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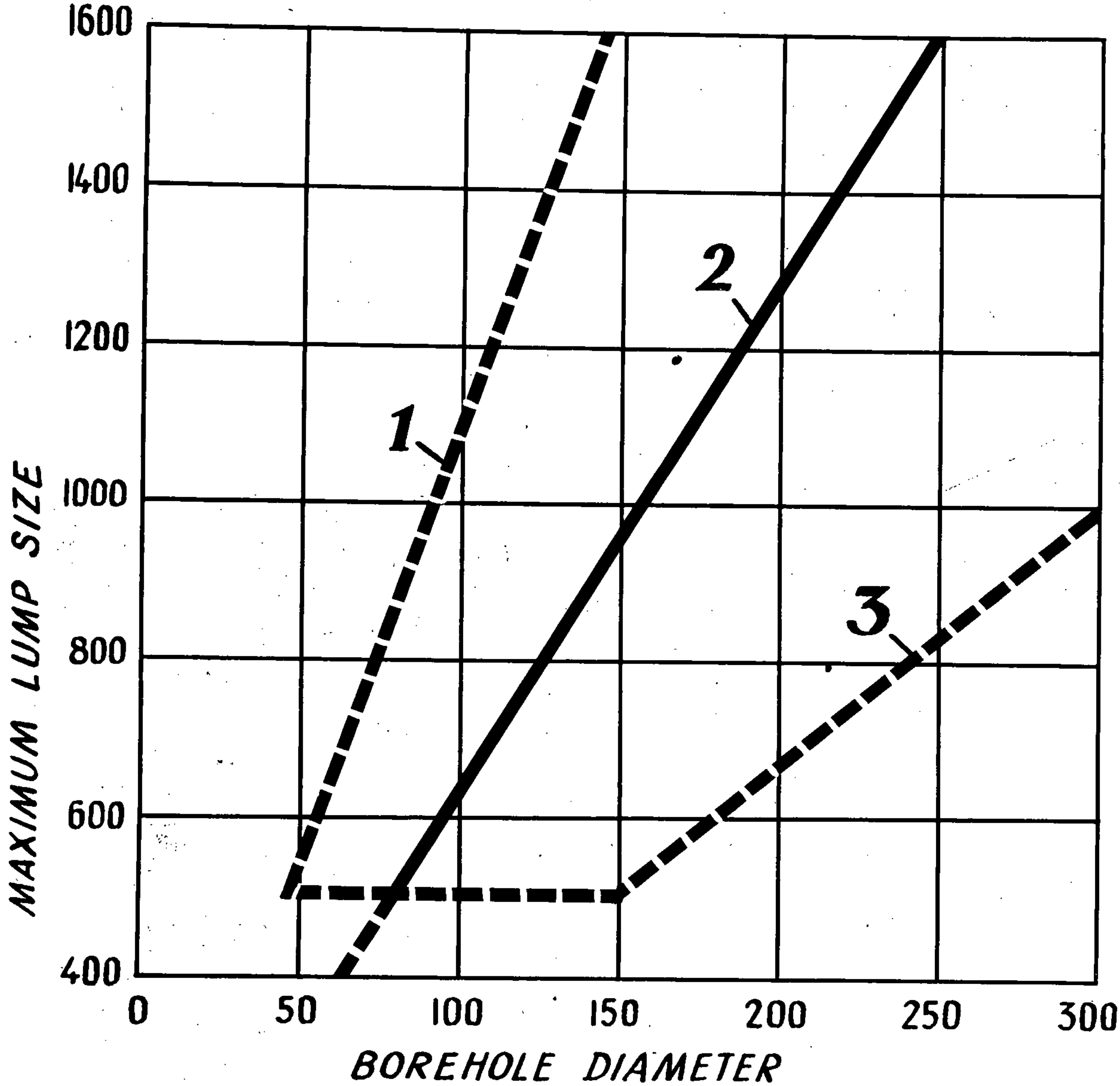

Fig. 1. DEPENDENCE OF MAXIMUM LUMP SIZE ON BOREHOLE DIAMETER.

1 — HARD-TO-CRUSH, 2 — MEDIUM CRUSHABLE, 3 — EASILY CRUSHABLE.

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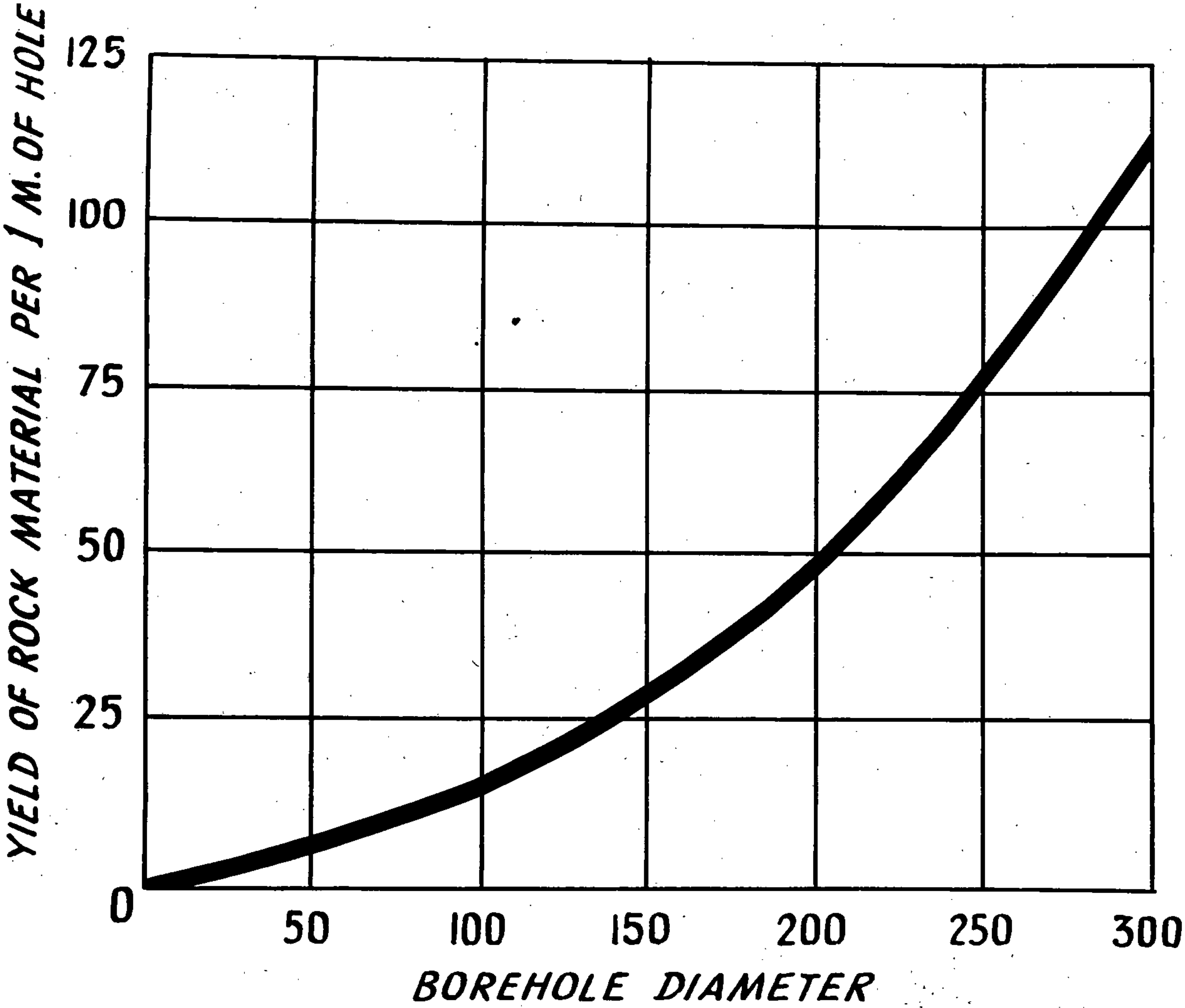

Fig. 2. DEPENDENCE OF ROCK YIELD PER 1 M. OF HOLE ON HOLE DIAMETER FOR MEDIUM CRUSHABLE ROCKS.

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CONTINUOUS DRILLING AND BLASTING

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In the U.S.S.R. line production methods are now being introduced for quarry mining of rock. These methods make it necessary to crush the rock to a size that will permit the use of continuous action equipment.

In modern quarries rock of average strength and crushability is generally broken to a maximum lump size of 1,200-1,300 mm with the yield of oversizes being up to 7 per cent. This makes for normal operation of excavators with 3-4 cu.m. buckets and crushers with a receiving opening sized 1,500 x 1,200 mm, both widely used in the U.S.S.R..

Calculations show that for rock to be handled by conveyors and other continuous action equipment its maximum lump size should not be over 500 mm; in this case normal operation of the aggregate equipment used for loading and secondary crushing is possible only if the content of oversizes is below 1-2 per cent. Hence, to use continuous equipment for rock it becomes necessary to decrease the lump size of blasted rock by 50-60 per cent.

The maximum lump size of the loosened rock depends to a great extent on its crushability, which in turn depends on the strength of the rock and also on the degree of dislocation of the massif. As regards its crushability, all rocks may

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conditionally be divided into three groups, which correspondingly yield grades over 1,200 mm in size to the extent of 1-2, up to 7 and over 7 per cent.

Practical experience has shown that boreholes of a diameter of 200x250 mm and corresponding patterns now used in drilling and blasting cannot yield rock of a lump size of 500 mm, even if the power of the explosive is used to maximum efficiency.

The required result may be obtained by providing a greater number of destruction centres in the rock massif. This will require a narrower pattern of smaller diameter holes. The correctness of this premise has been proved by the theoretical research of a number of Soviet scientists (G.I.Pokrovsky, A.F.Sukhanov, A.F.Belayenko, A.N.Khanukayev, and others) and by experimental work in several Soviet mines and quarries.

Since the degree to which rock is broken depends on the spread of the explosive in the massif to be blasted, while the sizes of the hole pattern in turn are a factor of the borehole diameter, the extent to which the rock is broken up may be considered a function of the borehole diameter.

A summary of the data on the parameters used and methods applied in drilling and blasting and an analysis of the practical results obtained have permitted a graph to be built showing the dependence of the maximum lump size of the blasted material on the diameter of the borehole for easily crushable, medium crushable and hard-to-crush rocks.

As will be seen in Fig.1, a borehole diameter of 150 mm yields a maximum lump size of 1,000 mm for medium crushable rock, while that from 200 mm wide boreholes is 1,200 to 1,300 mm in size.

An analysis of experimental data for small diameter boreholes suggests that for medium crushable rocks lumps of a maximum size of 500 mm will be obtained when the boreholes have a diameter of 80-100 mm.

As an approximation we may assume that for easily crushable rock a maximum lump size of 500 mm will be obtained from boreholes 125-150 mm in diameter, and respectively 50-70 mm for hard-to-crush rock.

Considering how difficult it is to drill deep holes of a diameter of 50-70 mm, crushing of rock to a maximum lump size of 500 mm by the drilling and blasting method to introduce line production methods should first be attempted with easily and medium crushable rock.

Drilling speeds increase with a decrease in the diameter of the borehole. The drilling speed, however, increases less intensively than the yield of the rock material per 1 m of hole (Fig.2) decreases; the total capacity of the equipment, expressed by the volume of rock material yielded, therefore, decreases when the borehole diameter becomes smaller. This means an increase in the drilling costs per 1 cu.m. of the rock material as the borehole diameter decreases. A decrease in the hole diameter leads to higher specific consumption of the explosive and also to an increase in the cost of the labour of blasting operations. As regards changes in the specific consumption of explosives for holes of decreased diameter opinions are controversial. The problem has yet not been studied sufficiently, therefore, to avoid underestimation of the cost of drilling and blasting holes of small diameter, it may be assumed that the specific consumption of explosives for holes having a diameter over 100 mm remains constant, while that in those having a diameter less than 100 mm rises with the decrease in the diameter. It follows that the decrease of the borehole diameter to obtain rock material crushed to required size boosts the total cost of drilling and blasting.

To make the transition to continuous quarrying methods of rock economically expedient, the following principle must be observed: the cost per 1 cu.m. of rock material (drilling and blasting, loading and conveying) with the new technology must not exceed the total cost of cutting, transport and the first stage of coarse crushing of rock quarried with the now prevailing periodic methods.

To make it possible to determine the cost of the crushing of rock by drilling and blasting to a maximum lump size of 500 mm, we must establish the dependence of the cost of drilling operations on the extent to which the rock is crushed, i.e. on the maximum lump size of the blasted material. Such computations should take into account new drilling equipment now being taken into production and the latest blasting methods (split-second blasting of multiple-row charges, divided by air gaps). With this purpose in view, the Giproruda Institute has calculated the cost of drilling different borehole diameters (50, 100, 150, 200 and 250 mm) for rock of average drillability of a strength of 10 according to Protodyakonov. The drilling and blasting cost per unit volume of rock material comprises drilling costs, blasting materials, explosives, crushing of oversizes, work on charging and tamping of holes.

From the cost of crushing per 1 cu.m of rock material and the graph showing the dependence of the maximum lump size on the hole diameter (see Fig.1) we see that crushing of rock by the drilling and blasting method to maximum lump size or 500 mm is expedient only if the cost of drilling and blasting is cut.

Analyses made show that the cost of breaking 1 cu. m of rock material by the drilling and blasting method in holes of 80-100 mm diameter consists of: drilling — 56 per cent, blasting materials — 18 per cent, explosives — 16 per cent, other expenses — 10 per cent. This shows that to lower the cost of blasting operations we should first and foremost lower the cost of drilling and of blasting materials.

The cost of drilling depends to a great extent on the capacity of the drilling rig and the cost per machine-shift. The drilling capacity can be increased by using more effective methods of rock fragmentation. Another important trend in the increase of the capacity of drilling rig and the lowering of cost per machine-shift is the creation of mechanised and automated multiple-spindle drilling sets, which considerably raise the productions of labour. Considering that medium and easily crushable rock can be broken to the required size by exploding charges of different diameters, it is expedient to consider drilling equipment separately for 80-100 and 125-150 mm holes.

For a single rotary gathering arm loader of a capacity of 6,000 cu.m per shift with two drilling machines, the total capacity of the drilling set should be 375 m per shift for holes up to 100 mm in diameter and 110 to 160 m per shift for holes 125-150 mm in diameter. The capacity per hour of each drilling machine should be not less than 25-30 m for holes of a diameter of 80-100 mm and 10-15 m for those of a diameter of 125-150 mm. At this capacity it is enough to have 2-4 drilling machines on one multiple-spindle drilling set.

The cost per machine-shift of modern drilling equipment comprises mainly: for rotary bit drilling—expenses for drilling tools (30 to 40 per cent and up); for rotary-percussion drilling—expenses for compressed air and drilling tools (over 50 per cent). This makes it very important to make drilling tools stronger and cheaper.

Since the cost of compressed air is considerable, it is important to change to electricity-powered drilling rig. To lower the cost of blasting operations, cheap explosives of medium and high power should be developed, hole charging should be mechanised and the labour used for charging be made more effective. Free-flowing explosive materials, made of trotyl and a solution of ammonium nitrate with a thickener added may become important, because they allow dense placement of the charge and hole charging is easily mechanised.

Charging sets to be constructed should meet the following requirements: 1) should charge simultaneously not less than 2-3 holes; 2) have a capacity of not less than 10-20 t/h; 3) be serviced by a single operator; 4) ensure complete mechanisation of all work connected with the charging of holes, creating air gaps, ensuring reliable spread of the detonation along all parts of the charge.

Table giving the technico-economic indices of the proposed blast drilling methods

	Rock	
	medium crushable	easily crushable
Capacity of multiple-spindle drilling set, m/shift	375	110
Number of drilling machines per set	3-4	2-3
Service personnel per set	1-2	1-2
Height of bench, m	30	30
Hole diameter, mm	80-100	125-150
Hole pattern, m	3x3	4x6
Angle of hole inclination from vertical, deg.	70	70
Hole depth, m	34	34
Yield of rock per metre of hole, cu.m	8-12	19-27
Yield of grades larger than 500 mm,%	1-2	1-2
Width of rock massif fragmentation, m	45	35

To increase the number of borehole charges that are fired simultaneously (up to 1,000), the methods and means for firing must be vastly improved. During recent tests explosions have been set off with radio waves. This method permits one to fire the charges with short delays according to preliminarily worked out patterns.

Conclusions

1. The introduction of continuous methods in quarrying rock will lead to a re-distribution of the cost of various processes making up the total cost of quarrying. It unavoidably leads to an increase in the cost of drilling and blasting which should be compensated for by a lowering of the cost of loading, conveying and coarse crushing of the rock at the factory.

2. A maximum lump size of the blasted material of 500 mm (with oversizes within 1-2 per cent) may be obtained by using a denser pattern of small diameter boreholes (80-100 mm for medium crushable and 125-150 mm for easily crushable rock).

3. To improve the technico-economic indices of drilling and blasting automated multiple-spindle drilling sets should be set in. These sets should have a capacity of 110 m/shift for holes 125-150 mm in diameter and 375 m/shift for 80-100 mm diameter holes.

4. Cheap explosives of medium and increased power, permitting mechanisation of hole charging must be developed.

5. Required is a set for hole charging that will ensure

complete mechanisation of the placement of charges of required construction and the spreading of the detonation to all parts of the divided charge.

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COMPREHENSIVE AUTOMATION OF DEGTYARSK COPPER MINE (URALS)

By A. Shilin, Chairman of the Sverdlovsk Administration of the Nonferrous Metal Scientific and Engineering Society

The development and investigation of new automation schemes, instruments and other facilities for the Degtyarsk Copper Mine (Urals) were started in 1956 in collaboration with the Nonferrous Metal Automation Designing Department, the Nonferrous Metal Designing Institute, the Central Exploration Institute, the Mining Machinery Research and Designing Institute in Moscow and the Urals Copper Research and Designing Institute.

A designing department was set up at the mine in 1957 and an automation workshop in 1960 for designing, building and assembling new mechanisms and automation setups. In 1957 the electric-instrument laboratory at the mine expanded into an automation instrument laboratory in which new automation setups and schemes were adjusted and improved when in service, trouble-free performance of automatic equipment maintained and instrumentation and equipment repaired.

By 1962 the over-all mechanization, automation and dispatching control of the mine had been completed.

The following equipment has been developed, constructed and introduced to put truck-exchange to automatic and remote control: an electromagnetic upper- and lower-action rope pusher, an air-operated pusher, a batching stopper air motor, air-operated cage landing cams, etc. Altogether 116 units of truck-exchange equipment have been constructed and installed at the surface and at the levels of three shafts.

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Air-operated hatch locks and truck-discharging vibrators have been introduced, and an automatic coupler for VG-5 trucks developed. The electric locomotives have been modified to introduce remote control at the loading and discharging places. Double-truck ring-type dampers and 100-m^3 compressor plants have also been modified. New types of output controllers have been developed in collaboration with the Thermal Power Department of the Urals Polytechnic Institute.

Portable air-operated saws, timber handling monorails and drill carriages have been developed, constructed and introduced for the mine.

Many surface processes have been mechanized: the surface transport has been electrified, timber fixing mechanized, etc.

The underground electric transport has been equipped with signalling, interlocking and blocking systems. The equipment in hand includes 10 newly developed units (motor-driven switches, rail transducers, dispatcher's indicator panels, etc.). A switch remote-control circuit (by locomotive drivers) has been introduced. At present there are in operation 46 remotely controlled switches (including 33 driver-controlled ones). A two-way high-frequency communication between locomotive drivers has been put in service. All electric locomotives (35 in all) now in service are equipped with H.F. communication units. On one level locomotives at the loading and unloading places were transferred to remote control.

Shaft skip lifts of 2,800 kW total capacity have been automated. Nothing but automatic lifts are employed for raising ore. In the hoisting shafts the cage landing cams are automatically controlled, automatic cage bafflers are fitted and the hoisting cages are all remote-controlled. Type ACK-1130 high-frequency equipment is used for communication and signalling from within the cages when running. Truck exchange at the surface is automated at all shafts.

Automatic rock hoist installation is in operation, and automatic rope defect detectors fitted. More than 400 motors are provided with built-in thermal trip circuit-breakers. All main and sump drainages (14 units in all with a total capacity of 4,900 kW) are automated. All general shaft ventilating units are remote-controlled. In addition, all ventilating doors in the air headings, air-stream heating plants, shaft boiler units, water-pumping stations and others are automated.

Over-all dispatching control has been introduced in all mine shafts, which has led to sweeping changes in the nature and level of production control. The equipment which the dispatcher has at his disposal, including industrial television, has enabled him to handle easily and adequately a large volume of information, thereby managing and organizing the shaft production as an integral process.

Automation Economics

Higher labour efficiency, and lower production costs are a major indicator of the effects that comprehensive automation has produced.

Owing to the comprehensive automation, 672 operators have been released at the mine from 1956 to September, 1962, while the total amount of ore raised registered a considerable increase. The efficiency of a face worker has risen 70 per cent, of a shaft worker 93 per cent, and of a mine worker 95 per cent (as compared with 1955).

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in 1962 by 18,000,000 kW-hr (including 9,000,000 kW-hr saved as a result of automation). Depreciation showed a 160,000 rouble increase annually, and an entire saving in running costs amounted to 1,503,000 roubles annually. This figure does not take into account savings in conventional permanent costs, repair costs and trouble-prevention and stand-by costs, although all these savings have been accounted for when measuring the lowest possible total production costs. When estimating savings, accont has also been taken of higher efficiency of face workers owing to better ventilation, more stable compressed air pressure, better lamp and saturator maintenance, etc. as a result of automation.

Moreover, the highly efficient mining systems in use has had a great impact on ore production costs.

Comprehensive Automation Costs

Between 1956 and September, 1962, 1,996,000 roubles were spent on mechanization and automation. To this figure we must add 67,000 roubles spent on the depreciation of the equipment scrapped as a result of the comprehensive automation in the mine. Thus, the total amount will make 2,063,000 roubles. The capital invested into mechanization and automation will be recouped within 2063:1503=1.4 years.

The introduction of comprehensive automation has had a great impact on the improvement of labour conditions. For example, the signalling and interlocking of the underground haulage and locomotive remote control has reduced industrial accidents at the loading places by more than 60 per cent as compared with 1956.

The automation of the truck exchange on the levels and standings as well as the shaft machinery control, the over-all interlocking of the sage landing cams, shaft doors and hoists and the loud-speaking communication have completely eliminated the hazards of large trouble and accidents. There have been 30 per cent fewer case of disease and 85 per cent fewer accidents over this period. Also, there have been fewer case of catarrhal diseases in the shafts as a result of the automation of the heading ventilation, shaft airway doors, as well as the automatic heat control of all local ventilator motors, and calorifer units (used for keeping a preset constant temperature in the shaft).

Further Advances in Comprehensive Automation

Auxiliary processes have largely been mechanized and auto mated in recent months.

When the automation of the surface shaft equipment and auxiliary processes underground has been completed, further efforts will be made to introduce automatic and remote control into the work faces in order to go without human element, speed up the efficiency and make the work of operators easier.