TOWARDS 40,000 TONNES/DAY WORKINGS

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ABSTRACT

Longwall faces producing in the order of 20,000 tonnes per day of raw coal are presently running in the Houillères du Bassin de Lorraine of Charbonnages de France.

Production levels of twice that level pose some technical and economic problems of coal cutting, ventilation, coal and equipment transportation in the face and from the face to the surface.

The peak productions of the equipment in use today, the results of trials with new equipment, the utilisation of forecasting models (some in the form of software), allow us to foresee feasible technical solutions to these problems.

They will first be considered in the best possible context (thickness, dip, gas, strata, faults). This will give the necessary conditions to reach daily output of that level.

A sensitivity analysis of these figures to the effective natural conditions will then be undertaken and the implication of such a project on health and safety (dust, climate ...) will be carefully considered.

A concise technical and economical approach will permit us to forecast the economical interferences of such production units and specially the consequences of sudden variations of the output (downtimes, failures, etc).

From this investigation, goals for technological development and research areas will be outlined to be able to reach this production target in the XXIst century.

INTRODUCTION

In recent years daily productions of about 20,000 tonnes of raw coal per day have been achieved from underground workings using longwall face techniques in several countries. These outputs have all been achieved in peak productions; but it is obvious that current technology can produce output of this magnitude. In addition, with good organisation and management of the coal face, and more generally, of all the down stream processes from actual coal-winning, mean figures can approach these peak values.

Doubling these peak outputs (i.e., 40,000 raw tonnes per day) raises two types of questions:

- does current mining technology permit daily outputs of this order?
- how must the entire production process, from coal face to surface, be organized to provide mean figures of this magnitude?

Before describing how, without technological revolution, this 40,000 tonnes-per-day working can, in our opinion, be achieved, let us briefly consider the reasons why we have chosen a caved longwall face with a sharer for outputs of this level.

To achieve such production using continuous miners which, under ideal conditions produce 200 tonnes per hour, would necessitate using a dozen or so much machines.

An output shared by such a number of machines would of course render the whole operation less vulnerable to working hazards and breakdowns; but it would also pose problems of very complex organization, particularly those of product transportation.

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Furthermore, the operation would have to be limited to a pillar layout without pillar extraction, which is an operation too delicate to sustain this level of production. Thus, under these conditions, recovery rates would be severely reduced.

Coal winning by ploughed longwall faces could provide hourly outputs from the face comparable to those obtained using drum shearer but the difficulty of coordinating on a regular basis the roof support and cutting operations in a ploughed face make repetitive systematic operation difficult, when this would be essential in a face to obtain these production levels.

The major technical problems of achieving 40,000 tonnes daily were studied, using a typical working where the current state of technology suggested that this production level could, a priori, be achieved without insurmountable difficulties.

The formula relating daily production to the principle parameters of a longwall face may be expressed as follows:

\[ P_j = \frac{L \times o \times p \times d \times T}{(L/v)+t} \]

where:

- \( P_j \) = daily raw coal production, in tonnes
- \( L \) = face length
- \( o \) = working thickness/seam thickness
- \( p \) = width of web (shearer drum width)
- \( d \) = density of coal, in situ (assuming a density of 1.2 effectively converts gross production to net production, i.e., postulating a dirt free coal seam)
- \( T \) = daily work period
- \( v \) = average cutting speed at work face
- \( t \) = average time for face-end operations.

Two face models will be considered: a single longwall having a face 300 to 400 meters long (faces of this length exist, or have existed); a double-unit comprising two faces/panels each equipped with its own shearer. The shearsers advance simultaneously and evacuate their outputs via a central maingate.

Seam thickness between 3 and 4 meters were selected. Experience has shown that, in such thickness, coal winning is possible with a two-directional double-drum shearer of suitable diameter. Such thicknesses exist in the major coal producing countries and constitute a significant percentage of their reserves.

Web widths of 0.8 m, 1.0 m and 1.2 m were selected.

A daily working time of 20 hours was chosen based on the output of a longwall face worked by four uninterrupted shifts, with a productive working ratio of the order of 80%. Then production rate of 2,000 tonnes per hour for the 20 hours during which the face is effectively worked, is necessary to achieve the 40,000 tons/day.

A duration of 16 minutes was chosen for the face end operations, based on performance figures achieved in high-output workings, in particular those from the La Houve mine in the Lorraine Collieries.

Finally, so as not to over-complicate the problem, the site was considered to be as follows:

Situated in a favourable geological context; at a depth of about 500 meters, with no tectonic faults on the panel, with good surrounding rock, having a dip of less than 10°, a coal containing low methane levels and a single vein, without other veins in the surrounding terrain, either along-side or above.

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DETERMINATION OF THE PRINCIPAL DIMENSIONAL AND ORGANIZATIONAL PARAMETERS OF A 30,000 TONNE-PER-DAY WORKING

The principal dimensional and organizational parameters will be obtained by:

a. carrying out a study of sensitivity of production rate to changes in the principal parameters (length, thickness, web width) and

b. using a computer simulation of the mining operations.

INVESTIGATION OF THE FACE DIMENSIONS AND SHEAHER SPEEDS REQUIRED FOR THE PROPOSED PRODUCTION LEVEL

Single longwall face, with a single shearer

Using the formula relating output to the principal parameters, and assuming a density of 1.2 and a face-end operation time of 16 min, a study of output sensitivity was carried out by considering:

- two seam thicknesses, 3.5 and 4 m
- three web widths, 0.8, 1.0 and 1.2 m
- three panel lengths, 300, 350 and 400 m.

The results of this study are shown in the graphs in figures 1.a,b,c and 2.a,b,c.

It can be seen that, for 3.5 m thick seams, an output of 40,000 tonnes of raw coal per day can only be obtained with a web width of 1.2 m, and that assumes an average shearer speed of 10 meters/min (corresponding to a peak rate of 12.5 m/min), which is in excess of known current rates.

For coal faces with a seam thickness of 4 m, the required level of production can certainly be obtained with a web depth of 1 m, but this is at the limit of maximum attainable shearer speeds. It is therefore only possible to attain a 40,000 tonne-per-day level with a web depth of 1.2 m using shearer speeds of 8 m/min.

These solutions presuppose using a shearer close to the extreme limits of normal practice, thereby leading us to consider a double unit face, i.e., to share production between two shearsers.

Double-unit face (with two identical shearsers)

Under the same conditions as above, the target production becomes quite attainable

- for web depths of 1 m and panel lengths of 200 and 250 m in a 3 m thick seam (figs 3a, 3b, 3c)
- for faces between 200 and 250 m long with a web depth of 0.8 m in a 3.5 m thick seam (figs 4a, 4b, 4c).

In conclusion, the target output can be achieved from two double unit faces of length 200-250 m and a web depth of 1 m; the required average shearer speed would be between 6 and 8 m/min.

INVESTIGATION OF REQUIRED DIMENSIONS AND ORGANIZATION OF DOUBLE UNIT FACES TO ENABLE THE PROPOSED LEVEL OF PRODUCTION

The choice of double-unit face presents two problems:

- the simultaneous arrival of the two shearsers at the central maingate of the faces would prolong the face end operations for both machines
- during the simultaneous advance of two faces, the development of an offset between the advance of each face, in excess of one web width would probably cause serious problems of coal transfer and roadway maintenance.

These two problems were studied using computer simulations of the operation of the two faces, with the following characteristics:

- duration of simulation = 20 days
- the shearer speed on each pass obeyed normal rules
- a maximum offset of one web depth between the faces

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SINGLE FACE FIG. 1
3.0m Thickness

DAILY PRODUCTION Metric Tons
50000
40000
30000
20000
10000

SPEED m/min
0 4 6 8 10 12
Fig 1a Web 0.8m

DAILY PRODUCTION Metric Tons
50000
40000
30000
20000
10000

SPEED m/min
0 4 6 8 10 12
Fig 1b Web 1.0m

DAILY PRODUCTION Metric Tons
50000
40000
30000
20000
10000

SPEED m/min
0 4 6 8 10 12
Fig 1c Web 1.2m

SINGLE FACE FIG. 2
4.0m Thickness

DAILY PRODUCTION Metric Tons
50000
40000
30000
20000
10000

SPEED m/min
0 4 6 8 10 12
Fig 2a Web 0.8m

DAILY PRODUCTION Metric Tons
50000
40000
30000
20000
10000

SPEED m/min
0 4 6 8 10 12
Fig 2b Web 1.0m

DAILY PRODUCTION Metric Tons
50000
40000
30000
20000
10000

SPEED m/min
0 4 6 8 10 12
Fig 2c Web 1.2m

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. a 2-minutes increase in the face-end operations time (common unloading point) whenever the shearsers arrive there simultaneously (interference)

- face length : 200 metres.

General organization of the faces

The barmonogram in Fig 5 shows that organisation of the work in the face presents no major problems. For a face equipped with integrated electro-hydraulic control, both chocks, in groups of 4, and roof shields can be moved at a speed greater than 10 metres/min for an average shearer speed of 7 metres/min.

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35.
Simultaneous working of two units

The simulation shows that an average difference of 10% between average speeds of the shearsers causes an advancement offset of 3.5 webs per day and produces an average of 5 shearsers interferences at the common face end. A difference of 20% between the shearsers speeds causes an advancement offset of 7 webs per day and, as before, an average of 5 shearer interferences.

These data show that the above working characteristics are unacceptable, and that it is necessary to allow sufficient reserves of shearer power to enable any accidental losses to be made up.

To this effect, it has been shown that, in a face 3.5 m thick, an increase in average cutting speed of 1 metre/min would compensate for stoppages amounting to 9 minutes per run in a 200 metre face with a web depth of 1 metre.

In conclusion, this study of the organization of the operation indicates that only a working 3.5 thick, comprising two units 200-250 metres long, each equipped with double-drum shearsers with drum diameters of 1 metre, giving average speeds of up to 8 metres/min, but possessing sufficient reserves of power to allow for short-term acceleration, would enable sufficiently reliable daily productions of 40,000 tonnes.

SPECIFICATION OF FACE EQUIPMENT AND RELATED INFRASTRUCTURE

THE DRUM SHEAVER

Our studies indicate that the appropriate sheaver should have the following configuration:

- double-drum sheaver
- bi-directional cutting-height greater than 3.5 m
- a cutting speed in excess of 8 m/min
- shearer-drum width of 1 m
- drum-diameter suited to the seam thickness.

These basic parameters have been used in the simulation program PC DRUM to determine the associated operating conditions required to draw up the machine specification.

PC DRUM (2) is a shearer simulation and evaluation program developed by the Ecole des Mines of Paris and Cof group which can be used to give the relationships between:

- the speed of advance of the shearer
- machine specifications (mass, no. of drums, no. and power of motors, max. haulage effort, output) as a function of:
  - the machine's environment
  - drum specifications (diameter, speed of rotation, pick-arrangement, cutting disc type)
  - picks (type and wear incurred)
  - rock characteristics (compressive strength, internal angle of friction).

The simulation, Fig 6, indicates that a cutting-power of about 900 KW is necessary to cut coal of compressive strength 80x10^6 Pascal at a speed of 8 m/min in a seam 3.5 m high.

However, if the picks are slightly worn, or if the strength of the coal approaches 100x10^6 Pascal, 1,100 KW is required to achieve the same production.

An adequate reserve power margin is then required to cope with any necessary changes in the environment or in the shearer condition.
The selected shearer should provide high performance and long-term reliability combined with safe operation and have the main following technical specifications:

- shearer power between 1,100 and 1,500 KW (each cutting head having its own motor)
- 150 KW electric or hydraulic haulage
  - cutting speed from 0 to 12 m/min
  - haulage effort of 700 KN
- shearer operating on 5,000 V
- shearer arms long enough to permit use of drums up to 2.5 m diameter, 1 m wide, rotating at 25 rev/min
- modular machine design, easily interchangeable modules in mobile housings
- high component accessibility, minimising breakdown repair times
- machine operations by on-site remote control or true remote control
- machine operation monitoring for preventive maintenance and aid in breakdown diagnosis
- cutting-drums fitted with safety devices preventing frictional ignition of methane and reducing dust emission during cutting
- optimisation of helix design for product transfer to the armoured face conveyor
- pick distribution to suit the prevailing geological conditions, and optimising product output size distribution
- optimised pick design for wear resistance (diamond picks).

The shearer, being the critical equipment to achieve the goal of production, should work continuously and have a continuous condition monitoring system on line (type Telsafe CA) (3). This monitoring system should also include an expert system for fault diagnosis and quick assistance at breakdown repairs.

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37.
Roof support

Roof support specifications must take into account:

- General geological conditions including structural analysis
- Seam characteristics:
  - Seam thickness including amount and position of dirt beds
  - Nature of the surrounding rock
  - Coal quality
  - Depth
- Working method:
  - Advancing or retreating face
  - Face length
  - Coal winning method
  - Treatment of goaf
  - Roadways dimension.

These elements are used to define the geometry of the support, the specifications of the props and their compatibility with the other face equipment shearer and armoured conveyor.

The following requirements have been considered in this study:

- The equipment of a high-production face with simple yet robust machinery
- Conveyor and support advance speeds sufficient for a shearer speed of over 8 m up to 12 m/min
- Panel opening of 3.5 m or more
- Web width of advance of 1 m
- A retreating face
- Bolted rectangular roadways.

In addition, the experience of Charbonnages de France with the new type of 2-leg (5,000 kW) chock-shield with variable opening from 2.1 to 4.5 m, having a load-bearing capacity of 800 kW/m(2), suggests that such a support can greatly facilitate overcoming the specific problems of supporting a high-output face (17,000 tonnes daily output from a 200 m face of working thickness 4.3 m, using a 550 kW shearer) (7).

The characteristics of this chock are (Fig 7):

* One step-back, size 1.5 m.
* The setting load-bearing capacity after advance is the same as the load-bearing capacity during movement after a face advance.
* Mass 16.4 tonnes.

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monobloc floor-plate

(2,350 x 1,250) with a 280 mm channel for the advancement jack (inverted advance system)

5. floor pressure of
29x10^5 Pascal at the front of the sole and 4x10^5 Pascal at the rear, derived from the sliding pressure

2-section shield with locked articulation permitting withdrawal of the front section during transport. Connection to the back shield is by means of a jack

   - shield dimensions: 2,350 x 1,300 mm
   - side shields, with lateral displacement, on one side only
   - frontal shield-support manoeuvred by two jacks fixed at the front of the roof canopy
   - passageway for personnel inside the chocks behind the line of props
   - overall dimensions of the support for transport: 4,55 x 1,40 x 1,50 m. The height of 1.6 m is obtained by retracting the feet of the props during transport.

A chock may be used if the roof is stable immediately after shearer passage. Moreover, using a two-leg shield gives a high load-bearing capacity of 800 KN/m(2) with two 2,500 KN props and a maximum reduction in mass and overall dimensions, providing easier transport and working of smaller faces.

A two-leg chock can be used up to a load-bearing capacity of 1,100 KN/m(2) if the hydraulic pressure during advance can be reduced to 450x10^5 Pascal.

The same chock arrangement is possible with a opening of 1.75 m, having a maximum load-bearing capacity of 900 KN/m(2).

This solution simplifies face equipment and gives a reduction of advancement time of more than 15%.

With electrohydraulic machine control, these times can be of the order of 8 or 10 seconds per chock.

The rate of advance is thus the same as the speed of the shearer. Electrohydraulic machine controls are also required because of their capacity for:

   - simultaneous operation
   - programmable sequences
   - remote control.

An interface permits the advance to be regulated according to shearer position.

A face-management programme permits monitoring of:

   - chock position and face alignment
   - operating parameters. (Fig 8)

Behind the face, the entries are supported by self-advancing roof support. The intersection is set up to facilitate the discharging of the face conveyors onto the advancing armoured conveyor of the main gate and the circulation of personnel.

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35.
PRODUCT REMOVAL

The armoured face conveyor is designed so as not to restrict shearer advance-ment and yet be able to remove coal at peak production rates.

It has to be robust and reliable and to require no maintenance throughout the entire working of the face.

It has then to be made up of:

- a horizontal section comprising wide pans with medium thick plates and a closed base. The 880 mm cross-section will be large enough to ensure removal at a production rate of 2,000 t/hr.

The mobile assembly could consist of a double chain of either 34 mm or 38 mm diameter chain, depending on the face length.

The connections between pans will be an integral part of the front side ramp (2,500 KN) and of the trailing-side ramp (3,000 KN).

The Dynapac face haulage system consists of a captive chain, made of alternate long 38 mm diameter links and short forged links 44 mm in diameter.

The shearer is guided and anchored on its trailing side by two heavy hook-shaped guides.

- the two motorized conveyor heads will have power ratings matched to the working conditions (680 KW for the main head and 180 KW for the auxiliary head, for a 250 m face producing 1,200 tonnes/hr) (Fig. 9). The head motors should have variable speed (up to 1.5 m/sec) and be equipped with gradual start-up devices, cut depth limiters, and an automatic chain-tensioner.

Fig. 9: Face chain conveyor Power of the drive-heads

The maingate product removal system has to be designed to assure the transportation of a production rate of 2,500 t/hr.

It shall consist of a large conveyor (x 7 m) onto which the two face conveyors discharge.

This conveyor will overlap a second conveyor equipped with a crusher, which processes the random-sized products.

Both conveyors are mounted on caterpillar tracks.

The crushed coal will be transported on a belt conveyor which has to be 1,400 mm wide to remove the coal at a speed over 3 m/s. The evaluation of the required power has to be done according to the length of the successive conveyors and inclination of the gallery (Fig. 10).
This flow rate can be expressed by the formula:

\[ Q_m / s = (1 - *) \cdot 0.8 \cdot i \cdot P_0 / (86400 \times l) \]

where:
- \( * \): drainage ratio (taken as 10 %)
- \( i \): irregularity coefficient (taken as 1.6)
- \( P_0 \): daily production at each face: 20,000 tonnes/day
- \( l \): maximum permissible content (taken as 1.5 %)
- \( S \): specific methane released per tonne out giving a value of \( S = 2.5 \text{ m}^3/\text{tonne} \).

Based on the assumed average stratigraphic conditions for the surrounding rock, we can deduce from previous studies (5,6) linking specific emission and concentration, that this type of coal winning operation can be achieved without major problems of firedamp. If the coal has a maximum desorbable methane content of 2.3 \( \text{m}^3/\text{tonne} \).

Nevertheless, it should be noted that:
- this method of calculation has only been tested for face thicknesses of less than 3 m and at low advancing speeds (4 metres/day), but it is known that a high rate of advance is an advantage since the specific emission is in general lower in such cases
- the presence of a thick layer of impermeable rock blocking the migration of firedamp in the surrounding strata would indicate that coals with a higher maximum desorbable methane content may be worked.

Climatic conditions can be examined using a provisional model (7) to predict the temperature changes within the circuit comprising the air intake at the central maingate, the coal face and the air return, as a function of the initial air temperature, ground temperature, and electric power consumption throughout the circuit.
Fig 11 shows the changes in the "resultant temperature" (TR) throughout the ventilation circuit for various initial ground temperatures.

![Diagram showing changes in resultant temperature (TR) throughout the ventilation circuit](image)

Fig 11: Evolution of resultant temperature along the ventilation circuit

It can be seen that the generally accepted limit of "TR" (28°C) would not permit this type of working without recourse to ground refrigeration, whenever the unworked ground temperature exceeds 30°C.

At the targeted high production level, there will be dust production associated with the coal winning by shearer and with transportation of the product, especially unloading at the face end. Effective means will therefore be required to minimize such emissions (spraying, capping).

Personnel can be protected from dust emissions caused by cutting and support-moving operations by using remote control systems controlled by operators situated upwind of the dust-source.

"Resultant Temperature" TR is a standard term in France. It is calculated from the dry-bulb temperature (td), wet-bulb temperature (th) and air speed (Va) using the formula:

\[
TR = 0.3td + 0.7th + Va
\]

INFRASTRUCTURE OF THE FACE AREA

The central maingate will be a single large gate acting as an air inlet; air outlet is effected at the panel ends. These may be double or triple gates, depending on the local regulations. In the case of multiple entries, the entry situated nearest the face will be abandoned (by caving) as the face advances; the other entries, separated by sufficiently large pillars and linked by crosscuts, can be kept behind the face. The choice of panel end configuration (single or multiple) will, in turn, influence the choice of ventilation system. In the case of a single gate system, the obligatory loss of the end gate behind the advancing face requires a H-type ventilation system (Fig 12a). In a multi-entry system, a U-type layout (Fig 12b) may be used by keeping entries open the whole length of the work area with a possible air supply at the face-head. This solution is preferable when there is no risk of spontaneous combustion.

![Diagram showing the layout of the double unit system](image)

Fig 12: Layout of the double unit system

The multiple-entry system at the face ends provides additional advantages in working logistics. The speeds of face advancement and the requirement of speed for the dismantling-transportation-resetup operations at the face require rubber-tyred vehicles.
The existence of multiple en-\textcircled{es} provides certain advantages for \textcircled{en}, such as the separation of trans-\textcircled{port} flows, which facilitates the dis-\textcircled{mantling}-setting-up operations.

\textbf{ECONOMIC CONSIDERATIONS}

An accurate economic evaluation of a 40,000 tonnes/day unit would be unrealistic as the results will mainly be related to the reliability of the whole equipment. It is obvious that each unit would, in most cases, be the only producing unit of one mine and that any failure on any of the links of the production chain will cause an important loss of income: loosing 2,000 tonnes/hour is roughly equivalent to loosing 100,000 $ per hour ...

It is then important, according to the demand for coal at the mine, to integrate in the production chain some capacities (silos, bunkers) to absorb variations of production or demand.

To do an accurate economic evaluation, the cost of investment is of course one of the main parameters. The cost of the equipment described above and estimated in conjunction with manufacturers reaches about 30 million $ for a double unit 2 x 200 m working a 3.5 m thick seam. The cost of the main pieces of equipment are given as follows:

- Supports : $96,000
- Coal face $96,000
- Entries/roadways $7,000
- Hydraulie plant and flexible hoses $2,500
- Shearsers and accessories $22,000
- Face armoured conveyors $21,000
- Transfer conveyors and crushers $4,000
- Face lighting, communication and monitoring systems $1,700
- Power supply system, including transformers $9,800

\textbf{CONCLUSIONS}

In conclusion, this approach to the problem of producing 40,000 tonnes per day from a face shows that, technically, this output level is indeed possible using existing equipment operating at above its normal rated limits. Such apparent optimism should however be tempered with the realization that such operating techniques may well not become a reality until the 21st century or perhaps shortly before, depending on certain conditions.

Without listing all these conditions, a few of them which are clearly essential to the setting up of such a working are:

- Improved equipment reliability and the employment of zero-fault maintenance and preventive maintenance policies
- Use of high voltage in the working (\(5,000 \text{ V}\)) for machinery (drive units, shearsers), to cope with power requirements
- Use of operational cutting tools (diamond picks) whenever economically possible
- Use of fully automated (robotized) more powerful product-removing machinery
- Improvement of ventilation systems, particularly at the face end, which are currently the most high risk areas for methane
- Use of remote electro-hydraulically controlled self-advancing roof supports coupled to shearer progress
- Development of automated and robotized systems to optimize the overall operation, as outlined in Fig. 13.
There is, therefore, a challenge of both productivity and safety. It is nonetheless within the reach of the coal industry, given good coordination between mine management, research centres and equipment manufacturers.

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ACHEIVING 435-2540 TONNES PER SHIFT FROM CONTINUOUS MINER SECTIONS, A REPORT ON THE USA'S MOST PRODUCTIVE COAL MINES

By

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ABSTRACT

The U.S. Department of Interior's Bureau of Mines has the opportunity to visit many U.S. mines, including some that claim 2 to 5 times the national average of 33⅓ raw tonnes/unit shift (tphp) from a continuous miner section. In a project to assess the reasons for this high production, the Bureau conducted two studies.

The first study focused on the 25 highest producing continuous miner sections. What stood out during the visits was the quality of the labor-management relations. In each of the mines there was a positive labor-management relationship based on mutual trust and a sense that the employees were the company's most valuable resource. Discussions with labor revealed a universally positive attitude toward the company. Interviews with the mine superintendents invariably ranked the quality and attitude of workers as more important than anything else.

Also during the mine visits of the 25-mine study, no engineering factors obviously stood out. Time at the face was greater than average, most were in coal over 1.52 m and the roof was generally good. Also, none of the 25 mines had to contend with high methane levels. Because this study found that a high percentage of mines were in seams less than 1.52 m thick, the second study examined thin seam mines only. In the thin seam study, the U.S. Bureau of Mines visited continuous miner sections in seams of 1.22 m or less and which exceeded an average production of 33⅓ tphp. Ten mines were found where average shift productions ranged from 435 to 1952 tphp and averaged 781 tphp. Many similarities were found between these thin seam mines and the 25 mine study. The same positive labor-management relationship existed that was seen in the original study. Outside of the seam height restriction in this study, the Bureau observed the same factors with one exception. Technological factors appeared to play a more important role in thin seam operations.

25 MINE STUDY

Productivity is the single most important consideration for coal mine operators. This emphasis on productivity has generated in the literature some remarkable claims of mines with exceptional unit shift productions.

The U.S. Department of Interior's Bureau of Mines, in the course of its research programs, has had the opportunity to investigate some high production U.S. mines, mines with tonnages of 2 to 5 times the national average of 33⅓ raw tonnes/unit shift (tphp).

Discussions were held with both management and labor at these mines. Records were examined, and assurances were obtained that the tonnages claimed were in fact produced on a regular long term basis. Mine design, equipment, geology, management practices, local unemployment and other pertinent factors were also recorded.

SOME SURPRISES

The information obtained gave some surprises. The following are considered surprises because they refute much which is considered as conventional knowledge.

1. Half of the mines were unionized (UMWA).
2. Only 4 mines cut 1.13 m or more before place changing.
3. Only 5 mines used diesel haulage, and their tonnage was only 13 per cent better than the others.
4. Ten of the 25 mines were less than 1.52 m thick, 4 were under 1.22 m, and 1 managed to extract 1089 tonnes from an 0.84 m seam.

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