THE ADOPTION OF A MODIFIED STOOP-AND-ROOM METHOD OF PRODUCTION WITH MECHANICAL CONVEYING AIDS IN SCOTTISH OIL SHALE MINES.

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SUMMARY.

A historical survey of underground systems of working Scottish oil shales from the commencement of production in 1859 to the present date discloses the progressive increase of output per man shift. This is due successively to a more intimate knowledge of the oil shales and a closer acquaintance with the geological peculiarities of the area, the introduction of explosives suited to the needs of the industry, the adoption of electricity as a source of power supply, and finally the application of mechanical aids to lighten the more arduous tasks. In consequence of the exhaustion of the richer seams, where once a few tons of output per man employed provided an economic basis, it gradually became imperative to increase the outputs. This necessity eventually resulted in intensive mechanization where conditions were suitable, and outputs per man shift have risen, the accident rate has decreased, ventilation to the working faces is improved, while production costs are lower.

The method adopted consists of panels 400 by 120 yd each of which is developed and immediately followed by a system of total extraction. The cycle of operations at the working faces consists of drilling, blasting with compressed gunpowder, and hand filling the shale on to a shaker conveyor. The conveyors extend the entire length of each roadway leading to a face, and at the lower end the shale is transferred to a troughed belt conveyor running the whole length of the panel, and again transferred to a similar conveyor working at an inclination of 1 in 3.6 uphill for delivery into hutches on the regular haulage system. When filling at the face is sufficiently advanced, the shaker conveyor is extended and the cycle of operations repeated.

Approximately 100,000 tons is won from each panel, and with the double-shift system the life of a panel is about nine months.

INTRODUCTION.

The working of the Scottish oil shales has been carried on continuously from 1859 to the present date by a variety of methods, all of which have been recognized as good mining practice.

Factors governing the method of work are: the inclination of the strata ranges from flat to vertical; the thickness of the workable beds varies from 3 to 10 ft or more; the thickness of the intervening strata (see Fig. 1);
the roof conditions; the presence of inferior bands, the proximity of faults, and the intrusion of igneous rocks.

Information obtained from the working of seams, from exploratory borings, and from the correlation of exposed outcrops, etc., provided data

for the development of suitable methods of working. As these methods progressed other factors became manifest, such as the nature and quality of explosives, the production and distribution of electricity, improvements in ventilating machines, pumps, etc., and so new approaches to increased production were continually being made.

**History.**

Generally speaking, the early method of working varied between stoop and room and the longwall advancing, the former being normal practice
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where seams of 5 ft thickness or more were to be mined. The latter method usually prevailed where thin seams of high oil yield justified the excavation of inferior-quality roof or floor strata to the height necessary for transport and to partly support the "waste" or excavated area as extraction proceeded. Modifications of each method were applied where necessary or convenient.

Where stoop-and-room methods were applied, a miner and his helper would operate one or two faces, holing or undercutting being done by pick in the lower part of the seam, advantage being always taken of soft plies. The upper portion of the seam was then blasted down in layers. The material thus produced was filled into hutchcs by the helper and transported to a common gathering point for further transportation by pony or mechanical haulage. The output per man shift was between 3 and 3½ tons.

In the longwall system, with a length of advancing face of probably 100 yd, a 16-yd portion of which was allocated to a miner with two helpers, the actual operation of winning the material was similar to that for stoop and room, but the output was only between 2 and 2½ tons per man shift.

In both instances shotholes were drilled by hand-operated ratchet machine, and loose gunpowder was the blasting medium. The longwall method, with its relatively low output per man employed, fell into disfavour and was abandoned for many years, but the stoop-and-room method continued, and by intensifying the use of explosives the output per man shift was stepped up to between 5 and 6 tons.

Nearly forty years ago, electrically driven drilling machines were installed, but proved unwieldy in highly inclined seams. Mechanical failure was frequent, and the nature of oil shale did not lend itself to easy drilling. Their use did not continue beyond a fair trial.

The stoop-and-room method of working with local variations was accordingly continued until some fifteen years ago, when mechanical drilling was again tried. Special tipped drilling bits overcame the inherent toughness of the mineral to be bored, improved type drilling machines gave satisfactory results, and output increased to nearly 8 tons per man shift. With the further development of mining equipment to undercut the seams and to convey the mineral from the face to common loading points, the longwall-advancing system was again introduced in 1941 (see "Longwall working of the Broxburn oil-shale seam with coal-cutters and conveyors.").

In 1945, with total production falling due to labour shortage as a result of the war, it was felt that in view of the experience gained and in consideration of the aspects outlined below, some modification of the stoop-and-room system could be evolved to use conveyors for primary and secondary haulage. The existence of narrow roadways over the whole area to the boundaries called for repeated costly renewal of timber supports and considerable wastage of manpower during the many years before total extraction. Roof and side conditions continually deteriorated, and long and tortuous intake and return airways required to be maintained. Haulage-ways along the line of strike, with many deviations, necessitated primitive or complicated arrangements, and usually-proved a costly method of transport. Accidents, fortunately relatively low in number and in the main of a minor nature, were capable of reduction and, by elimination of the source of many injuries, valuable man shifts could be saved. It was felt, therefore, that moderniza-
Fig. 2.
ELECTRICAL TRANSFORMERS AND DISTRIBUTORS.

Fig. 3.
HUTCH LOADING POINT WITH STORAGE AND ASSEMBLY ROADS.
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tion would improve the output per man shift from 8 to at least 12 tons. Mechanized mining was, therefore, tried where the thickness of the seam was 8 ft, and it lay at an inclination of 1 : 3.6 with a strong, shaley roof and a floor of fairly smooth bituminous blaes. The depth from the surface is over 600 ft and ironstone ball plies are present in the seam. Water-bearing strata forms the upper roof measures, but is not exposed until total extraction permits of roof breaks in the "waste."

PRELIMINARY LAY-OUT AND DEVELOPMENT.

The adoption of such a method of working necessarily calls for a sufficiency of power and distributing switchgear, with ample ventilation, suitable loading facilities, and well-laid roadways ensuring the rapid assembly of hutches. Fig. 2 shows the electrical transforming and distributing centre constructed early in the scheme and nearby to it, with incoming voltage of 3300, which is transformed to the working pressure of 440 V by either or both of the duplicate 300 kVA transformers.

The loading point is capable of loading a hutch in 45 sec, and is shown in Fig. 3, together with the storage and assembly roadways adjacent to it.

The lay-out (Fig. 4) of the area for the development of a series of panels for eventual total extraction consists primarily of developing one roadway to the full dip, complete with permanent supports, hutch roadway, manholes, and pumping column. These facilities are required later, first, for development work, and then for the conveyance of timber and steel supports, machinery, and material to the working panels. All haulage work is carried out by a permanent direct-rope haulage installation at the rise end of this haulage-way (see Fig. 5). A companion roadway 35 yd distant from and parallel to this is driven uphill from the panel levels. In this companion a 30-h.p. 26-in broad, six-ply rubber-and-canvas troughed belt conveyor is installed to convey uphill the mineral delivered from the level road conveyors (as shown in Fig. 6) to the loading point in Fig. 3.

To form a working panel, a "level" belt-conveyor road, together with a parallel companion road 20 yd to the rise, are developed as straight roadways to a total distance of 400 yd, with inter-connecting headings driven to the full rise at 10-yd centres. It is arranged that the levels have a slight gradient in favour of the load where possible but, should the seam dip locally, at least a level course is maintained by roof ripping, and if a rise is encountered the roadways follow the rise. By this means the unsatisfactory and expensive procedure of pumping accumulations of water from the hollows is avoided. During the development of the lower level, recesses 4 ft deep are formed in the full thickness of the seam on the low side of the roadway, and corresponding in number and position to the connecting headings. In these are housed ultimately the distributing and control switchgear unit for the individual working faces. At this stage the development of the panel is carried out without mechanical help other than shot-hole drilling machines, but later may be expedited by the use of shovel loaders at the faces. The mineral is hand-filled into hutches and transported to the main incline by means of a portable electric hauler. Such preparatory work is carried out continuously in three 8-hr shifts of a miner and two helpers per shift. The maximum man shifts worked per week are 49 ½, giving an advance of approximately 20 to 25 yd weekly. On reaching the
limit of 400 yd in the direction of strike, preparation for the mechanical conveying is begun by installing a 20-h.p., five-ply belt conveyor of similar construction to that already provided in the companion incline, and six electrically-operated 20-h.p. shaker conveyor units in the first six inbye

connecting headings between the levels. Auxiliary portable ventilating fans of the axial flow type and of 5 h.p., capable of producing 2600 cu. ft. of air per minute at the ultimate developed limit of the working headings, i.e., 120 yd, are provided, as well as two 1½-h.p. rotary drilling units driving drill rods at 600 r.p.m. The rods are fitted with tungsten-carbide-tipped removable bits.

The power-distributing switchgear units are mounted on wheels for
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FIG. 5.
THE DIRECT-ROPE HAULAGE INSTALLATION.

FIG. 6.
LEVEL-ROAD CONVEYORS.
transport along the hutch tracks. When in use these are accommodated in the recesses, corresponding to the working faces they supply, on the dip side of the lower level and with sufficient clearance for the passage of material and machinery to inbye or outbye places. The unit supplies power at 440 V through flexible wire-armoured, rubber-sheathed, four-core cables to the shaker conveyor motor, and to the auxiliary fan motor by a similar three-core cable. The section of the unit feeding the two drilling machines receives at 440 V and transforms to 125 V, this being conveyed to the drill motors by five-core trailing cables. A small 300-VA transformer is attached to the end of the unit, and supplies power for electric lighting at 110 V through wire-armoured two-core cable.

![Fig. 7. TRANSFERRING SHALE FROM THE SHAKER CONVEYOR TO THE LEVEL-BELT CONVEYOR.]

Lighting is provided at the lower end of each shaker conveyor, where the mineral is transferred by a special hopper arrangement (see Fig. 7) to the level-belt conveyor. Experiments were carried out on face lighting with standard 60-W flame-proof units and with the newer type 30-W fluorescent tubes 18 in long. Both systems gave satisfactory illumination in addition to that supplied from the electric cap lamps carried by each workman. Due to the need to withdraw the lighting units to a safe distance when shotfiring, it was felt by the operators that the extra labour involved did not justify its continuance.

The existing mine hutch was 3 ft high from rail to top of body, and when hand-filled could be so loaded that the shale extended to about 12 in above the body top, the weight of mineral per hutch thus sent to the surface
being about 20 cwt. As conveyor loading would not permit of such a well-trimmed hutch, the body of some of the rolling-stock was increased in height by 9 in—permitting about 18 cwt of shale to be loaded. Any alteration was of necessity confined to the height, since any change in length or width was undesirable, if not impossible, on account of existing clearances, tippler widths, etc.

It may be here noted that sequence control of the starting and stopping of all conveyors is a feature preventing delivery from any one to another while the latter is immobilized. It is effected by electrical interlocking through speed switches, which also allow for belt breakage or slip. In the belt-driving gear a device is incorporated which prevents the starting up of a belt in the reverse direction and, in the case of the uphill conveyor, also prevents any run back when the power is switched off.

**Operation of Winning the Mineral.**

With the completion of the preliminary lay-out, the development of the inbye heading is commenced by forming a 5-yd-wide working-place driven to the full rise, Fig. 8. The method of production of the mineral adopted

![Fig. 8. The Working-Place.](image-url)
is the normal practice of drilling shotholes known as "holing shots," seven in number, from a height of 3 ft from the floor and dipping to reach the floor at a length of 6 ft; three mid and three top holes are then drilled to the required depth. This completes the drilling operation, which is followed by blasting out the mineral using cartridges of compressed gunpowder in a quantity determined by the shotfirer. A maximum of four shots are fired at any one time, the source of ignition being by powder fuse, which must be ignited by not less than two workmen. The rotation of firing is as shown in Fig. 9, which also shows the position and direction of the holes. In the charging of all shotholes, the drillings of the mineral itself were previously used as stemming, and were rammed behind the charge to the mouth of the hole. For many years this was accepted as a good medium, but, as a result of experiment, the stemming to be used in future will consist of a mixture of granulated, spent shale and hydrated lime in the proportion of 40:1 with the addition of sufficient water to form a plastic mix; this is formed into random-length cartridges by an ordinary moulder's core-making machine. These cartridges are stored for drying.
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and then supplied to the workmen. This practice conforms very nearly
to the recommendation of the Safety in Mines Research Board of 1934. The
more efficient stemming material results in a decreased use of explosives
and a saving of man hours in the charging operation. Its use also reduces
the smoke produced in the blasting, with a resultant improvement in the
ventilating conditions. An average advance of 4 ft is made per round of
shots fired, producing some 24 tons and a 14-cycle is expected from each
shift, i.e., a production of 36 tons.

The filling of the mineral produced proceeds continuously as far as possible
by either one, two, or three men. Three men, a miner and two helpers,
employed in each place, has been proved to give the maximum output per
man shift. The miner is responsible for the safe and systematic cycle of
operations, but all three are required to form a team with no specific duties,
and all, in time can interchange in drilling, shotfiring, timbering, extension
of conveyors, etc. The average weekly advance in such a heading is 22 yd,
equivalent to a total excavation of 7920 cu. ft. and producing approximately
400 tons.

A series of time studies of the various duties carried out by one group
revealed the following duration of each operation in one shift:

<table>
<thead>
<tr>
<th>Operation</th>
<th>%</th>
</tr>
</thead>
<tbody>
<tr>
<td>Filling</td>
<td>26</td>
</tr>
<tr>
<td>Drilling</td>
<td>12</td>
</tr>
<tr>
<td>Shotfiring</td>
<td>12</td>
</tr>
<tr>
<td>Supports</td>
<td>10</td>
</tr>
<tr>
<td>Fitting cables, etc</td>
<td>15</td>
</tr>
<tr>
<td>Conveyor extensions</td>
<td>13</td>
</tr>
<tr>
<td>Meals and other duties</td>
<td>12</td>
</tr>
</tbody>
</table>

Successive headings are so commenced to give a stepped series of advancing
faces, which in turn permits of a systematic total extraction along a definite
line and promotes a good control of the roof in the working places. On
reaching the limit of the panel to the rise the inbye ribs or long stoops thus
formed are extracted in a series of "lifts" downhill, a retreatng method of
extraction being practised until the upper level roadway is reached, when
the shaker unit and ancillary appliances are moved outbye to continue the
rotation.

Due to the rapid development and subsequent extraction of the panels,
and to the fact that ventilation is of a high standard, the original timber
supports remain in sufficiently good condition to be withdrawn in retreating
and to warrant their re-erection in a later development. Thus a consider­
able saving in what is undoubtedly a very costly commodity, is effected.
Recovery of supports used at the stage of total extraction is nearly 100 per
cent, and is mainly due to the strong nature of the roof, which on occasion
can be broken down only with difficulty.

The quantity of explosives used per ton of shale produced is variable
within small limits, but 1 lb can be considered an average. It should be
observed that where twenty shots per shift is required a total of nearly
25 lb is required, and it was found impracticable to conform to the regulat­
tions governing the use of explosives. An exemption from the provisions of
the Explosives in Coal Mines Order was therefore obtained, and Special
Regulations now apply to the conveyance and storage underground of the
considerable quantity of gunpowder used per shift in this particular district.
The oncost personnel per shift employed on a fully working panel is as follows:—

1 man for assembly of empty hutches.
1 man for despatch of full hutches.
1 man for loading operation.
1 man for attendance at transfer point.
1 man for all conveyor maintenance.
2 men for general purposes.
1 man to act as overseer and fireman.

These eight men, together with the eighteen producers, thus contribute approximately 7\(\frac{1}{2}\) tons of output per man shift worked in the panel. All distribution of explosives, supports, and the dismantling, fitting, and reassembly of units is carried out as required by additional manpower.

**Fig. 10.**

**AUXILIARY VENTILATION FANS.**

**VENTILATION.**

Each working face is adequately supplied with intake air, delivered from the auxiliary fans (Fig. 10) already described, and conducted to within 15 yd of each face by a series of 12-in-dia collapsible canvas air-tubes which are suspended by hooks from a taut wire rope, and so arranged to be readily retractable when blasting is in operation to prevent damage from projected material. Immediately shotfiring is completed the air tubes are re-extended along the suspension wire to near the face, and the fan started up. The products of combustion are speedily dispersed to the return airway, and the workmen are quickly free to make inspection for any release of firedamp, damage to supports, loose material on roof or face, and so to continue the
cycle. Recirculation of air to any one face or the circulation from one to
another is obviated by the method adopted, and a surplus into the panel is
always ensured to dilute and render harmless any noxious gases likely to be
produced.

Special care is taken to permit examination for firedamp at all edges of
areas totally extracted, as it is within the roof cavities thus formed that such
potential danger normally lies.

TRAINING OF PERSONNEL.

With the introduction of such an intense method of work the personnel
are given special training, etc., as follows:

1. Miners and helpers employed in development are gradually accustomed
to the use of drilling machines, the care of same, and the intelligent use and
protection of trailing cables and distributing units. This is stressed as
these men are to be eventually drafted to the more mechanical production
in the panels. Breakdown due to careless handling means more than mere
repair, and is a source of loss of valuable man hours.

2. Surveying and planning staffs need to realize that where once the
workings were carried out before the plans were completed, planned road­
ways must now be carefully predetermined to enable belts to convey in
straight lines with systematically supported roof and sides.

3. Plant supervisors, mechanical and electrical alike, are taught that
much of the success expected depends on their correct assembly, alignment,
and maintenance of the costly plant installed.

4. Management must realize that manpower is valuable, it must be used
intelligently to get results, and a team spirit must be promoted with
interchange of ideas.

Such close supervision by officials as is possible with this system gives to
to all a safety consciousness which is reflected in the average accident rate.
For example, the accident rate for the year 1949, per 10,000 tons raised from
a group of four mines working a normal stoop-and-room system and under
the control of the same management, was 3-454 accidents and 78-606 man
shifts lost, while from the mechanized section described it was reduced to
1-068 accidents and 21-211 man shifts lost.

PAYMENT.

Payment for work done is on a ton-rate basis, thus enabling each
individual group in a working face to benefit from the return for extra skill
or effort made. A rate per ton being fixed by agreement between the
employer and the workmen's representatives, each face is so measured
weekly that the cubic capacity excavated is calculated, and the sum of each
separate excavation so related to the total output delivered by the conveyor
belt that any one group is allocated its correct proportion of the whole.

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