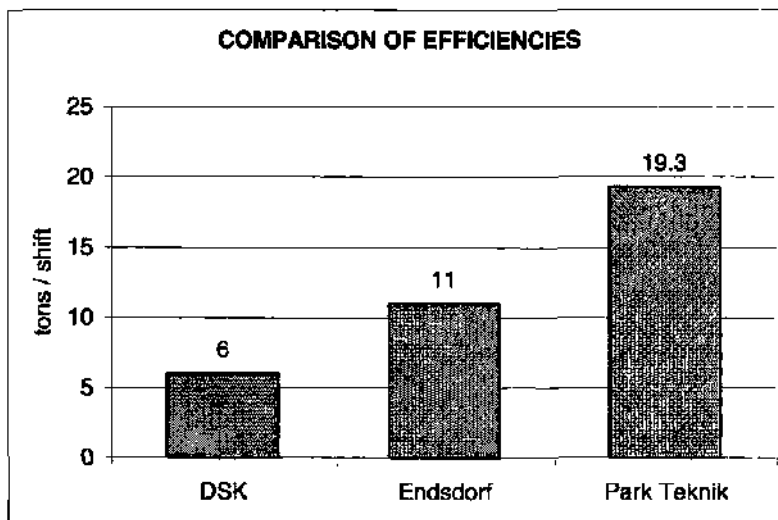Figure 8. Overall managerial efficiencies of year 2000.

Figure 9.Comparison of average efficiencies in 2000.

Local Sustainable Development in Districts with Current Exploitation of Lignite Fields

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ABSTRACT: This paper deals with the expected impact of investing the recently established Compensation for Lignite Fields Exploitation (CLFE) on the development of counties in Greece where lignite mines are in operation, with emphasis on sustainability, in the sense of steady-state growth in the long term. Two models are considered, both with the regional Gross Domestic Product (GDP) Y_t as a dependent variable. After proving that these models may reduce to a unique model in the form of a complete second order difference equation, we use numerical data available for the three counties where lignite is produced to obtain the time path of Y_t. Quantitative analysis of the results shows that the impact of investing the CLFE on Y_t is almost negligible as a macroeconomic magnitude, and a multi-dimensional (economic, social, environmental, hydrological) integrated approach should be adopted in order to influence sustainable development.

1 INTRODUCTION

'Sustainable development* İs development that meets the needs of the present without compromising the ability of future generations to meet their own needs. The term embodies two key concepts (World Commission on Environment and Development, 1987, often referred to as the 'Brundland Report*): (1) the concept of 'needs', to which overriding priority should be given, and (2) the idea of limitations imposed by the state of technology and social organization on the natural resources availability and the capacity of the environment to meet present and future needs. For large-scale projects and activities, like the exploitation of a lignite field for electricity production, it would be appropriate to consider the anticipated impacts in terms of their implications for sustainable development.

A necessary condition for sustainable development at local level is the continuation of operation of the main network of activities in the region under consideration after the cessation of operation of an industrial complex or a mine of major influence. In cases where the impact of such a cessation on the rest of local activities is decisive, leading to discontinuity of economic/social progress, a vicious circle may become established: increase in unemployment —* decrease in demand for goods and services —* decrease in new capital supply for investment —* • decrease in productivity —* • decrease in sales —* decrease İn production —> increase İn unemployment —*...; the same process can become established at a regional or national level, with several localities suffering the same problem.

Evidently, a breakdown of this vicious circle can be brought about only via exogenous action, mainly in the form of proper investment to activate local resources. For example, the German coal industry had been in decline since the second world war and when cheap foreign coal became available, many mmes closed down; since most of the industrial enterprises İn the Ruhr area were dependent on or connected with coal mining, the crisis extended to several industrial units which closed, leaving abandoned and derelict sites. In order to stop the catastrophic decline of the industrial sector in this region, the government decided in the early 1980s to stimulate the development of new industrial units, in part with public funding, on condition that they were established on derelict land left by the demise of the coal mines and associated industries. This example and some other similar cases have been analysed by Bell & Genske (2000) and Bell et al. (2000), who showed that the rehabilitation of abandoned sites may require significant investment, even though most of these sites have immediate access to pre-existing infrastructure.
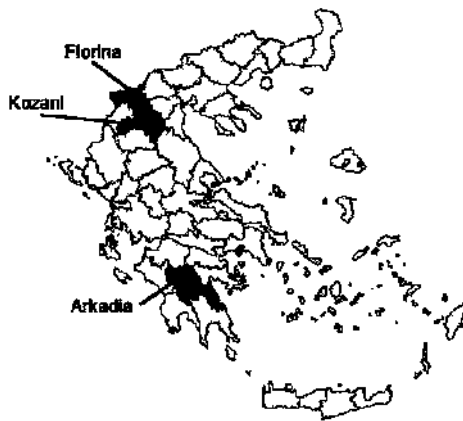
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Figure 1. The three counties with lignite fields under exploitation

In Greece, there are lignite fields currently undergoing exploitation by the Public Power Corporation (PPC) in three counties; Arkadia, Florina and Kozanı (see map in Figure 1). The PPC contributes to sustainability at local level by providing (i) technical services and construction equipment for small works within nearby communities, (ii) thermal energy to the small cities of. Kozani and Ptolemais, and (iii) economic resources for archaeological excavation in the vicinity of mines so that ancient ruins or items may be unearthed and their loss prevented, (iv) advice to local authorities on methods of environmental protection. Moreover, the Greek Parliament issued Law 2446 in December 1996, which states in article 20 that the PPC is obliged to give 0.4% of its gross income annually (named herein, for convenience, Compensation for Lignite Fields Exploitation - CLFE) for the development and protection of the environment İn these three counties.

This fund is allocated in accordance with the proportion of electricity production based on lignite supplied by the thermal/electrical power stations of each county. A ministerial order issued on 22.07.97 gave guidelines for the choice of sectors where money coming from this fund should be invested: the development of each county according to its comparative advantages, creation of new jobs, vocational training of the unemployed, upgrading of human specialization by providing the means for development of new skills, increase in competitiveness of the primary, secondary, and tertiary production sector, development of local infrastructure, and conservation of the environment. The committee appointed by the Minister of Development to manage this fund consists of representatives of various chambers (economical, technical, industrial and commercial, geotechnical), ministries and local authorities. The characteristics of the members of this committee in combination with the guidelines of the above-mentioned order give the impression that emphasis is put on regional development rather than on environmental management. This impression led the Ecological Movement of Kozani to express the view that CLFE, which had been a demand initially for environmental concern of the people of Kozam for 10 years and was established finally by the state as a form of subsidy for investment, was intended to contribute to local development.

The aim of the present study is to examine the macroeconomic consequences of investment based on the CLFE fund, regardless of the kind or the purpose of the physical capital purchased with money from this fund. More specifically, we concentrate on the expected changes in regional Gross Domestic Product (GDP) to determine whether it is worthwhile investing through CLFE in order to increase this macroeconomic magnitude, or it is better to make another choice that will enhance sustainability in the long run.

2 A SIMPLE MODEL

To investigate the economic impact of investing CLFE in a region with lignite mmes in operation, we use the so-called *acceleration principle* (Samuelson, 1939), according to which the induced private investment I, in any period t is proportional to the increase in consumption Q of that period over the preceding C_t.i. In this relation, we introduce an amount of exogenous investment Hi equal to CLFE:

$$I_t = \beta [C_t - C_{t-1}] + H_t \qquad (1)$$

The regional GDP Y, of the district where CLFE is applied is given by the following definition, which is widely accepted in macro-economic theory and practice:

$$Y_t = C_t + I_t + G_t \qquad (2)$$

where the governmental expenditure G, is exogenously determined and the consumption expenditure Ct, in any period t, is proportional to the regional GDP of the preceding period:

$$C_t = \alpha Y_{t-1} \qquad (3)$$

The coefficient a of proportionality in Equation 3 is called the *marginal propensity to consume* and represents the proportion of a small increase in regional income which will be spent by the inhabitants to cover their needs. According to the "fundamental psychological law" of J.M. Keynes, the coefficient a falls when regional income rises, i.e.,

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the richer the people in a region, the greater the proportion of increase in their income which would be saved.

By substituting Equations 3 and 2 in 1, we obtain:

$$Y_t = \alpha (1 + \beta) Y_{t-1} - \alpha \beta Y_{t-2} + G_t + H_t \qquad (4)$$

Inasmuch as the increase in demand for electric power is negligible and no new power plants based on lignite are introduced into the national system of electricity production, there İs a constant rate of lignite mine exploitation. Consequently. H, is independent of time t. Under steady state conditions (implying also Gi-const.), the complete second order difference Equation 4 is rearranged to give:

$$Y^2 - a(1 + \beta)Y, -, + a\beta Y, = G + H \qquad (5)$$

The form of the complete solution of this equation depends on the type of roots of its homogeneous part. For real and unequal roots:

$$Y_t = C_1 m_1^k + C_2 m_2^k + \frac{G+H}{(1-\alpha)}$$

For real but equal roots:

$$Y_t = (C_1 + C_2 k)m_{12}^k + \frac{G+H}{(1-\alpha)}$$

For complex conjugate roots:

$$Y_t = Ar^k \cos(k\Theta + B) + \frac{G+H}{(1-\alpha)}$$

where mi,m.2 are the real roots of the auxiliary equation irf - ct (I + ß) m + a ß = 0, when its discriminant is positive; r,0 are given hy the complex conjugate roots of the auxiliary equation (when its discriminant is negative) with polar forms: r (cos© ± i-sin©); mi.$_2$ is the double real root of the auxiliary equation when its discriminant is zero; C|, C$_2$, A, B are constants.

As the auxiliary equation is of the form m~ + aim + ci2 = 0, with ai = -a (1 + ß) and a$_2$ = aß, there is a necessary and sufficient condition for the solution of the homogeneous part of Equation 5 to converge to zero, independently of the initial values of the regional income Y$_0$ and Y|.This condition requires that both the roots of the auxiliary equation be less than I in absolute value. If the complete form of Equation 5 has a constant value as solution, then putting Yif=Y =const., we obtain,

$$Y^* = \frac{G+H}{(1+a_1+a_2)}$$

which is an equilibrium or stationary value of Y. A necessary and sufficient condition for this equilibrium value to be stable is p<1, where p=max { [mi I, Im? I} and m₁, m$_2$ are the roots (either real or complex) of the auxiliary equation. The conditions for these restrictions to be valid are:

$$\begin{vmatrix} 1 + \alpha_1 + \alpha_2 > 0 \\ 1 - \alpha_1 + \alpha_2 > 0 \\ 1 - \alpha_2 > 0 \end{vmatrix} \quad \text{or} \quad \begin{vmatrix} 1 - \alpha (1 + \beta) + \alpha\beta > 0 \\ 1 + a (1 + \beta) + \alpha\beta > 0 \\ 1 - \alpha\beta > 0 \end{vmatrix}$$

which usually hold as a, $\beta > 0$, $\alpha < 1$, $\alpha\beta < 1$, in most cases of economic reality.

The stable solution $Y^* = \dfrac{G+H}{(1+a_1+a_2)} = \dfrac{G+H}{(1-\alpha)}$ is

identified as the economic 'multiplier' because of the

property to increase the regional GDP Y by $\dfrac{G+H}{1-\alpha}$

times as a result of the investment G + H.

However, if conditions of full employment prevail in the local labour market, this multiplier is unlikely to increase the regional income to such an extent; in this case, the demand for services/goods related to investment will cause some rising of prices which will not permit real income to reach the level estimated by means of the multiplier action. The labour markets of the three districts under consideration (Kozani, Arkadia, Fiorina) where lignite mines are in operation do not give evidence of near-full employment; nevertheless, the difference of the levels of employment among them introduces an element of discrimination as regards the consequences of exogenous investment: the lower the level of employment, the more beneficial the multiplier action in regional GDP.

3 AN EXTENDED MODEL

The multiplier-accelerator model, which was used above to investigate the economic impact of investing CLFE in a region with lignite mines in operation, is a relatively simple one. Further analysis is required by means of more sophisticated modelling. This can be done by incorporating CLFE into a dynamic model where the income and capital stock of one period determine investment and consumption for the next period, and the changes İn these last economic magnitudes determine the former ones for the following period, and so on. As a basis for such interaction over time, we use Duesenberry's model (Duesenberry. 1958), where we introduce the governmental expenditure G, and the exogenous investment H$_t$ in the definition of Y$_t$, as follows:

$$Y_t = C_t + I_t + G_t + H_t, \quad I_t = I_{bt} + I_{ht}$$

where Ibt and IM stand for business and housing investment, respectively; the variables on the right-band side of these definitions are given by the following expressions:

$$C_t = f_1 (Y_{dt-i}, C_{t-1}) \tag{6}$$

$$I_{bt} = f_2 (Y_{t-1}, K_{bt-1}, E_{t-1}, P_{t-1}, D_{t-1}, R_{t-1}) \tag{7}$$

$$I_{ht} = f_3 (Y_{t-1}, K_{ht-1}) \tag{8}$$

$$P_t = f_4 (Y_t, K_{bt-1}) \tag{9}$$

$$d_t = f_5(P_{t-1}, d_{t-1}) \tag{10}$$

where R=capital consumption allowances;
  Kb=business capital stock;
  Kh=stock of houses;
  P=profits (including those undistributed business and farms);
  d=dividents and entrepreneurial withdrawals;
  E=retained earnings of business;
  D=business debt;
  Yd=disposable regional income available to individuals, which is given by the following definitional relationship:

$$Y_{dt} = Y_t - P_t + d_t - R_{bt} - R_{ht}$$

Capital consumption allowances for business and housing are proportional to the stock of capital in the corresponding sectors:

$$R_{bt} = x K_{bt}, \quad R_{ht} = y K_{ht} \tag{12}$$

where the capital stock is given by the following isomorphic relations give, which are simple balance identities:

$$K_{bt} = K_{bt-1} + I_{bt} - R_{bt}, \quad K_{ht} = K_{ht-1} + I_{ht} - R_{ht} \tag{13}$$

By definition, we also have:

$$D_t = D_{t-1} + I_{bt} - R_t - E_t, \quad E_t = P_t - d_t \tag{14}$$

i.e., business debt is the algebraic sum of one-period-lagged debt, business investment, capital consumption allowances (depreciation), and retained earnings (profits minus dividends).

To be in agreement with Duesenbeny's modelling: (i) we disregard the housing equations so as to let the system of equations deal only with business investment, as this influence prevails; (ÍÍ) we eliminate lagged consumption from the consumption function (Eq. 6) and lagged dividends from the dividend Equation 10; and (iii). we make investment depend on profits and profits on capital stock at the end of any period rather man at the start,

implying $1^\wedge = x$ KM only for the expressions that give It and Ki (i.e., Equations 7a and 13a, respectively) but not Ci. Under these simplifying assumptions, die corresponding equations are rewritten as follows:

$$C_t = f_1 (Y_{t-1} + d_t - P_{t-1} - R_{t-1}) \tag{6a}$$

$$I_t = f_2 (Y_{t-1}, K_{t-1}, P_{t-1}, R_t) \tag{7a}$$

$$P_t = f_4 (Y_t, K_t) \tag{9a}$$

$$d_t = f_5 (P_{t-1}) \tag{10a}$$

$$K_t = K_{t-1} + I_t - R_t \tag{13a}$$

By substitution, we obtain the following incomplete system of equations:

$$C_t = f_1 [(Y_{t-1} - f_4 (Y_{t-1}, K_{t-1}) - x K_{t-1}), \quad f_5 (f_4(Y_{t-1}, K_{t-1}))] \tag{6b}$$

$$I_t = f_2 [(Y_{t-1}, K_{t-1}, f_4 (Y_{t-1}, K_{t-1}), x K_{t-1}] \tag{7b}$$

$$K_t = K_{t-1} + I_t - x K_{t-1} \tag{13b}$$

By expressing diese equations in linear form and taking into account the basic definition of regional GDP Y,, we obtain die complete system:

$$Y_t = C_t + I_t + G_t + H_t \qquad C_t = \varepsilon Y_{t-1} + \zeta K_{t-1}$$
$$I_t = \gamma Y_{t-1} + \delta K_{t-1} \qquad K_t = (1 - x) K_{t-1} + I_t$$

By substitution, we obtain the relations for regional GDP Y, and capital K,, under constant H, G:

$$Y_t = (\gamma + \varepsilon) Y_{t-1} + (\delta + \zeta) K_{t-1} + H + G \tag{15}$$

$$K_t = \gamma Y_{t-1} + [\delta + (1 - x)] K_{t-1} \tag{16}$$

By eliiiiinating KM from these equations, we obtain:

$$(\delta + \zeta) K_t = (\delta + 1 - x) Y_t + [\gamma (\delta + \zeta) - (\gamma + \varepsilon) \tag{17}$$
$$(\delta + 1 - x)] Y_{t-1} - [\delta + (1 - x)] (G + H)$$

In order to deduce a difference equation as regards tiie regional GDP, we combine Equations 15 and !7 to eliminate K,. The final result is:

$$Y_t - [(\gamma + \varepsilon) + (\delta + 1 - x)] Y_{t-1} - [\gamma (\delta + \zeta) - \tag{18}$$
$$(\gamma + \varepsilon) (\delta + 1 - x)] Y_{t-2} = (x - \delta) (H + G)$$

This is a rather complete dynamic model of regional GDP, according to Duesenberry's reasoning, where we have incorporated I, and G, to account for exogenous investment coming from the PPC to offset lignite mines operation and from die state as a contribution to regional development. By adopting

the simplifying assumptions made by Kooros (1965), i.e., very low depreciation and negligible connection between consumption and capital stock (implying $x=0$ and $\zeta=0$, respectively) and $\delta = -1$, we obtain the deduced expression:

$$Y_t - (\gamma + \varepsilon) Y_{t-1} + \gamma Y_{t-2} = G + H \qquad (19)$$

which is similar in form to Equation 5 and identical to this equation in the special case that e = a and $y = a\ \beta$. Therefore, Equation 5 can be used to investigate the influence of the investment H, which equals tie share of the lignite tax given to each lignite-producing region. We should also bear in mind that we can extend mis model to include governmental expenditure G given for regional development as well as other macroeconomic magnitudes of local interest.

4 IMPLEMENTATION AND RESULTS

To investigate the expected impact of investing CLFE on regional GDP over time, we apply Equation 5 twice for the lignite-producing counties of Arkadia, Kozani, Fiorina: the first time by including CLFE, and the second time by excluding it Then, by subtracting, we obtain the difference ΔY_t in monetary value, which represents the expected impact. The results are given in Figure 2. The initial values Y_0 and Y_i are taken from the regional statistical tables for 1995 and 1996, respectively. Therefore, the first year for which GDP is estimated is 1997. All monetary values have been reduced to this year, which was also the latest year for which analytical statistical data at regional level were available by 31-12-2000.

It is worthwhile noting that ΔY_t exhibits a sinusoidal form, as it is the difference of two complete second order difference equations with constant coefficients and the same auxiliary equation which has a negative discriminant. The necessary and sufficient conditions for stability are fulfilled, since a, $\beta > 0$, $a < 1$, $\alpha\beta < 1$. In practice, all investment, even for specific large works, is distributed within the corresponding year, while there is a tendency for the annually invested capital to increase; mis implies a smoothing effect on Y_t, and consequently on ΔY_t, over time, resulting in a corresponding curve with negligible oscillation and usually a positive slope. The smoothing effect may be further enhanced if empirical evidence suggests a time period shorter than one year, in which case equilibrium is achieved earlier. Figures 3-6 present 2D and 3D graphs of one-parameter and two-parameter sensitivity analysis, either over time or in cross-section, i.e., for the same year (here, 2005 is chosen as an example).
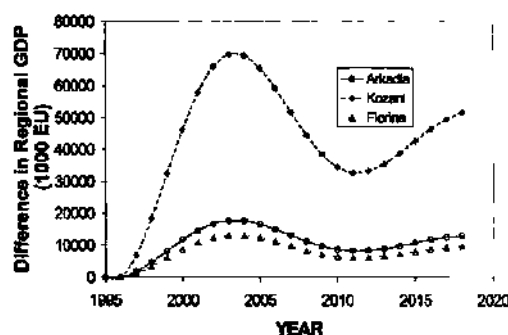

Figure 2 Expected impact of investing CLFE over a long time period (a=0.85, 0=1.00).
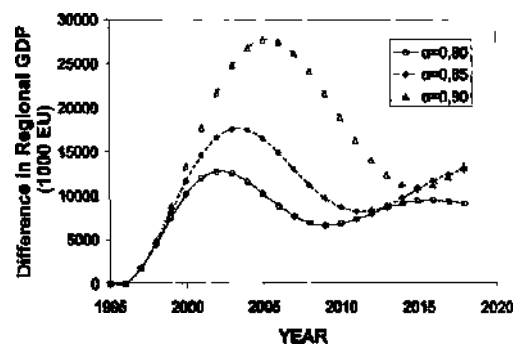

Figure 3. Dependence of expected impact of investing CLFE on the marginal propensity to consume a (ß=1 00)
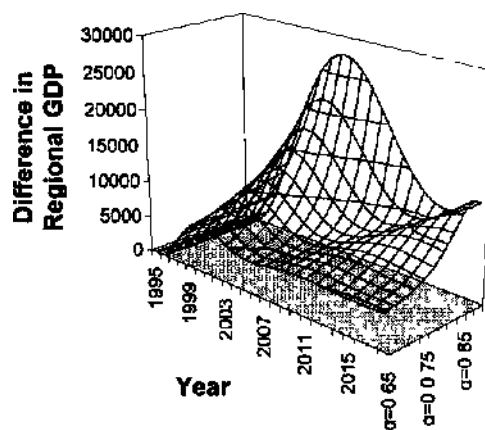

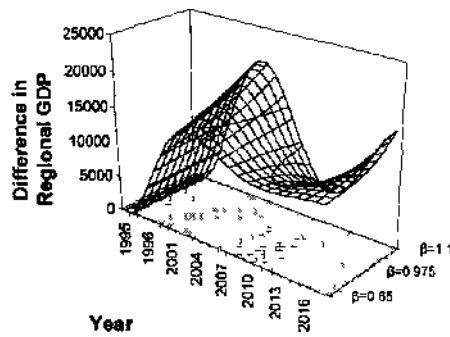Figure 4. One-parameter sensitivity analysis of expected impact, with varying a-values over time

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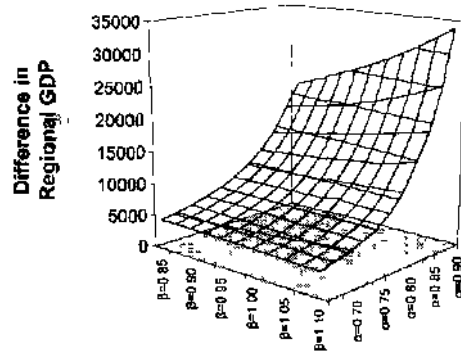Figure 5. One-parameter sensitivity analysis of expected impact, with varying ß-values over lime

Figure 6 Two-parameter cross-sec non (year 2005) sensitivity analysis of expected impact, with varying values of both a & ß

5 DISCUSSION

The first point that should be discussed is the scope for establishing the CLFE. Is it a tax to finance regional development or an environmental policy? If both of these apply, is there a rule for sharing the fund? If the committee decides without a consistent rule, what are the criteria for such decision making? Finally, how are all these issues connected with sustainability at local level?

bHhe introduction section, we mentioned a doubt expressed by ecologists about the real purpose of the fund. There are two reasons that at least partially justify this doubt: (i) the financial support of social activities that have nothing to do with investment and (ii) the use of some part of the fund for land reclamation, although all restoration works should be included in the PPC's liability to return the mined land in a condition suitable for rehabilitation. The existence of this last reason is due to the fact that the PPC, when negotiating with the state's licensing authorities, does not submit a detailed long-term plan with a specific reclamation/recultivation schedule, in

contrast to what happens in other lignite/coal-producing countries of the EU. For example, Rheinbraun, the sister company of RWE Energy, Germany's biggest power utility, spent a decade designing the Garzweiler II mine extension in cooperation with local authorities. According to this design, water, agricultural and forestry management plans were included for as far ahead as 2080, with detailed specifications aimed to balance the interests of agriculture and local recreation, while also ensuring that wildlife displaced by lignite mining could be re-established (Ballay 1996). It seems that this message has been received by PPC, as this corporation recently submitted a very detailed plan to obtain the license for the new mine at Mavropighi (which means "black spring") in the Lignite Centre of Prolemais-Amyndeon - LCPA.

Another point that deserves attention is the distribution of CLFE among the lignite-producing counties. The criterion of contribution to electricity production introduces a measure of participation in the product while the philosophy of CLFE should be to offset land loss and environmental deterioration, especially in the vicinity of lignite-fired power plants, due to fly ash in the atmosphere and polycyclic aromatic hydrocarbons together with heavy metals in the surface soils (Batzias & Roumpos, 2000a; Stalikas et al., 1997). If the offset criterion is adopted, CLFE can be distributed according to lignite production. In such a case, Arkadia will enjoy a better share as it has the poorest lignite (Table 1).

Table 1. Production data for the three counties under consideration (Source[1] PPC)

Electric Power Production from Lignite			
GWh			
	1996	1997	1998	1999
Florina	4018	3480	3445	3911
Kozani	17606	18623	20930	20021
Arkadia	4629	4699	5315	5678
total	26253	26802	29690	29610

Lignite			
million tonnes			
	1996	1997	1998	1999
Florina	7.84	6.79	7.2	8.28
Kozani	36.48	38.08	39.45	39.37
Arkadia	12.61	11.52	12.06	13.3
total	56.93	56.39	58.71	60.95

Electric Power Production / lignite			
GWh / million tonnes			
	1996	1997	1998	1999
Florina	513	513	478	472
Kozani	483	489	531	509
Arkadia	367	408	441	427

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As regards the impact of investing the CLFE, the results shown in Table 2 prove that the difference in regional GDP AY(due to this investment for the three counties under consideration is almost negligible. Actually, it ranges between 0.9% (valid for the optimistic scenario for Arkadia) and 2.20% (valid for the pessimistic scenario for Kozani) or 3.26% (valid for an estimation of regional GDP based on Equation 5, with ct=0.85 and ß=1.00, for Kozani).

Consequently, the amount given as CLFE must increase substantially if our intention is to significantly influence regional GDP, which seems to follow a time path closer to a rather pessimistic scenario (Fig. 7). Probably, such a favourable influence might be realised with an integrated multi-dimensional approach, rather than with a pure economic approach. Bellmann (2000) has shown the advantages of multi-dimensional modelling for integration of ecological, hydrological, economical and social components of regions disturbed due to extended surface coal mining. The same methodology can be followed when mining is still in progress, as modern techniques allow for some restoration works while mining is ongoing, often up to 150 feet from the edge of the mine (Phillips, 1993). It has been proved (Batzias & Roumpos, 2001) that under certain conditions parallel restoring is more beneficial than serial restoring (where all mining works must be finished before the restoration process begins).

These multi-dimensional integrated solutions enhance local sustainability, as they contribute to long-term steady-state development. In addition, it is worthwhile noting that by putting emphasis on the environmental dimension, we can go beyond the pure economic magnitudes as criteria for assessing the value of a lignite mine. As it has recently been shown (Batzias & Roumpos, 2000b), the environmental dimension can be used as a basic factor for choosing the optimal lignite field for exploitation, by means of multiple criteria analysis. This technique also allows other factors/criteria (like agricultural land loss, creation of new jobs, redistribution of national income for regional development, know-how and technology diffusion) to play a significant role in multi-dimensional decision making, thus contributing to local sustainable development.
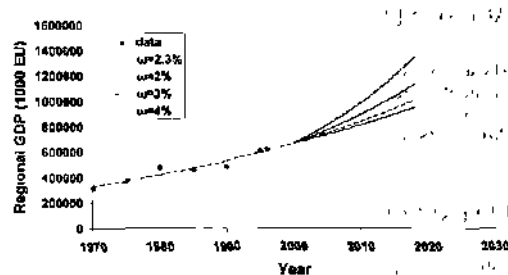

Figure 7. Forecasting the GDP for Arkadia, under ati optimistic, a moderate and a pessimistic scenario. The dashed line gives a • non-linear projection based on data for 1970-1996. •

6 CONCLUSIONS

To investigate the expected impact of investing.thç. recently established Compensation for Lignite Fields Exploitation (CLFE) on the development of the counties where lignite mines are in operation, we can apply a multiplier-accelerator model twice in the form of a complete second order difference equation. The results obtained show mat the impact, as a macroeconomic magnitude, is almost negligible. Consequently, the amount given as CLFE to the local management committee must be increased substantially, while any kind of financial support coming from the CLFE should be directed towards investment opportunities that enhance sustainability in each county. On the other hand, the Public Power Corporation (PPC) should undertake specific liability to return the mined land in a condition suitable for rehabilitation, according to an a priori determined schedule, regardless of the way the CLFE capital is invested in the time period.

This clarification will permit the local committee to orientate and coordinate its activities properly in the long run, without being obliged to invest in land restoration. The PPC's restoration plan will be considered given and will form the main corpus for sustainable development. These specifications seem to have been adopted by the PPC, as similar reasoning is contained in the detailed plan that this corporation submitted recently to obtain the licence for the new mine at Mavropighi in the Lignite Centre of Ptolemais - Amyndeon (LCPA).

It is also suggested that the distribution of CLFE among lignite-producing counties according to the criterion of contribution to electric-power generation İs not compatible with the target of sustainable development. Instead, the distribution in proportion to lignite production strengthens sustainability at local level. By extending this reasoning, we propose a multi-dimensional (economic, social, environmental, hydrological) integrated approach, in order to stabilize development in the long run.

Table 2. Regional GDP Y, and percentage increase AY, in ihe year 2015, under an optimistic, a moderate, and a pessimistic

County:	Arka d i a		Kozani		Fiorina	
Parameter	Y, 10^6EU	AY, %	Y, 10" EU	AY, %	Y, 10* EU	AY, %
a=0.85	1009.1	1.07	1308.9	3.26	551.78	1.45
<o=2%	897.3	1.20	1942.7	2.20	459 96	1.74
(0=3%	1038.7	1.04	2248.9	1.90	532.45	1.50
o>=4%	1200.7	0.90	2599.6	1.64	615.49	1.30
Projection	944.1	1.14	-	-	-	-

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Technical Efficiency Measurement and Occupational Safety Level. The Case of Greek Lignite Mining

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ABSTRACT: The purpose of this paper is to measure technical efficiency in Greek lignite mining using not only conventional input-output data, but also mine accident data over the period 1970-96. Technical efficiency is measured using the basic Data Envelopment Analysis (DEA) model, which takes into account conventional data (i.e., real output, labor and fixed capital) and the number of disabling injuries. The latter are treated as input and alternatively as a joint 'negative' output in alternative applications of the basic DEA model. In the light of the combined results, the treatment of disabling injuries as either input or output provides fairly similar results (i.e., for one and a half decades, the Hgnite production system at the sectoral level operated efficiently, reaching the <u>maximum</u> potential output level only over the period 1989-91 and in 1996).

1 INTRODUCTION

1.1 *Technical efficiency measurement - Data Envelopment Analysis*

Work that has been done on measuring efficiency can be categorized into techniques that use a parametric or stochastic frontier production approach and those that use a non-parametric or linear programming approach (Forsund et al., 1980). Data Envelopment Analysis (DEA) is a modern technique of operations research which is used for efficiency measurement and belongs to the non-parametric approach.

The measuring of efficiency using a non-parametric approach began essentially with the pioneering work of Farrell (1957). Then, Chames et al. (1978) revived Farrell's efficiency measure as a technique and coined the term DEA. The new technique was based on a linear programming problem for measuring the efficiency of decision management umts (DMUs). DMUs are different production units which operate under similar conditions, employing the same inputs and producing the same outputs.

Most studies so far have used cross-section data evaluating the performance of various production units, but it should be noted that Burley (1980) used the technique for measuring efficiency at the sectoral level. DEA was applied İn the <u>mining</u> industry by Byrnes et al. (1984) to a sample of Illinois strip mines.

The present paper uses time series of data for measuring the technical efficiency in Greek lignite mining over a long period. The term technical or productive efficiency is used mainly by economists in order to describe how well an organizational unit is performing in utilizing resources to generate outputs or outcomes. In our case, DEA sheds light on the composition of productivity differential among compared yearly activities, using not only conventional input-output data (Burley, 1980; Tsolas, 1995a; Tsolas, 1995b; Tsolas & Panagopoulos, 1996), but also aggregate mine accident data over the period 1970-96. Technical efficiency İs measured using a DEA model which takes into account, in addition to real output (i.e., excavated lignite) and labor and capital inputs (i.e., man-shift hours paid and fixed capital respectively), the number of disabling injuries. The latter are treated as input and alternatively as a joint 'negative' output in different input-output DEA models. The treatment of mine accident data is similar to that of the environmental data in DEA applications (Tsolas, 1996).

DEA is based on mathematical programming principles and provides the technical efficiency scores of each one of the yearly activities over the period 1970-96. Activities are deemed efficient only if technical efficiency scores are equal to unity; the difference between unity and technical efficiency score yields the percentage of potential output loss due to inefficiency.

1.2 Occupational safety

Unlike technical efficiency, safety does not readily lend itself to definition in a quantitative manner; rather it is a qualitative, judgmental factor related to the acceptability of risks (Zabetakis, 1981).

Various studies on productivity and safety (Zabetakis, 1981; Tsolas & Panagopoulos, 1995) are based on the assumption that there İs *a priori* a negative correlation between productivity and safety, and present some empirical evidence at the sectoral and mine level respectively through the derivation of regression models. According to the results of a recent study (Tsolas & Petrakis, 2000), the above assumption seems to be unrealistic in the case of Greek lignite mining as far as the disabling injuries are concerned. In the light of these results, disabling injuries are treated as input and alternatively as a joint 'negative' output in different input-output DEA models.

The study aims to present some evidence regarding productive efficiency in Greek lignite mining in relation to occupational safety level For this, an extended input-output data set is used in which, apart from real output, labor and capital, disabling accidents are also included.

2 METHODOLOGICAL FRAMEWORK

2.1 Occupational safely as a joint 'negative ' output

Work-related accidents are considered as a joint 'negative' output of the production process. Given a set of n inputs, say $X = (X_1 , X_2 , ..., X_n)$, firms are considered to be able to produce along a product transformation frontier marketable output (Q) and accident output (A) combinations. Using a dual-output production frontier of the form F(Q, A, X) = 0, a product transformation curve, like the one in Figure 1, in which input levels are assumed to be constant, summarizes the transformation possibilities of the firms; in other words, the trade-off between occupational safety and marketable output Movements along this curve are achieved by reallocating inputs from output-producing to accident-reducing activity. The slope of this curve is positive because:

$$dQ/dA = -(\partial Q/\partial X)/(\partial A/\partial X) \qquad (1)$$

where dQ/dX >0 because the production of marketable output Q requires input X and $\partial A/\partial \lambda$ < 0 because the reduction of accidents requires input λ.

This curve defines a positive opportunit; cost for reducing accidents.

The discussion above i based on the assumption that firms can alter the incidence of accidents and hi

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faced with a technological trade-off between safety and marketable output. If changes m accidents are instead random, then il would be expected that a negative relationship between accidents and output would be observed. Tn this case, a reduction of accidents would be beneficial for the firm in terms of fewer disruptions to production, and thus increased productivity.
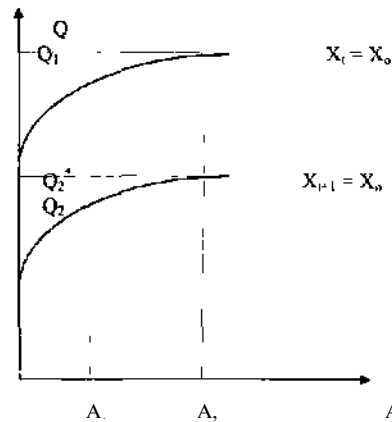

Figure 1 Product transformation fronuer, see also Sider (1983)

2.2 Technical efficiency measurement using DEA

Technical efficiency is obtained from the solution of the following problem, which is known as the basic DEA model (Chaînes et al., 197S):

Given a set of n yearly activities (YA) YAj (j=1,2,...,n), each with a set of m inputs x_{ij} (i=i,2,. .,m) and a set of M outputs ykj (k=1,2,...,M), determine for one particular YA, say YAp, with inputs x_{ip} and outputs yk, whether it is efficient; in other words, whether w equals one, which stems from the following linear programming problem under the assumption of constant returns to scale:

MUX \V subjretto

$$\sum_{j=1}^{n} x_{ij} \lambda_j \le x_{ip}, \qquad i = 1,...,m$$

$$\sum_{j=1}^{n} y_k \lambda_j \le y_{kp} w. \quad k = 1,.. , M$$

$$w, \lambda_j \ge 0$$

Activities are deemed efficient (i e.. DEA scores = 1.00) if they are not dominated by any other pure activity or a.iy of their subset

3 DATA SET

Real output, labor input, capital input and occupational mine data are considered here. The data are drawn from the National Statistical Service of Greece and the Ministry for Development, Directorate of Policy for Mineral Resources (see also Tsolas & Petrakis, 2000).

3.1 Real output

Real output is measured in terms of excavated lignite tonnage. An unadjusted measure of physical output is more accurate for a homogenous product (Darmstadter, 1997). In the case of Greek lignite, a decrease in the heating content of excavated lignite implies a downward adjustment in the level and rate of change in productivity and vice versa, but limitation of the data restricts the analysis to that direction.

3.2 Labor input

Labor input is measured in terms of aggregate (white and blue collar) total man-shift hours paid (including those for illnesses, accidents and days off).

3.3 Capital input

In order to keep track of changes in capital equipment, a perpetual inventory method is used. Thus, whenever there is a new investment in equipment, it is added to the capital stock and remains there until it is declared 'retired' from assets. The average useful service life is taken to be equal to 20 years and is used as the basis for this 'retirement'.

Capital input is the physical use of machinery and equipment, with depreciation taken as an approximation of the capital consumed in the production process.

The model applied here for the estimation of fixed capital is a 'service flow model' because physical inputs are converted into drachmas that are payments for services provided by capital inputs. For tfie conversion in constant 1970 drachmas, a deflator constructed by the Ministry of National Economy's services (Ministry of National Economy, 1998) was used (Tsolas, 2000). This model views productivity as the measure of the efficiency of the conversion process (Green & Green, 1985; Tsolas & Petrakis, 2000).

3.4 Occupational mine data

The occupational mine data include disabling injuries over the period 1970-96.

4 INPUT-OUTPUT MODEL FORMULATION

Following the analysis presented above, three input-output (I/O) models and respectively three DEA models are considered.

i) Model 1 uses excavated lignite as output, and man-shift hours paid, fixed capital and disabling injuries *as inputs*.

ii) Model 2 uses excavated lignite and disabling injuries as outputs, and man-shift hours paid and fixed capital as inputs.

iii) Model 3 uses excavated lignite as output, and man-shift hours paid and fixed capital as inputs.

The occupational mine data are incorporated into model I as input under the assumption that they represent the inflated monetary values of an extra cost and alternatively, into model 2 as "negative" output (reciprocal values are entered into the model).

In the first model, they are treated as an extra cost which reflects the cost of improving working conditions, workforce hygiene, etc. The application of DEA therefore uses the disabling injuries as input and discriminates the activities using lower cost to produce a particular yearly output level.

In the second model, they are treated as a 'negative' output; according to DEA, the more the outputs (more lignite produced and less injuries occurred), the better the efficiency.

In the third model, only conventional input-output data are used.

5 RESULTS

The estimated technical efficiency scores for each one of the input-output models are presented in Table 1. Spearman rank order correlation coefficients are presented İn Table 2.

On the whole, the DEA scores show among others that the yearly activities over the period 1989-9! and in the year 1996 had relatively high efficiency. Therefore, the performance of the sector at the end of the 90s and in the final year of the period studied can be considered satisfactory.

Both input-output models I and *3* give fairly similar results. This is due to the fact that there is a positive correlation between disabling injuries and production (Tsolas & Petrakis, 2000), and as a result, the agreement between the rankings in models I (conventional data) and 3 (conventional and occupational data) is very satisfactory.

Model 2, in which occupational data are treated as negative output, was used as an alternative to model 1. A comparison of these results with those of model 3 shows that the treatment of disabling injuries as a negative output provides similar results.

For a discussion of the results, the risk level of Greek lignite mining and its effect on total factor productivity (TFP) measurement should be taken into account.

Table 1. PEA scores.

	Model 1	Model 2	Model 3
1970	86,90	100.00	82.51
1971	100.00	100.00	100.00
1972	99.82	97.75	95.49
1973	94,25	93.97	90.06
1974	83.73	85.03	83.73
1975	94.15	95.90	93.90
1976	100.00	100.00	100.00
1977	100.00	100.00	97.80
1978	88.66	88.59	82.19
1979	85.31	85.89	80.05
1980	74.90	74.82	70.41
1981	78.17	76.37	73.76
1982	69,28	69.57	67.27
1983	69.43	95.63	68.52
1984	72.46	100.00	72.44
1985	76.25	89.64	76.23
1986	80.10	96.71	79.62
1987	80.09	91.10	79.90
1988	95.30	95.30	95.30
1989	100.00	100.00	99.60
1990	100.00	100.00	100.00
1991	100.00	100.00	100.00
1992	89.32	88.59	88.56
1993	86.19	86.19	86.19
1994	90.18	90.18	90.18
1995	97.98	97.98	97.98
1996	100.00	100.00	100.00

Table 2. Spearman rank order correlation coefficients.

	Model 1	Model 2	Model 3
Model 1	-	0.682*	0.980*
Model 2	-	-	0.673*

Source: Table 1.

* Correlation is significant at the level of .01 (2-tailed).

As far as the incidence rate (i.e., injuries per 200,000 man-hours) is concerned, there was a notable decline during the 70s, but since 1981 this decline has not accelerated further. Moreover, it seems that the risk levels associated with disabling injuries have not changed markedly; this was especially true during the 80s (Tsolas & Petrakis, 2000).

Moreover, a comparison of productivity growtfi, as it is measured either using conventional inputs and output (Tsolas, 2000) or accounting for changes in the working environment, shows a similar trend

for most years of the time period studied (Tsolas & Petrakis, 2000).

The points above reinforce the combined results of this paper; all three models provide fairly similar results. It should be noted that the model which treats occupational safety data as output (model 2) provides better DEA scores for 1983-87 than the oüer models, though both models which use occupational safety data (models 1 and 2) provide better DEA scores especially during the second half of the 70s as compared with me model which uses conventional data (model 3). This is probably due to the continuous decline in the incidence rate over the 70s.

6 CONCLUSIONS

In the present paper, disabling injuries are considered as an input and alternatively as a joint 'negative* output for the lignite production system in order to measure the technical efficiency of the whole system over the period 1970-96. The technical efficiency is measured through the application of the classical DEA model.

From the combined results of various input-output models considered here, the agreement between me rankings could be deemed satisfactory, especially in the case where occupational data are treated as input. This is reinforced by the fact that comparison of the productivity growth, as it is measured either using conventional inputs and output or accounting for changes in the working environment, shows a similar trend for most years of the time period studied (Tsolas & Petrakis, 2000).

Although the incorporation of disabling injuries into the input-output models provides similar results, there is slight evidence that treating occupational data as "negative" output improves the discriminating power of the basic DEA model applied here.

However, it is worth noting that the present paper has some limitations that could be explored by future research, such as the scale efficiency (Byrnes et al., 1984). It is considered that, if more detailed and comprehensive data (e.g., waste volume processed, another 'negative' output imposed by geological factors, intermediary inputs, etc.) are made available and the assumption of constant returns to scale is relaxed, die results of newly formulated DEA models will be more reliable.

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The U.K. Coal Industry since Privatisation in 1995

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ABSTRACT: The United Kingdom coal industry was privatised on I[st] January 1995 when the production assets of the British Coal Corporation passed into private ownership. The majority of deep coal mines were sold to one major operator and to ease the transition from public to private ownership, privatisation was preceded in some instances by a short period during which independent operators were allowed to take over coal mines under 'lease and licence' arrangements. For the first 3 years of privatisation, most of the mine operators were supplying coal at a price determined before privatisation. This paper details the UK. mine operators, the effect of the electricity-producing companies and their influence on the price of coal for new contracts, and the significance of competition from gas power generation.

1 INTRODUCTION

The deep coal mines -that were fonnally under the ownership of the British Coal Corporation passed into private hands on 1st January 1995. In the past 6 years of private ownership, the mining industry has faced several challenges which have largely been overcome because of willingness, ingenuity and the determination to adapt to market forces, and the successful lobbying of the U.K. government.

The U.K. İs now a small player in terms of coal production on a global scale. Table 1 details world coal production from major suppliers in 1999.

Table I. Major producers of hard coal

PR China	1,029 Mt
USA	914 Mt
India	290 Mt
Australia	225 Mt
South Africa	224 Mt
Russia	163 Mt
Poland	112Mt
Ukraine	81 Mt
Indonesia	74 Mt
Kazakhstan	56 Mt

Table 2 lists current operating deep coal mines in the U.K. and their 1999 production figures, while Table 3 illustrates total U.K coal production since 1989. The latest production figures published by the Department of Trade and Industry indicated that deep coal production in the U.K. in 1999 was 20.9 Mt (U.K Energy in Brief, 2000)[121].

2. CURRENT DEEP COAL PRODUCERS IN THE U.K.

2.1 *RJB Mining PLC*

The major coal-producing company in the U.K. is RJB Mining PLC[11], which currently produces 17 million tonnes from its deep mines and over 4 million tonnes from surface mines. In December 1994, the company successfully completed the purchase of the core mining activities of the English coalfields previously managed by British Coal at a cost of £814 million. At this time, RJB reopened four collieries closed by British Coal. Rossington, Clipstone and Calverton (ceased production in April 1999) were reopened under lease and licence and Ellington was reopened in the spring of 1995. In its first full year of operation, RJB invested heavily and spent £300m on underground mining developments, including the drivage of over 160 kilometres of underground roadway, £55m on new plant and equipment and £11m on modernising general colliery infrastructure.

Table 2. Current Deep Mine Coal Producers in the U.K. (1999 figures) '

RJB Mines	Location	Manpower	Annual Output (Mt)
Clipstone	Nr. Mansfield, Nottinghamshire	222	0.5
Thoresby	Edwinstowe, Nottinghamshire	533	1.7
Welbeck	Nr. Mansfield, Nottinghamshire	521	1.2
Daw Mill	Nr. Coventry	657	1.4
Ellington	Nr. Morpeth, Northumbria	433	0.8
Harworth	Nr. Doncaster, South Yorkshire	623	1.6
Rossington	Nr. Doncaster, South Yorkshire	369	0.9
Maltby	Nr. Rotherham, South Yorkshire	532	1.3
Prince of Wales	Pontefract, West Yorkshire	580	1.3
Kellingley	Knottingley, West Yorkshire	558	1.4
Riccall	Selby, North Yorkshire	418	1.7
Stillingfleet	Selby, North Yorkshire	882	2,0
Wistow	Selby, North Yorkshire	570	1.5
Non-RJB Mines			
Longannet	Edinburgh, Scotland	738	1.0
Hatfield	Doncaster, South Yorkshire	200	0.4
Tower	Aberdare, Mid Glamorgan	350	0.4
Betws	Ammanford, Dyfed	100	0.1
Others		-	1.7
	Totals	8286	20.9

Table 3. U.K. Coal Production Since 1989.[14]

Years ➔	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Deep-mined	79.6	72.9	73.4	65.8	50.5	31.8	35.1	32.2	30.3	25.1	20.9
Opencast	18.7	18.1	18.6	18.2	17.0	16.8	16.4	16.3	16.7	14.8	15.3
Other Sources	1.5	1.8	2.2	0.5	0.7	0.4	1.5	1.7	1.5	1.4	0.9
Total	99.8	92.8	94.2	84.5	68.2	49.0	53.0	50.2	48.5	41.3	37.1

Proven and probable coal reserves were increased by 138 million tonnes to 488 million. In 1996 further investment was made and by the end of 1997, £1000m had been spent in three years on capital projects at RJB sites.

RJB identified 450 million tonnes of reserves in Nottinghamshire and Lincolnshire, and in September 1996, lodged an application with the Coal Authority for a licence to further develop proposals covering a 200 square kilometre area bounded by the towns of Newark, East Retford, Tuxford and the City of Lincoln. Code-named the Witham Prospect, RJB envisage the development of a new mine, employing about 500 people and producing up to three million tonnes of coal a year in an area with excellent links to several of Britain's large coal-burning power stations. However, the scheme has not proceeded due to the collapse in world coal prices.

Early in 1997, RJB, together with Texaco, announced plans for a joint study to explore the feasibility of developing Britain's first large-scale state-of-the-art "clean coal" power station on a site adjacent to RJB's Kellingley Colliery in West Yorkshire. Shortly afterwards, the U.K.'s biggest electricity generator, National Power, teamed up with RJB and Texaco, stating that if the Kellingley project was successful, they had other suitable sites for die development of further clean-coal plants. Taking about two years to construct, the new 400MW Kellingley station would incorporate clean coal technology with a proven track record for efficiency and emission reduction performance, and generate enough power to supply a large city.

In 1995, RJB made a pre-tax profit of £173m on a turnover of £1,461m, repaid £313m of bank debt, produced 37 million tonnes of coal and sold 41.9 million tonnes. In 1996, RJB made a pre-tax profit of £189m on a turnover of £1,308m, paid off the final £160m of bank acquisition debt, produced 34.9 million tonnes of coal and sold 37.6 million tonnes. In 1997, RJB made a pre-tax profit of £172.5m on a turnover of £1,124m, produced 31.8 million tonnes of coal and sold 31.2 million tonnes. The contracts RJB acquired on the privatisation of British Coal for the supply of coal to electricity generators National Power, Powergen and Eastern, expired in March 1998. This meant that for the first three years of operation, RJB had a relatively 'easy' time. Replacement contracts were subsequently agreed for the supply of up to 109 million tonnes by 2003.

However, the contracts were for lower tonnages and with tighter margins. As a result, in 1998, RJB reported a profit of £50m, a turnover of £822.5 million, and sales of 25.9 million tonnes. In 1999, the company reported a profit of £11 million on a turnover of £700 million, but in the first six months of 2000, made a loss of £10.2 million on a turnover of £381.7 million.

RJB Mining is considering plans to revive the Thome colliery near Doncaster with the creation of hundreds of jobs, and has also identified 450m tonnes of *coal reserves* at Witham on the Nottinghamshire border.

2.2 *Mining (Scotland) Ltd.*

Longannet mine is located in Fife in Scotland. It is the only operating deep coal mine in Scotland and is known to suffer from very difficult mining conditions. The mine currently employs around 730 people and its sole market is the nearby Longannet power station owned by Scottish Power Pic. Mining (Scotland) Ltd signed a 6-year contract with Scottish Power, which came into force on 1st April 1998. The pit ran into major difficulties in March 2000 when new reserves[l*l], mostly under the River Forth, ran into severe geological problems that needed seismic investigation. Without the injection of £1 million of funds by a minority shareholder who subsequently became the new chairman, the mine may have had to close.

In April 2000[m], the U.K.*s Trade and Industry Secretary Stephen Byers announced that £100 million was being made available to aid Britain's 17 remaining coal mines. It was hoped the funding would help combat the massive subsidies foreign pits receive and make the U.K.'s coal mines more competitive. The U.K. government, which also announced that it was to end the policy of opposing new gas-fired power stations (implemented in late 1997), insisted that the funding was a temporary measure to help the industry adapt to changes in the electricity market. The aid package[l8*] was approved by the European Commission in November 2000 and in December 2000 it was announced that Longannet was to receive £17.5 million in aid[l*l].

The Longannet aid application was the first to receive approval under the U.K. Coal Operating Scheme, whereby the EC has to approve each subsidy payment on an individual basis. The aim of the U.K. Coal Operating Scheme is to ensure the survival of the industry in the short-term; aid will only be given to those companies which can demonstrate that they will be able to compete in the international marketplace in the long-term.

2.3 *Tower Colliery*

Tower Colliery is currently the U.K.'s only employee-owned coal mine and is situated in Hirwaun in the Cynon Valley in South Wales. In late 1994, workers at the mine who wished to be included in the buy-out were asked to pay £2000 each into a "deposit" fund in order to prove to the government that this was a serious attempt to purchase the colliery. On 3rd January 1995, with a procession of men accompanied by their families, a brass band and the union banner in the lead, Tower Colliery reopened. Of the 320 men employed at Tower before closure, 200 became shareholders in the new company[ll0l].

During 1995, Tower produced 450,000 tonnes from a single working coal face. In 1995, the mine made a pre-tax profit of £4 million and production has been steady over the past 4 years. In October 1999, however, the colliery suffered a minor earthquake[lml] that initially caused minor underground damage but subsequently more serious problems developed and cracks caused by the tremor caused methane gas to flood into the mine. Some £500,000 was spent on tackling the problem, including the provision of a methane drainage system, and production was halted for 9 weeks[l2l].

2.4 *Betws Colliery*

Betws Colliery, in Ammanford, Carmarthenshire, Wales is considered to be one of the most modem mines in Europe[l3l]. Opened in 1978 and privatised in 1994, it produces and processes premium-quality anthracite mined from a single coal seam, ensuring a consistent top-quality product. A purpose-built drying and sizing plant was recently completed to enable the mine to produce exceptional quality filter media for use by the water treatment industry.

2.5 *Other Mines*

Other mines that have closed since 1995 include Hem Heath, Coventry, Markham Main (Coal Investments), Silverdale and Annesley/Bentinck (Midland Mining) in England and Monktonhall in Scotland.

3 THE INFLUENCE OF THE ELECTRICITY GENERATORS

The main or perhaps the ultimate purpose of a business is to produce a profit for its shareholders. In doing so it may offer a service, manufacture goods or produce a commodity. At the same time, the business employs people and so will hopefully benefit the community at large. In the case of coal,

the principal use in the U.K. is for electrical power generation and since none of the mining companies own large power stations (with the exception of Combined Heat and Power - CHP - initiatives), they are reliant upon selling coal to the electricity producers. This inevitably puts them at a disadvantage since in a free market the power generation companies are able to purchase coal from whoever they wish. There is no mora! obligation for them to support the U.K.. coal industry unless they have an ethical policy in place to deal with UK-based suppliers, even at a possible reduction in profit. This obviously can create bad feelings between the coal producers and coal buyers, especially since mining in the U.K. has long had, and continues to have, a strong trade union presence and sense of comradeship.

The ability of the mining companies to stay in business depends on the price they can command for their coal, and there are numerous factors that determine this. In the early days of electricity privatisation, the U.K. saw a move towards gas as the preferred fuel for power generation. The *'Dash for Gas'*, as it was called, was seen to be the answer to several problems, largely concerned with the environment and the U.K.'s commitments to reducing the emission of greenhouse gases.

Since none of the U.K. coal operators produce electricity, it follows that the power generation companies have a great deal of influence over the price of coal. For example, in November 1997[JJ] the electricity generator Powergen announced it would stop purchasing coal from RJB unless it reduced its price by 15%. At the time, Powergen said that it could meet most of its coal needs through imports and from smaller suppliers in the U.K. This coincided with a DTI report to the European Commission which warned of large-scale job losses if the coal industry contracted.

Soon afterwards, miners lobbied the U.K. Parliament[15] to try to persuade the government to prevent large-scale pit closures since fears were abound that the coal industry was facing a massive, and perhaps final, run-down. Mining unions, Members of Parliament (MPs) and coal producers joined forces to fight the threatened shutdowns. The Coalfield Communities Campaign recommended that the government establish an energy policy that ensured there was a market for coal, otherwise deep mining could become extinct by the year 2005. In November 1997, the Energy Minister, John Battle, commented that the government was pursuing a number of initiatives to support the continuation of markets for coal, and the Department for Trade and Industry also stressed that the government did not close pits, as it did not own them. Politics, therefore, have played a major role in the fortunes of the U.K. privatised coal industry and the current Labour

government reminded the press that it was a previous Conservative government which privatised the coal industry, power stations, grid system and electricity-purchasing system.

Whether the U.K. has an energy policy is a matter for strenuous debate, and it is not within the scope of this paper to discuss this topic. However, it is worthwhile considering in brief the implications that the privatisation of the electricity-generating capacity in the U.K. has had on the coal industry. The U.K. has three distinct elements of electricity supply. First, there are the generators using coal, oil. gas and nuclear power. Then there are the supply companies, some of which own power generation capability, but who generally contract to buy power from the generators via a complex contract system. Finally, the National Grid is a separate companj whose function is to distribute the electricity. At the lime of power generation privatisation in 1990. it wasn't feasible for British Coal, a state-owned and state-operated industry, to purchase an inieresl in power generation. However, mining union involvement is an interesting, if possibly contentious, question. The privatised coal industry today may well have been radically different if controlling interests in power generation were controlled by mining-friendly organisations.

The end of 1997[""1] was marked by hundreds of miners making representations to Parliament, and the government announced an action plan for protecting the industry, which aimed to level the playing field between coal and other forms of electricity generation such as gas and nuclear power. Essentially, the coal contracts whose prices had been negotiated before the privatisation of British Coal were due for renewal, and at this time natural gas was seen as a more environmentally friendly and cheaper fuel than coal. The move towards electricity produced by gas resulted in what has become known as the 'Dash for Gas*.

3.1 *The Dash for Gas*

The *Dash for Gas,* namely, the building of gas-fuelled power stations, had been proceeding at a rapid rate since 1990, and could be considered as a major threat to. the livelihood of the U.K. coal industry. In December 1997, the government finally announced that it was putting a hold on new gas-fired power stations in a bid to save the coal industry, and tfìàt it would review its policy on approving new power station developments" '. The Confederation of U.K. Coal Producers was of the opinion that past approvals for gas-fired power stations were to blame for the current problems in the negotiation of new contracts for coal. However, it quickly became apparent that the future of the U.K. coal industry was still at risk, despite the

government's decision to freeze the development of new gas-fired power stations""[1]. Coal mining companies considered that the government's ban on new gas stations was a step in the right direction, but that it might prove to be too little too late and argued that subsidies should be offered to mining industries to avoid pit closures.

Concern was subsequently expressed in June 1998 on the logic behind the moratorium on planning permission for gas-fuelled power stations even though the ban was intended to protect the coal industry until the government completed its energy review[1IV]. Further argument ensued when the developers of one of the biggest industrial sites in Britain criticised a government minister for trying to persuade them to switch from gas to coal as their source of power[1TM1]. A scheme at the Bağlan Energy Park, near Swansea, Wales, was intended to create several thousand jobs in an area of high unemployment, and local councils and businesses had started to attract industry with the promise of cheap energy from a gas-fired power station. Critics claimed that the government was protecting miners" jobs at the expense of employment elsewhere in the economy.

It took until October 1998 for the government to unveil major reforms of the energy market that marked the end of the *Dash for* Cos'[2]. The review outlined plans to restrict the growth of new gas-fired power stations and cut wholesale electricity prices, and concluded that the comparative costs of new gas-fired stations as against the existing coal-fired power stations could not justify the scale and speed of the *Dash for Gas*. Since then 15 new gas-fired power stations have been refused the go-ahead.

The major review of U.K. energy policy was due to be completed in October 2000 when changes to the electricity market were to be proposed to allow new entrants into the market and enable new power station developments to go-ahead[1TM1]. In response to claims that France, Germany and Spain paid £3 billion a year in subsidies to their coal industries, the government announced an aid programme for the U.K. coal operators, which was subsequently approved by the European Commission.

4 THE FUTURE OF THE U.K. COAL INDUSTRY - CONCLUSIONS

Now that the U.K. government appears to be developing a coherent plan for energy production, it is likely that the U.K. coal operators will remain competitive. The recent subsidies allocated to the operators are only intended to be short-term in nature and it is unlikely that long-term state subsidies are politically viable in the U.K. However, the cost to the U.K. taxpayer of £100 million pounds in subsidies so far could represent exceptional value for money compared to the subsidies that other EU countries provide for their coal industries. Concerns over world oil prices, how many years of North Sea gas reserves will be available, the true cost of nuclear power and the contribution to power generation from renewable sources are all factors in whether coal will continue to be mined in the U.K. During 6 years of privatisation, (he coai industry has adapted and operators arc expanding on a global scene, all of which is an indication of the likelihood that coal will be mined in the U.K.. at a profit for several more years to come.

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Effect of Indicator Kriging on Recoverable Ore Reserves of El-Gidida Iron Deposit

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ABSTRACT: Geological studies carried out on the El-Gidida iron deposit revealed that the ore body can be divided structurally into different zones. A three-dimensional geostatistical model was constructed for each zone with the aid of the powerful software "Vulcan". The objective of the present paper is to prove the efficiency of indicator kriging in estimating the recoverable ore reserves of the El-Gidida iron ore deposit above a certain cut-off grade. The recoverable ore reserves were estimated using the ordinary kriged model and the cumulative reserves were estimated without considering weighting factors. A comparison between cumulative ore reserves resulting from the indicator kriged model and ordinary kriged model was performed. The indicator approach helps to keep the tonnage above the cut-off grade approximately constant without distinct changes until the cut-off grade value is close to the average value of iron content. The tonnage above the cut-off grade, with the ordinary kriged model, started to decrease at lower iron content values.

1 INTRODUCTION

When constructing variograms, samples must belong to one zone in order to be able represent the spatial variability of that zone. These zones can be defined by studying the geology of the ore body, including its structural, mineralogical, and chemical properties. A database is then created for a specific zone which has the same geological features and similar assay values. Hence, vertical and horizontal variograms are constructed for each zone separately (Gouda et al., 1995).

The ore body of each zone was divided into three-dimensional small blocks of sizes 50 x 50 x 3 m where 50 x 50 m is the plan view and 3 m is the vertical thickness of each block. Ordinary kriging was used to calculate the estimated iron content of the small blocks to produce the three-dimensional kriged model for each zone.

When using the ordinary technique, the cumulative process was carried out without considering weighting factors. Depending on the grade/tonnage distribution of the iron content, the sum of the tonnage above each cut-off represents the cumulative tonnage.

In this case, the average grade was taken as the arithmetic mean of the grades above the cut-off.

Indicator kriging was used to estimate the proportion of the ore body above a certain cut-off grade by considering weighting factors for the data

above the cut-off grade. In this technique, indicator variograms were constructed, based on the available data which are above a specific cut-off grade, to be used with the indicator kriged model (Lemmer, 1984). Hence, the distribution, average grade and amount of the recovered tonnage at any cut-off grade can be obtained. In order to show the efficiency of indicator kriging, its cut-off grade-cumulative tonnage curves were compared with that produced from the ordinary kriged model.

2 GEOLOGY AND ZONES OF ORE BODY

The El-Gidida orebody is an oval-shaped depression and its main structural elements are a major anticline striking NE-SW and plunging to the NE, and normal faults trending NE-SW, N-S and NW-SE, as shown in Figure 1 (El-Aref & Lotfy, 1989). The ore body occurs more or less as a horizontal bed with a difference in elevation from roof to foot of up to 76 m. The ore body also includes lenses and lens-shaped intercalations, with thickness varying from 1 to 3 m and slightly mineralized barren rocks represented mainly by ferruginous clays and rarely by sands (El-Akkad & Issawi, 1963).

The stucture geology of the ore body, where the presence of major faults reflects the possibilty of dividing the ore body into seven zones, is shown m Figure 2. These zones are: C and M in the high

central area and NW, N, E, NE and SE in the wadi area. A computerized database was established for each zone based on grade information derived from 100 x 100 m test pits/drill hole grid system.
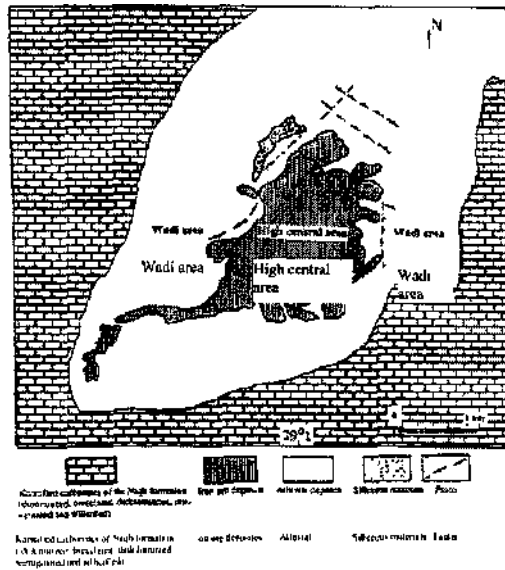

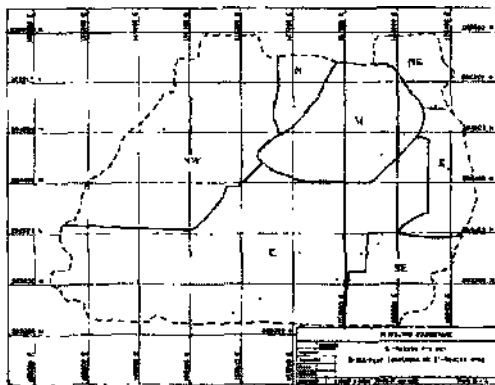Figure I Geological map of El-Gidida area, (simplified after EI-ArefandLotfy,l989)

Figure 2. Zone divisions and location of drill holes at El-Gididaarea.

3 ORDINARY KRİGED MODELING

The ore body of each zone was divided into small blocks. The size was 50 x 50 x 3 m for each block (50 x 50 m through the horizontal direction and 3 m through the vertical direction).

3.1 *Vanograms*

Variogram parameters are required in order to carry out kriged modeling. In this task, for each zone, vertical and horizontal variograms were constructed to express horizontal and vertical variability. These variograms were created for five zones; C, NW, M, SE and E where insufficient samples found in zones N and NE.

Table 1 illustrates the differences in the fitted variogram spherical model parameters: the nugget effect (C_0) and sill (Q both horizontally and vertically, and the range of influence (a) vertically to prove die presence of zonal anisotropy within the ore body. The results indicate that there is a difference in the mineralization characteristics of each zone; hence, each zone must be modeled separately.

Table I. Variogram parameters.

Zooe	Vertical			Horizontal		
	Co	C	a	Co	C	a
c	21.9	73.8	3	22.7	36.8	300
NW	14.5	100	7	12.9	29 5	300
M	37,1	135	4	25.1	44.3	300
SE	18.9	92.4	3	51.3	66.2	300
E	19.4	90.8	4	2t>.9	47.3	300
EIGidida	23.9	78.6	3	21.4	40.3	300

3.2 *Kriged models*

The ordinary kriging technique was used to calculate the estimated iron content of the small blocks within the different zones to produce the three-dimensional kriged model for each zone (an example of these models is given in Figure 3).

Table 2. Statistical parameters of the kriged models.

Zone	Iron content (%)		
	Mean	Si.D	COV
C	53.5	6 4	!! 9
NW	42.9	7.5	17.7
M	48.6	6.6	13.6
SF	51.6	3.6	6.9
E	49.17	4.87	9.9
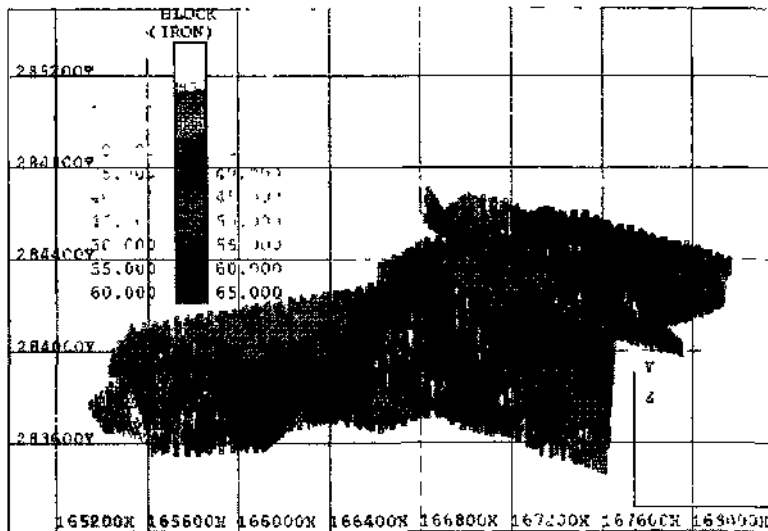
Figure 3 Three-dimensional knged model of zone C
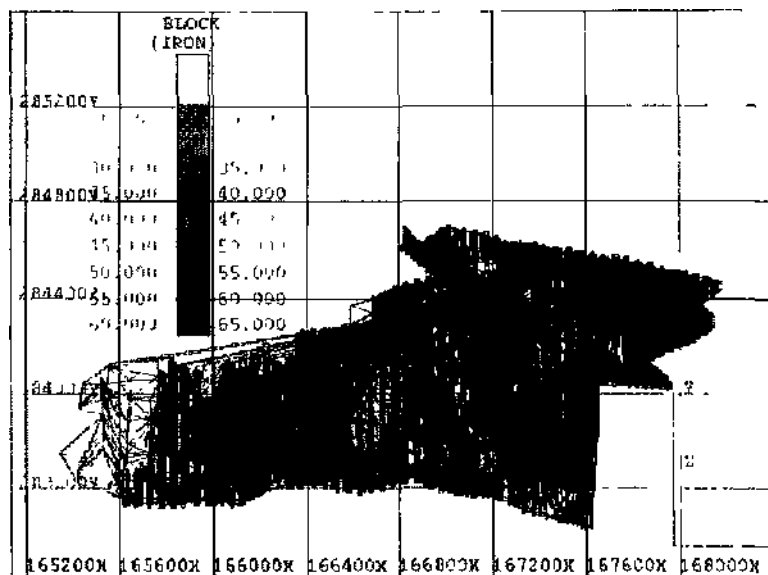
Figure 4 Indicator knged model of zone C al 60% cut-off

These models should also be used with the indicator kriging technique.   .

The kriged models illustrate the distribution of iron content within the different zones, characterizing the poor and rich zones, as shown in Table 2.

The concept of dividing the ore body into different zones could be supported by this analysis where the statistical parameters clearly vary from one zone to another.

4 INDICATOR KRIGING

Indicator kriging allows the construction of a local or global histogram which can be considered unbiased estimators of precisely defined probabilistic distributions; hence, full account is given to the underlying continuity. Such estimates of local distributions can be used in mine planning for the estimation of local recoverable reserves (Jourael, 1984).

4.1 Indicator variograms

Ordinary kriging requires a model of the variogram of the variable being estimated, which is iron content in our case. Indicator kriging requires an indicator variogram, which is different from the variogram used with ordinary kriging.

The indicator variogram depends on the determining of the proportion of iron content values above each cut-off grade z. Each one of the iron content values can be transformed into an indicator by considering different weights for the proportions below and above the cut-off grade (Issaks & Srivastava, 1989). The indicator variogram can be defined as (Journel, 1982):

$$\gamma(h, z) = 0.5 E \left[\left(I(x+h, z) - I(x, z) \right)^2 \right] \qquad (1)$$

Indicator kriging can be performed at several cut-offs using a separate variogram model for each cut-off. In the present study, an approximation is introduced to indicator kriging by using the same variogram model for estimation at all cut-offs.

The variogram model chosen for all the cut-offs is developed from the indicator data at a cut-off close to the median as defined İn statistics. It İs the middle value of a set of numbers arranged in order of magnitude. Indicator variograms can be constructed based on the introduced approximation. Since the variogram used in this approximation to indicator kriging is based on the median indicator, it is usually referred to as "median indicator kriging".

Table 3 presents the fitted indicator vanogram spherical model parameters to be used in indicator kriging modeling. These indicator variogram parameters seem to have different values from those of the variogram used with ordinary kriging due to the characterisitjcs of the indicator approach. The deduced ranges of influence are less than those used with ordinary kriging and vary from one zone to another due to the cut-off grade taken into account. A nugget effect was also found in indicator variogram models within different zones, confirming that the ore body of the El-Gidida area is characterized by a wide variety of mineralogical composition and distribution of main ore-bearing and gangue minerals.

Table 3 Indicator vanoRram parameters.

Zone	Vertical			Horizontal		
	C_n	C	a	C_0	C	a
C	0.03	0.09	3	0 03	0 07	270
NW	0.03	0 11	5	0 03	0 07	280
M	0 05	0 13	3	0 04	0 12	300
SE	0 04	0.12	3	0 02	0.08	250
E	0.02	0.08	3	0 03	0.09	250

4.2 Indicator kriged models

Indicator kriging (IK in short) was introduced to infer a model for the conditional cumulative density function (cdf), e.g., the recovered tonnage and quantity of metal above the cut-off (Perez, 1988).

DC introduces indicator step functions in the grade $z(x)$ at x with the mean value over an area A within a deposit D:

$$\phi(A, z) = \frac{1}{A} \int_A i(x, z) dx \qquad (2)$$

The proportion of the tonnage recovered and the corresponding recovered quantity of metal after applying cut-off z is given by:

$$t(A, z) = 1 - \phi(A, z) \qquad (3)$$

The integral (2) can be accomplished by numerical integration since all indicator values are known in the deposit A:

$$\phi(A, z) = \frac{t}{|A|} \sum_{\alpha=1}^{n} t(x_\alpha, z_\lambda) \qquad (4)$$

This equation can be approximated by the weighted linear combination:

$$\phi^*(A, z_k) = \frac{1}{|A|} \sum_{\alpha=1}^{n} w(x_\alpha) I(x_\alpha, z_k) \qquad (5)$$

If n samples are available over A, the weights u^J are defined according to some criterion. For example, the polygonal method can be used to determine these weights. In this case, the weights are proportional to the polygonal area of each sample location x_α.

Indicator kriging should be accompanied by ordinary kriging. In this case, two groups of variogram parameters were used. In the first, the deduced variogram parameters listed in Table 1 were considered with ordinary kriging, and in die second group, the indicator variogram parameters recorded in Table 3 were taken into account.

The indicator kriging technique was developed to calculate the cumulative tonnage above each cut-off. The cumulating process is dependent on considering the weights at each cut-off. The three-dimensional indicator kriged models (an example is given in Figure 4) illustrate the distribution of the iron content within the oreb ody after taking a specific cut-off grade.

The richest parts of the ore body of each zone can also be identified through indicator kriged models at higher cut-offs. Zones C and M contain the largest amounts of rich ore in comparison with the other

zones. This information can be considered helpful for the blending process.

5 CUT-OFF GRADE TONNAGE CURVES

In order to illustrate the effect of considering weights when indicator kriging is used and its efficiency in estimating the recoverable tonnage, a comparison was made between the recovered tonnage, at different cut-offs for each zone, derived from the ordinary kriged models (i.e., without taking weighting factors into account) and that derived from indicator kriging (Figure 5).

5.1 Cumulative tonnage

In the ordinary technique, the cumulative process was carried out depending on the grade/tonnage distribution of the iron content. The sum of tonnage above each cut-off represents the cumulative tonnage and the average grade was taken as the arithmetic mean of the grades above the cut-off.

It is obvious that the cumulative tonnage above the cut-off grade, when indicator kriging was used, gave better results than the ordinary kriging technique. The cumulative tonnage above the cut-off does not clearly decrease until it reaches about 55% cut-off for zone C, 45% for zone NW, 50% for zone M, 55% for zone SE and 50% for zone E. These values are approximately as the average value of the iron content within the different zones. However, the resulting cumulative tonnage above the cut-off starts to decrease clearly at about 40%, 30%, 35%, 45% and 40% cut-off grades with ordinary kriging. The recoverable ore reserve of El-Gidida ore body could change according to the technique used to estimate it. Dividing each zone into small three-dimensional blocks and considering weighting factors with the indicator approach made the high iron content values play an important role in keeping the tonnage above the cut-off approximately constant until the grade value was close to the average.

This reflects the importance of considering weighting factors when the cumulative process is performed as in indicator kriging to overcome the wide variabilty of iron content within the El-Gidida ore body. Consequently, the ability to control the mine planning and production processes could be improved. In addition, indicator kriging could enable the use of rich and poor parts of the ore body.

5.2 Relationship between average grade and cut-offs

It is important to show how the average iron content of the recoverable tonnage at each cut-off is affected by the application of indicator kriging. As shown in Figure 6, the average iron content within the different zones seems to be higher with indicator kriging than with ordinary kriging. This is another way to prove the efficiency of the indicator technique in calculating the cumulative tonnage.

Calculating the average grade on the basis of taking weights for the different cut-offs into account revealed that the low iron content values have only a small effect on the recoverable ore estimate, i.e., the low iron content values do not represent any major problem when determining such an estimate.

6 CONCLUSIONS

This following conclusions can be made from this study:
1 The vertical and horizontal variogram parameters showed that the ore body has zonal anisotropy and it should be not be considered as one zone so as to avoid a lack of information on its mineralization characteristics.

2. The ordinary knged modeling of the iron content showed the importance of using three-dimensional kriged modeling where the horizontal and vertical variability are taken into account at the same time.

3. The use of indicator kriging is important in calculating cumulative tonnages of the El-Gidida iron deposit according to a cut-off grade.

4. The cumulative tonnage within the different zones does not decrease clearly until the grade value is close to the average value due to the distribution of the three-dimensional values of iron content.

5. The low iron content values do not have a clear effect on the cumulative tonnage where high values tend to have high weighting factors, giving high tonnage above a high cut-off.

6. Indicator kriged models could be used for the blending process to provide the charge to the blast furnace at a constant rate of average iron content and to use poor and rich mineralized parts of the ore body.

7. This approach may solve problems due to the high variability and mineralization characteristics of the ore body.

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(a) Zone C

(b) Zone NW

(c) Zone M

(d) Zone SE

(e) Zone F.
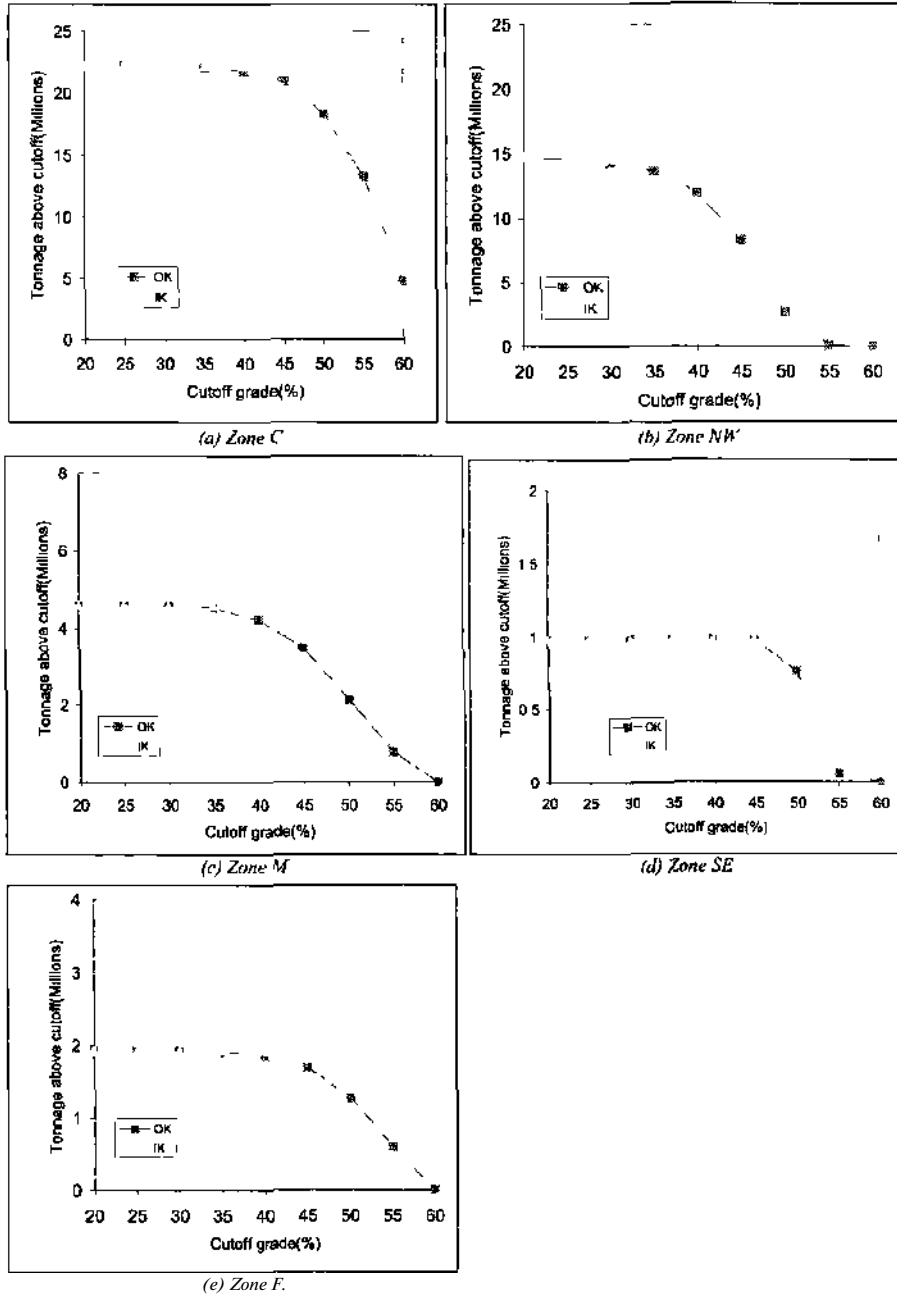
Figure 5. Relationship between cut-off grade and tonnage above it within different zones

(a) Zone C

(b) Zone NW

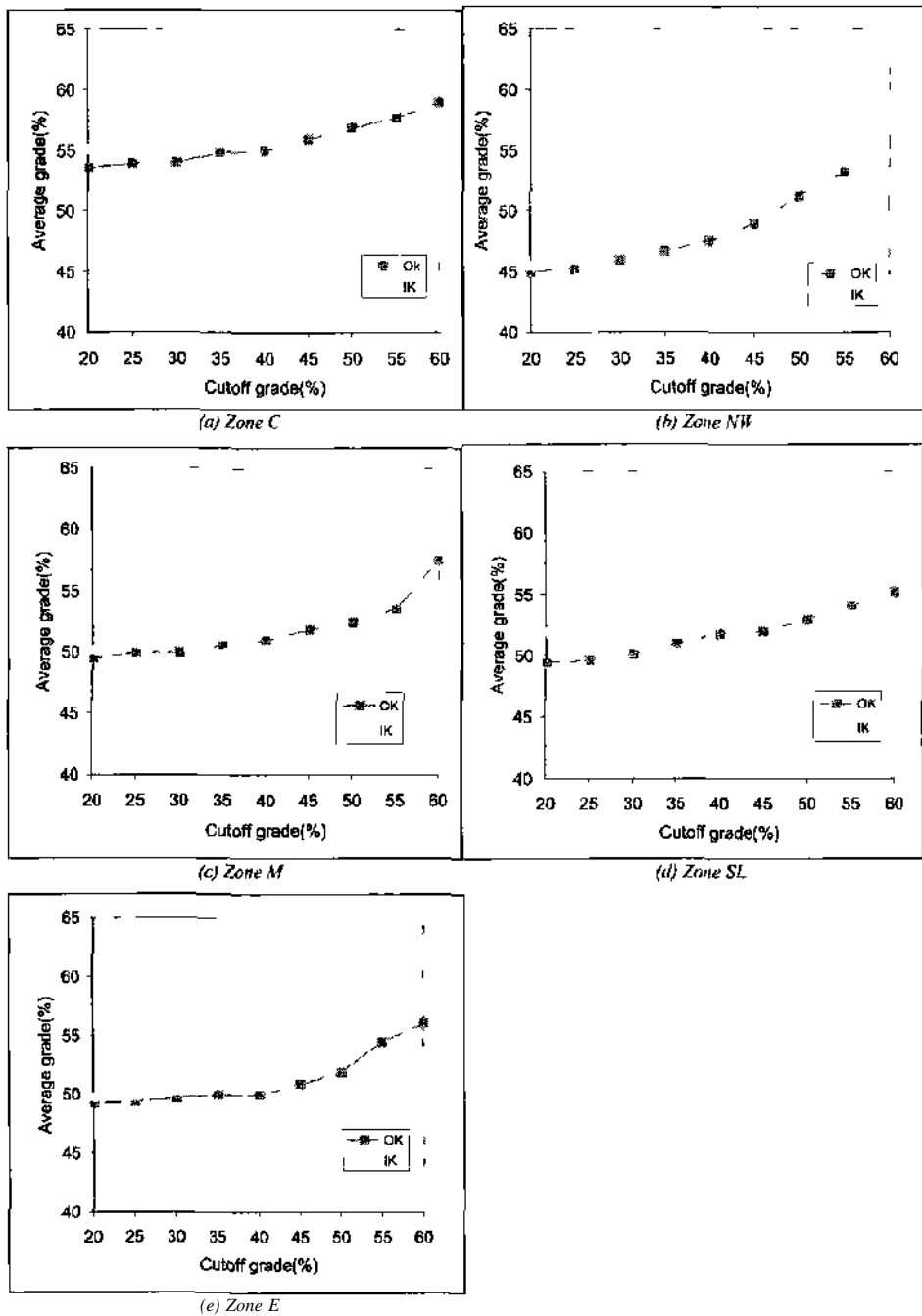*(c) Zone M*

(d) Zone SL

(e) Zone E

Figure 6 Relationship between cul-off grade and average grade within different zones

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Anisotropy Problem in Geostatistical Simulation by Orthogonal Transformed Indicator Methods

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ABSTRACT: This paper explores the manner in which orthogonal transformed indicator methods (OTIM) handle with anisotropy in stochastic simulation. In orthogonal transform, three decomposition algorithms are considered: Spectral, Symmetric and Cholesky-Spectral. Using a simulated deposit with anisotropy ratio 1/3, all three algorithms are evaluated in terms of grade variogram and grade-tonnage curves reproduction.

1 INTRODUCTION

Conditional cumulative distribution function (ccdf) plays an important role in geostatistical estimation and sequential simulation. Indeed, data-dependent optimal estimations are computed from conditional distribution functions and sequential simulations are obtained randomly drawing from conditional distribution functions. A variety of method for estimating conditional distribution functions is suggested. These are classified as parametric and nonparametric. This study İs concerned with nonparametric approach, especially orthogonal transformed indicator method of this approach. The conditional distribution functions and their nonparametric estimation are described in Goovaerts (1997) and Tercan and Kaynak (1999).

Orthogonal transformed indicator method (Tercan, 1999) İs a compromise between the two extremes of indicator cokriging and indicator kriging. It requires less estimation and modelling over indicator cokriging and uses more information over indicator kriging. The idea behind this approach is to transform the indicator functions into a set of spatially orthogonal functions (factors) and to use the autokrigeability property of these functions. Orthogonalization of indicator function relies on principally the decomposition of the indicator variogram matrices as a matrix product.

Despite the aforementioned advantages of OTIM, the approach may be problematic when the variable studied reveals an anisotropic structure. Indeed, it İs not possible to construct the indicator variogram matrices anisotropically because they are estimated either omnidirectionally or in a particular direction. The purpose of this study is to investigate how the conditional distribution functions obtained using orthogonal transformed indicator methods work in geostatistical simulation in the presence of anisotropy. In the estimation of conditional distribution function, three decomposition algorithms are considered; Spectral (SPEC), Symmetric (SYMM) and Cholesky-Spectral (CHSP) decomposition. For comparison, indicator knging (INDI) is also used.

Geostatistical simulations are mainly used for generation of equi-probable alternative realizations of mineral grade and geologic features with specified histogram and variogram. Typically, these realizations are fed into a transfer function developed as a logical equivalent of mineral deposit. By processing multiple equiprobable realizations through the transfer function, an equivalent number of responses are obtained, i.e. a response distribution. This distribution of responses provides a probabilistic assessment of the uncertainty associated with the input variable (Jourael, 1989). In the present study, the decomposition algorithms are evaluated in this setting. First of all, the ability of the decomposition algorithms in reproducing anisotropic variograms is examined. Two transfer functions are defined; proportion and average of grade values above a specific cutoff, yielding grade-tonnage curves as responses. The uncertainty of grade tonnage curves is assessed by the distribution of responses for each algorithms.

2 SEQUENTIAL SIMULATION

Consider the simulation of variable grade Z at N grid nodes x_n conditional to the data set [z(Xa), ct=l,...,n].

Sequential simulation (Journel and Alabert, 1988; Gomez-Hernandez and Srivastava, 1990) amounts to modelling the conditional distribution function then sampling it at each of the grid nodes visited along a random sequence. When a nonparametric approach is considered an indicator-based method is used. To ensure reproduction of the grade variogram model, each ccdf is made conditional not only to the original n data but also to all values simulated at previously visited locations. Multiple realizations are obtained by repeating the entire sequential drawing process Sequential simulation starts with the transform of an indicator vector into the spatially orthogonal factors.

3 CASE STUDY

Figure 1 shows the spatial distribution of 2500 conditionally simulated values on a 50 \ 50 m regular grid at a level of Kure (Asikoy) copper deposit and hereafter considered as reference values for subsequent work.
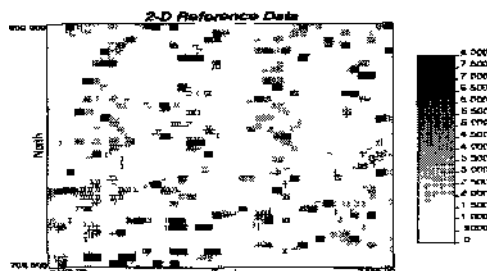

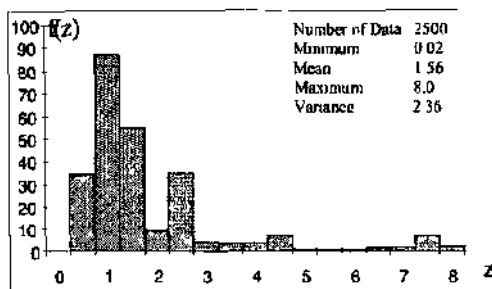Figure 1 The spatial distribution of reference data sel

Figure 2 A frequency distribution of reference data set

The simulated annealing algorithm given in GSLIB (Deutsch and Journel, 1998) was used to force the realization to match an anisotropic spherical vanogram with nugget effect 0.3, partial sill 2.1 and range 38 m in NS direction and 12 m m

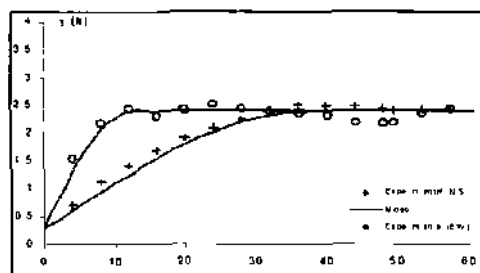EW direction. Figure 3 and 4 show frequency distribution and variograms of the reference data set.

Figure 3 The experimental and input vanograms of the reference data set m the NS and EW directions

The purpose of this study is to evaluate the decomposition algorithms in the presence of anisotropy under perfect conditions. So the problem of statistical inference of the variogram functions will not be addressed here Instead, the variogram models deduced from reference (2500 data) information are used.

As there are no economic and technical restrictions, the nine cutoff values corresponding to the nine deciles of the reference distribution are used: these are; 0.46, 0.60, 0.73, 0 82, 1.10, 1.20, 1.44,2.15 and3.21.

Factor variograms were computed for nine cutoff values. As the order of the CHSP and SPEC factors gets higher, the range of spatial correlation decreases and essentially vanishes for the sixth factor in the NS direction and for the third factor in the EW direction. However, the variogram of the SYMM factors does not display any decrease in spatial correlation. All the factor variograms with a spatial correlation were modelled with a geometric anisotropy model with a larger range in the NS direction (these variograms are not shown here).

One hundred realisations of grade were generated using each of the three decomposition algorithms and also indicator simulation. The first realizations are shown in Figure 4. SISIM given in Deutsch & Joumel (1998) is modified in order to handle with OTTM.

Figure 5a-b show experimental variograms of the 100 realizations for each of algorithms. These figures indicate that there are large discrepancies (fluctuations) between input and realization variograms for all algorithms. This is an expected result since simulation from an indicator-derived (either indicator or orthogonal transformed indicator kriging) ccdf guarantees reproduction of indicator variogram for the cutoffs Zk considered not the grade variogram. In theory, the reproduction of the grade variogram is guaranteed only if indicator cokriging

with infinite number of cut off values are used as pointed out by Armstrong & Dowd (1994) and Deutch & Joumel (1998).

Neither sequential orthogonal transformed indicator simulation (SOTIS) nor sequential indicator simulation (SIS) seems to reproduce anisotropy well (Figure 5). This may be related to smaller number of grid nodes (2500) and smaller size of simulation area respect to the longer range of input anisotropic variogram model (ratio being 1/5 only). However, one obvious result is that SOTIS works as good as SIS in reproduction of anisotropic variogram.

Also note that realization variograms have higher nugget effect than the input variogram model. This high nugget is a result of the discretization procedure used in indicator approach. Indeed nonparametric techniques are applied at K discretization cutoffs Zk and they provide ccdf s for these cutoff values only. In stochastic simulation, it is necessary to complete ccdf for all values other than K. discretization cutoffs z_k The conditional distribution functions are completed by interpolating between ccdf values and extrapolating beyond them. However, the interpolation / extrapolation of ccdf values is done independently from one location to another. This makes simulated grade values within the same class (z u, Zk) spatially uncorrected. Consequently, whatever the number of cutoffs, the sequential realizations have high nugget variances as a result of this artificial noise within classes. OTIM is no exception and shares the same property.

Estimated conditional distribution function may not satisfy the order relations of a valid distribution function. For example, cdf value may be less than 0 or greater than 1 or they may be decreasing with increasing cutoff values. When order relations violations occur, they must be corrected. These corrections for order relations do not impact the reproduction of the grade variogram but the indicator variogram. Indeed experience shows that the SYMM algorithm produces significantly less order relations than other algorithms but the SYMM fluctuations do not differ much from those of other algorithms.

3.1 The response distributions

The proportion and average of the values above the nine cutoffs corresponding to the nine deciles of the reference distribution are calculated first for the reference data set and then for each realization. Figure 6 shows the response distnbution for the proportion (tonnage) while Figure 6 presents the response distribution for the average (mean grade) for SOTIS and also SIS. In these figures, the response distributions at each cutoff are presented using box-plots. In addition, the true proportion and
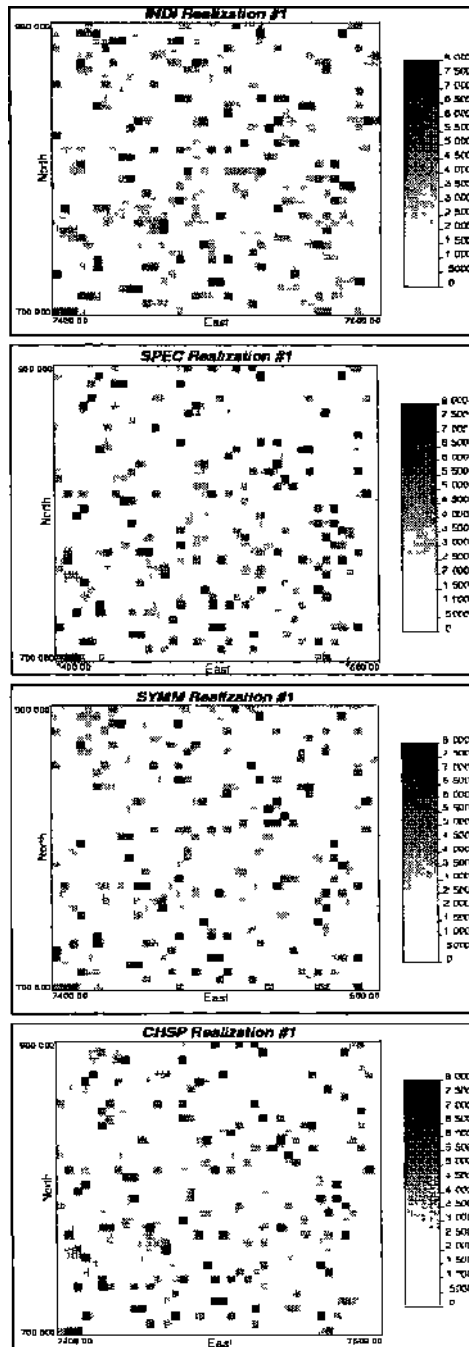

Figure 4 First realizanon of ihe spatial distribuions generated by three decomposition algorithms and indicator kngmg

average values calculated from the reference distribution are marked with ' * ' in these figures.

AH the algorithms can be said to be accurate since the response distibutions of both proportion and average contain the true values. Common to all the decomposition algorithms and indicator approach is an increase in variability of the averages with increasing cutoff. However, for the distribution of the proportion, the increase is seen only at and around median cutoff (1.1). Above all, all the algorithms yield similar uncertainty distributions.

One reason for this is the high number of conditioning data (2500) used in simulation. As this number decreases, one may expect largest differences between algorithms.

4 CONCLUSIONS

Although the conclusions that can be drawn from this study are specific to the data set studied, it is clear that sequential simulations based on orthogonal transformed indicator method reproduce anisotropic variograms as good as sequential indicator simulation algorithm. There seems to be no big difference between the decomposition algorithms when considering the distribution of grade tonnage curves. This is due to the high number of conditioning data. A similar study should be done with no conditioning data in order to see the differences between algorithms.

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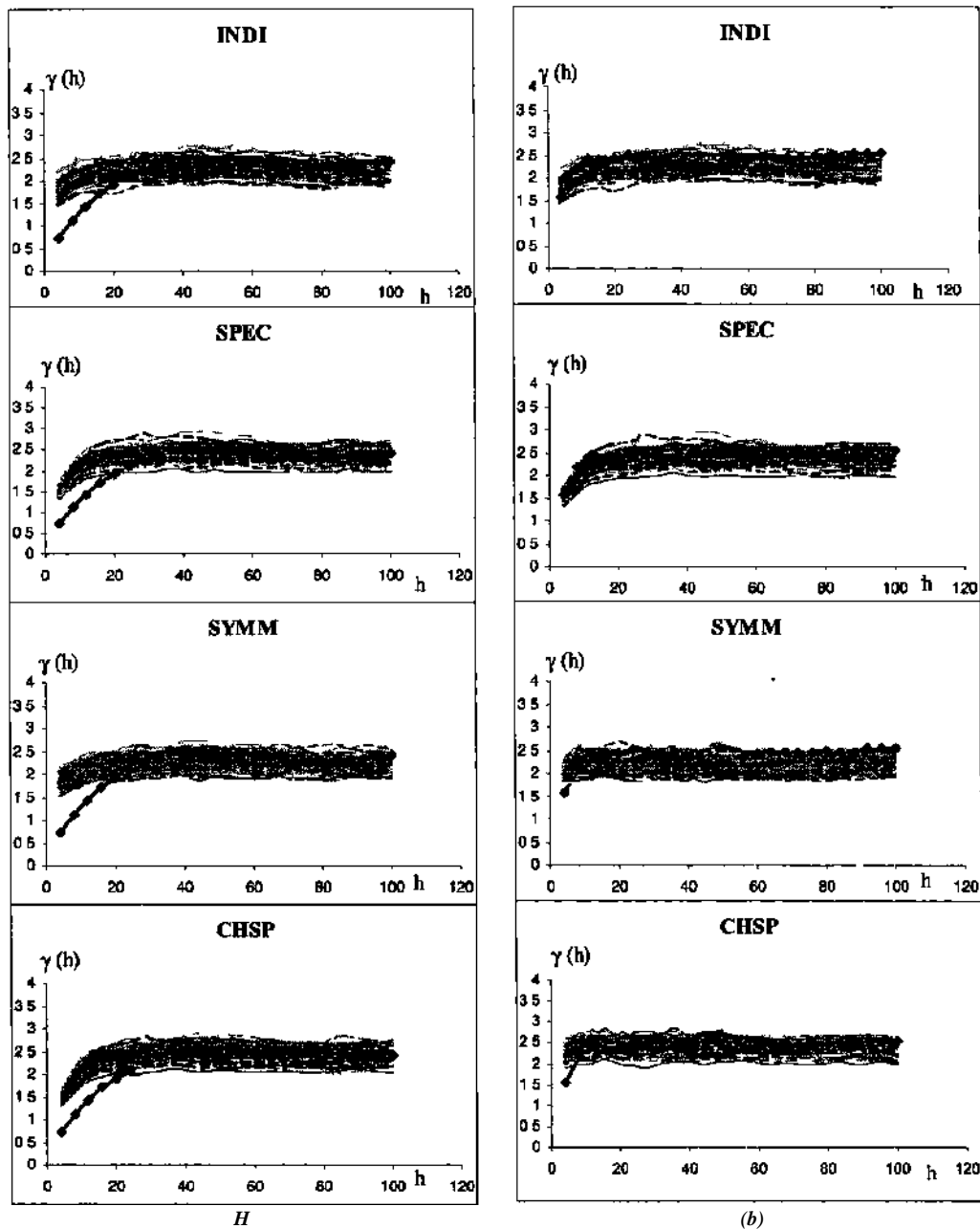
Figure 5 (a) Vanograms of the realizations for INDI, SPEC, SYMM and CHSP in NS direction, (b) Vanograms of the realizations for INDI, SPEC, SYMM and CHSP in EW direction
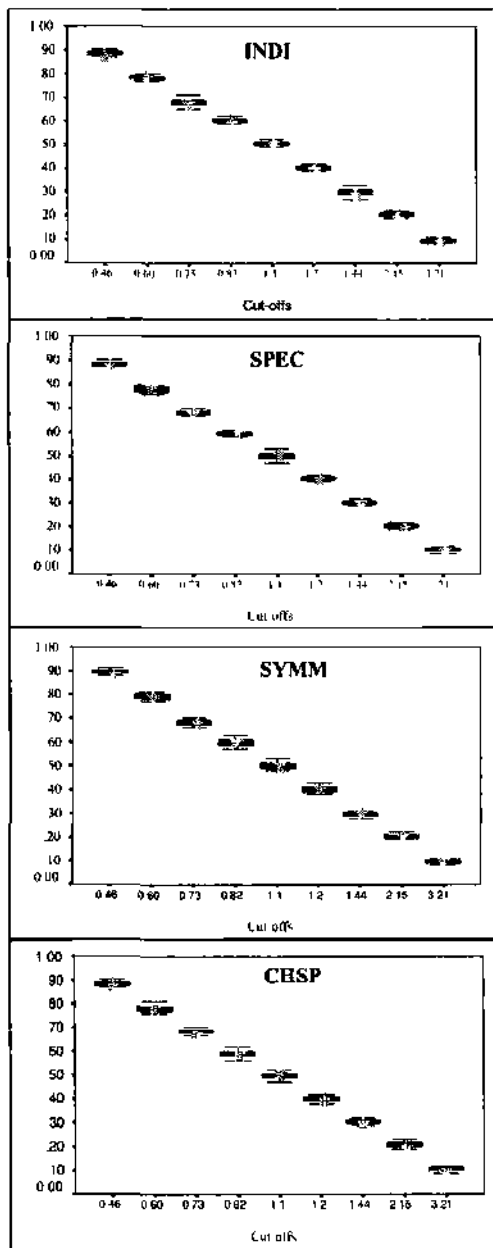
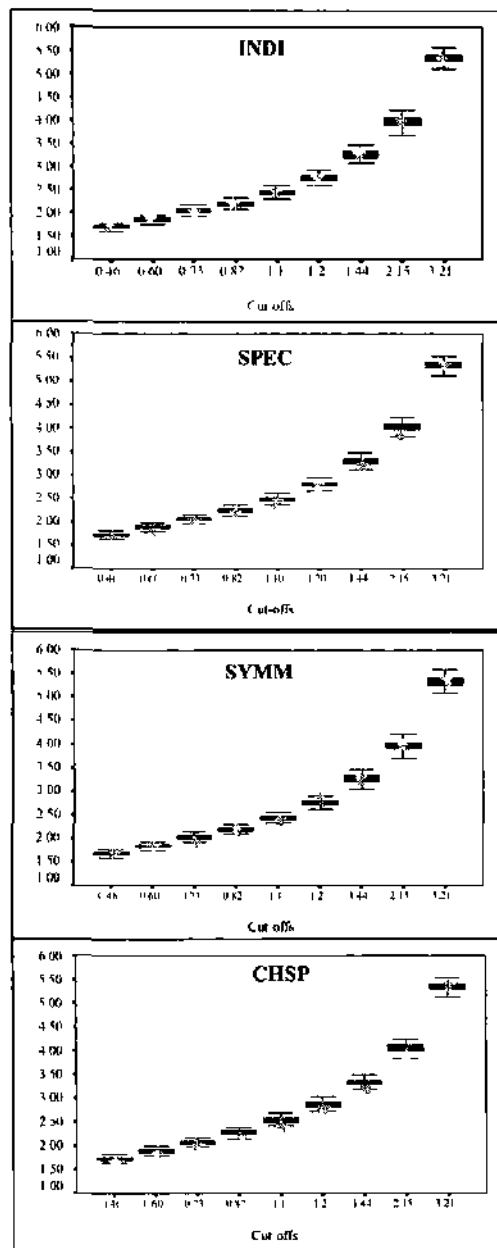Figure 6 The uncenainity distribution for the proportion

Figure 7 The uncertamity distribution for the a\erage

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Geostatistical Simulation of a Quarry

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ABSTRACT: Mine production scheduling requires quantification of the variability of the attributes of the mined product as delivered to the facility. Geostatistical simulation is a method of generating, on any specified scale, realisations of these attributes. Geostatistical simulation provides a set of values that can be used in mine production planning. In this paper, after the importance of geostatistical simulation is briefly reviewed, the sequential Gaussian simulation, which has been widely accepted for the simulation of in-situ mineral properties, is introduced in a case study.

I INTRODUCTION

Data on the in-situ characteristics of the mineral product are obtained from drilling and other sampling programmes such as grade control and blast hole sampling. The grades of planning blocks from the grades of drill holes and/or the grades of blast hole cuttings are commonly estimated by linear kriging (e.g., simple or ordinary kriging). These estimators, based on the least squares method, have significant drawbacks:

1. They are conditionally biased because they implicitly assume a cost or loss function that equally penalises underestimation and overestimation. Unless a Gaussian model for errors is assumed, linear kriging methods yield an inadequate measure of local accuracy.

2. Linear kriging methods yield smoothed results, which cannot be used for these applications because of their sensitivity to the existence of extreme values and their patterns of continuity. Geostatistical simulation can assess uncertainty in production scheduling by use as a "transfer function". To put it another way, risk arising from simulation is assessed by multiple realisations.

3. Linear kriging methods can estimate on the basis of fixed support (e.g., point or block). As the size of the support changes, it is necessary to use cumbersome and assumption-dependant correction methods.

Conditionally unbiased methods are, in general, non-linear. These methods do, however, require significantly more assumptions, which are often unverifiable, and they can be prohibitively time-consuming. Furthermore, these methods provide a sense only of local uncertainty because each conditional cumulative distribution function deals with a single location. Notice that single point ccdfs (cumulative conditional distribution functions) do not ensure the quantification of spatial uncertainty. For example, the prediction of grade fluctuations for various mining and processing decisions (e.g., extraction method, production schedules, milling and stockpiling) requires the assessment of spatial uncertainty rather than that of local uncertainty.

2 GEOSTATISTICAL SIMULATION

Geostatistical simulation provides a set of values that conform to the following criteria (Dowd, 1993):

(i) At all sampled locations they coincide with the actual values,

(ii) They have the same spatial dispersion, i.e., same variogram, as the true values,

(iii) They have the same distribution as the true values.

(iv) They are co-regionalized with any other simulated variable in the same way as the true values.

A set of values conforming to these criteria is called a conditional simulation, i.e., a simulation that is conditional on the simulated values coinciding with the true values at the sampled locations. A non-conditional simulation has attributes (Ü), (iii) and (iv), but not (i).

Conditional simulation does not create data; it simply provides one possibility (among an infinite

number) of what may actually be present at non-sampled locations. This approach amounts to considering the true values as one particular realisation of a random function; each conditional simulation then provides another.

Estimation and simulation are two separate and distinct procedures with different objectives and different results. The objective of estimation is to provide the best (however defined) estimate of a variable at any location. The objective of simulation is to provide a set of values that conform to the criteria listed above, i.e., values that reproduce the characteristics, or behaviour, of the phenomenon as observed in the available data.

Stochastic simulation methods are used:

a. To assess the impact of uncertainty. Stoc&astic simulation provides a means of assessing risk (Dowd, 1994 and Goovaerts, 1999). In this kind of study, many alternate models are generated and processed to construct a distribution of possible values for specified attributes. This distribution is used to evaluate the risk associated with the uncertainty at unsampled locations- Simulation models can also be used for decision-making under uncertainty.

b. To honour heterogeneity. Stochastic simulation reproduces spatial variability. In some cases, only one outcome is used as a basis for performance prediction. Stochastic simulation methods enhance the ability to produce a realistic level of heterogeneity.

c. To obtain complex information. Geostatistical simulation can incorporate an increasingly broad range of information that cannot be accommodated by more conventional methods.

Figure 1 illustrates differences between the use of linear kriging and geostatistical simulation. Geostatistical simulation can be used to assess uncertainty in the production scheduling. The estimation of block model, a posterior scheduling is evaluated as a "transfer function" (Rossi, 1998). Geostatistical simulation provides a response distribution by generating multiple images, i.e., a series of schedules based on possible realisations of the block grades.

2.1 Sequential Approach to Simulation

The basis of sequential simulation is that conditioning is extended to include all data within a neighbourhood that includes the original data and all previously simulated values (Joumel, 1989). Sequential methods are based on an application of Bayes* Theorem:

$$P(A_1, A_2, ..., A_n) = P(A_n | A_1, ..., A_{n-1}).$$
$$P(A_{n-1} | A_1, ..., A_{n-2}) . .... P(A_2 | A_1).P(A_1)$$

(1)
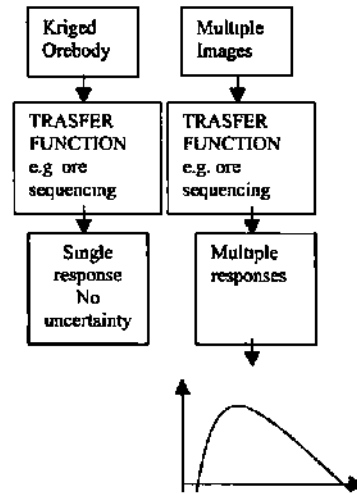
Figure 1 Comparison of linear kriging and geostatistical simulation.

EMBED

Given the joint simulation of z-values at k locations surrounded by n data, the realisations can be generated by drawing from a conditional cumulative distribution function:

$$F(u'_1, u'_2, ..., u'_k; z_1, z_2, ..., z_k | (n))$$
$$= F(u'_k; z_k | (n+k-1)) * ...$$

(2)

$$* F(u'_2; z_2 | (n+1)) * F(u'_1; z_1 | (n))$$

where $|(n+k-1)$ designates conditioning to the n data values and to the k-1 previous realisations. This decomposition makes it possible to generate a realisation of a random vector $\{Z(u'_j), j = 1, ..., K\}$ in K successive steps:

1. Determine the conditional distribution at the first location, which is conditional on the available n data:

$$F(u'_1; z | n)) = Prob[Z(u'_j) \leq z | (n)]$$

(3)

2. Draw a value z; from the conditional distribution of Z; given the n data
3. Draw a value Z2 from the conditional distribution of Z2 given the n original data and that $Z_t = z_s$

K. Draw a value z_K from the conditional distribution of Z_K given the n original data and that $Z_t = z_t$, Z2 = z3, ..., $Z_{K1} = z_{K1} n$.

Sequential simulation is used for the generation of conditional realisations of either a multi-Gaussian random function or any non-Gaussian random function as long as its conditional distribution can be derived. The type of sequential simulation depends upon the way in which the local conditional probability distribution is estimated. For instance, multi-Gaussian kriging yields an estimate of the local conditional probability distribution (lcpd) by assuming a normal distribution and estimating the mean and standard deviation. If multi-Gaussian kriging is used in simulation, the algorithm is known as sequential Gaussian simulation. If indicator kriging is used to estimate the lcpd, the algorithm is known as sequential indicator simulation.

2.2 Sequential Gaussian Simulation

Parametric non-linear geostatistical methods require the conditional probability distribution of the random variable, which, in practice, is impossible to obtain. For a Gaussian random function with known mean, the conditional distribution of Z-, is Gaussian, with mean $/$,- and variance $o £y$, where $Z_{\bar{r}}$ is the simple kriging estimator of $Z_{\bar{r}}$ and $G^{\wedge}$; is the associated kriging variance. The multi-Gaussian model overcomes this problem by using the normal scores transform of the grades:

$$Z(x) = \phi(Y(x)) \tag{4}$$

$$Y(x) = \phi^{-1}(Z(x)) \quad x \in Orebody \tag{5}$$

One of the most important advantages of multi-Gaussian kriging is that there is no restriction on the type of grade distribution, provided İt is first transformed to a normal distribution. Prior to the simulation, exploratory data analysis helps to reveal whether lack of stationarity, the presence of outliers, clustering of data or spiking exists. Provided that the random function $Y(x)$ is also multivariate normal and strictly stationary, sequential Gaussian simulation can be implemented. Otherwise, other procedures should be considered. The hypothesis of multinormality requires each random variable $(Y(x), x$ e $Orebody)$ to be normally distributed (Olea, 1999). To put it differently, multivariate normality among all variables at all possible spatial locations is assumed. Not only is normality of the one-point conditional distribution function (cdf) required, but also the random function should be multivariate Gaussian. If transformed data are not multivariate normal, the simulated data will not reproduce the characteristics of the original data. Ideally, a transform which is Gaussian all two-point cdfs should be fulfilled. Such a transform is very difficult. A simpler, and more readily verifiable, assumption is bivariate normality. This can be checked from bivariate histograms of sample pairs for a fixed lag during variogram calculation.

The inverse, or back, transformation is performed by linear interpolation. Extreme simulated normal values that lie outside the range of the conditioning data are interpolated using two pre-specified extreme transform pairs. The range is set to the minimum and maximum constraints on the original data values or to narrower values if these constraints are unlikely to be exceeded. The main advantages and disadvantages are as follows (Dowd, 1992):

Advantages:
1. Sequential Gaussian simulation guarantees that data are honoured at their locations because kriging is an exact interpolator and, therefore, yields a zero kriging variance when a datum is estimated. The simulated value is thus drawn from a normal distribution with zero variance and a mean equal to the datum itself. As the conditioning is an integral part of the simulation, no additional step İs necessary.
2. Anisotropics can be handled automatically as part of the kriging process. Kriging with an anisotropic semi-variogram ensures that the anisotropics are imparted to the kriged values, which, in rum are imparted to the simulated values drawn from distributions with means equal to the kriged values.
3. Any covariance function can be implemented.

Disadvantages:
1. The main drawback rests on the assumption of using the intermediary Gaussian distribution. In practice, it is impossible to guarantee a multivariate normal distribution.
2. There is some evidence to suggest that sequential Gaussian simulation produces less variation between successive simulations than other methods such as turning bands.

The selection of application parameters, such as the use of ordinary kriging versus simple kriging, the maximum number of simulated nodes retained for kriging, octant search parameters, and upper and lower extrapolation values can profoundly affect the procedure. Non-stationarity can be taken into account by using kriging with a trend model instead of simple kriging. Chiles & Delfiner (1999) reported that a Gaussian stochastic process with an exponential covariance model is a special case where the method could be applied easily without any approximation for any set of data points or simulated points, if the mean is known.

3 CASE STUDY

Sequential Gaussian simulation is implemented for which 616 data values are available from 27 drillholes and simulated values at 10000 locations

are required. The data are CaO contents taken from the quarry of a cement plant The performance of the method is assessed by its ability to reproduce the specified model parameters and statistics. A stochastic realisation should reproduce the declustered sample histogram and the semi-variogram model, and should coincide with data values at their locations {Figure 2 for semi-variograms and Figure 3 for histograms).
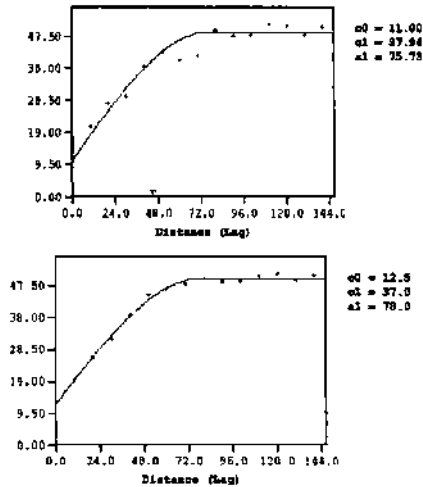

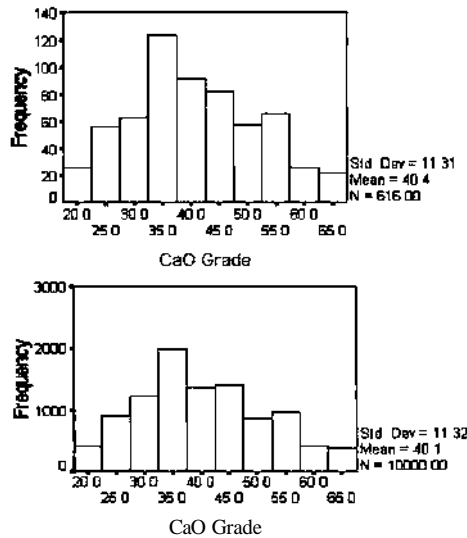Figure 2. Semi-variograms of dala (above) and simulated field (below).

Figure 3. Histograms of data (above) and simulated field (below).

In practice, exact reproduction is impossible because:

- Model statistics are inferred from sample information, which is assumed to represent population parameters. As more data are used in the conditional realisation, the realisation statistics become increasingly similar and closer to the desired statistics.
- Data are subject to measurement error and, thus, measured values are not necessarily the same as the real, in-situ values. Furthermore, the part of the nugget effect that arises from these errors is not reproduced by the realisation semi-variogram.
- When the semi-variogram range is larger than the size of the simulation area, ergodic fluctuations of the semi-variograms of the realisations are especially important. Searching is limited to a maximum of the eight closest original data and the eight closest previously simulated values so as to reduce the computation time and to avoid round-off errors in the kriging matrix. But theoretically, the search should be extended to the semi-variogram range.

Discrepancies between the semi-variograms of the data and those of the simulated values may also arise when:
a. there are no data or previously simulated values for kriging,
b. there are insufficient data and previously simulated values for kriging,
c. the kriging matrix is singular.

4 CONCLUSIONS

As has been seen from the case study, 10 000 locations have been simulated. Histogram and semi-variogram parameters have been fairly reproduced. Selective mining units and planning blocks are obtained by averaging the values of these supports over me larger volumes. The averaging procedure is effectively a change of support operation. The set of blocks obtained by this process constitutes a 3-D block model of me ore body. Simulated values on planning blocks are ready to be submitted to the production scheduler. Note that geostatistical simulation generates only one possible realisation of block grades. Given that the fundamental technical parameters are the ore reserve block grades and tonnages, geostatisical simulation allows the assessment of technical risk arising from these parameters. The simulation procedure is repeated many times. Each simulation produces different block grades and tonnages and thus a different pit shape and size that have a significant effect on the risk analysis.

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Technical and Economic Possibilities of Using Low Temperature Geothermal Sources in Croatia

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ABSTRACT: Geothermal sources are exploited in two basic forms, i.e., the primary form of thermal energy and that converted into electrical energy by using an adequate thermodynamic cycle. There is a possibility of applying several processes for the conversion of thermal into mechanical or electrical energy which depend on the thermodynamic characteristics of the geothermal water. Geothermal sources in Croatia are mostly so-called low-temperature geothermal sources with water temperatures lower than 100°C. The low-temperature potential is a major disadvantage when it comes to power generation, since state-of-the-art geothermal power plants require reheated steam or hot water to operate. There is a common belief that the optimum way of exploiting low-temperature geothermal sources is heat generation. This paper examines the technical and economical possibilities of using low-temperature sources for conversion of thermal energy into mechanical work and electrical energy.

I INTRODUCTION

Geothermal sources in Croatia do not contain steam of relatively high temperature like in Italy, although both countries are in almost the same position with respect to geothermal belt. Nevertheless, the low-temperature difference Stirling engine can be successfully applied even when hot water is of a moderate temperature. Calculations based on the recently confirmed geothermal wells indicate there is a potential for generation of about 46 MW of electrical power. The feasibility of the low-temperature difference Stirling engine for exploitation of the existing geothermal potential in Croatia is analyzed on the ground of energy and economic advantages.

A binary process is used for geothermal reservoirs at relatively low temperature. In this process, the geothermal fluid in the heat exchanger only transmits heat to the secondary highly volatile fluid used to drive the turbine, to be reinjected into the reservoir through the injection well.

The Stirling cycle seems to be a better and more practical solution resulting in considerably higher efficiency because it is thermodynamically equivalent to optimum Carnot's cycle, while Clausius-Rankİne

follows this cycle only partially. The development of Stirling's engine with flat plate heat exchangers has shown that low-temperature geothermal reservoirs may also be successfully used for conversion of heat into mechanical work or electrical energy. The temperature difference of not more than several degrees Celsius which enables the process makes the flat plate low AT Stirling engine particularly attractive for exploitation of this kind of geothermal source. Hot water from the well circulates through a number of Oat boxes connected with a crankshaft driven by a generator. After transferring its heat to the plant, the cooled water is returned to the reservoir by means of the injecting pump. In addition, a geothermal plant using the Stirling cycle has considerable technical and economic advantages over the classic Clausius-Rankine process because there is no evaporator, condenser, FW pump or numerous other mechanical elements. This makes the Stirling cycle technically simpler and less investment- and cost-intensive.

For this reason, this paper gives a comparison of technical and economic efficiency for binary and Stirling facilities using a geothermal reservoir *with* known reserves, constructed wells and geothermal water quantities.
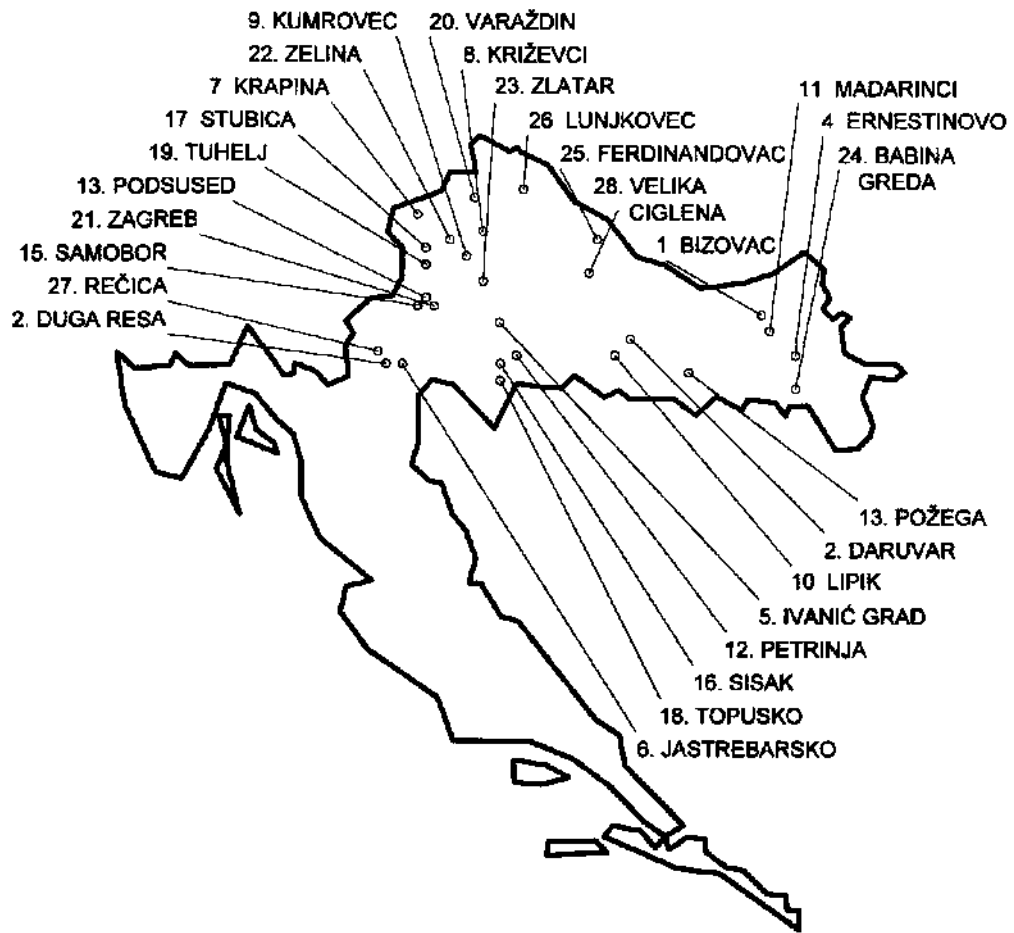
Figure 1. Locations of the geothennal fields in Croatia.

2 GEOTHERMAL ENERGY POTENTIAL IN REPUBLIC OF CROATIA

All the geothermal reservoirs in the Republic of Croatia can be classified in two groups:

A) reservoirs with temperature of geothermal water below 100°C (1 to 23 in Figure1);

B) reservoirs with temperature of geothermal water above 100°C (24 to 28 in Figure1).

The geothennal reservoirs are situated in the central and Panonian regions of Croatia, as shown in the geographical map. The central region comprises the areas from Kordun and Banovina to Medimurje. The Panonian region extends through the Panonian basin and Medimurje to the eastern border of Croatia.

In Croatia, there are 14 locations where geothermal energy is used mostly for balneology, recreation and space heating. Still, this is only a half of the total of 28 geothennal reservoirs situated İn the northern part of Croatia. Therefore, techno-economical analysis of direct geothennal energy use is provided for all fields. The main parameters in the technical part of the analysis are the number of geothennal wells and related flows and temperatures as the basis for installed thermal power calculation.

3 TECHNICAL AND ECONOMIC POSSIBILITIES OF USING GEOTHERMAL ENERGY

3.1 *Direct geothermal energy usage*

Existing capacities are presented in Table 1, where the flow (q) and temperature (t) of geothermal water at production wells are actually utilized.

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The temperature difference (At) is the average value of the differences between geothennal water temperatures (t) and the outlet temperature at relevant heat exchanger, given by user. These data are the basis for thermal power (Q) calculation, which is the sum of power at each well at one location. The total geothennal capacity used at the 14 locations in Republic of Croatia is only 42 MW thermal.

A possible increase in geothennal energy use can be realized in three ways. The first is use of the maximum peak flow at existing wells. The next step is utilization of the maximum temperature difference, with an outlet temperature from the heat exchanger of 20°C. The third power increase can be realized by involving more wells in heat generation. When all these conditions are put together, the maximum geothennal potential could reach 740 MW,.

The unit costs (cGE) of direct geothermal energy use are calculated by adding the unit capital cost, unit maintenance and unit electrical cost for pumping energy at every reservoir, as presented in the diagram below.

Unit costs (cGE) are different for the various geothermal locations. The lowest energy cost is at Velika Ciglena, while the highest is at Krizevci, as shown in Diagram 1. However, the reference value of importance for future geothennal development is an average price of 0.0166 USD/kWh,, which should be taken as the same for all the geothermal fields in distribution.

In Diagram 1 it can be seen that exploitation of most reservoirs is below the average line, and is profitable. The other 10 reservoirs with unit costs higher than the average price are not competitive economically.

The most valuable are higher temperature resources. They can produce the largest capacities with increased capital cost and the lowest energy price.

Table 1. Installed capacities for direct geothermal energy use.

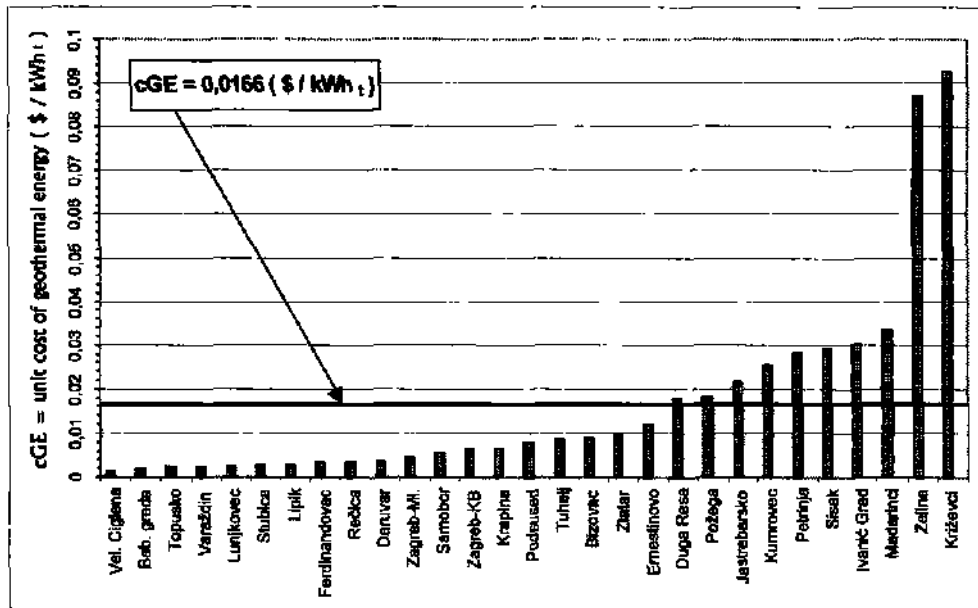| No.	Name of reservoir	Geothermal well	Flow q (m³/h)	Temp. t (°C)	Delta T Δt (°C)	Power Q (kW,)
1	Bizovac	Biz-4	14	98	42	1560
		Slavon-1	18	86		
2	Daruvar	Antunov ı.	34	47	27	2000
		Ivanov iz.	30			
3	Ivanic Grad	Iva-Tl	10	60	26	300
4	Jastrebarsko	izvor	190	26	6	1320
S	Krapina	KRT-1	145	42	13	2200
6	Lipik	B4a	30	61	26	900
7	Pofcga	izvor-1	10	27,5		
		izvor-2	MO	25	5	740
		trofi-1	7	24,5		
8	Samobor	STB-I	108	28	8	1000
9	Stubica	izvor	35	48	20	3720
		B-l	126	57		
10	Topusko	TEB-1	45	63		
		TEB-2	35	63	26	9200
		TEB-3	225	63		
II	Tuhelj	izvor 1	145	32	12	3760
		izvor 2	125			
12	Varazdui	BI	72			
		B2	7	58	23	2600
		B3	18			
13	Zagreb-Ml.	MIadost-1	11	70	50	3420
		MIadost-3	50	80		
14	Zlatar	Sutinka-1	210	26		
		Sutinka-2	30	26	10	9420
		Sutinka-3	570	39		

Diagram 1. Geothermal reservoirs sorted according to unit energy cost.

3.2 *Binary process*

By conversion of thermal into electrical energy (Table 2) in a binary cycle, it is possible, with the existing water yield and temperature, to ensure gross power of Nb=3.8 MW$_e$. The power necessary for station service consumption is 0.5 MW$_e$, so the net power (delivered to the power grid) is N$_n$=3.3 MW$_e$. For the economic analysis, die selling price of electricity is p=72.2 USD/MWh. Assuming the facility would be in operation for 8,000 hours a year, the annual electricity output would be:

E= N$_n$*8,000 = 3.3*8,000 = 26,400 MWh/year.

The total revenue from electricity sales would be:

UPe= E*p$_e$ = 26,400*72.2 = 1,906,880 USD/year

With regard to downstream heat exchangers, the water temperature is 70°C, and its thermal energy could be used by numerous consumers (greenhouses, fish ponds, dryers, *the* tourist industry, and die like). The available temperature difference being At=30°C, İt is possible to ensure maximum heat generation of Q=22.6 GJ/h. Assuming the heat delivery is planned for 2,000 hours, and a with selling price of PQ= 2.76 USD/GJ, die total revenue of thermal energy sales would be:

UPo=Q*2,000*po=22.6*2,000*2.76=124,807USD/year

The total annual revenue from selling thermal and electrical energy from the binary cycle would be:

SUP = UPe +UPQ = 2,030,887 USD/year

and it would remain constant for me calculation period of 20 years.

3.3. *Stirling process*

In all thermodynamic processes in which heat is converted into mechanical work or electrical energy, and particularly when low-temperature sources are involved, ambient temperature plays an important role. Therefore, use of the Stirling process in a hypometical geodiermal reservoir would change energy potential throughout the year relative to changes in average ambient temperature.

Calculation of the possible power generation from geothermal reservoir "X" - cascade process.

Well yield: D= 146 l/s = 525,600 kg/h
Outlet temperature: Ti= 136°C = 408°K
Ambient temperature: T$_0$= 20°C = 293°K
Mean temperature: Ts= (Ti+T$_0$)/2 = (136+20)/2
                        =78°C = 351°K
Temp, difference: ΔT= Ts-T$_0$ =408-293 = 58°K

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Camot's efficiency: $\eta c = \Delta T/Ts = 58/351 = 16.5\%$
Thermal power: $Q = D \cdot cp \cdot \Delta T =$
$\approx 525,600 \cdot 1/860 \cdot 58 =$
$= 35,447\ kWt \sim 35.5\ MWt$
Electrical output: $N = Q \cdot \eta c = 35.5 \cdot 0.165 = 5.9\ MWe$
Power generation: $E = N \cdot z = 6 \cdot 8,760 \cdot 0.8$
$= 42,048\ MWh$

The values calculated could be achieved in a cascade process and by use of a flat plate Stirling engine, with a yield of 146 l/s, geothermal water outlet temperature of 135°C and assumed constant ambient temperature of 20°C.

The calculation basis was a gross plant power of $Nb = 5.9\ MW_e$. As for the binary process, the station service consumption is assumed to be 0.5 MW, and the net power (to be delivered to the grid) is $N_n = 5.4\ MW_e$. Assuming 8,000 operating hours a year, the power generation would be:

$E = N_n \cdot 8,000 = 5.4 \cdot 8,000 = 43,200\ MWh/year$

The total income from power sold would be:

$UPe = E \cdot p_e = 43,200 \cdot 72.2 = 3,119,040\ USD/year$

The calculations presented above show that by use of the Stirling process, under thermodynamic conditions in a hypothetical geothermal reservoir, it would be possible to achieve a total output which is 50% higher man that achieved in a binary process. This results in equal increases in generated power and total revenues.

4 CONCLUSIONS

Economic evaluation of the geothermal reservoir "X" pilot project includes comparison of the binary and Stirling process. The basic input evaluation data are given in Table 2.

Table 2. Basic input data for geothermal reservoir :X_pilot proiecL

Input data		Binary process	Stirling process
Construction duration	year	1	1
Project lifetime	year	20	20
Plant capacity	MWe	3.8	5.9
Specific investment	USD/kW	2,926	1,319
Internal rate of return (TRR)- predetermined	%	10.00	10.00
Operation duration	h/year	8,000	8,000
Power generation	MWh	26,400	43,200
Heat generaüon	GJ	45,200	#
Employees	personnel	13	8
Generation costs			
Depreciation			
Depreciation rate for civil works	%	5.00	5.00
Depreciation rate for equipment	%	6.67	6.67
Depreciation rate for intangibles	%	20.00	20.00
Other material expenses (% of total revenue)	%	4.50	4.50
Other non-material expenses (% of salaries)	%	11.60	11.60
(% of total revenue)	%	2.00	2.00
Maintenance costs (% of investment)	%	3.00	3.00
Insurance costs (% of investment)	%	0.58	0.58
Gross salaries	USD/employee	7,680	7,680
Royalties (% of total revenue)	%	2.50	2.50

All expenses include investment, operational and maintenance costs, and any other expenses related to power generation which reduce the final financial outcome (taxes, royalties, contributions, membership fees, etc.). Total investment includes all the resources necessary for preparation for geothermal energy exploitation. For the binary process, this is assumed to be 12,120,000 USD, and for the Stirling process, 8,484,000 USD. The technical simplicity and smaller number of parts (no boiler, condenser, FW pumps or other related mechanical parts) make the Stirling process considerably cheaper than the classic binary process. For this caLulation, it is assumed that the total investment in the Stirling process is 30% lower than that in the binary process. The cost estimate is important for capital investment efficiency evaluation. Individual input standards were taken into account, along with the total

quantity of each particular input, input unit prices, purchase value of equipment, number of employees and gross salaries.

The lifetime planned for both processes is 20 years. Dynamic methods were used in economic evaluation:

payback period,
net present value (NPV) method,
internal rate of return (IRR).

The economic evaluation results are much better for the Stirling process than for the binary process. With considerably higher power, annual output and total revenue, the Stirling process results in twice as short a payback period. With a predetermined discount rate of 10%, the net present value for the binary process is negative, and according to the economic evaluation criteria, the project is not feasible.

For the Stirling process, the net present value is positive and therefore the process is feasible. The internai rate of return is therefore a relative criterion of project efficiency which offers information on the average annual rate of return. For the binary process project, this is only 9%, which is below a feasible rate. The internal rate of return for the Stirling process would be 29%, and this makes it feasible.

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Regulation of the Mode of Mining Operations in Open Cast Mining of Steeply Dipping Fields

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ABSTRACT: The methods of mining operation mode control are described. An example of forming a temporarily non-working board in the Kachary quarry is given together with the advanced small quarry of Cokolovsky open cast mine in the planning of mining operations.

1 INTRODUCTION

The performance of the manufacturing and economic activity of mines using an open excavation method is in many respects instituted by the extent of the optimality of accepted solutions İn planning and production management, assets, property, investment and financial activity. These are bound to a conjuncture in the markets of the commodity produced, the materials and machinery obtained, money and capital. The production and ecomonic policy of a firm should allow for stability or the possibility of the changing of the bulk, nomenclature and quality of the commodity, estimation and collation of the most effective directions of mining operations, progress and technology of production, decrease in costs, upgrading of assets and choice of logical sources of collecting information, material and financial resources.

For the maintenance and extension of the productive capacities of mining operations of mineral resources, research into and acceptance of such solutions at the choice of a direction of progressing up for the mode and time planning of mining, which ensure the optimum allocation of bulks of production of an overburden, is indispensable.

Only those bulks of an overburden and in certain directions should be effected which provide continuity of mining and production in given bulks with demanded qualitative characteristics and in particular periods.

2 THE MODE OF MINING OPERATION BY TEMPORARY NON-WORKING BOARD

The regulation of Strippings ıs carried out as follows:
- Variation of open pit boundaries or development cycles, if it causes a decrease in overburden coal ratios without impairment of the qualitative performances of useful mined mineral;
- Installations of a direction of mining operations progressing, supplying sequential drawing in mining leases of deposits, which İn quahtative characteristics and bulks makes it possible to obtain the commodity foreseen for implementation with minimum first mining values of overburden ratios;
- forming of technological zones: in quarry with rational arguments;
- usage of opening schemes with half-stationary and temporary openings;
- application of mining methods in steep layers that allow mining with magnification of the angles of acclivities of working boards;
- of phased mining of the deposits with forming of temporarily non-working boards, small advanced open casts, twinning or threeing of benches that provides conservation and carrying on later periods of considerable bulks of an overburden, with allowance for secure subsequent iterating of mining operations on them and complete improvement of pillars.

Thus, the proposed strategy of regulation of intensity direction, mode and time schedules of mining operations, including in primal time for a decrease of peak loads on overburden, rigid technologies of mining of deposits İn steep layers with forming of temporarily non-working boards and advanced small open casts enables:
- extraction of reserves of an open pit field in periods with service factors of overburden lower or close to a mean overburden ratio in an open pit field at low prices for commodity and rises during high prices;
- systematic mining of ores at complete loading of preparation plant powers;
- possibility of regulation İn time of ore bulks and of corresponding stripping bulks on the basis of the

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prognosis of supply and demand on firm's commodity and prices for it, and also on consumable materials in markets;

- continuous mining of ores in required bulk and qualitative performance, forming of optimum ore and rock flows, supplying rise of useful reductants in a concentrate.

A temporarily non-working pit wall represents a tank circuit of open cast, which is reshaped inside an open pit field and Is a temporary boundary, up to which the mining operations in the defined operating periods are carried out. It Is eliminated at the approach of the sustainable tendency of an abatement o*" stripping bulks and cutting of an excavation front. Time and the bulks of an electrode spacing of temporarily non-working boards are determined by concrete accounts in the design or long-term planning of mining operations.

In some cases, as the temporarily non-working board becomes a pit wall or its separate lease, formed by the way of a final board according to the project, acting on mat moment, and which by virtue of new accepted solutions embodied in the new project, envisioning the extension of open pit boundaries has appeared inside the new boundaries of an open pit field and is considered in it as the temporarily non-working board.

In design, the temporarily non-working board represents a stepping surface derived by collection of acclivities of benches and berms (transport, periodic clearing, safety). The arguments for a temporarily non-working board should meet the requirements of stability and mine safety in their formation and improvement. The lean angles of temporarily non-working boards, depending on concrete geological conditions, deformed states of rock in board massifs and periods of existence of boards, are usually in limits from lean angle of a working pit wall up to the lean angle of a pit wall in its end position. The steeper the angle of its slope, the better the performance of its application. The acclivities of benches on non-working boards, as their design element should have for a standing period of one year a stability margin of 1.1-1.2. For 20 years, this should be 1.2-1.5, and for more than 20 years, It should be 3 or more.

The number and width of berms in a temporarily non-working pit wall depends on the system and opening schemes concrete and from opening up of working levels, arguments of the mining method, and the type of applied open cut transport. The width of safety berms and periodic clearing is regulated by "uniform safety rules at digging of deposits by an opened way". For supply of the uniform mode of mining operations and uninterrupted operation of open cast in ore mining when applying temporarily non-working boards, it is necessary to provide" sufficient penetration feed of mining operations in the moving of this board.

Temporarily non-working boards in open casts have found a use as one of the tools of regulation of the mode of stripping, decrease of peaked bulks of overburden and carrying them on later periods at mining deep seated acclinal, steeply dipping and unformed deposits by sequential queues (Bukeikhanov, 1991).

The technology of improvement of open cast with temporary conservation of boards in the area of the former USSR has continued for more than 50 years. The idea of separation of open cast improvement on stages for the first time was put forward by the designers of the Unipromedy Institute and is applied in practice m the designing of deep open cast mining. In 1945, the scheme developement by stages proposed for Sibay open cast was accepted in the project and is carried out in practice. Then, in different years mining by stages with temporarily non-working boards was applied at the Kovdorski, Nikolaevski, Jitikarinski, Korshunovski, Krivorojski, Kalymakirski and Bajenovski open casts.

Known examples of the use of temporarily non-working boards outside the former USSR are listed below.

The Kessiay open cast (Canada) is divided into nine stages of development. The altitude of the temporarily non-working board is 300-320 meters with an opencast depth of 490 meters.

The planning of mining operations at once in outlines of stages is carried out on the basis of an inadmissibility of termination or decrease of bulks of mining at transferring from stage to stage. The number of simultaneously fulfilled benches and width of safety platforms depend on the output of the excavators. The working area is divided into leases. While the bench of a lease is working, the others are in temporary conservation. The angles of the acclivities of temporarily non-working boards, as well as practically on all foreign opencasts, which are carried out In stages, are close to the angles of the acclivities of the final boards.

At the Kacharski open cast of the "Sokolovsko-Sarbaiski dressing manufacturing association" use of temporarily non-working boards for regulation of the mode of stripping Is stipulated. The project capacity of open cast on ore in 2000 has increased 8.0 million tonnes and on overburden is 21 million m^3. An exit on total output In 15 million tonnes is envisaged in 2020. The mining method is one-onboard with transport of empty rock mass on external dump by a combined auto-railway transport.

At present, the open cast has reached a size of 2190 meters in diameter. The depth of the open cast for 1.01.2000 has increased 282 meters. The western board during 2100 meters is set in an end position up to a mark of-427 m. (63 meters in altitude).

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prognosis of supply and demand on firm's commodity and prices for it, and also on consumable materials in markets;

- contmuous mining of ores in required bulk and qualitative performance, forming of optimum ore and rock flows, supplying rise of useful reductants in a concentrate.

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In some cases, as the temporarily non-working board becomes a pit wall or its separate lease, formed by the way of a final board according to the project, acting on that moment, and which by virtue of new accepted solutions embodied in the new project, envisioning the extension of open pit boundaries has appeared inside the new boundaries of an open pit field and is considered in it as the temporarily non-working board.

. In design, the temporarily non-working board represents a stepping surface derived by collection of acclivities of benches and benns (transport, periodic clearing, safety). The arguments for a tempo^- rarily non-working board should meet the requirements of stability and mine safety in their formation and improvement. The lean angles of temporarily non-working boards, depending on concrete geological conditions, deformed states of rock in board massifs and periods of existence of boards, are usually in limits from lean angle of a working pit wall up to the lean angle of a pit wall in its end position. The steeper the angle of its slope, the better the performance of its application. The acclivities of benches on non-working boards, as their design element should have for a standing period of one year a stability margin of 1.1-1.2. For 20 years, this should be 1.2-1.5, and for more than 20 years, it should be 3 or more.

The number and width of berms in a temporarily non-working pit wall depends on the system and opening schemes concrete and from opening up of working levels, arguments of the mining method, and the type of applied open cut transport. The width of safety berms and periodic clearing is regulated by "uniform safety rules at digging of deposits by an opened way". For supply of the uniform mode of mining operations and uninterrupted operation of open cast in ore mining when applying temporarily non-working boards, it is necessary to provide suffi-

cient penetration feed of mining operations in the moving of this board.

Temporarily non-working boards in open casts have found a use as one of the tools of regulation of the mode of stripping, decrease of peaked bulks of overburden and carrying them on later periods at mining deep seated acclinal, steeply dipping and unformed deposits by sequential queues (Bukeikhanov, 1991).

The technology of improvement of open cast with temporary conservation of boards in the area of the former USSR has continued for more than 50 years. The idea of separation of open cast improvement on stages for the first time was put forward by the designers of the Unipromedy Institute and is applied in practice in the designing of deep open cast mining. In 1945, the scheme developement by stages proposed for Sİbay open cast was accepted in the project and is carried out in practice. Then, in different years mining by stages with temporarily non-working boards was applied at the Kovdorski, Nikolaevski, Jitİkarinskİ, Korshunovski, Krivorojski, Kalymakirski and Bajenovski open casts.

Known examples of the use of temporarily non-working boards outside the former USSR are listed below.

The Kessiay open cast (Canada) is divided into nine stages of development. The altitude of the temporarily non-working board İs 300-320 meters with an opencast depth of 490 meters.

The planning of mining operations at once in outlines of stages is carried out on the basis of an inadmissibility of termination or decrease of bulks of mining at transferring from stage to stage. The number of simultaneously fulfilled benches and width of safety platforms depend on the output of the excavators. The working area is divided into leases. While the bench of a lease is working, the others are in temporary conservation. The angles of the acclivities of temporarily non-working boards, as well as practically on all foreign opencasts, which are carried out in stages, are close to the angles of the acclivities of the final boards.

At the Kacharski open cast of the "Sokolovsko-Sarbaiski dressing manufacturing association" use of temporarily non-working boards for regulation of the mode of stripping is stipulated. The project capacity of open cast on ore in 2000 has increased 8.0 million tonnes and on overburden is 21 million m^3. An exit on total output in 15 million tonnes is envisaged in 2020. The mining method İs one-onboard with transport of empty rock mass on external dump by a combined auto-railway transport.

At present, the open cast has reached a size of 2190 meters in diameter. The depth of the open cast for 1.01.2000 has increased 282 meters. The western board during 2100 meters is set in an end position up to a mark of-427 m. (63 meters in altitude).

board {from the upper horizon up to a safety berm with the broken-up rockmass arbor inclusively) ceases with the beginning of delivering of the upper horizon. From the limits of a delivered lease of the temporarily non-working board all machinery is removed and access for servicers İs forbidden.

Other methods of regulation of the mode of stripping are small advanced open casts, which allow in short times preparation of the front of mining operations on ore and to start winning operations, and, on the other hand, they can provide forming of padding front of mining operations on overburden. With the use of small open casts in an open pit field, die working area of main opencast and small advanced opencasts is reshaped, i.e., not one working area operates, but a few, which subsequently merge partially or completely. In some cases, the small open casts can be laid out in the limits of a working area of open cast and intensify mining operations on its particular leases. The small advanced open casts, as one of the effective methods of regulation of the mode of mining operations, are applied for development of disconnected deposits in large and deep open casts with the purpose of the creation of a padding front of operations both in mining, and on an overburden and rising of their intensity, and also rising of a complete withdrawal of mineral resources from entrails and decrease of their dilution, if the ore bodies have pitch angles with different directions. Thus, between main and small advanced open cast, the pillars, folded, as a rule, by overburden rock mass are reshaped. They are reshaped in the limits of boundaries of an open pit field and consist of a small number of benches, and also supply the transport link of working levels with a transportation network of open cast, uncovering and preforming developments. The small advanced open casts are usually laid out along the contact of conditioned ores with overburden rock mass on the part of hanging walls of the deposit. The lean angles of working boards and on small advanced open casts can be differenL As a rule, they are from 16° to 30°-45°.

In the planning of mining operations at Sokolovski open cast, for quick input in working off of ores with veins and regulation of the mode of stripping, the creation of advanced small opencast is proposed.

As part of the authorized project (SSGPO, 2000), the output of Sokolovski opencast on ore till 2005 is on level 3 million tons. The ores with veins began development in 1998, as padding to magnetite up to a level 3 million tons. In the project, exploitation of the new reserves of magnetite ores will begin in 2007. The maximum output from these ores accepted by the project is 2 m tons. The complete damping of mining operations İs planned in 2026.

At present, the mining of magnetite ores and extraction of rocky overburden rock mass is conducted from a southern (Main) lease of open cast. Thus, the large part of overburden rock mass is located at an internal automobile dump located in the northern part of the open cast.

Simultaneously with it delivering of East bead with deleting of soils of a friable overburden and mining of ores with veins of ores is prolonged. In a delivering zone, the open cast has achieved the final outlines up to level -125 m., southern and southeast up to level -200m (375m from a daylight area).

The communal depth of opencast to the end of 2000 has reached 420 m (the mark of bottom - 245m). The dimensions of the open cast on the surface are: length - 3400m, width - 2000m.

Expansion of the communal front of mining operations İS 13.2 km, and including on ore - 1.0 km.

Expansion of the fissile front of operations is 6.5 kms, including on ore - 0.7 kms.

The transport of mined rock from the working faces of the east board is carried out by railway transport, and from the working faces of the main open cast by a combined auto-railway transport.

At Sokolovski iron ore open cast at planning of mining operations in the period 2001-2005, the problem on creation of small advanced opencast, which is reshaped İn opencast of a northwest part of opencast, is reviewed. Its creation provides quick access to the ore deposit production of winning operations, without the working off of all strata of overberden rock mass (-35m) / (-80m) on the part of the main opencast and magnification as ore - on 900m, and of overburden - on 1200m fronts. (Fig. 2.)

Thus, the experience of designing, planning and handling of mining operations at Kazakhstan opencasts has shown that the most effective tool of regulation by stripping works and supply of regularity of winning operations when mining large opencasts fulfilling a steeply dipping field are the of creation in them of temporarily non-working boards and advanced small opencasts. It is necessary to allow for these positions İn the creation of systems of design automat projecting and planning of mining-transport systems in deep opencasts (Bukeikhanov et. al., 1989).

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Forming of temporary non working board (TNWB) to end position

Completion of the forming of TNWB to end position

Forming of TNWB to the intermediate position

-Bulks of rockmass operated at the I stage

Real position of mining operations

Position of mining operations at the forming process of TNWB

Position of mining operations at the formed TNWB

Bulks of rockmass operated at the II stage

I stage

II stage

I stage

Figure 1 The arguments of TNWB

Ore body

advanced small opencast

2001 year
2002 year
2003 year
2004 year
2005 roa
2005 year

overburden rock transmited on the
latest period of mining operations
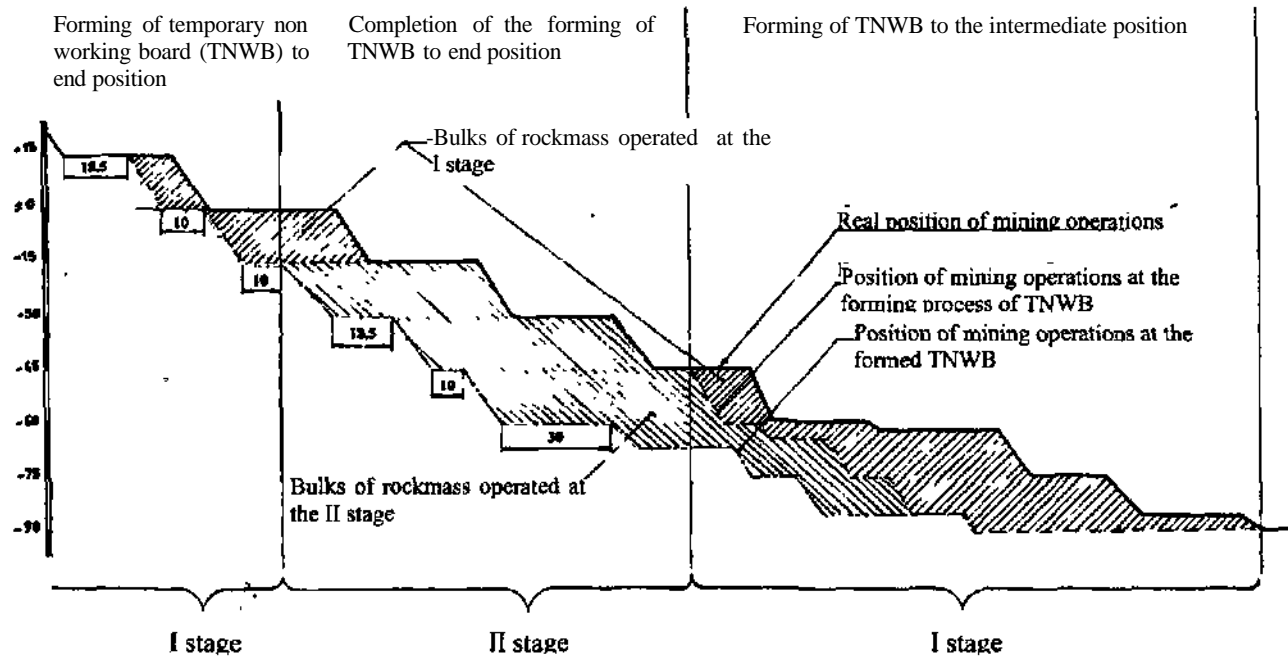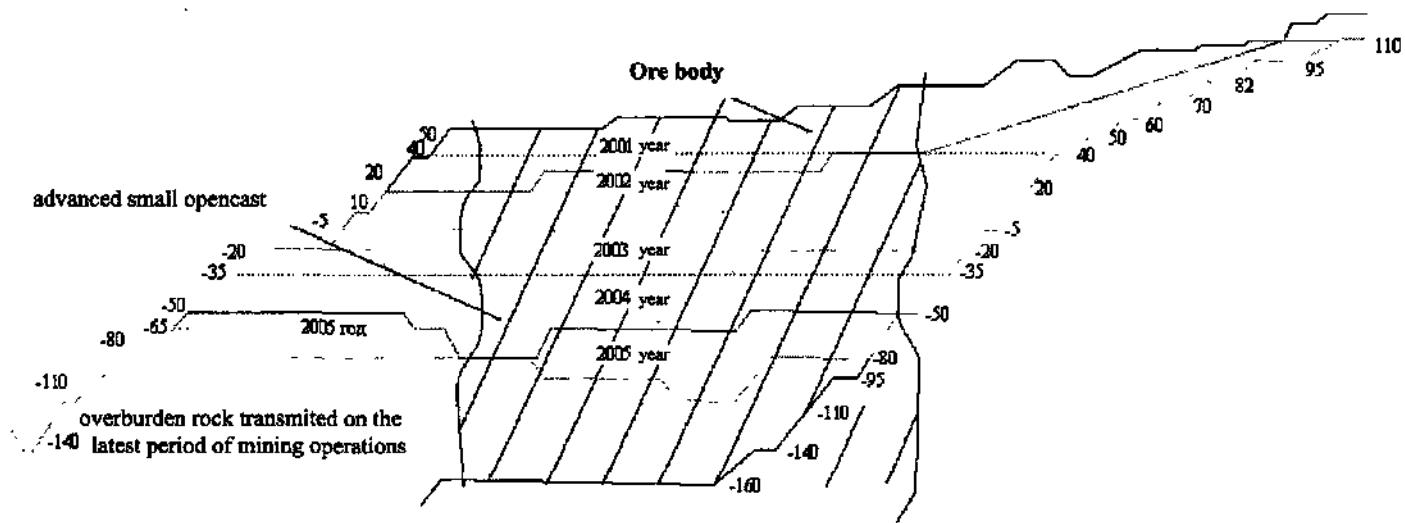
Figure 2. Fonning of advanced small opencast at mining of Sokolovslci opencast.

The History of the Evolution of Salt Working Methods in Romania, from Antiquity to the Present

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ABSTRACT: The presence of many salt massifs has favoured the appearance and development in the area of continuous mining activity. This has generated an evolution in exploitation methods in the following order: exploitation of the wood civilisation from the Dacian-Roman period, the bell type, systematic exploitation in rooms and small pillars and exploitation in solution.

1 INTRODUCTION

A review of documents and old works, as well as the study of many salt exploitations, some of which are very old, allow any attentive researcher to come up with new ideas regarding salt exploitation in this region from ancient times to the present. In this study, we focus on the development of exploitation methods used over thousands of years. Salt exploitation in the Romanian Carpathian area has been possible due to the presence of about 200 salt massifs with special characteristics: some are in outcrop or are very close to the surface, and the majority have superior resources of NaCl, > 97-98%, some of them with large reserves: Ocnele Man (Bozasca), 9,200 mil t; Reghiu-Andreiasu, 13,250 mil t; Sic, 18,300 mil t; Ocna Mures, 23,500 mil t; Turda, 38,750 mil t; Praid, 50,000 mil t; Ocna Sibiului, 61,000 mil t; and Sarmasel, 100,000 mil t. As a result of our studies, we have managed to identify for the first time 46 salt massifs under exploitation (Fig. 1), out of 200 which exist, and as part" of the massifs we have found traces of documents which certify the existence of many former exploitations, from antiquity to the present

2 METHODS OF EXPLOITATION

We ordered the methods of salt exploitation as follows: the wood civilization exploitations from the Dacian-Roman period, bell exploitations, systematic exploitations, and exploitations with rooms and small pillars and in solution.

2.1 *Methods of wood civilization exploitation*

In valleys and ravines, through the washing of salt massifs by aquifers, areas of waterlogging with salty

efflorescence often appeared. Through ordinary evaporation, either in situ or in clay or wooden pots, different quantities of salt were obtained. Near Ocnele Mari in citadel 1 (Cotofeni Culture), thousands of tronconic "glasses" specially made for the evaporation of brine were discovered. In salt valleys, salt outcrops often appeared, especially due to landslides and the erosion due to rain. When they could not break salt rocks on the surface, people had to dig through rudimentary holes using wooden wedges and hammers. Depending on the digging depth, the salt was carried on the back to the surface, or a human chain was formed and the salt blocks were passed from hand to hand. As the salt was very heavy, sometimes a sort of sledge pulled by men or horses was used. The main problem was water infiltration, against which they used linings of animal skins and sticks with clay on walls and ceilings. Some researchers have revealed very useful information about this. From Wollmann (1996), we have Figures 2, 3 and 4; after Berciu, from the Verbicioara culture (1800-1300 BC), heavy axes used for <u>mining</u> exploitation and the breaking of salt blocks in the area of Dacian Buridavei (near Ocnele Mari) have been preserved. Parvan reported that a hammer, a pickaxe (Pickel) with a polygonal profile and gathering "ailerons" (together with more chisels in the form of axes) were found at Gusterita, which Goos compared with an analogous object found in a salt mine from Hallstatt to which he attributed the same usage. It was dated as being from 900-500 BC. In the area of the salt massifs, native tombs with cremation urns were found. At Uioara de Sus (globated at Ocna Mures), seven clearly outlined tombs were found - cremation "in the urn" from the VI-V centuries BC belonging to the Agatarsi Scythians. At Ocnita-Teaca, Bistrita Nasaud, a bronze mirror was found with the head of a ram at the end of the

handle and six more arrows. In 1871, 1846 and 1847, at Ocna Slatina in "the King's Valley", traces of Dacian exploitations (Bronze Age, probably die 2nd Iron Age) were found. These had irregular outlines with a depth of 10-13 ra, a diameter of approximately 13 m and a height of excavation which rarely exceeded 5 m. It is interesting that such exploitation was equipped with a "tank" dug in the salt used to collect waters from infiltration. Some

wooden tools and jude ropes have been found, which leads to the conclusion that salt was brought up through a system of levers and pulleys. At Valea Florilor, some hand mills and other household objecte have been found. At Dacic Ocnita-Buridava feathers of a hammer and some tools specific to salt exploitation dating from the 1st century BC have been found.
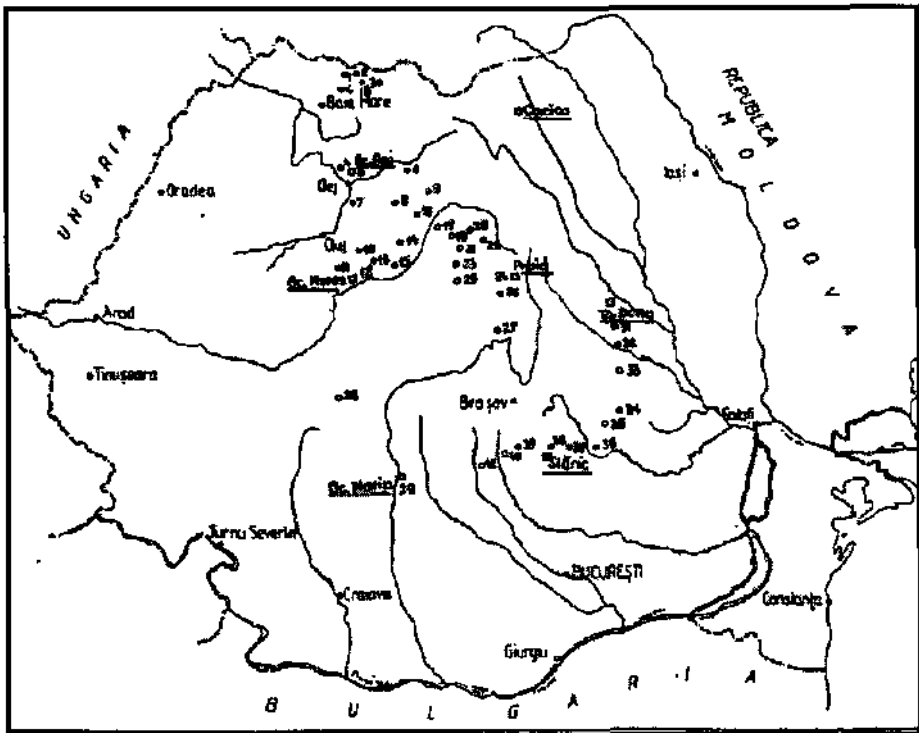

Figure 1. Ancient mines: 1. Ocna Slatina 2. Cojocna 3. Sic 4. Rogna 5. Ocna Dej 6. Saratel 7. Sic 8. Ciceu 9. Iliusa 10. Manastur 11. Turda 12. Ocna Mures B.Domnesti 14. CaianuMare 15. Cuzdioara 16. Vireag 17. Releag 18. OcnaSugatag 19. Sintereag20. Chiua 21. Tau 22.Beclean 23, Sovala 24. Mogosmat 25. Martinis 26. Sanpaul 27. Rupea 28. Ocna Sibiului 29. Boizasca 30. Rcma de Sus Middle Ape mines: 31. Grozesti 32. Nord Vaiea Sarii - Sud Valea Sarii 33. Reghiu - Andreiasu 34. Bisoca(Picineaza)35. Rusavat (Sarea lui Buzau)36. Aricesti 37. Ghitloara (AnInisul-Sarani) 38. Teisani 39. Telega 40. Doftana41. Baicoi Actual mines: Cacica, Tg. Ocna, Slanic Prahova. Ocnele Mari, Ocna Mures, Ocna Dej, Praid.

2.2 Methods of exploitation from the Daco-Roman period

It is generally considered that salt mining developed during this period through the appearance of new exploitations, through the enlargement of die resulting gaps and through the memods used. There were two methods of salt exploitation: the semi-dried method and the dried method.

2.2.1 The semi-dried method

In this memod, salt was obtained through a procedure which allowed the rocks to be separated from the salt blocks by hitting with a hammer in tracks created as a result of dissolution by water brought through a system of eaves from the surface. In the free Dacians from Ocna Slatina as well as in the area occupied by the Romans at Valea Florilor and Ocna Dej, similar wooden eaves have been found. These were assembled at 0.3 m from the surface up to a

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maximum of 10 m in depth on wooden supports and had the role of ensuring water transport from the surface to the working front. For the assurance of the water flow or for raising it, the Dacian Romans used "hydraulic wheels". The eaves were preserved from place to place with pearces plugs, through which there were unlignified wires of lime-tree trunks. By moving these eaves and by closing or opening the plugs, the water spurt could by led to somewhere away from the exploitation. At the head of the eaves. Hat eaves provided with plugs of water distribution were often used. Through dissolution, the water created a ditch in the desired direction. As matter of fact, this was a vertical and horizontal undercutting. The water was collected in a concave area and was evacuated. Through hitting with hammers in the feathers and with the help of crowbars, the salt was detached in blocks. In 1902. some wooden tools, which could even originate from before the Romans arrived in Dacia, were found at an ancient exploitation at Turda. Among them, a shovel with a short handle of 46x 13 cm used in the scraping of salt obtained through evaporation was found.
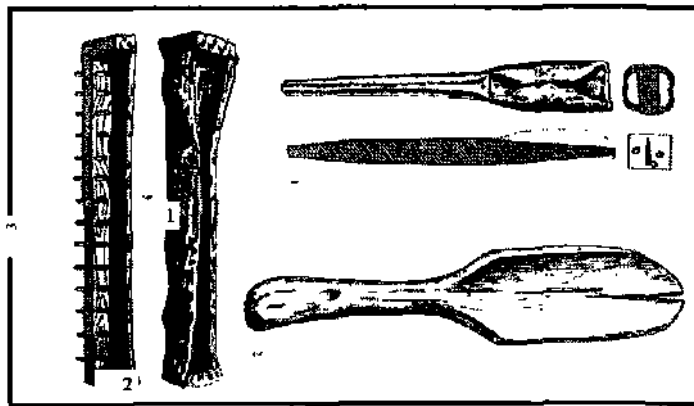
Figure 2. Tools and installation dating from the wood civilization. 1. Sharp pickaxe of the "Lappeinbein"-Spalnaca (Ocna Mures) 2. Wooden shovel - Turda 3. Trough with orifices for undercutting salt in fresh water.
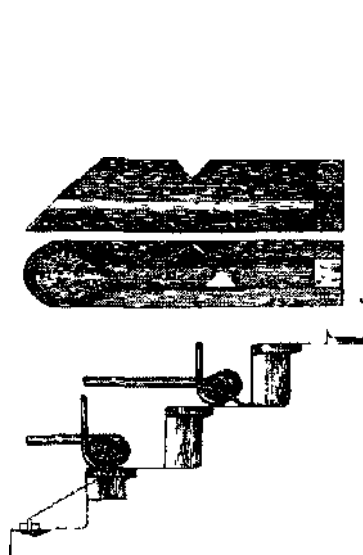

Figure 3-The component parts of on instulliition of conducting fresh water tor undercutting salt - Ocna Slutina.
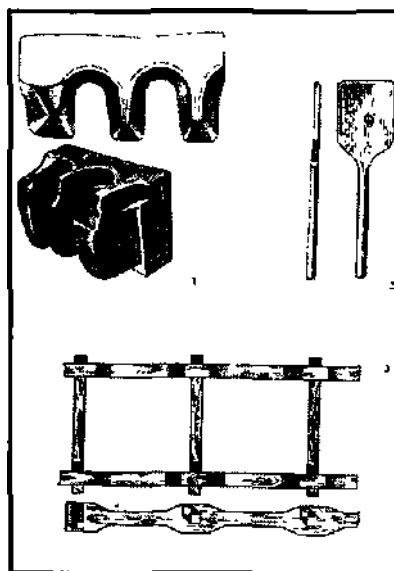

Figure 4. The component parts of on installation of conducting fresh water for undercutting salt - Ocna Slatina.

2.2.2 The dried method

In the second method, rooms of well-defined dimensions, lengths of 15-30 m and widths of 4-8 m, were obtained. One piece of proof is the exploitation from the surface with square sections (7 x 7 m) provided with reinforcements of moulded beech or oak. Salt exploitation was carried out by descending from the surface, detaching being done only in the sole. For üiis, iron hammers were used for the creation of tracks in the exploitation sole, after which through the utilisation of feathers and crowbars salt blocks were cleaved. The access to the exploitation sole, as well as the bringing of the salt to the surface, was accomplished with stairs and ropes. Aeration was through natural circulation, and the lighting was also natural, creating a semi-dark atmosphere. At the surface, a roof was made for protection from rain and snowfall. Around this, a system of drainage was built with ditches. When they could not evacuate the water infiltrating the exploitation, it was abandoned and another one was opened in the nearby area, at a distance of 15-20 m. In order to support what has been said before, we can turn to ancient texts, and to epigraphic and archeological sources. From ancient text, it follows that near Potaissa (Turda) was the city of Salinea, where "the Salt Headquarters-Collegium salinariorum" was, and at Sarateni was the "castrum salis". Of the epigraphic sources, only the epigraph from Domnesti dedicated to the health of Aelius Marus can be mentioned - "conductor pascui et saünarum", the inscription, near Sanpaul, from the village of Martinis, dedicated to the health of C. Iul. Valentinus c(onducator) sahnar(um). Soldiers were stationed at Micia-Vetel with P. Ael. Euforus - salinary agent. Of the archaeological sources, there are roman coins, die votive altar worshipping Jupiter and Terra Mater from Domnesti, and bricks with cohort signs from Ocnele Mari, Buridava, etc.

2.3 The bell-type exploitation method

Up to the 17th century, the exploitation technique for salt in Romania did not progress significantly. The first attempt at standardization of the exploitation and administration of salt mines was to be completed by the Austrian mountain forum, by the detachment from the metalliferous exploitation of some topographers, who drew the first mining maps. These present the geometrical form of the intracarpatic mines - conical excavation - regularly called the bell-type exploitation (Figure 5). It seems that this system dates back to the 14th century, but the information is not conclusive. However, the last exploitation in which the digging system of simple surface holes was abandoned and the use of the bell-type exploitation was abandoned was at Praid in 1780. The mining technique consists at first of the making of some fairly primitive boreholes with depths of 10-15 m. If salt was found, the next step was the making of two wells wim a section of approximately 1.8 x 2 ra, up to 4-5-m depths in the salt layer. From this level, they were enlarged to 4-6 m at a depth of 2-3 m. In order to prevent water from entering, the back of die key signature was filled with buffalo skin, clay, wool or chaff. After arrangement of these wells, the excavation continued in a conical shape, up to approximately 8 m in depth, where the junction with the other well occurred and the exploitation was given a conical shape. From the second part of the 18th century, the use of four wells for the mine opening began. In the exploitation process, the purpose was to detach the salt in blocks which could be transported over long distances and for which a lot of money was paid. They did not have much use for small salt. In diese exploitations, pointed-end hammers were used to separate tracks and detach beds. The beds were 2-5 m in length, 0.6-1.5 m in width and 0.3-0.5m İn depth. This size varied according to the bed inclination, the thickness of the white or black salt bed and the bed's position İn the required outline. After die creation of a track 15-20 cm in widtii with an approximate small-base trapezoidal outline, the hammer edge was introduced at a distance of 5-10 cm from one another for detachment of the bed. For detachment of a bed approximately 2 m in length, 40-50 hammer edges were used. After detachment of the bed, which was sometimes difficult, breaking again occurred with hammer edges and throygh lulling 20-30 kg balls were obtained. The salt balls were transported by hand, or by barrow up to a farding bag moved up and down by a pulley called a 'crivac', as used at the exploitations of Slanic Prahova, Telega, Doftana, Ocnele Man, Turda, Ocna Dej, and Ocna Sibiului. It was made of a rope at the end of which a platform from bar guzly buffalo skin, solid nets on which the salt was put. The rope wrapped a cylindric tambour fixed on a principal vertical axis, in such a way tiiat during the rotation of the tambour axis, one end of the rope to rose and the other descended into the mine. The tambour was put in more with the help of four or six 'tovele\ at die end of which a pair of horses were usually harnessed. Following white salt zones, most of the time steps of 4-6-m exploitations remained on which people climbed and descended on planks equipped with transversal laths in order to prevent slipping. These exploitations reached 96 ra in depth in Ocna from Slanic Hill Prahova, 145m in depth in Ocna from Slanic Valley Prahova, 147.5 m in Ocna Cojocna, 152 m in Ocna Superioara, Turda, and 160 m Ocna Mare and Sibiu. Aeration occurred by natural circulation, and sometimes fresh air was also introduced in buffalo skins. When the air was no longer breathable, that is, when the candles that provided the light went out, straw was burnt at the opening of the salt mine. In 1739, a manual air hole was introduced; it was a kind of turbine introduced in a wooden frame that was used at the exploitation.
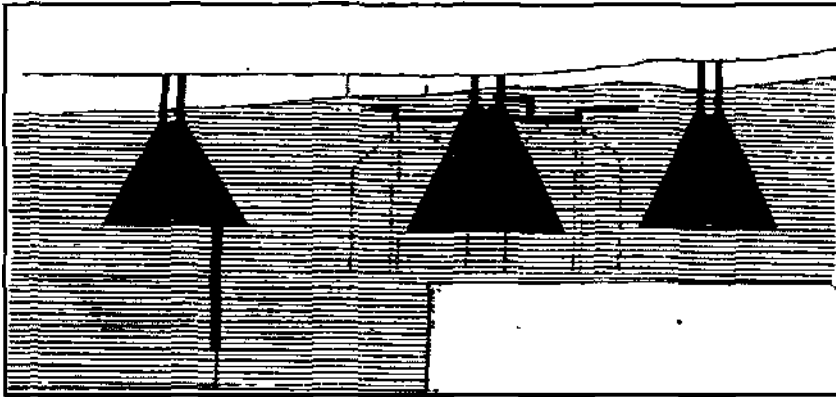
Figure 5. Section through the mine of Turda

2.4 *The systematic exploitation method*

The exploitation methods developed from the conical excavation type, with a single room, to a system with several connected rooms, called systematic rooms. Using the information found, the following chronology of the development of this method can be-made: from Vielicyka (Poland), this method passed to mines in Maramures Costiui, Sugatag, Ocna Slatina (1777, at the proposal of the engineer Iosef Grosychimed), then to Ocna Sibiului, then to Ocnele Mar (1845, by the Austrian engineer Foit) Slanic (I860), Doftana (1865), and Targu Ocna (1870). The advantage of this method was that an exploitation field with a larger surface could be created, and a larger quantity of salt could be extracted through work of the same extension. Within this method, two variants can be distinguished: the first one retains the transversal section with a bell appearance, and the second one has a trapezoidal aspect of the upper part of the exploitation rooms.

2.4.1 *The systematic exploitation method with the aspect of a bell*

The first systematic mine were realized by separating directional and transversal galleries through salt massifs. After some short soundings, descending digging of the exploitation rooms took place through the same system of carving and detaching of the furrow, except that the walls of the rooms had had a tilted and plain aspect. Through this exploitation system, working conditions were improved, aeration could be ensured through natural circulation, and through galleries and wells over long distances. More workers were used. At Slanic there were 160-220 'ciocanasi', that is, those who cut the salt, and there were 180-220 'maglasi' those who worked by
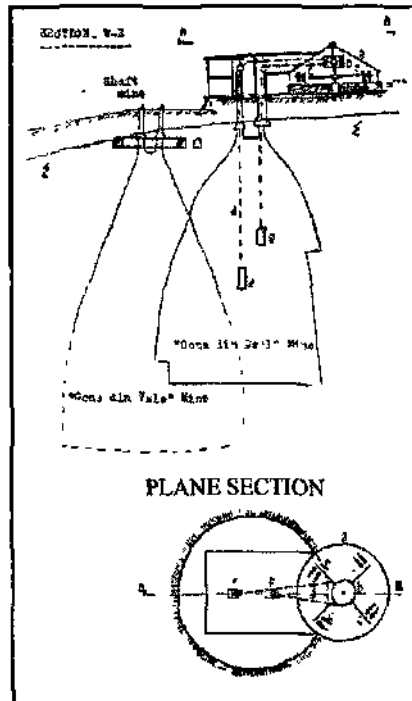
Figure 6. The section N-S in the old mine at Slanic Prahova.a)building of'crivac" b)cylinderc) hors, d) cable, e) hamper ,0 shaft

hand. At the salt mines free men were used as well as prisoners. Mechanical ventilation was introduced for the first time in a salt mine in Slanic, at the Systematica mine by the engineer Carol Craciunul (Caracioni) İn 1854. Electric current Was introduced in Slanic İn 1883, and was used on a large scale from 1910.
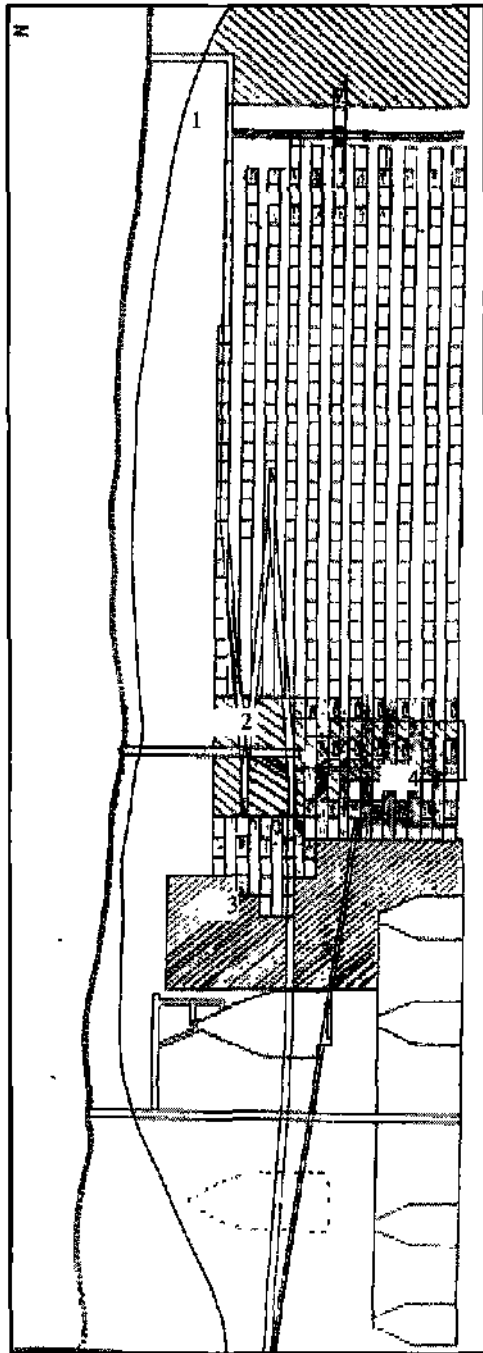
Figure 7. The section N-S in the old mines at Slanic Prahova (1-Victoria Mine ,2- Carol Mine. 3-Mihai Mine and 4-Unirea Mine).

2.4.2 The systematic exploitation method with ceiling for trapezoidal rooms

The person who conceived this method was the engineer Stamatiu Mihai. He designed a system of exploitation at Slanic with a central pillar surrounded by rooms and pillars at 150m and under ancient bell-like exploitation and systematic with a bell aspect (Carol mine and Mihai mine, Figure 7); after constructing climbing parts with a compartment for stairs made of wood the ceiling galleries of future rooms were opened.

They initially had a section opening of 4 x 2.5 m, but were enlarged to 10 x 2.5 m. During digging, haveye were used, and wooden balconies were installed laterally in the salt. The exploitation of the work fronts was carried out in the order: undercutting, perforation, loading with explosives of the holes obtained after perforation, the blasting of the holes, either pyrotechnically or electrically, the loading of the salt in trolleys, transport underground and the extraction of the salt in the well to the surface. For lighting, bulbs of 300 W were used and in the 1960s these were replaced with fluorescent tubes. On the surface, exploitations gained an industrial aspect through the building of annexe maintenance workshops, pipe stations, the building of the modification of locomotives, warehouses of ground salt or clods, of packaging for coal, cement, and clay, cloakrooms, canteens and offices. Mine working has diversified, and new jobs such as havators, perforators, pyrotechnists, electricians, mine locksmith engineers, locomotive mechanics, undercutters, etc. have appeared- Durihg the operation of systematic exploitations, progress was made from obtaining a single type of salt balls to obtaining several other types of salt. The exploitation of subterranean surfaces has substantially expanded: only the Unirea mine in Slanic Prahova, closed down in 1970, had a room surface of 72,000 mp outlined by 15 rooms with a 35-m opening, a ceiling of 60 degrees, and 11 pillars with a 50-m side line. The height of the room was 55 m.

2.5 The method of exploitation with rooms and small pillars

After hundreds of years of exploitation by the bell-type system and after about 200 years of exploitation with the systematic type system, the exploitation of salt passed to a multi-flood system with rooms and smal pillars (Slanic Prahova, Tg. Ocna, Ocna dej, and Ocnele Man). The method was tested for the first time in Romania at a pilot mine in Tg. Ocna between the September 1966 and 1968, when two floors were opened. The sizes of the rooms and the pillars were adapted to each massif. The biggest mine opened in this system was the Victoria mine in Slanic Prahova (1971-1992), which comprised a

multi-floored structure of 11 floors. The sizes of the supporting pillars were 14 x 14 m, 15 x 15 m, 36 x 16 m, 17.5 x 17.5 m and the heights were 8 m. The ceilings between the floors were 7 and 8 m The opening of such an exploitation was accomplished creating two wells of ventilation, an auto-inclined plan and ventilation galleries. The exploitation of the rooms was İn the following order: undercutting, perforation, loading the holes with explosive, blasting of the ventilation fronts, loading the salt from stopes by means of transport, transport to the underground crusher equipped with a somersault, crushing of the salt underground and its transport an a relay of converger lanes to the surface.

2.6 *Methods of exploitation in solution*

For massifs which had tradition and prospects for the chlorine sodium industry and which had difficult geological conditions in the deposit, in the 20th century the derricks exploitation was used. One advantage of this method was the fact that areas of the massif with low qualitative parameters could be exploited, and sterile elements, most of them insoluble (calcium and sodium sulphates) remained as sediment in the dissolution rooms. Salt in solution İs

exploited today at Ocna Mures, Ocnele Mari, Tg. Ocna and Cacica.

3 CONCLUSIONS

According to the study, it was found that 46 salt massifs had the geological, technical, economical and social conditions for mining activity. An untold history has built up around them over the years. The present study is the starting point for anyone wishing to broaden global salt research in this area.

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Salt in Turkey

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ABSTRACT: NaCl salt, which is an important substance in the chemical industry and in food preparation, is produced from four main sources: lakes, seas, salt rock and underground salt water. Salt from lakes has the lowest cost and is of high quality. Turkey is surrounded by the sea in three directions, and salt production from the sea is gaining in importance, while production from salt rock is decreasing because of high production costs and other conditions. Turkey has sufficient production capacity in its seas. In this paper, the salt potential of Turkey is evaluated.

1 INTRODUCTION

Sodium chloride or common salt is the chemical compound NaCl. Salt occurs naturally İn many parts of the world as the mineral halite and as mixed evaporites in salt lakes. Seawater has a lot of salt; it contains an average of 2.6% (by weight) NaCl, or 26 metric million tons per cubic kilometer (120 million short tons per cubic mile), an inexhaustible supply. Underground salt deposits are found in both bedded sedimentary layers and domal deposits. Some salt is found on the surface as the dried-up residue of ancient seas, like the famous Bonneville Salt Flats in Utah. Salt even reaches earth from space.

Sodium chloride crystals are cubic in form. Table salt, when viewed under a magnifying glass, can be seen to consist of tiny cubes tightly bound together. The salt crystal is often used as an example of crystalline structure and many online science pages offer instruction on growing salt crystals. Salt crystals have been photographed under a microscope. Different types of crystal have different uses, such as in food (Anon., 2000).

It varies in color from colorless when pure, to white, gray or brownish, typical of rock salt (halite). Chemically, it is 60.663% elemental chlorine (CI) and 36.337% sodium (Na). The atomic weight of elemental chlorine is 35.4527 and that of sodium is 22.989768 (Anon.,2000).

Sodium chloride is sold in several different particle sizes (gradation) and forms, depending on the intended end use. Discrete crystals can be seen in rock salt used for deicing. Fine granules are typical of table salt and popcorn salt is even finer. Kosher salt, pickling salt and ice cream salt are slightly coarser. Small compressed pellets are used in water softeners and large salt blocks are used as salt licks for livestock. When viewed under strong magnification, all sodium chloride is crystalline. Very large cubic crystals, of two, three or more inches in size, can be seen in some salt mines. They are transparent and cleave into perfect cubes when struck with a hard object (Anon., 2001). The properties of salt are given in Table 1.

Table 1 Properties of salt.	
Chemical Properties	
Formula	NaCl
Atomic weight - Na	22.989768 (39.337%)
Atomic weight - CI	35.4527 (60.663%) ·
Eutectic composition	23.31% NaCl
Crystal form	Isometnc, cubic
Color	Clear white
Physical Properties	
Specific gravity	2 165 gr/cm
Hardness (Moh's Scale)	2.5
Critical humidity at 20 ˚C	75.3%
Melting point	800.8 ˚C
Boiling point	1,465 "C

2 SALT IN THE WORLD

2.1 *The global balance of salt*

The history of world salt production and consumption parallels the history of humankind. From primitive to modem times, every human being has had an association with salt Although not aware

of their physiological need for salt, prehistoric human beings obtained their salt primarily from the meat of the animals they hunted. These animals were often found congregating around salt springs or salt licks to satisfy their innate salt cravings. With the beginning of an agricultural society, humankind found the need to supplement vegetable and cereal diets with extra quantities of salt (Kostick, 1993).

The quest for salt became more important with the advance of civilization. Because of its solubility, surface deposits of salt were scarce, and new methods of obtaining salt were needed. This was the beginning of the world salt mining industry. The basic concepts of salt mining have changed very little since early times; only refinements in the techniques of extraction and processing have evolved (Kostick, 1993).

Of more than 170 nations in the world today, the Bureau collects or estimates production data from 98 different countries, which represents virtually all world output. In 1990, world salt production was 193 million tons, whereas, in 1980, world salt production was only 9 million tons. During the twentieth century, world population increased from 1.6 billion to 6 billion people. The increase in world salt production compared with population growth was significantly greater because new uses for salt were discovered that would change demand patterns.

Salt had been universally used for food flavoring and for food preservation, but it wasn't until die mid-19' century mat salt was regarded as an important raw material for die chemical industry. In the early 1860s in Belgium, the Solvay process was developed to make synthetic soda ash from salt. Later, salt became essential in the electrolytic process to make chlorine gas and sodium hydroxide. Today, salt is the largest mineral feedstock consumed by the world chemical industry (Kostick,1993).

2.2 *World Production Reserves and Reserve Base*

World resources of salt are practically unlimited. Almost every country in the world has salt deposits or solar evaporation operations of various sizes.

There are no economic substitutes or alternatives for salt. Calcium chloride and calcium magnesium acetate, hydrochloric acid and potassium chloride can be substituted for salt in deicing, certain chemical processes and food flavoring, but at a higher cost.

The world's salt reserves are large. Economic and subeconomic deposits of salt are substantial in principal salt-producing countries. The oceans comprise an inexhaustible supply of salt.

World production in 1998 and in 1999 is given in Table 2.

Table 2. World sail production

	Years (* 1000)	
	1998	1999'
United Stales	41,200	41,400
Australia	8,880	8,800
Brazil	5,500	5,700
Canada	13,300	13,400
China	30,800	31,000
France	7,000	7,100
Germany	15,700	15,200
India	9,500	9,500
Italy	3,600	3,600
Mexico	8.400	8,400
Poland	3,900	4,000
Russia	2,000	2.100
Spain	3,500	3,600
Ukraine	2,500	2,400
United Kingdom	6,600	6,600
Other Countries	23,600	37,200
World total (May be rounded)	186,000	200,000

estimated

3 SALT IN TURKEY

3.1 *Deposits*

Turkey has large salt deposits because of its geological structure. Rock salt deposits in Turkey are given in Table 3.

Although sea salt deposits in Turkey (surrounded by the sea in three directions) are unlimited, they depend on two sea salt facilities, which have a production capacity of more than 620,000 tons.

Table 3. Rock salt deposits (million tons).

Site of Facilities	City			
Sekili	Yozgat	107	359	300
Çankın	Çankırı	821	358	-
Gülşehir	Nevşehir	75	96	959
Tepesidelik	Kırşehir			20
Tuzluca	Kars			100
Kağızman	Kars			60
Oltu	Erzurum			30

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Lake salt deposits can be calculated with the simple method given below.

Surface of salt lake $= 1200$ km^2 (production area)
Salt thickness $= 8$ cm (3-20 cm)
Density $= 2.2$ t/m^3

From this, the deposit of the salt lake is approximately:

0.08 x 1200.10^6 x $2{,}2 = 210.10^6$ million   tons. However, according to research, the deposit of the lake is much larger than this.

3.2 *Production methods*

Salt production is generally carried out by one, or a combination of any of three methods.

3.2.1 *Solar evaporation*

Solar evaporation is the oldest method of salt production. It has been used since man first noticed salt crystals appearing in trapped pools of sea water. Its use is practical only in areas with warm climates in which the evaporation rate exceeds the precipitation rate, either annually or for extended periods, and ideally where there are steady prevailing winds (Calvinaco, 1990).

Solar salt production is the capturing of sea water in shallow ponds İn which the heat of the sun evaporates most of the water. The concentrated brine containing impurities is then discarded by mechanical harvesting machines. Two types of ponds are used. First is the concentrating pond. Here the salt water from the ocean or salt lake is settled and concentrated. The second is the crystallizing pond, in which the salt is actually produced.

Crystallizing ponds range from 15 to 90 acres with a foot-thick floor of salt that has been deposited over the years. During the salt-making season of four to five months, a saturated brine solution flows continuously through these ponds. There is as much salt in the water as it can hold, and, therefore, most of the pure salt crystallizes out of the solution as the water evaporates (Calvinaco, 1990).

The ponds are then completely drained. Mechanical harvesting machines gather the loose salt and carry it to piles.

Solar evaporation depends on certain factors.
- Large areas with small slope.
- Low salt water permeability.
- Low rainfall rate.
- High evaporation.
- Dry wind.
- Long summer season for evaporation (Calvinaco, 1990).

3.2.2 *Rock salt mining*

Salt mines are among the safest of mines and are also the most comfortable to work in. While mine temperature varies with depth, the average temperature remains about 70 degrees year-round.

Salt may appear in veins like coal. Veins are the original bedded salt deposits. Salt may also be found in domes. Salt domes are formed when earth pressures force salt up through cracks from depths as great as 9 or 12 km. These domes are roughly like plugs of an almost circular shape and are a few hundred meters to a kilometer across. Both domes and veins are mined in a similar way (Calvinaco, 1990).

Salt is mined by the room and pillar method. Salt is removed in a checkerboard pattern to leave permanent solid salt pillars for roof support. The room height may average from 3.5 meters in a bed, to 27 meters in a dome (Calvinaco, 1990).

The above-ground processing of the rock salt consists of sorting the mined salt into various marketable sizes by screening it over mechanically operated screens. When separated, each size is conveyed to its individual bulk storage bm to await packaging for shipment or to be loaded as bulk salt into railroad cars, trucks, river barges, or lake boats for *shipment to* customers (Calvinaco, 1990).

3.2.3 *Vacuum pan evaporation*

Another method of salt production used is the evaporation of salt bɪɪne by steam heat m large commercial evaporators. This method yields a very pure salt mat is fine in texture and is principally used in those applications for which quality is of primary importance.

The first part of the operation is known as solution mining. Two wells are sunk 150 to 300 meters apart into the salt vein. Water is forced down one well under great pressure and seeks the easiest way out, namely up me other well. Once the two wells are connected, the operation beings. Water is pumped down one well. The salt below is dissolved and the resulting salt brine is forced to the surface through the other well. It then goes into huge tanks for storage (Calvinaco, 1990).

Next, the brine is pumped into vacuum pans. These are huge closed vessels about three storeys high and normally İn a series of three, four, or five, with each one in the line under greater vacuum than the preceding one (Calvinaco, 1990).

!n the vacuum pan process, steam is fed to the first pan. This causes the brine in the pan to boil. The steam from the boiling brine is then used to heat the brine in the second pan. The pressure in the second pan is lower, allowing the steam made by the

boiling in the first pan to boil the brine in the second pan.

The pressure is reduced still further in each succeeding pan. This allows the steam made by the boiling brine in the previous pan to boil the brine in the next pan. The boiling operation could be done with just one pan. Several pans in a series, however, produce more salt per pound of steam, and, thus, are more energy efficient (Calvinaco, 1990).

The heating of brine actually takes place in the middle section of the vacuum pan. Two large metal sheets act as the top and bottom of this section. The sheets are connected by hundreds of copper tubes through which the brine can flow freely between top and bottom. The pan is filled with brine to just above the top of the tubes. Steam is fed around the tubes and is hot enough to keep the brine boiling.

In most pans, the brine goes up through the tubes and down through a well in the center. There, an agitator and natural circulation keep it flowing. Vigorously boiled and agitated brine produces salt in the shape of small cubes. The salt settles continuously into the narrow bottom of the pan. Here it is pulled off as slurry - a mixture of brine and salt. The slurry goes from the vacuum pans to the filter dryer, in which heating removes the moisture and dries the salt (Calvinaco, 1990).

3.3 Production

In Turkey, crude salt production is made from three sources, namely, the sea, salt lakes and rock salt. The production capacity of these salt facilities and production quantities In the East few years are given in Table 4 and Table 5.

Table 4. Salt production capacity in Turkey.

Place	Production Capacity	Rate
Kayacık	500,000	25
Kaldırım	400,000	19
Yavşan	400,000	19
Lake Total	1,300,000	63
ÇamalD	600,000	29
Ayvalık	20,000	1
Sea Total	620,000	30
Çankırı	60,000	3
Sekili	20,000	1
Tepesidelik	15,000	1
Tuzluca	20,000	1
Kağızman	12,000	1
Rock Salt Total	127.000	7
Others	21,000	1
Total	2,068,000	100

Table 5. Salt production (1998-1999).

Type of Production	1998	1999
ÇamaKı (Sea)	430,513	557,262
Ayvalık (Sea)	8,340	9,843
Sea Total	438,853	567,105
Kaldırım	458,196	483,728
Kayacık	597,055	404,145
Yavşan	562,750	490,564
Lake Total	1,618,001	1,378,437
Rock Total	93,931	89,377
Others	18,846	19,458
Total	2,196,631	2,169,631

3.4 Sales

The 1999 reported distribution of salt by major end use was, chemicals, ice control, distributors, general industrial, agricultural and food, and exports. The selling quantities from 1995 to 1999 are given in Table 6, and exported salt İs given in Table 7.

Table 6. Salt sales.

Years	Sales (Tons)
1995	1,619,000
1996	1,825,000
1997	1,999,000
1998	1,916,000
1999	1,850,000

Table 7. Salt exports of Turkey (1995-1999).

Years	Quantity (Tons)	Value (USD)
1995	3,418	21,000
1996	3,568	35,680
1997	13,021	156,240
1998	12,737	101,8%
1999	11,564	92,512

Although the quantity of exported salt is not sufficient, there has been progress in recent years. Turkey has exported salt to the Turkish Republic of Northern Cyprus, Bulgaria and Azerbaijan.

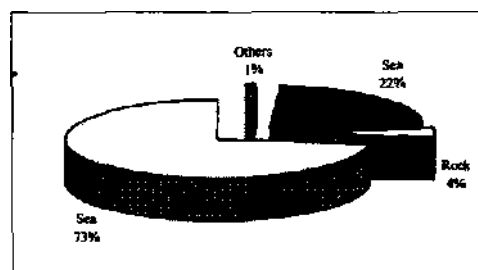Salt sales according to production area for 1999 are given in Figure 1.

Figure 1. Salt sales according to variety.

3.5 Consumption

Chemical: the greatest quantity of salt used in the chemical industry is by the chloralkali sector. Traditionally, the chloralkali sector included salt consumed for chlorine, coproduct sodium hydroxide, and synthetic soda ash. Chlorine and caustic soda are considered to be the first generation of products made from salt. These two chemicals are further used to manufacture other materials, which are considered to be the second generation of products from salt.

Food processing: every person uses some quantity of salt in their food. Salt is used in meat packing, canning and the dairy industry.

General industrial: the industrial uses of salt are diverse. They include, in descending order, oil and gas exploration, other industrial, textiles and dyeing, metal processing, pulp and paper, tanning and leather treatment, and rubber manufacture.

Agricultural industry: since prehistoric times, humankind has noticed that animals satisfy their hunger for salt by locating salt springs, salt licks, or playa lake salt crusts (Kostick, 1998).

Water treatment: commercial and residential water-softening units use salt to remove the ions causing hardness (Kostick, 1998).

Ice control and road stabilization: the second largest use of salt is for highway deicing. The developer of the Celsius temperature scale discovered that salt mixed with ice {at temperatures below the freezing point of water) creates a solution with a lower freezing point than water by itself (Kostick, 1998).

3.6 Quality of Salt

In obtaining high quality in sea salt production, the important thing is the cutting process and not letting other minerals down. However, for lake salt and rock salt, the salt naturally comes into existence itself. For this reason, İn these kinds of salt, people have little chance of determining the quality. Thus, the quality of lake and rock salt is better than sea salt and other types. The qualities of salt produced in Turkey are given in Table 8.

Table 8. Sail qualities.

Analyze	Kayacık	Kaldırım	Yavşan	Çamaltı	Çankırı
Humidity	5.4950	16710	1.1320	8.1212	0.1710
Melting material m water	0 0780	0.1100	0.2840	0.2998	0.2040
$CaSO_4$	0.6807	0.4766	0.6870	0.8848	12610
$CaCl_2$	-	-	-	-	0 0205
K_2SO_4	0.2467	0.4629	0.4371	-	-
MgSO«	0.0430	0.3481	0.0348	0.2794	-
MgClî	-	-	-	0 2553	-
NaCl (Humid)	93.2566	96.7314	97.2314	89.9595	98.1435
NaCl (Dry)	98.7516	98.4024	98.3634	98.0807	98.3145

3.7 Costs

The average production quantity per worker was 520 tons in 1995, 774 tons in 1996, 899 tons in 1997, 785 tons in 1998, and 774 tons in 1999. Labor costs make up a large part of overall costs. Hence, modernization studies and the use of artificial intelligence in the salt industry have been growing in importance. The relation between production quantity and number of workers is given in Figure 2. As shown in this figure, in spite of reductions in the number of workers, production quantity has increased since 1995.

The production costs are given in Table 9; as shown in the table, the most expensive salt category is others, followed by rock salt and sea salt.

The price is determined by the general management, which is known as TEKEL, three times a year. The crude salt sale price was set at: 3700 TL/kg ($10.58/ton) for the first part of the year, 4000 TL/kg ($9.89/ton) for the second part of the year, 4500 TL/kg ($10.48/ton) for the third part of the year, and 6000 TL/kg ($11.76 /ton) for the last part of die year (Anon., 1999).

Table 9. Production costs QJSD/ton).

Type Of Crude Salt	1997	1998	1999
Sea salt *	8.78	10.54	996
Lakesalt	4.33	5.10	6 28
Rock salt	19.93	20.27	28.21
Others	62 83	60.86	64.39
Average	6.37	7.35	8.80
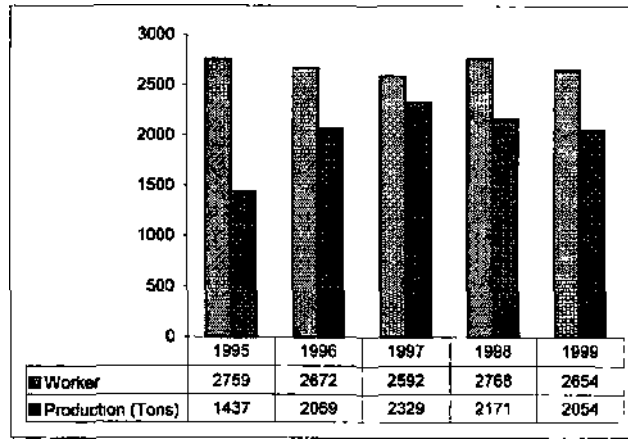
Figure 2 Relationship between worker and production

	1995	1996	1997	1998	1999
Worker	2759	2672	2592	2768	2654
Production (Tons)	1437	2069	2329	2171	2054

4 DEVELOPMENTS IN SALT INDUSTRY

TEKEL has been investing in a number of projects recently to achieve this goal. Some of the recent investments are given in Table 10.

Table 10 Important investments by Tekel

Name of Project	Place
• Investment of Çama İt ı Salt Washing Facilities and Improvement of Production Ponds	Izmir
• Kaldırım Saltworks Dockage System (Completed)	Şerefi i kochi sar-Ankara
• Construction of dykes for Lake Saltworks Production Ponds (Yavşan - Kayacık) (completed)	Cıhanbeylı-Konya Şeref] i koçh i sar-Ankara

Salt washing facilities are among the most important investments. The salt washing facilities are designed to work as two production lines, each producing 50 tons per hour washed salt from the seawater. The raw salt carried by conveyors and trucks from the stock area is transferred into a bunker. It is then transferred from the bunker through a crusher/grinder to a pickling brine mixing tank where calcium sulfate and salt crystals are separated from each other. All of the magnesium sulfate dissolves in water. From here they are transferred respectively to the first washing tank, hydraulic cyclone and the second washing tank, and calcium sulfates and thin solid materials left in the salt crystals are separated from the salt. Salt crystals in the washing tanks move down by gravity, water and thin solid materials are carried by water move up and poured from overflow wessel.

In the process before stacking, washed salt is dried with centrifugal extractor. During this process clean water sprayed into the centrifugal extractor removes the main solution including soluble materials in it from the salt.

There are two important reasons to found washing facilities in Çamaltı Saltwork.

First reason is that chlorine is the main input material in Petkim Chlorine and Alcohol Facilities, which purchases nearly the half of raw material production of saline and it is obtained here. So Petkim needs washed salt having required conditions.

Firstly the yearly-required quantity of salt (220-250 thousand ton washed salt) will be supplied by Tekel, construction of which has been completed and yearly production capacity is 300 thousand ton/year.

The second important point is that raw salt produced in all EU countries is taken to the stock areas after being passed through washing facilities. This application is realized also in other countries than EU ones like Egypt and Jordan in which production of salt is high. By founding salt washing facilities in İzmir Çamaltı Saltwork, Tekel will be the first one who realizes this in the industry in Turkey.

There is a need for washed salt in our developing economy of Turkey and the important part of our industry establishments are growing every year. This need increases continuously especially in leather, textile, paint and oil chemical industries.

This facility has also the capacity to satisfy these needs.

By using the latest developments in the world sea salt production and washing technologies in Çamaltı Salt Washing Facilities,

- The production can be realized with less cost,

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- The more quality washed salt can be obtained,
- Modern technologies can be used,
- The environment İs not polluted,

The salt with %99,5 purity, which does not include heavy metals and which İs required by chemical and oil chemical industries, can be produced. (Tekel, 1999)

5 CONCLUSIONS

Turkey has potential to become one of the world's leaders in salt production and export by increasing the quality and efficiency of die salt production.

Turkey produces approximately 2 million tons of salt, which counts for only 1% of the total world salt production. Turkey exports almost 12000 tons of salt a year.

Salt has strategical importance in industry and the chemical sector. Turkey is surrounded by the sea in three directions, and has large deposits in Salt Lake, so salt production becomes important in evaluating these large sources.

Modernization studies are being carried out and efforts to increase capacity are being made by Tekel in order to produce better quality and cheaper salt. With a view to more efficient salt production, some suggestions are given below.

• To increase exports, all commercial and political possibilities should be explored.

• More production areas for sea salt production should be examined and evaluated.

• Produced crude salt should be sent to processing facilities directly wimout stocking so as to decrease salt losses due to weather and ground conditions.

• Environmental pollution surrounding salt lakes should be stopped immediately for the production of clearer and better quality salt.

• Costs in other salt production plants and rock salt mines should be decreased by modernization or by decreasing the number of workers; otherwise, these place should be closed until salt prices cover salt costs,

• Turkey has no potash deposits and the annual demand for potash, which is 90,000 tons, is satisfied by imports from other countries. However, in many countries, dissolved potash is extracted from lake and sea water. Potash production directly from lakes or from the processing of bitterns remaining after solar salt production is very common. After the increase of the annual production capacity to 1,000,000 tons at the Çamaltı sea salt plant, which is the largest plant İn Turkey, approximately 30,000 tons of potash, 170,000 tons of magnesium, 100,000 tons of NaïS04 and 3,760 tons of Br2 will be extracted from remaining bitterns.

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